

TNT MINES LIMITED

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RL10/1988

MOINA

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ABSTRACT

TNT Mines commissioned Jacobs to carry out a desktop scoping study into an 800,000 tonnes per annum open pit mining operation. Capital cost for a contract mining operation were estimated at \$97M and operating costs, excluding logistics to Burnie and beyond, at \$34.60 per tonne.

A metallurgical test work program has been commenced and is looking at a flow sheet to produce the optimal mix of magnetite, fluorspar and scheelite concentrates. The first phase of this program is almost complete and indications are that a near acid-grade fluorspar concentrate can be produced, albeit at a very fine particle size, and an acceptable scheelite concentrate can be produced. Further work is required to make an acceptable magnetite concentrate.

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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).

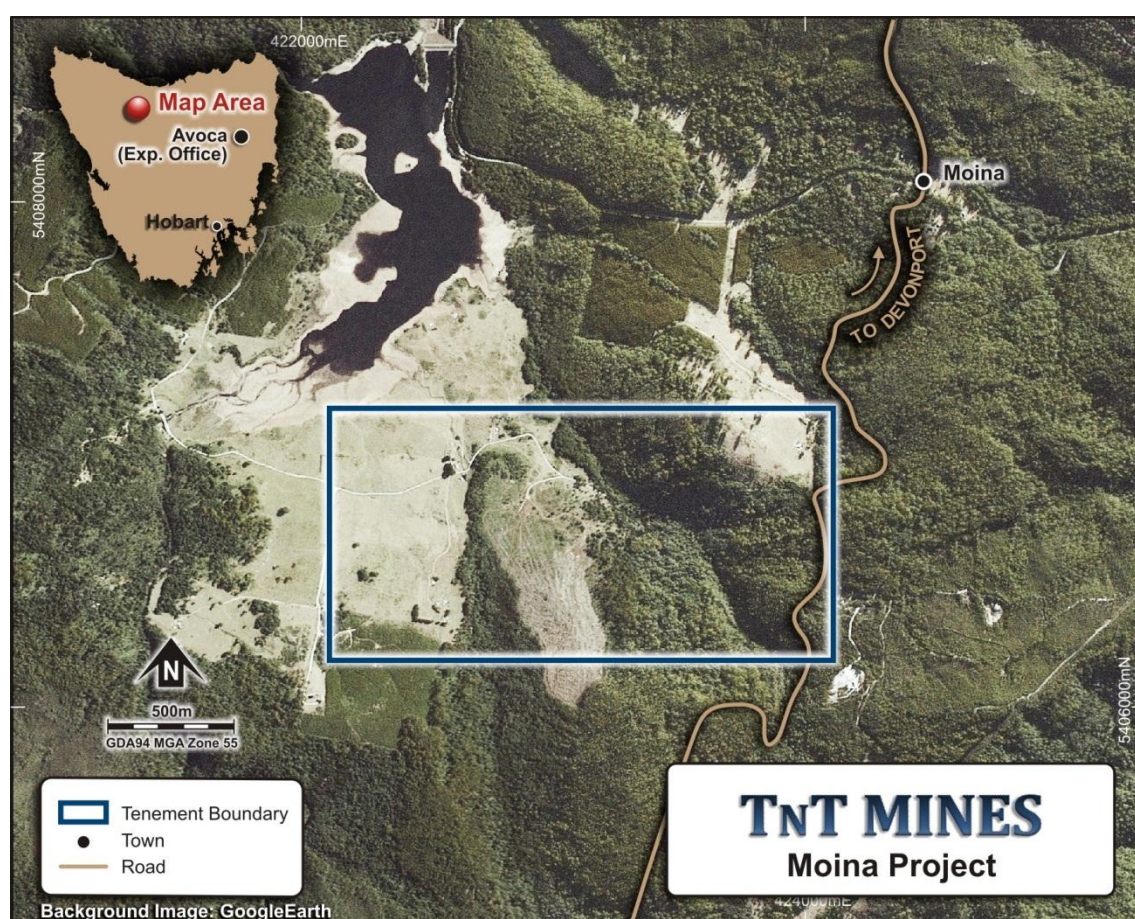


Figure 1: Tenement location plan

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 wriggilite 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.

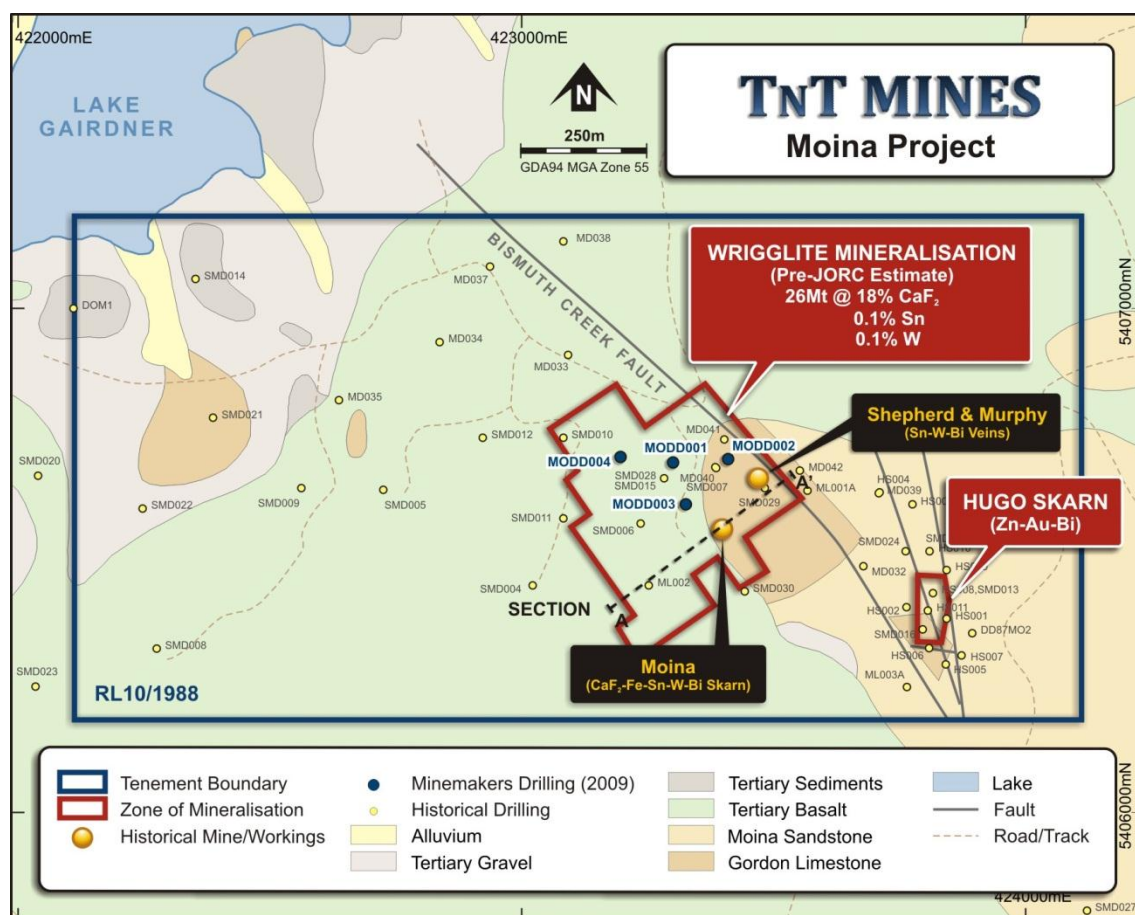


Figure 2: Tenement geology

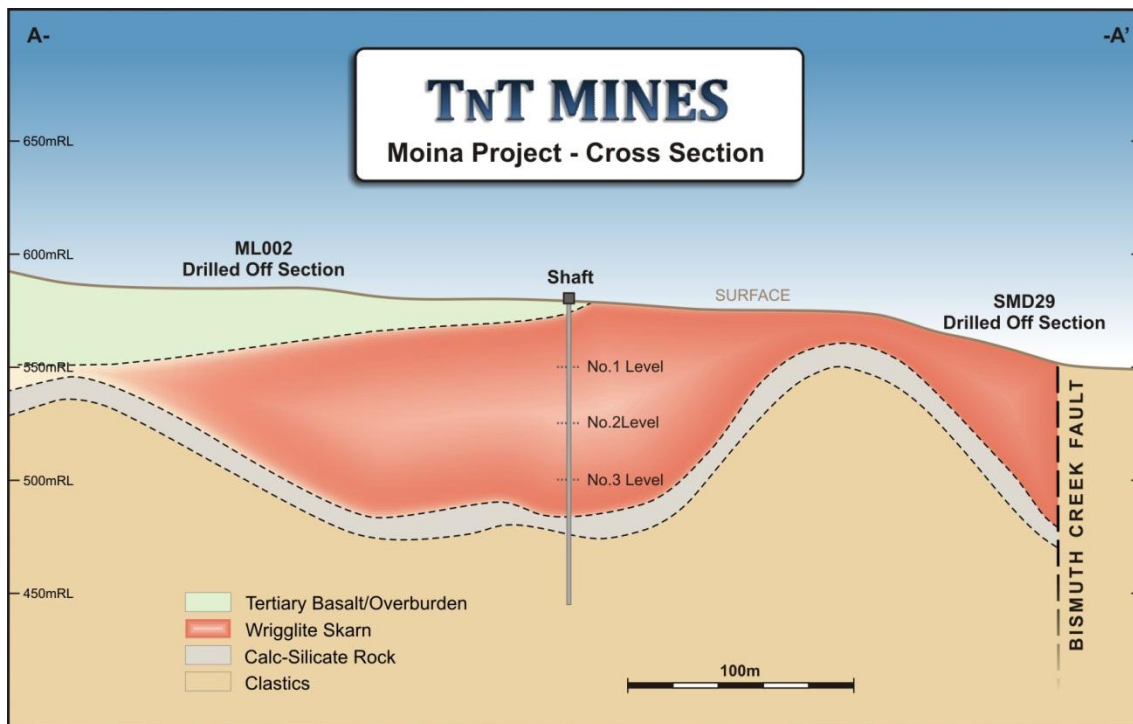


Figure 3: Section through Moina deposit

1.3 Exploration Rationale

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

The Moina fluorspar deposit has a pre-JORC resource estimate of 26.5Mt @ 18% fluorspar, 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 fluorspar 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 proposes 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 fluorspar, 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 since 2006:

- 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 wiggilite 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.

3.0 WORK COMPLETED DURING THE REPORTING PERIOD

3.1 Review of 2008 Davis Tube Recovery work

In April, TNT Mines engaged Geos Mining to undertake a review of the Davis Tube Recovery work undertaken by Minemakers in 2008. The review is attached as Appendix 1.

The testing was carried out on historical core and only two holes were available at that time, one from the Hugo Skarn and one from the main Moina Skarn.

3.2 Metallurgical test work

A new metallurgical test work program was commissioned in May. This study is being carried out at ALS-Ammtec Burnie, formerly the Burnie Research Laboratory, and ALS-Ammtec Perth under the management of John Glen. Metallurgical advice and interpretation is coming from Brian Povey of Mintrex and Ron Goodman. This is the first integrated study to look at producing magnetite, fluorspar, and scheelite concentrates. The first stage of the work is nearing completion and a report is expected in mid-late October.

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%. Following in from the production of three concentrates, the next phase will be to determine the optimal balance between these three potential products, or even if there is one for all three products. Oxide flotation and gravity work is being carried out at Burnie and detailed DTR work was carried out in Perth. Some magnetite separation work has also been carried out at Burnie. All available data is included as Appendix 2.

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)

3.3 Mining study – Jacobs

Jacobs, a large international engineering company, were engaged to carry out a desktop scoping study on a mining operation at Moína. The study generate CAPEX and OPEX on an 800,000 tonnes per annum open pit mining operation producing magnetite, fluorspar and scheelite concentrates. The study is included as Appendix 3.

4.0 DISCUSSION OF RESULTS

4.1 Review of 2008 Davis Tube Recovery work

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 DTR testing under stringently controlled test conditions using the 2009 core.

4.2 Metallurgical test work

The notes below are based on conversations between TNT personnel and John Glen and on interim reports provide by Mintrex on DTR work carried out at Ammtec – Perth. A final interpretation report will be available in mid-late October.

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 from 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.

4.3 Mining study – Jacobs Engineering

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 Contacting, 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.

A review of the Jacobs study will be made by Mintrex and Ron Goodman following the completion of the metallurgical test work program.

5.0 CONCLUSIONS AND FUTURE WORK

This year has seen a significant advance in the development of the Moina project. Mining and process plant costs have been established and a first attempt at a capital cost estimate has been made. An integrated metallurgical test work flow sheet is being developed to allow production of three saleable concentrates; magnetite, fluorspar and scheelite.

Work for next year will include further metallurgical studies to find the optimal mix of magnetite, scheelite and fluorspar concentrates to produce the highest economic return. Work will commence on agglomeration of fluorspar as the final Moina product size will be too fine to meet current commercial specifications. Marketing studies into fluorspar and magnetite will also be initiated. A JORC compliant resource will be completed in late October.

If sufficient funds are able to be made available then a program to drill out the deposit to an Indicated/Measured JORC status will be considered. This program is essentially dependent on the outcome of marketing studies for magnetite and fluorspar.

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 – Initial Study of Magnetite Recovery, Moina Skarn, Tasmania.
Geos Mining Mineral Consultants**



Initial Study of Magnetite Recovery

Moina Skarn, Tasmania

TNT

Job No. 2469-01

Report Date: 24 May 2012

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Executive Summary

The Moina magnetite skarn deposit, in Tasmania, contains a number of potentially valuable minerals, including fluorite, tungsten, bismuth and tin. The skarn is fine grained and metallurgically complex. Previous metallurgical testwork has not focussed on magnetite recovery, but current economics favour magnetite as a valuable product. This study examines the available data for the potential to recover magnetite into a saleable product. The most likely product market is the iron ore fines for pelletisation, with the major alternative market being dense media for coal washery use.

Significant work relating to potential recovery of magnetite includes petrological work, QEMSCAN analysis and Davis Tube testwork.

QEMSCAN analysis of a sample composited from recent drilling of the main wriggilite skarn showed that 60% of the magnetite is liberated in the minus 20 micron fraction. The QEMSCAN indicated that if an initial magnetic separation was done at minus 50 micron, efficient separation could potentially remove around 30% of the magnetite as liberated or free grains, then a second pass after regrinding to minus 20 micron could potentially remove another 20 – 30% of the contained magnetite. Inevitably some locked grains would be recovered, increasing yields and diluting the grades.

QEMSCAN also confirmed the presence of significant iron bearing silicates in the skarn, such that chemical analysis for iron cannot be relied upon to determine magnetite content of ore or concentrates.

The Davis Tube testwork showed that concentrate yields averaged less than 10% for calc-silicate and siltstone lithologies, but over 25% for wriggilite. The chemical analysis of the wriggilite concentrates showed that only the minus 20 micron fraction gave acceptable iron content of 62%, with the 20 – 53 micron fraction averaging 50% Fe. However wriggilite from the main skarn (SMD11) had significantly worse chemistry (lower iron and undesirably high levels of silica, sulphur and potash) than wriggilite from the Hugo skarn (SMD16), which produced a concentrate that would be likely to satisfy most customers' chemical specifications. This difference, if confirmed in future testwork, is potentially significant both geologically and metallurgically. The Davis Tube Recovery (DTR) concentrates from the coarser fractions contained too much dilution from silica and other undesirables to be considered acceptable for iron ore. None of the concentrates produced would be recommended for coal washery magnetite due to the content of diluting silicates.

However, the Davis Tube results may have been the result of sub-optimal test conditions and the single stage grinding used. There is little information available about the feed grade of the samples used, or about the test conditions. As the QEMSCAN liberation results indicate that better results could be achievable, further Davis Tube testing is recommended, using multiple stage grinding and carefully monitored test conditions. Magnetite has been recovered magnetically from feed sizes down to 10 microns. Although such fine grinding would not be economic for recovery of magnetite alone, should the recovery of other minerals require a fine final grind, additional magnetite recovery may assist in defraying the costs of this grinding.

There is potential for production of magnetite for iron ore production but at present there is no evidence that the higher grades required for coal washery magnetite could be achieved. Yields will be relatively low. On the current evidence only 55% - 65% of the magnetite is likely to be recovered into a concentrate, probably in the order of 10% of the feed. Given the grind sizes expected to be required to recover this magnetite, this would only be economic as a co-product or by-product.

On the current evidence, it seems likely that a saleable fluorite product is likely to be required before the Moina project can achieve economic viability.

Additional DTR tests are recommended to clarify uncertainties in the current testwork .

Disclaimer

While every effort has been made, within the time constraints of this assignment, to ensure the accuracy of this report, Geos Mining accepts no liability for any error or omission. Geos Mining can take no responsibility if the conclusions of this report are based on incomplete or misleading data.

Geos Mining and the authors are independent of TNT, and have no financial interests in TNT or any associated companies. Geos Mining is being remunerated for this report on a standard fee for time basis, with no success incentives.

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Introduction

This report documents the results of a brief initial review of processing testwork carried out on the Moina skarn deposit. It was prepared at the request of Mike Hannington and Russell Fulton of TNT Mines Limited. The objective of the review was to provide geological input into a potential process flow sheet for comminution and test work design.

The focus of this initial report has been on the evaluation of the potential to recover magnetite from the wrigglyte skarn, as the potential to recover other minerals has been addressed previously (Delta Minerals Limited, 2007).

PROJECT BACKGROUND

Moina is located 40 km SW of Devonport in northern Tasmania. The deposit is held under retention licence.

The Moina skarn deposit is a magnetite skarn and has a 1970s pre-JORC resource estimate of 26.5Mt @ 18% CaF₂, 0.1% tungsten and 0.1% tin with Bi, Mo and some patches of Au. The geology of the deposit is fully described in a number of reports, including (Kwak & Askins, 1981) (Gifford, 2006) etc. and will not be repeated here.

TNT's focus has been on evaluating the wrigglyte skarn, a fine grained, chaotically banded rock composed of alternating light (vesuvianite and fluorite) and dark (dominantly magnetite) lamellae (Kwak & Askins, 1981). The wrigglyte is cut by later veinlets which may carry tin (predominantly as cassiterite) and tungsten (mainly as scheelite, with minor ferberite (wolfram) reported) and minor bismuth (as bismuthinite) (Sammer, 2008). The wrigglyte itself may also carry minor tin, tungsten and bismuth. About half of the tin is in silicate form (associated with garnet and as stokesite) and is not expected to be recoverable.

Potentially valuable components of the wrigglyte are fluorite, magnetite, and tin, tungsten and bismuth. Minor garnet in the wrigglyte is not likely to give a major return to the project. Potentially valuable sphalerite, gold, wollastonite and garnet are located in other phases of the skarn; these other skarns are not considered in this report. Other associated sedimentary and skarn lithologies are likely to only be mined as dilution to the wrigglyte and hence are not considered in detail here.

Although over 45 holes have been drilled into the Moina deposit, only 12 have been drilled in the area of the main wrigglyte resource (Figure 1). The wrigglyte is interpreted to be folded. The thickest drill intersection of wrigglyte is reported to be about 50m (Sammer, 2008).

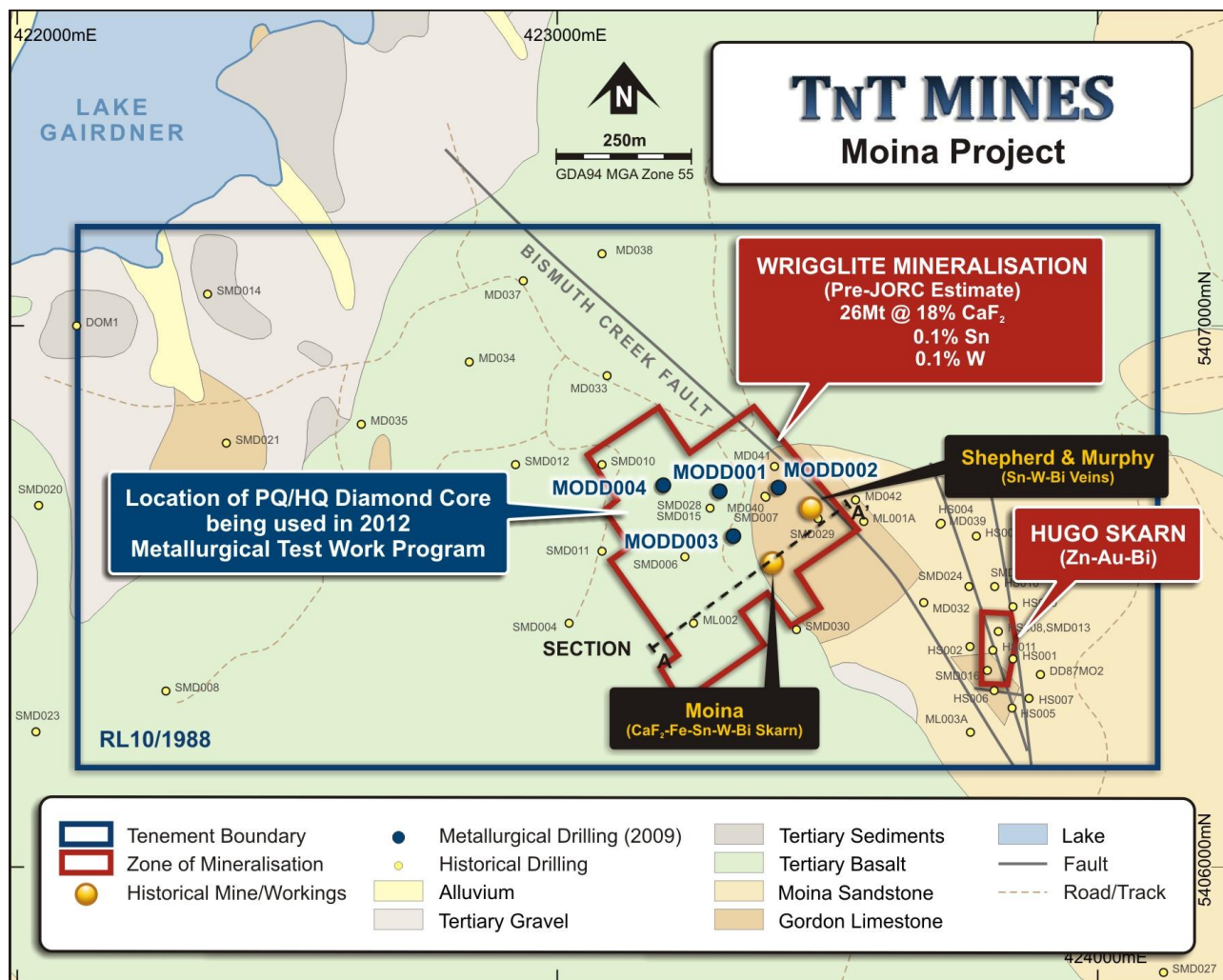


Figure 1: Moina deposit with drill locations

DATA PROVIDED

A number of reports were provided by the client, with the most significant being a report on the potential to process Moina ore (Delta Minerals Limited, 2007), two reports on QEMSCAN work (SGS Minerals Services, Feb 2011) and (SGS Minerals Services, March 2011), a petrographic report (Sammer, 2008) and a report on preparation of sample composites, labelled as Comp 1 - 20 (Deberner, 2008).

Background material on the project included a paper in Economic Geology (Kwak & Askins, 1981), Minemakers and TNT prospectus reports and a TNT corporate presentation (Hannington, 2012).

Data provided included logs and analyses of the older drilling, including holes SMD 11 and 16, which were resampled into Comp 1 – 20, and the sampling details for these holes. The Comp samples were crushed to 100% passing 1.18mm, and ground to 80% passing 53 micron, then screened into three size fractions, plus 53, minus 53 plus 20, and minus 20 micron fractions. These size fractions were analysed by ICP scan and then submitted for Davis Tube testing. Additional data provided were the weights in each of the size fraction, the ICP analyses and Davis Tube magnetic recovery results.

Magnetite Specifications

Specifications for **magnetite iron ore** are now quite varied, but typical specifications of magnetite iron ore being offered for sale by various traders on the internet would be:

Fe - Varies from 58% (low grade) to over 66%; 60- 61% is a common standard.

Impurities – up to 7% silica, 0.15% P, 0.1% S, 3% alumina.

For blast furnace use magnetite fines must be sintered to form synthetic lump ore. For production of steel by direct reduction in an electric arc furnace, fines must be pelletised. The optimum chemistry for direct reduction is (Anon, 2011):

- $<0.03\%S$, $<0.05\%P$,
- $Na_2O+K_2O <0.1\%$
- $TiO_2 <0.2\%$
- B_2 basicity = CaO/SiO_2 , 0.9 – 1.1
- Minimal content of tin, copper, chrome.

The basicity is often controlled during pelletisation, by the inclusion of limestone as well as bentonite. Obviously reduction in silica will reduce the amount of limestone required, which reduces the cost of the pellets and increases the iron content of the final pellets.

Specifications for magnetite for use in dense media separation (mainly in coal washeries) are generally stricter than for iron ore both chemically and in terms of sizing. For this market the magnetite needs to be relatively fine (less than 45 micron, often finer) and at least 90% (although most specifications are for over 95%) magnetite (Gungor, 2010). Ninety percent magnetite equates to 65.1% Fe (if there is no iron in any other mineral) while Fe must be in excess of 68.7% for a concentrate to contain 95% magnetite. A constant grade and high magnetic susceptibility are critical for this use.

Alternate markets for magnetite include use as a weighting agent, high density concrete (recommended for radiation shielding), cement manufacture, abrasive agent, natural pigment, in electronics, catalysts, environmental clean-up agent etc.

In the high density markets, for weighting drilling muds, and in concrete for use in dams, underwater pipelines, and foundations, magnetite competes against barite and ilmenite. There are a number of natural products that can be used as abrasives for air-blasting, including garnet, ilmenite and slag, so this market is quite competitive. Most magnetic materials used in electronics are synthetic, as are most pigments, so natural magnetite is only a niche product in these fields. Most cement manufacturers prefer hematite to magnetite, although there may be some markets here – iron oxide forms about 2 – 3% of the feed for a typical cement plant. The remaining markets are expected to be relatively small and specialised, and time has prevented any detailed consideration of them here.

Review of Test Results

SAMPLES USED

The QEMSCAN analysis was conducted on a single bulk sample (**PO268 Modd Comp**, or the MODD sample) composited from recent drilling. The composite was all wriggilite mineralisation, and intervals used are given in Table 1, with hole locations shown on Figure 1.

Hole	Meterage	Interval length
MODD001	25.0-50.0m	(25m total)
MODD001	62.0-71.0m	(9m total)
MODD003	31.0-79.0m	(48m total)
MODD004	52.0-74.0m	(22m total)

Table 1: Sample details of QEMSCAN composite

The head grades of the PO268 Modd Comp are listed below.

F	S	Sn	Bi	W	Zn	Mo	Fe
7.90%	0.20%	0.16%	0.05%	0.06%	0.02%	0.005%	18.50%

Table 2: MODD sample chemistry

The **Comp samples** used for the Davis Tube testwork were derived from quarter core resampled from two older drillholes, SMD 11 and 16. Hole SMD 11 is located just west of the main wriggilite resource area, while SMD 16 is located to the east in the Hugo Skarn (Figure 1), a zinc rich skarn which includes calc-silicate and wriggilite skarns. In this hole, near the contact of the calc-silicate, the wriggilite contains significant sphalerite (up to 2.85% Zn). The Comp samples include samples representing wriggilite, mixed wriggilite plus other lithologies, calc-silicate and one metasilstone sample (Table 3).

Sample Number	Hole	Depth	Lithology
COMP 1	SMD11	31.0-37.0	Mixed Wriggilite and metaseds
COMP 2	SMD11	62.0-71.0m	Mixed Wriggilite and calc-silicate
COMP 3	SMD11	37.0-42.0	Wriggilite
COMP 4	SMD11	52.0-74.0m	Wriggilite
COMP 5	SMD11	42.0-47.0	Wriggilite
COMP 6	SMD11	47.0-52.0	Wriggilite
COMP 7	SMD11	52.0-57.0	Metasilstone
COMP 8	SMD11	57.0-60.7	Wriggilite
COMP 9	SMD16	30.0-34.0	Calc-Silicate skarn
COMP 10	SMD16	34.0-38.0	Calc-Silicate skarn
COMP 11	SMD16	38.0-42.0	Calc-Silicate skarn
COMP 12	SMD16	42.0-45.0	Wriggilite

Sample Number	Hole	Depth	Lithology
COMP 13	SMD16	45.0-49.0	Wrigglite
COMP 14	SMD16	49.0-61.0	Calc-Silicate skarn
COMP 15	SMD16	61.0-65.0 and 104.2 – 105.0	Wrigglite,
COMP 16	SMD16	105.0-110.0	Wrigglite
COMP 17	SMD16	110.0-115.0	Wrigglite
COMP 18	SMD16	115.0-120.0	Wrigglite
COMP 19	SMD16	120.0-125.0	Calc-Silicate skarn
COMP 20	SMD16	125.0-129.0	Wrigglite

Table 3: Details of Comp samples

MINERALOGY AND LIBERATION

BACKGROUND

QEMSCAN is a technique using an electron microprobe to study the mineral composition of grains in a crushed sample of rock. Particle map analysis produces a picture of the composition of individual grains, and by studying various size fractions it is possible to determine the liberation size with some confidence, permitting prediction of the behaviour of minerals under different processing techniques.

Work done by SGS (SGS Minerals Services, Feb 2011 and Mar 2011) included XRF chemical analysis of the size fractions (which was reconciled to the QEMSCAN analysis), XRD analysis for mineral identification, and QEMSCAN.

TEST RESULTS

The QEMSCAN analysis (SGS Minerals Services, Feb 2011) shows the MODD sample contains:

- 7.92% F (16.2% CaF_2), of which 92.1% is in fluorspar and 7.9% is unavailable for recovery.
- 0.07% W (0.088% WO_3), of which 94.8% is present as scheelite and 5.2% is unavailable for recovery.
- Of the 0.16% Sn measured in the sample, only 48.3% is present as cassiterite and 51.7% is unavailable for recovery. Most of this is the tin locked in silicates, believed to be extremely fine cassiterite. This very fine cassiterite is not included in the recoverable cassiterite.
- Bismuth. The sample contains 0.046% Bi, which is all present as Bi-sulphide or native bismuth and is all available for recovery.
- 0.5% pyrite and 0.03% other sulphides. Sulphur is undesirable in iron ore.
- Iron containing silicate minerals include vesuvianite (13.8%), garnet (5.6%), actinolite (5.8%), other iron amphiboles (15.8%), biotite (8%) and chlorite (1.7%). SGS report an average of 10% Fe in the vesuvianite and an average of 4% Fe in calc-silicates, while Kwak and Askins report over 20% FeO in garnet, 25 – 20% in amphibole, and 36% FeO in a biotite (annite). Together these would carry a significant quantity of iron, and emphasise the danger in attempting to estimate recoverable magnetite from chemical analysis of iron.

The XRD did not record any iron bearing phases other than magnetite, and SGS assumed that all iron oxides were magnetite, which seems reasonable, based on the drill logging and petrography.

In the MODD sample, iron oxides are relatively evenly distributed between the size fractions studied (plus 63, 63 – 45, 45 – 20 and minus 20 microns). Calcite, phyllosilicates and iron rich amphiboles all show a tendency to slime (concentrate in the minus 20 micron fraction).

QEMSCAN study of the magnetite in the MODD sample indicates that 60% of the magnetite is liberated in the minus 20 micron fraction. This is in line with previous observations that most magnetite crystals are around 20 microns (Delta Minerals Limited, 2007). If an initial magnetic separation was done at minus 50 micron, efficient separation could remove around 30% of the magnetite as liberated or free grains (a concentrate yield of around 5 – 7% of feed), then a second pass after regrinding to minus 20 could remove another 20 – 30% of the contained magnetite; i.e. a further ~5% of the original feed mass. The initial grind to minus 50 could also be used to recover any scheelite (SGS Minerals Services, March 2011).

Inevitably, separation would not be efficient, even after low intensity concentrate cleaning, some locked grains would be recovered, so yields may be higher and grades would be diluted. The unliberated magnetite is mainly locked with silicates and fluorite.

DAVIS TUBE TEST RESULTS

BACKGROUND

The Davis Tube is designed to recover magnetic material (commonly but not exclusively magnetite) from a pulverised sample. It reports the concentration, expressed as weight per cent, of the magnetic grains in a rock able to be separated by a magnetic field. The Davis Tube equipment consists of an extremely powerful electromagnet which can generate a magnetic field intensity of up to 4,000 gauss, a glass separation tube and a motor driven agitation mechanism. The tube is positioned between the poles of the magnet at an angle of approximately 45 degrees (the angle is adjustable). During the operation of a small electric motor drives the agitating mechanism that supports the water filled glass tube. The tube moves forward and backward while it rotates simultaneously. Any magnetic particles present in the product sample inside the tube are collected in the zone of intense magnetism. A vigorous washing action by agitation is applied to these magnetics. Eventually all non-magnetics are flushed from the tube. A clean concentrate of magnetics is then available for determining the per cent recovery (Sepor Inc., n.d.).

Parameters in the Davis Tube Recovery test (DTR) are given in Table 4. One important variable to be set in the test is the magnetic field intensity which may range from 500 to 3,000 gauss (Dworzanowski, 2012) (O'Donnell, 2010).

Variable	Standard setting
Stroke Frequency	60/minute
Stroke Length	38 mm
Magnetic Field Strength	3000 gauss
Tube Angle	45 degrees
Tube Diameter	25 mm
Washing Time	15 minutes or until water is clear
Water flow rate	540ml/minute

Table 4: Standard settings of parameters in Davis Tube recovery
From (O'Donnell, 2010)

Size fractions usually tested by DTR range from minus 150 micron to minus 10 micron, but a common standard for initial testwork is minus 45 micron (O'Donnell, 2010). Finer fractions are used where liberation is incomplete at the coarser sizes. Recovery of fines by DTR may be enhanced by magnetic flocculation, and Dworzanowski (2012) achieved magnetite recovery in excess of 55% into a high grade (70% Fe) concentrate using a minus 10 micron fraction. However, if the fine particles do not flocculate (and it is possible that high water flow may inhibit flocculation) they are likely to be entrained into the non magnetic rejects and lost.

In addition, the grind procedures are critical to performance of the DTR (Povey & Leather, 1997). A two stage grinding procedure is recommended, as this can produce much cleaner DTR concentrates, with only a small loss in concentrate yield. In this case only a one stage grind was used, so the results may not be optimal.

Although full details of the sample preparation for the DTR results are available, there are few details on the parameters for the DTR tests conducted on the Comp samples, which limits the interpretation of these results.

More critically, there are no records of any analyses of the head grade of these composites, so the recovery of iron from these remains unknown. The original analyses provided for these holes do not include iron analyses, and ALS Perth has disposed of all sample pulps (Abbott, pers. comm., May 2012). Given that these were quarter core samples from previously sampled holes, it seems unlikely that iron analyses can now be carried out, unless there is retained sample at Burnie from the original concentrates, or a portable XRF can be used on the remaining quarter core (and this will only give a rough guide as to iron grade).

TEST RESULTS

DTR concentrate yields for the different size fractions are given in Table 5 and analyses of these concentrates are given in Table 6. Yields from calc-silicate skarns and metasilicites are low, and comps 1 and 2 show the effect of dilution of these lithologies on the yields from the wriggilite. This is useful information for assessing the effect of mining dilution. Within the wriggilite, yields average 28%, being higher in SMD 11.

	Lith	(-20µm)	(+20µm)	(+53µm)	Weighted Average %
Comp-9	CS	4.14	3.85	7.46	5.65
Comp-10	CS	5.58	5.94	7.36	6.05
Comp-11	CS	4.89	5.30	5.76	5.18
Comp-14	CS	14.67	21.10	17.54	17.13
Comp-19	CS	10.63	16.70	10.98	12.48
Comp-7	Silt	8.10	11.68	4.03	8.44
Comp-1	Mixed	17.60	23.23	30.22	22.16
Comp-2	Mixed	25.20	32.94	39.45	31.05
Comp-3	W	21.09	35.47	44.48	31.76
Comp-4	W	23.97	42.81	51.31	37.33
Comp-5	W	26.60	38.08	45.22	33.72
Comp-6	W	22.98	28.71	34.69	27.46
Comp-8	W	22.33	29.22	38.22	29.10

	Lith	(-20µm)	(+20µm)	(+53µm)	Weighted Average %
Comp-12	W	17.23	25.66	24.20	22.01
Comp-13	W	16.11	19.92	21.05	18.35
Comp-15	W	21.73	28.49	23.87	25.31
Comp-16	W	26.32	36.61	44.62	34.19
Comp-17	W	25.07	32.13	40.32	30.46
Comp-18	W	20.28	29.69	35.09	27.51
Comp-20	W	23.66	16.79	31.19	22.78
Average		17.91	24.22	27.85	22.51
Max		26.60	42.81	51.31	37.33
Min		4.14	3.85	4.03	5.18
Average (Wrigglite - SMD11)	W	23.39	36.27	43.93	32.57
Average (Wrigglite - SMD16)	W	21.49	26.80	32.75	25.88
Average (Wrigglite)	W	22.28	30.59	37.22	28.56

Table 5: DTR concentrate yields

Table 6 shows that for all lithologies, the iron content in the concentrate increases as the size decreases, as there are less composite grains in the concentrate.

The wriggilite concentrates from SMD16 (Hugo skarn area) are significantly better quality (higher in iron) than those from SMD11 (main resource area). In addition, comp 5 averaged 1% sulphur over all size fractions (the average for all samples was 0.2% S), and the SMD11 wriggilite composites averaged over 7% silica for the minus 20 micron fraction, which is on the high side of desirable for a magnetite iron ore concentrate. Taking the data at face value, the SMD11 minus 20 size fraction is barely acceptable in iron content, and a larger top size would not give an acceptable concentrate without some cleaning to reject composite grains. In contrast, for SMD16 there is potential for a minus 30 or minus 40 micron fraction to produce a saleable grade of iron.

The calc-silicate concentrates are high in iron (average 54%) while the metasediment, as expected, contains less iron. Dilution of wriggilite with calc-silicate will reduce yield of magnetite but not impact severely on quality, but dilution with siltstone may also impact product quality and would need to be minimised.

	Lith	(-20µm) Fe %	(+20µm) Fe %	(+53µm) Fe %	Weighted Average %
Comp-9	CS	63.5	42.6	22.2	44.73
Comp-10	CS	59.5	52.3	35.3	51.06
Comp-11	CS	60.5	52.7	36.9	52.85
Comp-14	CS	64.6	58.6	37.8	56.80
Comp-19	CS	65.3	65.9	59.7	64.39
Comp-7	Silt	53.3	30.3	24.5	39.50
Comp-1	Mixed	58.8	44.5	31.5	47.36
Comp-2	Mixed	61.3	46.5	28.1	49.62
Comp-12	W	65.9	61	38.3	57.77
Comp-13	W	65.1	53.7	31.8	55.46
Comp-15	W	66	60	34.6	56.80
Comp-16	W	63.6	52.3	31.8	53.87

	Lith	(-20µm) Fe %	(+20µm) Fe %	(+53µm) Fe %	Weighted Average %
Comp-17	W	63.1	49.6	31.6	50.73
Comp-18	W	64.4	57	35.1	56.16
Comp-20	W	65.5	56.4	40.5	56.94
Comp-3	W	57.9	44.1	31	48.59
Comp-4	W	57.6	42.3	30.8	47.98
Comp-5	W	57.5	41.1	30.3	46.19
Comp-6	W	59.2	41.6	30.9	47.86
Comp-8	W	58	40.1	26.1	44.74
Average, all samples		61.53	49.63	33.44	51.47
Max		66.00	65.90	59.70	64.39
Min		53.30	30.30	22.20	39.50
Average (Wrigglite - SMD11)	W	58.04	41.84	29.82	47.07
Average (Wrigglite - SMD16)	W	64.80	55.71	34.81	55.39
Average (all Wrigglite)	W	61.98	49.93	32.73	51.93

Table 6: Iron analyses for DTR concentrates from Comp samples (ICP analyses).

Critical impurities in the minus 20 micron DTR concentrates are summarised in Table 7. This indicates that SMD16 minus 20 mesh would potentially satisfy most customer's requirements for magnetite iron ore pellets. Although potash and possibly chromium is higher than desirable, the remaining chemistry is good.

%	Calc silicate	Siltstone	Wrigglite – SMD-11	Wrigglite – SMD-16
-20 Average SiO ₂	4.75	10.95	7.4	3.69
-20 Average Al ₂ O ₃	1.21	2.75	2.68	1.4
-20 Average CaO	2.38	5.76	5.03	2.31
-20 Average TiO ₂	0.14	0.13	0.1	0.11
-20 Average K ₂ O	0.27	0.50	0.48	0.32
-20 Average Na ₂ O	0.06	0.40	0.2	0.03
-20 Average P	0.01	0.012	0.009	0.005
-20 Average Sn	0.1	0.149	0.08	0.15
-20 Average Cu	0.07	0.048	0.02	0.02
-20 Average Cr	2.3	1.88	0.56	0.51
-20 Average S	0.07	0.22	0.41	0.02

Table 7: Impurities in minus 20 micron DTR concentrates

The SMD11 minus 20 micron concentrate would fail most specifications for silica, sulphur and potash. The calc-silicate concentrate contains unacceptably high chromium, while the siltstone is too high in silica, alkalis and chromium. All minus 20 concentrates are low in titania and phosphorus which could enhance their marketability for blending.

All the plus 20 and plus 53 concentrates are too high in silica (and have other undesirable impurities) to make readily saleable concentrates without further cleaning.

The comments above may be premature, as some improvement may be possible if staged grinding is used (Povey & Leather, 1997), and we do not know if the test procedures were optimal.

Discussion

Interpretation of the DTR results is limited by:

- Lack of any feed grade analyses
- Single stage grinding potentially leading to sub-optimal results
- Lack of details on the DTR test procedures

The difference between the DTR results for SMD11 and SMD 16 is pronounced. Kwak & Askins (1981) indicate this is probably due to alteration in the vicinity of the Bismuth Creek fault. As a full review of the geology is outside the scope of this work (and would require inspection of outcrop and cores), Geos Mining cannot comment on this. Any variability of the wriggilite will limit the confidence in any conclusions reached from metallurgical testwork unless ore types are well characterised. If SMD 16 is typical of the Hugo skarn, and SMD 11 typical of the main wriggilite resource, the implication is that a saleable magnetite concentrate will be more readily produced from the Hugo skarn, potentially at a coarser grind size, but the main resource may prove difficult even at finer grind sizes.

Neither the core logs, the petrology (Sammer, 2008) nor the XRD tests (SGS Minerals Services, Feb 2011) mention any iron oxide other than magnetite, but (Kwak & Askins, 1981) mention hematite in SMD16, near the Bismuth Creek fault. Although minor hematite/goethite may be present, it seems likely that iron lost in the DTR is largely in the form of locked grains and silicates. Routine logging of drill samples and any composites for metallurgical testing with a well calibrated magnetic susceptibility meter is recommended to assist in determining the deportment of the iron oxides

The results from the QEMSCAN are more promising than those from the DTR results, as the QEMSCAN implies that clean magnetite concentrates (up to 70% Fe) may be theoretically possible at sizes between 50 and 20 microns. The DTR for SMD16 wriggilite implies that saleable iron grade concentrates may just be possible at a grind size less than 53 microns (although the impurities would be less than desirable), but the DTR for SMD11 (which should, from its location, be more similar to the MODD sample tested by QEMSCAN) implies that only a marginally saleable concentrate could be produced from the minus 20 micron fraction.

A primary grind to 20 microns is considered unlikely to be economic for a magnetite deposit, but in this case recovery of fluorite may enhance the economics. Finer grinding to recover other minerals may permit scalping of additional magnetite, as magnetic recovery of magnetite is possible to less than 10 microns (Dworzanowski, 2012).

There is potential for production of magnetite for iron ore production but at present there is no evidence that the higher grades required for coal washery magnetite could be achieved.

This study has at this stage not addressed:

- The integration of magnetite recovery with recovery of other minerals
- Pelletisation of a magnetite concentrate
- Marketability of the concentrate – the data are not yet sufficiently well known to warrant any detailed comparison of the potential product with the specifications of competing products
- Alternate markets for magnetite – weighting agent, pigment, etc., as these are expected to be relatively small and specialised markets.

Recommendations

1. Check if there is any retained sample from the 2008 Comp 1 – 20 samples (BRL report that they stored the remainder of the sample). If so, they should be measured for magnetic susceptibility and then analysed for iron.
2. Assess the variability of the wriggilite, to confirm whether SMD 11 is typical of the main resource and SMD 16 typical of the Hugo skarn. If there is significant variability within either area, the ore should be classified into different types (possibly using Sammer's classification) for future metallurgical testwork.
3. Holes SMD11 and SMD 16 may require re-logging to determine the reason for the difference in behaviour in the DTR tests, unless this is already obvious to those familiar with site geology. Use of a calibrated magnetic susceptibility meter while logging (or re-logging) core may assist in any classification. Additional petrology/whole rock chemistry/XRD may also be required.
4. Prepare some new samples for DTR tests, using a staged grinding process to ensure the best recovery and concentrate grades. If possible, the MODD sample should be one of those tested. Only a few composite samples maybe required initially, and these should represent the wriggilite ore types expected to form any significant proportion of the ore.
5. Future samples for DTR and metallurgical testwork should have magnetic susceptibility recorded.
6. Further work will then be required to determine the integration of magnetite recovery with recovery of other products from Moina, so that the economics of the whole project may be assessed.

Conclusions

There is potential for production of magnetite for iron ore production but at present there is no evidence that the higher grades required for coal washery magnetite could be achieved. Yields will be relatively low. On the current evidence only 55% - 65% of the magnetite is likely to be recovered into a concentrate, so if the MODD sample grade of 18% magnetite is typical of the wriggilite as a whole, this would be in the order of 10% of the feed. Given the grind sizes expected to be required to recover this magnetite, this would only be economic as a co-product or by-product.

On the current evidence, it seems likely that a saleable fluorite product is likely to be required before the Moina project can achieve economic viability.

Additional DTR tests are recommended to clarify uncertainties in the current testwork.

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APPENDIX 2 – Mintrex reports.

23rd May 2012

TNT Mines Ltd
Level 2, 34 Colin St
WEST PERTH WA 6005

Attn: Mr M Hannington

Dear Mike,

Proposal 1234 – Provision of Metallurgical Consultancy Services for the TNT Mines Ltd – Magnetite Section

Mintrex is keenly interested in assisting you with the works required for the development of the polymetallic project at Moina in Tasmania and help you to continue to refine the project – particularly in the magnetite area and with process integration with other processes.

At this stage we would like to offer metallurgical consultancy services of our Brian Povey to assist in this phase of the work – hopefully it will lead to more detailed studies in the future. Brian has talked with Ron and as a result of this discussion has suggested that Brian will develop a magnetite testwork programme for Ron to action in Burnie. This will cover material up to 5mm in size – and look for the best point to separate out the scheelite and magnetite – since the fluorspar will probably be evenly distributed between the magnetic concentrate and tails at coarser sizes. Brian can interpret such data and with Ron develop a single page process flow diagram in Visio combining the magnetite, scheelite and fluorspar circuits.

We can probably develop a first pass mass balance if required for refinement once testwork has been received.

We have estimated 35 hours for this work with the test programme developed by Friday evening.

Personnel

Key Mintrex Personnel for the programme:

Snr Principal Process Engineer – Brian Povey

Estimated Costs

The estimated costs are

Consultant 35 hours @\$285/hr	\$10,000
Expenses	\$200
Laboratory Costs	TBA

Mintrex People

Short form Curriculum Vitae for Brian, applicable rates and a copy of an ACEA short form contract conditions which Mintrex proposes are attached.

Costs and progress will be reported on a monthly basis with an invoice issued with the monthly report.

We would like to thank you for this opportunity to be considered to work with you on your project.

If this proposal is acceptable please sign the attached short form contract sheet and return to brian.povey@mintrex.com.au or fax to +618 9442 3399.

Yours faithfully,

MINTREX



Brian Povey
Snr Principal Process Engineer

May 2012

TNT Mines Ltd

SCHEDULE OF RATES

(GST and any withholding taxes must be added to all rates)

Personnel	Hour
Director/Senior Principal	\$285.00
Principal Engineer	\$250.00
Lead Engineer	\$220.00
Senior Engineer	\$200.00
Experienced Engineer	\$180.00
Principal Engineering Assoc.	\$180.00
Lead Drafter / Senior Engineering Assoc.	\$170.00
Senior Design Drafter	\$160.00
Graduate Engineer	\$150.00
Drafter	\$135.00
Senior Secretarial	\$120.00
Secretarial	\$95.00

Disbursements

All external payments (eg Accommodation, air fares, hire cars, printing)	cost + 15% (cost excludes GST) note – business class air travel for senior personnel
Computer, CAD and internal office machines	included in rates except colour copying colour copies at \$0.25/page
Company vehicle travel	\$1.20 per kilometre

RECOGNISED SPECIALIST EXPERT RESOURCES – For example structural dynamics or power system analysts are unlikely to be available at the quoted rate structure. Should they be required to be supplied, they would be supplied as disbursements at cost + 15%.

The rates quoted assume normal commercial trading terms (30 days) and we reserve the right to charge interest at 16% per annum on overdue payments from the date of invoice.

Mintrex Pty Ltd ABN 64 064 105 488 - PO Box 1932, Subiaco WA 6904

phone (08) 9442 3333 **facsimile** (08) 9442 3399 **website** www.mintrex.com.au

Curriculum Vitae

Brian C Povey**Qualifications:**

MBA, Tech Management, Deakin University, 1994
Diploma, Business Management, Deakin University, 1991
BSc Engineering (Metallurgy), Royal School of Mines, London
University, 1971

Professional Development:

Tasmania Open Cut Managers Certificate
Course in Static Radiation Gauges

Affiliations:

Fellow, Aus IMM
Member, Institution of Mining and Metallurgy
Member, Australian Institute of Managers
Chartered Engineer
CP(Met)

Experience:**March 2012 to present****Mintrex Pty Ltd****Senior Principal Process Engineer**

- Responsible for the development of the process engineering capability of the company covering due diligence work, direct consultancy services for grass roots, developing and operating mines and the process design of new plants

2000 - 2012**Promet Engineers Pty Ltd****Consultant Metallurgist/Director**

- Due diligence work has been carried out on magnetite and hematite operations in Mongolia, Canada, Peru, India and Australia. These have included QCM in Canada, ABM in Tasmania, Balla Balla, Cape Lambert and Hancock resources.
- Consultancy services to Crossland Resources, Gindalbie Metals, Atlas Iron, Australasian Resources, Windimurra Vanadium, Fox Resources, McArthur Minerals, Ironclad Resources, IronRoad and BHP Billiton on developing magnetite resources, magnetite related resources and hematite processing options. Recent work has included work on Sphere in Mauretania and Xstrata on the Zanaga project in the Congo. Also included consultancy and design work on Bootu Creek Manganese
- Consultancy services on the improvement and expansion of current operations to ABM in Tasmania and to Ferrexpo/Poltava Mining in the Ukraine. Consultancy services to Ernst and Young (Russia) and Severstal.
- Consultant Metallurgist for BFS levels studies, testwork, flow sheet development and process design for the Grange 7 Mtpa Magnetite project, Ferrexpo 10 Mtpa magnetite plant in the Ukraine, Karara projects studies to 8, 12 and 16 Mtpa of

magnetite, 10 Mtpa Extension Hill project and International Minerals 12 Mtpa magnetite project.

- Consultant Engineer on the OneSteel magnetite concentrator incorporating High Pressure Grinding Rolls and consultant on testwork, design and commissioning.
- Commissioning team member of a 3.0 Mtpa iron ore grinding and pelletising plant in India.
- Consultant to One Steel on treatment of low grade ores.
- Team member of a Woodward Clyde environmental group advising Bhilai Steel on future mine development and optimal material handling solutions for the iron ore.

1996 – 2000

Kvaerner Metals (now Aker Solutions)

Metallurgical Consultant

Australian Bulk Minerals – Prefeasibility Study

- Options for the development of a pig iron plant from a magnetite feed stock using HIs melt or Rotary Hearth/SAF technology.
- Design and costing of by-product vanadium plant using SX technology to produce 10,000 tpa vanadium pentoxide.

Westralian Sands (Iluka Minerals) – Option Study

- Detailed study of future strategic options for Iluka reviewing alternative kiln technology, leach plant improvements and alternative process and leach technologies.

Confidential Client – South Australian Steel Production

- Development of options and circuit design to produce 2.5 Mtpa of hot rolled coil from a South Australian iron ore source.

MISCELLANEOUS

- Design and development of a number of magnetite ore bodies including concentrator and pellet plant design. Projects also included testwork design and supervision.
- Option studies on a number of hematite concentrators to produce DR grade material. Operating and capital cost development on pellet plant and HBI plants using HYL technology to use the upgraded ores.
- Design of a drying and grinding circuit for the Jindal pellet plant in India.

1995 – 1996

Ghana Manganese Company

Operations Manager

- Contract work on rehabilitating a manganese operation in Ghana after its divestiture by the government.
- Set daily, weekly and monthly records in the second and third month of operation in the crushing plant.

1995

BHP Direct Reduced Iron Project
Consultant Metallurgist

- Lead Engineer on the concentrator design for the DRI plant planned at Port Hedland.

1993 - 1995

CSR Readymix
Brockman Mine Manager

- Responsible for the day-to-day operation of a 7 Mt/y iron ore mine, crushing and dry screening plant and the supervision of mining, laboratory and camp contractors.

1992 - 1993

Curragh Queensland Mining
Operations Manager

- Responsible for the day-to-day operation of a 6 Mt/y coal mine including the dragline operation, coal haulage, washery and maintenance departments.
- Participated in the commissioning of a fine coal washing plant and the introduction of the world's largest dragline, coal hauler and FE loader.

1977 - 1992

Savage River Mines (now Grange Resources)
Resident Manager

- Responsible for the coordination of all the departments on site including mining, mine engineering, ore dressing, maintenance, townsite, technical services and site administration through a group of departmental managers of an iron ore mine (magnetite) on the West Coast of Tasmania.

Chief Metallurgist

- Responsible for the technical development of the pellet plant and the concentrator. Developed the use of alternative fuels in the shaft furnaces until coal replaced 50% of the fuel oil and introduced heat recuperation techniques to reduce fuel consumption by 10%.
- Introduced computerised controls at the pellet plant and concentrator to reduce costs and improve productivity.
- Responsible for site environmental performance and rehabilitation.

Assistant Agglomeration Superintendent

- Responsible for the day-to-day operation of the pellet plant and ship loading facility for Savage River Mines including furnace refractory repairs and rebuilds and coordinating the production and shipping arrangements to meet the port requirements.

Plant Metallurgist

- Work mainly involved pellet quality control. Duties included the implementation of standardised operating procedures, test work to use coal as an alternative fuel in the furnaces and the development of a computer model of the shaft furnaces. This was developed further by CSIRO and was the basis of a major joint project which included furnace modelling and modelling of balling circuits.

1971 - 1976

Various Projects

Various roles in mineral process and metallurgical operations including tin, gold, bismuth and copper. Experience includes periods in several states in Australia, work experience in Zambia and in a steel strip rolling mill in the UK.

Publications:

- The Use of Mathematical Models in the Processing of Iron Ore - Batterham et al.
- Mixing Effects in Large Scale Packed Beds, Norgate et al.
- Development and Validation of Dynamic and Steady State Models for the Vertical Shaft Induration of Iron Ore Pellets, Norgate et al.
- Upgrading of Australian Magnetite Ore Bodies, B.C. Povey and K Leather.
- HPGRs and Wet Classification - B.C.Povey - Randol International 2005
- The use of HPGR for Crushing Magnetites - B.C.Povey -AusIMM Iron Ore Conference 2009

Contact Details:

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Phone: Bus Direct +61 8 9442 3316
Mobile: +61 (0)439 110 039
Fax: +61 8 9442 3399
E-mail: brian.povey@mintrex.com.au
Skype : brian.povey1
Post: PO Box 1932, SUBIACO, WA 6404
Office: Ground Floor, 1 Centro Ave, SUBIACO, WA 6008



ABN 64 064 105 488

CORPORATE PROFILE

Iron Ore Capability



Vision

To Be An Outstanding Resource Industry Engineering Company

EXPERTISE AND SERVICES

Mintrex is a Perth based consulting engineering and project management business focused on servicing the mining and mineral processing industry.

ENGINEERING CAPABILITIES

Mintrex is able to offer multi-discipline engineering design services for a wide range of resource development projects with experience in most states of Australia, Africa and Europe.

Mintrex is well qualified to complete a wide range of project work including, but not limited to:

- Feasibility studies
- Failure and other forensic engineering analyses
- Design, documentation and procurement
- Project engineering and construction management
- Planning studies and reports
- Engineering and asset integrity audits

CULTURE

Mintrex is committed to delivering value to our clients through providing sustainable and cost effective solutions. We seek to build our business by satisfying the needs of our clients at the same time as remaining commercial and competitive in the marketplace. To achieve this we seek commitment to the highest business standards, ethics and integrity.

Mintrex adopts an approach to our people, which is a partnership between employer and employee aimed at helping all to achieve their maximum potential. We seek to reward our people for their intellectual skills and the commitment they make. In return for the delivered value of the services our people provide, we seek a fair return from our clients.

Mintrex operates a quality management system to AS9001:2008, which is registered by SAI Global, certificate number QEC24281.

PROJECT REFERENCES

The Mintrex organisation includes personnel with experience from a wide range of companies – experience which considerably broadens the capability of Mintrex beyond the Mintrex named projects. Our senior people with iron ore experience – both magnetite and hematite are listed below

Project Management

Ian Kerr – General Manager Project. Ian has significant experience in iron ore projects: Project manager of a \$300m expansion of the Tom Price concentrator and Project Director for Mintrex's design and procurement contract to Cloudbreak III.

Process

Our Principal Consultant Metallurgist – Brian Povey, has worked on the development of many magnetite projects including OneSteel at Whyalla, a 2Mtpa plant which was built and designed with Brian leading the testwork and process team at ProMet. In addition he has worked on the many stages of the Gindalbie/Karara development, Poltava (Ukraine), Mt Gibson/Extension Hill, Southdown/Grange as well as concept studies such as Mt Jupiter, Mindax, Cashmere, Giralia, and Zenith. He has consulted on magnetite projects in Mongolia, Russia, India, Mauretania, Peru and Chile. His experience includes Resident Manager of Savage River mines (now Grange) – the only long term magnetite operation in Australia. He also has considerable experience in hematite upgrading (Boodarie, Cashmere, Whyalla LGO plant) and hematite operations (Brockman)

Mechanical and Project Engineering

Evan Platts our Principal Mechanical Engineer has working experience in BHP's shipping and material handling sections as both a maintenance engineer and as a design engineer in engineering services. His experience includes both material handling facilities and process facilities at Boodarie. He has also been involved with remote operations as Engineering Projects Manager for Territory Resources.

Naim Michel Abou-Rjeily our Senior Mechanical Engineer has practical experience in recommissioning the Northern Iron magnetite plant in Sydvaranger. His role included a project engineer refurbishing existing equipment and specifying and installing new equipment. He has worked on numerous other plants on a wide range of unit processes.

Mintrex, is a consulting engineering company which has worked on many projects in Australia and overseas. Of particular reference in the iron ore area are:

- Recommissioning of the Sydvaranger plant in Norway (magnetite) where Mintrex had the mechanical engineering design contract for all the new equipment.



- Development of facilities for Western Deserts at Roper River (PFS underway). This is the infrastructure and material handling facilities for a new iron ore project
- Project review of the Mbalam project in Cameroon
- Cloudbreak III (EP but halted)
- Iron Ore Storage (Darwin Port design for Territory Iron)

- POAGS expansion Port Hedland. Mintrex was involved with specifying equipment for the original Utah Point development and is now conducting early works on the project expansion
- Balla Balla – development of concepts for infrastructure and Port Development.

Projects

Tables 1 and 2 below list some recent Mintrex projects:

Table 1 Mintrex Selected Recent Study Experience

Project	Location	Study
Batie West (Au)	Burkina Faso	Pre-Feasibility Study
Ayanfuri (Au)	Ghana	Detailed Feasibility Study
Ayanfuri (Au)	Ghana	Pre-Feasibility Study
Nzema (Au)	Ghana	Detailed Feasibility Update
Tengrela (Au)	Côte d'Ivoire	Detailed Feasibility Study
Tengrela (Au)	Côte d'Ivoire	Pre-Feasibility Study
Tengrela (Au)	Côte d'Ivoire	Scoping Study
Bonikro Plant Upgrade (Au)	Côte d'Ivoire	Pre-Feasibility Study
Bonikro (Au)	Côte d'Ivoire	Detailed Feasibility Study
Grumesa (Au)	Ghana	Project Review
Mbalam (Fe)	Cameroon	Project Review
Mt Weld (La-Lu)	Australia	Detailed Feasibility Study
Garden Well (Au)	Australia	Detailed Feasibility Study
Sandfire - DeGrussa (Cu-Au)	Australia	Detailed Feasibility Study
Sandfire - DeGrussa (Cu-Au)	Australia	Pre-Feasibility Study
Sandfire - DeGrussa (Cu-Au)	Australia	Scoping Study
Duketon (Au)	Australia	Detailed Feasibility Study
Tomingley (Au)	Australia	Detailed Feasibility Study
Tomingley (Au)	Australia	Pre-Feasibility Study
DeGrussa (Cu-Au)	Australia	Pre-Feasibility Study
DeGrussa (Cu-Au)	Australia	Scoping Study
Wedgetail (Au)	Australia	Detailed Feasibility Study
Westonia (Au)	Australia	Detailed Feasibility Study
Woody Woody (Mn)	Australia	Detailed Feasibility Study

Table 2
Mintrex Selected Recent Project Experience

Project	Location	Role
Tengrela (Au)	Côte d'Ivoire	Implementation Contract Preparation
Bonikro (Au)	Côte d'Ivoire	Engineering, proc and support
Sydvaranger (Fe)	Norway	Mechanical design and procure
Nzema (Au)	Ghana	Implementation preliminaries
Sandfire - DeGrussa (Cu-Au)	Australia	Early Works Package EP
Warrego (Au)	Australia	EPCM
Magellan (Pb)	Australia	Upgrade design and procurement
Wedgetail (Au)	Australia	EPCM (project halted)
Woody Woody (Mn)	Australia	EPCM
Darwin Port Storage (Fe)	Australia	Engineering and construction support
Avoca (Au)	Australia	Client Rep and Superintendent
Gwalia (Au)	Australia	Upgrade engineering design
Cloudbreak III (Fe)	Australia	Design and const. mgt. (project halted)
Mt Weld (RE: La-Lu)	Australia	Superintendence and construction mgt
Garden Well (Au)	Australia	Engineering and procurement
Duketon (Au)	Australia	Engineering and procurement
Mt Cattlin (Li)	Australia	EPCM
Port Hedland Port (Fe)	Australia	Operational readiness and asset mgt
POAG Port Hedland (Fe)	Australia	Engineering, procurement and support
Ridges (Fe)	Australia	Project management and eng. support
Mt Weld (RE: La-Lu)	Australia	Operational readiness and asset mgt

DESIGN SERVICES

Mintrex can offer the following design services to its clients as a stand-alone design service or as part of its role in managing a larger project.

Process Engineering

Metallurgical Ore Body investigations	Operating Plant analysis
Testwork programme development	Plant due diligence investigations
Analysis of testwork data	Process Design and Equipment Selection
Process Circuit design	

Civil Engineering

Road and Road Infrastructure Design	Land Subdivision Design
Waste Management Design	Civil Drafting

Mechanical Engineering

Material Handling System Design	Underground Loading Station Design
Conveyor System Design	Underground Haulage Systems Design
Belt, Pneumatic and Screw Conveyor Design	Mine Dewatering
Dust and Fume Extraction Design	Slurry Handling Systems Design
Pump Selection	Solids and Liquids Sampling System Design
Piping Design	Crane Design to AS 1418
Compressed Air Systems Design	Commissioning
Crushing and Screening Design	Mechanical Drafting
Heat Transfer Systems Design	

Structural Engineering

Structural Design to AS4100	Belt Conveyor Structures – Suspended & Other
Structural Design to AS3600	
Steel Mining Structures	Steel Tank Design to API 650/AS1210
Mobile Plant Structures	Heavy Machinery Support Base
Silo Design to AS3774	Process Plant Structural Design

Structural Drafting

Electrical Engineering

HV Substation Design

Motor Control Design

Fault Level and Protection Studies

Power Reticulation System Design

Underground Power Systems Design

Process Plant Control

Instrumentation Design

Electrical Drafting

SHORT FORM CONTRACT

AGREEMENT FOR THE PROVISION OF CONSULTING SERVICES



Driving Business Success for Consulting Firms in the Built and Natural Environment

Business | Engineering | Design

BETWEEN

(the "Consultant")

AND

(the "Client")

- 1 The Consultant shall provide to the Client the consulting services described in the accompanying letter together with such other services as may be agreed from time to time (the "Services").
- 2 The Consultant shall provide the Services with such skill, care and diligence as is generally exercised by competent members of the consulting profession performing services of a similar nature, at the time the Services are provided.
- 3 The Services will be performed at either or both the site of the project (the "Site") or at other places reasonably required by the Client. Where the locations of the Consultant's work are not under the Consultant's control (including the Site), the Client must provide reasonable access to allow the Consultant to fulfil its obligations (including to provide the Services).
- 4 The Client shall, at its own cost, as soon as practicable make available to the Consultant all information, documents and other particulars relating to the Client's requirement for the project as is necessary for the Consultant to carry out the services as expressly set out in this Agreement (the "Requirements"). The Consultant is entitled to rely on such information, documents and other particulars as are provided by the Client pursuant to or in connection with this Agreement.
- 5 The Client shall pay to the Consultant:
 - a) the Fee and the Reimbursable Expenses as set out in the accompanying letter together with such other amounts in respect of other services agreed to be provided;
 - b) reasonable adjustments to the Fee and the Reimbursable Expenses to reflect the additional costs, expenses, liabilities, losses or other amounts incurred or suffered by the Consultant in the performance of the Services and arising out of or in connection with any event or matter beyond the Consultant's control; and
 - c) to the extent that amounts payable under this Agreement are not expressed to be GST inclusive, an additional amount for the GST incurred by the Consultant in relation to the supply of the Services ("GST").
- 6 The Consultant may claim payment in accordance with the times set out in the accompanying letter or, if no time is set out, monthly in arrears. The Client must pay to the Consultant, without set-off or deduction:
 - a) the amount payable under this Agreement for the Services provided during the relevant period, within 30 days of the Consultant's invoice; and
 - b) the GST payable under this Agreement for the Services provided during the relevant period, within 30 days of receiving a valid tax invoice.
- 7 If the Client does not pay the Consultant in accordance with this Agreement then, without prejudice to any other rights or remedies the Consultant may have, interest will be payable from the date of invoice until payment at a rate per annum equal to the Unsecured Personal Overdraft Rate as most recently published by Westpac, plus 1% per annum.
- 8 To the maximum extent permitted by law:
 - a) subject to paragraphs (b), (c) and (d) below, the Consultant's liability to the Client arising out of or in connection with this Agreement (including the performance or non-performance of the Services), whether under the law of contract, in tort, in equity, under statute or otherwise, shall be limited in aggregate to the amount specified in the accompanying letter or \$300,000, if no amount is stated in the letter.
 - b) the Consultant is not liable to the Client in respect of any indirect, consequential or special losses (including loss of profit, loss of business opportunity and payment of liquidated sums or damages under any other agreement);
 - c) the Consultant shall be deemed to have been discharged from all liability in respect of the Services whether under contract, in tort, in equity, under statute or otherwise, at the expiration of the period specified in the accompanying letter, or if no date is specified, on the expiration of 3 years from the completion of the Services;
 - d) if, and to the extent that, any of this clause is void as a result of section 68 of the Trade Practices Act 1974 (Cth), then the Consultant's liability for a breach of a condition or warranty is limited to:
 - (i) the supplying of the relevant Services again; or
 - (ii) the payment of the cost of having the Services supplied again.
- 9 Subject to the Client complying with its obligations under the Agreement, the Consultant grants to the Client a non-exclusive, royalty-free and irrevocable licence to use (and allow others to use) any intellectual property (including all drawings, reports, specifications, bills of quantity, calculations and other documents, including "works" as defined in the Copyright Act 1968 (Cth) created or produced by the Consultant) arising out of provision of the Services ("IP Rights") for the purposes of completing the Project. As between the Client and the Consultant, the ownership of the IP Rights vests in the Consultant.

- 10 Neither the client nor the Consultant shall disclose to third parties or use for any purpose (other than providing or benefiting from the Services) any information provided by the other unless:

- a) required by law;
- b) the information is already generally known to the public; or
- c) the other consents to the disclosure.

All documentation and materials containing confidential information provided by one party to the other shall be returned upon request.

- 11 Any dispute or difference ("Dispute") between the Client and the Consultant may be notified by a party to the other party and the parties shall:
 - a) firstly meet to negotiate, in good faith, resolution of the Dispute; and
 - b) secondly, if negotiation fails to achieve a resolution of the Dispute within 5 working days of the notification of the Dispute, attend mediation, administered in accordance with procedures as set out by the Institute of Arbitrators and Mediators Australia, provided that this provision shall not prevent the Consultant from instituting legal action at any time to recover moneys owing by the Client to the Consultant.
- 12 The Client may, without prejudice to any other rights or remedies it may have, by written notice served on the Consultant terminate its obligations under this Agreement:
 - a) if the Consultant is in breach of the terms of the Agreement and the breach has not been remedied within 28 days of a written notice served by the Client on the Consultant specifying the breach and requiring the breach to be remedied; or
 - b) upon the Client giving the Consultant 60 days' written notice of its intention to do so; or
 - c) if the Consultant informs the Client that it is insolvent, becomes bankrupt, or becomes subject to any official management, receivership, liquidation, provisional liquidation, voluntary administration, winding up or external administration ("Insolvency Event").
- 13 The Consultant may, without prejudice to any other rights or remedies it may have, by notice in writing served on the Client suspend its obligations under this Agreement:
 - a) immediately by written notice if the Client has failed to pay in accordance with this Agreement; or
 - b) if the Client is in breach of any of the other terms of the Agreement and the breach has not been remedied within 10 working days (or longer as the Consultant may allow) of a written notice served by the Consultant on the Client specifying the breach and requiring the breach to be remedied.
- 14 The Consultant may, without prejudice to any other rights or remedies it may have, terminate its obligations under this Agreement:
 - a) if the breach referred to in clause 13(a) has not been remedied within 5 days of a written notice served by the Consultant on the Client specifying the breach and requiring the breach to be remedied; or
 - b) if the Client is in breach of any of the other terms of the Agreement and the breach has not been remedied within 28 days of a written notice served by the Consultant on the Client specifying the breach and requiring the breach to be remedied; or
 - c) upon the Consultant giving the Client 60 days' written notice of its intention to do so; or
 - d) if an Insolvency Event occurs in relation to the Client.
- 15 If the Consultant considers it appropriate to do so, it may, with the Client's prior approval, which shall not be unreasonably withheld or delayed, engage other consultants to assist the Consultant in specialist areas. The other consultant shall be engaged at the Client's risk, cost and expense, and on its behalf.
- 16 Neither party may assign, transfer or sublet any obligations under this Agreement without the written consent of the other. Unless stated in writing to the contrary, no assignment, transfer or subletting shall release the assignor from any obligation under this agreement.
- 17 In the interpretation of this Agreement, no rule of construction applies to the disadvantage of one party on the basis that it put forward this Agreement or any part of it.

SIGNED by the parties on

20

CLIENT

CONSULTANT

Mintrex Pty Ltd

ABN 64 064 105 488

ACKNOWLEDGMENT OF ACCEPTANCE

TNT Mine Ltd
Level 2, 34 Colin St
WEST PERTH 6005

Attention: Mr M Hannington

Dear Mike,

PROPOSAL 1234 - Provision of Metallurgical Consultancy Services for the TNT Mines Ltd – Magnetite Section

We authorise the work to proceed on this contract under the conditions attached in the Consult Australia document.

Yours faithfully,
TNT Mine Ltd

Approver's Signature: _____

(Print Name): _____

Title: _____

Company Name: _____

Address for Invoices: _____

Accounts Dept Tel No: _____

Date: _____

Please signify this contract acceptance by signing this letter below and returning it to us by fax on 08 9442 3399 or by email to brian.povey@mintrex.com.au.

BUDGET ESTIMATE - IOQ12-106 METALLURGICAL TESTING MAGNETITE SAMPLE-TNT MINES MOINA PROJECT-MINTREX BENEFICIATION INVESTIGATION				
TESTWORK - RC CHIPS 60kg x 1 SAMPLE Core Type[TBA] 10kg x 1 SAMPLE	Unit Cost (\$)	No. of Units	No. of Samples	AUD \$
FIG TNT 001- CORE SAMPLE-PREPARATION AND TESTING				
1 Receive and handle sample	\$20	1	1	\$20
2 Crush to -32mm	\$60	1	1	\$60
3 Mix/split into 2 Samples-Sample A and B	\$45	1	1	\$45
4 Crush Sample A to -3.35mm for DTR testwork	\$30	1	1	\$30
5 Head Assay [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	1	1	\$70
6 Head Assay (CaF2)-Vendor testing -Estimate only	\$50	1	1	\$50
7 Mix/Split 150g-Head DTR P100 38µm	\$230	1	1	\$230
8 Milled Feed Size-Cyclosize (-38µm) Analysis	\$250	1	1	\$250
9 Assay Head DTR Mags [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	1	1	\$70
10 Assay DT Mags(CaF2)-Vendor testing -Estimate only	\$50	1	1	\$50
11 Mix/Split Sample B into 2 lots and Crush to 100% -4 and -2mm	\$30	1	1	\$30
12 Dry Screen (3 screen sizes) each Sample B to determine P80 and recombine	\$280	2	1	\$560
13 Dry LIMS -4 and -2mm @900 Gauss-approx 4-5kg each	\$350	2	1	\$700
14 Assay Dry LIMS mags and non-mags [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	4	1	\$280
15 Assay Dry LIMS mags and non mags(CaF2)-Vendor testing -Estimate only	\$50	4	1	\$200
16 Dry LIMS mags and non-mags sub sample 2 minute pulverise and DTW	\$150	4	1	\$600
17 Assay DT W Mag- mags and non-mags [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	4	1	\$280
18 Assay DTW Mag-mags and non mags(CaF2)-Vendor testing -Estimate only	\$50	4	1	\$200
19 Mix/Split -2mm Dry LIMS non-mags and put on Table (1-2kg)	\$450	1	1	\$450
20 Assay Table Cons (4-5) and Tail [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	5	1	\$350
21 AssayTable Cons (4-5) and Tail (CaF2)-Vendor testing -Estimate only	\$50	5	1	\$250
22 Recombine -4mm Dry LIMS mag and non-mags and crush to -2mm or -1mm (TBA)	\$55	1	1	\$55
23 Mix Split (Crushed -4mm) and Table (1-2kg)	\$450	1	1	\$450
24 Assay Table Cons (4-5) and Tail [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	5	1	\$350
25 AssayTable Cons (4-5) and Tail (CaF2)-Vendor testing -Estimate only	\$50	5	1	\$250
FIG TNT 002- RC CHIPS SAMPLE-PREPARATION AND TESTING				
1 Receive and handle sample	\$60	1	1	\$60
2 Crush 60 kg to -3.35mm for testwork	\$350	1	1	\$350
3 Mix/split into portions for testwork	\$250	1	1	\$250
4 Head Assay [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	1	1	\$70
5 Head Assay (CaF2)-Vendor testing -Estimate only	\$50	1	1	\$50
6 Asbestos Analysis	\$150	1	1	\$150
7 Mix/Split 150g-Head DTR P100 38µm	\$230	2	1	\$460
8 Milled Feed Size-Cyclosize (-38µm) Analysis	\$250	2	1	\$500
9 Assay Head DTR Mags [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	2	1	\$140
10 Grind Establishment (P80 500,250,150,75,45,32,25µm)	\$1,450	1	1	\$1,450
11 Mill to Size (P80 500,250,150,75,45,32,25µm)-1kg EACH	\$55	7	1	\$385
12 Milled Feed Sizing (P80 500µm)-Size confirmation	\$270	1	1	\$270
13 Standard DTW @3000 (P80 500,250,150,75,45,32,25µm)	\$110	7	1	\$770
14 Assay DTW Mags [XRF-24 elements+LOI100+W+Sn(low level)]	\$70	7	1	\$490
15 Assay DTW Mags(CaF2)-Vendor testing -Estimate only	\$50	7	1	\$350
 Supervision and Reporting (17.5%)	\$2,034	1	1	\$2,034
Courier -Estimate	\$500	1	1	\$500
 <u>Sample Storage</u>				
6 months of storage per pallet (\$85 per pallet per month)	\$85	6	1	\$510
 Assay Suite - "24 Element" XRF, LOI 371,650 and 1000°C AMMTEC Assay				
SUBTOTAL				\$8,789.00
GST 10%				\$878.90
TOTAL				\$9,667.90

Assay Intervals For Main Composite

Hole ID	Intercept, m		Lithology	F (%)	CaF2 (%)	Al (%)	Bi (ppm)	Ca (%)	Cd (ppm)	Cu (ppm)	Fe (%)	Mg (%)	Mo (ppm)	Pb (ppm)
	From	To												
MODD001	25.0	26.0	wrigglite skarn	10.6	21.78	3.94	781	15.4	10	119	16.4	1.46	<5	12
MODD001	26.0	27.0	wrigglite skarn	7.82	16.07	4.37	486	14.2	9	51	15.0	4.97	<5	<5
MODD001	27.0	28.0	wrigglite skarn	6.96	14.30	3.68	536	12.7	9	58	13.7	5.25	15	6
MODD001	29.0	30.0	wrigglite skarn	11.48	23.59	5.02	515	13.5	11	184	19.9	2.71	<5	6
MODD001	31.0	32.0	wrigglite skarn	7.55	15.52	4.70	601	17.6	9	23	15.3	1.08	<5	14
MODD001	34.0	35.0	wrigglite skarn	10.83	22.26	4.82	747	15.4	12	<2	22.1	0.99	<5	9
MODD001	37.0	38.0	wrigglite skarn	11.92	24.50	4.56	418	15.7	10	2	16.4	1.09	<5	11
MODD001	40.0	41.0	wrigglite skarn	11.46	23.55	4.87	224	18.1	8	<2	13.5	1.36	10	7
MODD001	41.0	42.0	wrigglite skarn	9.67	19.87	4.60	201	19.8	8	<2	14.0	1.06	<5	10
MODD001	42.0	43.0	wrigglite skarn	7.08	14.55	4.59	146	18.7	15	7	12.8	1.95	<5	16
MODD001	46.0	47.0	wrigglite skarn	10.38	21.33	4.26	510	15.8	9	<2	16.3	1.36	17	10
MODD001	48.0	49.0	wrigglite skarn	8.59	17.65	5.27	48	11.7	13	<2	23.6	1.54	62	6
MODD001	62.0	63.0	wrigglite/garnet skarn	9.1	18.70	4.26	536	12.5	13	<2	22.8	1.65	36	<5
MODD001	63.0	64.0	wrigglite/garnet skarn	7.07	14.53	3.79	586	17.0	10	<2	15.6	1.48	<5	18
MODD001	66.0	67.0	wrigglite/garnet skarn	13.75	28.26	4.50	577	18.2	9	87	12.9	0.55	26	15
MODD001	69.0	70.0	wrigglite/garnet skarn	8.89	18.27	4.91	442	18.3	10	3	16.6	1.13	236	5
MODD003	34.0	34.6	wrigglite skarn	7.67	15.76	4.57	603	12.9	8	8	24.3	0.72	<5	18
MODD003	36.1	37.0	wrigglite skarn	10.95	22.50	4.36	506	11.5	9	<2	28.3	0.58	<5	20
MODD003	38.0	39.0	wrigglite skarn	6.08	12.49	3.33	645	15.4	<5	23	11.6	1.07	<5	12
MODD003	40.0	41.0	wrigglite skarn	13.18	27.08	5.10	864	15.7	7	203	19.7	0.79	9	11
MODD003	42.0	43.0	wrigglite skarn	12.37	25.42	4.67	938	13.1	9	34	26.9	0.74	<5	17
MODD003	45.0	46.0	wrigglite skarn	12.92	26.55	4.29	839	14.8	6	17	18.8	0.61	15	10
MODD003	46.0	47.0	wrigglite skarn	10.89	22.38	4.93	636	14.4	7	61	19.7	0.98	12	10
MODD003	47.0	48.0	wrigglite skarn	11.08	22.77	4.73	867	15.8	6	104	17.5	1.00	7	19
MODD003	49.0	50.0	wrigglite skarn	8.89	18.27	3.86	713	12.7	7	8	22.1	3.38	<5	17
MODD003	51.0	52.0	wrigglite skarn	10.64	21.87	4.76	686	14.4	6	7	17.6	2.19	34	5
MODD003	52.0	53.0	wrigglite skarn	9.85	20.24	4.77	658	16.2	6	<2	17.2	2.31	16	6
MODD003	54.0	55.0	wrigglite skarn	12.02	24.70	5.10	668	20.0	5	28	15.9	2.10	15	17
MODD003	56.0	57.0	wrigglite skarn	10.10	20.76	4.43	711	22.4	5	9	16.0	2.24	17	6
MODD003	57.0	58.0	wrigglite skarn	8.82	18.13	4.50	713	12.1	6	10	19.0	6.63	9	8
MODD003	59.0	60.0	wrigglite skarn	9.37	19.26	4.49	611	14.1	6	3	18.0	4.46	<5	16
MODD003	60.0	61.0	wrigglite skarn	9.28	19.07	5.10	502	16.6	6	16	17.9	2.00	7	12
MODD003	62.0	63.0	wrigglite skarn	7.13	14.65	4.66	850	15.2	6	<2	20.4	1.21	<5	12
MODD003	64.0	65.0	wrigglite skarn	9.00	18.50	5.67	449	15.4	6	<2	15.7	1.44	14	6
MODD003	66.0	67.0	wrigglite skarn	11.38	23.39	5.49	621	20.2	6	48	18.0	0.61	19	11
MODD003	67.0	68.0	wrigglite skarn	12.36	25.40	5.51	726	21.0	6	36	18.3	0.60	<5	6
MODD003	71.0	72.0	wrigglite skarn	7.52	15.45	5.37	471	15.9	<5	57	12.0	4.16	17	6
MODD003	73.0	74.0	wrigglite skarn	8.70	17.88	5.14	580	15.8	6	89	15.2	1.60	27	16
MODD003	74.0	75.0	wrigglite skarn	10.14	20.84	5.37	591	17.6	6	32	17.3	2.16	11	9
MODD003	76.0	77.0	wrigglite skarn	10.68	21.95	5.57	534	17.1	6	27	17.5	1.23	36	9
MODD003	77.0	78.0	wrigglite skarn	7.73	15.89	4.78	769	18.5	6	53	17.2	1.44	73	18
MODD004	53.0	54.0	wrigglite skarn	14.47	29.74	5.17	998	19.1	<5	141	19.1	0.72	<5	13
MODD004	54.0	55.0	wrigglite skarn	5.22	10.73	4.33	258	18.6	<5	<2	19.1	1.26	<5	23
MODD004	56.0	57.0	wrigglite skarn	10.67	21.93	5.07	676	16.8	<5	84	18.8	1.69	<5	7
MODD004	59.0	60.0	wrigglite skarn	9.18	18.86	4.50	486	13.6	<5	80	20.2	2.62	6	9
MODD004	60.0	61.0	wrigglite skarn	9.28	19.07	4.51	484	14.4	<5	31	18.3	3.95	10	6
MODD004	62.0	63.0	wrigglite skarn	9.69	19.91	4.53	326	17.4	<5	32	15.1	3.99	23	12
MODD004	64.0	65.0	wrigglite skarn	10.05	20.65	4.84	368	15.1	<5	35	17.9	3.23	21	<5
MODD004	67.0	68.0	wrigglite skarn	10.31	21.19	5.46	289	19.1	<5	18	13.5	1.22	13	14
MODD004	69.0	70.0	wrigglite skarn	10.69	21.97	4.43	497	18.6	<5	<2	20.7	0.89	<5	11
MODD004	71.0	72.0	wrigglite skarn	6.69	13.75	4.93	171	16.6	<5	3	17.6	2.11	10	11

Assay Intervals For Main Composite

Hole ID	Intercept, m		Lithology	S (%)	Sb (ppm)	Si (%)	Sn (ppm)	W (ppm)	Zn (ppm)
	From	To							
MODD001	25.0	26.0	wrigglite skarn	0.50	54	9.30	1522	382	175
MODD001	26.0	27.0	wrigglite skarn	0.17	36	10.4	1116	1750	211
MODD001	27.0	28.0	wrigglite skarn	0.16	41	10.3	1059	631	272
MODD001	29.0	30.0	wrigglite skarn	1.13	17	10.4	1743	540	141
MODD001	31.0	32.0	wrigglite skarn	0.15	53	12.3	1987	752	121
MODD001	34.0	35.0	wrigglite skarn	<0.02	24	10.1	1576	544	167
MODD001	37.0	38.0	wrigglite skarn	0.06	30	10.8	1029	405	229
MODD001	40.0	41.0	wrigglite skarn	<0.02	21	13.3	1441	1122	159
MODD001	41.0	42.0	wrigglite skarn	<0.02	36	12.6	1618	792	277
MODD001	42.0	43.0	wrigglite skarn	0.11	73	13.1	1093	280	1863
MODD001	46.0	47.0	wrigglite skarn	0.08	21	11.3	1108	719	225
MODD001	48.0	49.0	wrigglite skarn	0.07	<5	9.45	626	1227	132
MODD001	62.0	63.0	wrigglite/garnet skarn	0.05	10	9.97	2569	3718	327
MODD001	63.0	64.0	wrigglite/garnet skarn	<0.02	38	11.4	1595	346	244
MODD001	66.0	67.0	wrigglite/garnet skarn	0.57	36	9.91	577	644	398
MODD001	69.0	70.0	wrigglite/garnet skarn	0.05	50	11.4	978	1750	514
MODD003	34.0	34.6	wrigglite skarn	0.11	17	11.7	2519	1000	103
MODD003	36.1	37.0	wrigglite skarn	0.20	<5	10.5	3422	633	134
MODD003	38.0	39.0	wrigglite skarn	0.11	7	19.4	1558	785	102
MODD003	40.0	41.0	wrigglite skarn	1.31	12	11.3	1840	1211	89
MODD003	42.0	43.0	wrigglite skarn	0.21	9	9.36	2100	861	124
MODD003	45.0	46.0	wrigglite skarn	0.08	14	14.5	1731	1347	103
MODD003	46.0	47.0	wrigglite skarn	0.38	10	13.1	1819	1399	105
MODD003	47.0	48.0	wrigglite skarn	0.57	18	13.7	1545	774	94
MODD003	49.0	50.0	wrigglite skarn	0.06	12	11.8	1936	988	119
MODD003	51.0	52.0	wrigglite skarn	0.16	19	11.4	1359	2328	133
MODD003	52.0	53.0	wrigglite skarn	0.02	37	11.4	1204	1365	117
MODD003	54.0	55.0	wrigglite skarn	0.13	39	11.4	1203	1316	149
MODD003	56.0	57.0	wrigglite skarn	0.06	77	9.78	765	2147	128
MODD003	57.0	58.0	wrigglite skarn	0.07	22	11.4	1024	1164	170
MODD003	59.0	60.0	wrigglite skarn	0.04	22	11.6	1179	768	191
MODD003	60.0	61.0	wrigglite skarn	0.09	29	13.6	1332	1138	151
MODD003	62.0	63.0	wrigglite skarn	0.07	12	14.8	2401	1389	109
MODD003	64.0	65.0	wrigglite skarn	0.19	14	13.4	1122	1128	123
MODD003	66.0	67.0	wrigglite skarn	0.22	27	11.7	1175	1659	134
MODD003	67.0	68.0	wrigglite skarn	0.17	28	11.3	1168	1472	119
MODD003	71.0	72.0	wrigglite skarn	0.26	28	14.9	1001	1250	136
MODD003	73.0	74.0	wrigglite skarn	0.48	12	13.2	1338	1091	110
MODD003	74.0	75.0	wrigglite skarn	0.20	25	13.2	1438	1845	109
MODD003	76.0	77.0	wrigglite skarn	0.19	10	13.7	1731	1551	100
MODD003	77.0	78.0	wrigglite skarn	0.35	22	14.7	1613	1494	111
MODD004	53.0	54.0	wrigglite skarn	0.77	49	10.2	1691	888	123
MODD004	54.0	55.0	wrigglite skarn	0.04	34	13.6	3411	334	116
MODD004	56.0	57.0	wrigglite skarn	0.55	34	11.8	1509	1036	126
MODD004	59.0	60.0	wrigglite skarn	0.49	26	11.3	1494	784	156
MODD004	60.0	61.0	wrigglite skarn	0.18	44	11.8	1288	966	167
MODD004	62.0	63.0	wrigglite skarn	0.22	53	11.1	1080	998	163
MODD004	64.0	65.0	wrigglite skarn	0.25	37	11.6	1165	1146	136
MODD004	67.0	68.0	wrigglite skarn	0.19	34	13.2	1182	1045	537
MODD004	69.0	70.0	wrigglite skarn	0.11	33	11.3	1896	1060	154
MODD004	71.0	72.0	wrigglite skarn	0.05	26	13.6	1610	708	144

Assay Intervals - Second Composite (DTR Only)

Hole ID	Intercept, m		Lithology	F (%)	CaF2 (%)	Al (%)	Bi (ppm)	Ca (%)	Cd (ppm)	Cu (ppm)	Fe (%)	Mg (%)	Mo (ppm)	Pb (ppm)
	From	To												
MODD001	30.0	31.0	wrigglite skarn	10.85	22.30	4.83	415	14.9	12	87	21.5	1.25	<5	10
MODD001	47.0	48.0	wrigglite skarn	10.72	22.03	4.78	484	14.8	9	<2	16.7	1.54	29	<5
MODD003	32.0	33.0	wrigglite skarn	9.98	20.51	4.08	1236	11.2	9	10	30.0	0.88	<5	21
MODD003	33.0	34.0	wrigglite skarn	5.59	11.49	3.90	628	7.45	13	<2	40.4	0.65	<5	33
MODD003	49.0	50.0	wrigglite skarn	8.89	18.27	3.86	713	12.7	7	8	22.1	3.38	<5	17
MODD003	75.0	76.0	wrigglite skarn	10.96	22.52	4.96	665	18.1	6	20	18.4	1.30	18	18
MODD004	64.0	65.0	wrigglite skarn	10.05	20.65	4.84	368	15.1	<5	35	17.9	3.23	21	<5

Assay Intervals - Second Composite (DTR Only)

Hole ID	Intercept, m		Lithology	S (%)	Sb (ppm)	Si (%)	Sn (ppm)	W (ppm)	Zn (ppm)
	From	To							
MODD001	30.0	31.0	wrigglite skarn	0.57	30	10.8	1629	424	138
MODD001	47.0	48.0	wrigglite skarn	0.95	12	11.3	1095	1127	140
MODD003	32.0	33.0	wrigglite skarn	0.11	6	9.01	2789	677	128
MODD003	33.0	34.0	wrigglite skarn	0.02	<5	7.33	2254	812	137
MODD003	49.0	50.0	wrigglite skarn	0.06	12	11.8	1936	988	119
MODD003	75.0	76.0	wrigglite skarn	0.14	10	13.2	2215	1239	97
MODD004	64.0	65.0	wrigglite skarn	0.25	37	11.6	1165	1146	136

PROGRESS REPORT

ABN 64 064 105 488

To: Michael Hannington **Date:** 15th June 2012
From: Brian Povey **cc:**
Subject: **TNT — Metallurgical Consultancy**
Mintrex Progress Report 1— Period Ending Saturday 16th June 2012

Dear Mike,

The testwork programme has been developed and agreed

- 7 x 1 kg samples will be ground to different sizes down to 80%-25micron and concentrated in a Davis Tube
The feed and concentrates will be assayed to allow estimates of the distribution of various elements to magnetics and non magnetics and at what size
- In a second set of tests samples have been crushed to 4mm and 2mm — these will be tested by dry magnetic separation to again check the distribution at coarse sizes. A sample of -2mm will also be tabled to see if a W, and Bi and Fe rich stream can be separated. In a second test a sample of non magnetic -2mm material will be tabled

What is trying to be done here is see if a richer W or Bi stream can be produced at 2mm — which might represent the discharge of a rod mill. We are seeing if we need to remove the magnetics first — or whether it can be done as a single stream.

The current status is that the RC chip/crushed core from AMMTEC Burnie is now being split into subparts to do the grind work.

The Core that was received had a couple of samples taken for DTR and the rest is being used to develop separation data at 4 and 2mm. We did plan to do more DTR tests on these but we probably won't get much of a feel about variability yet. The samples from Burnie were too fine for the coarser dry magnetic separation work and AMMTEC here assumed that when they received the core that was what it was for.

I have talked to Ron and provided some assistance with equipment sizing, capital and operating costs and in the test programme we are testing some design concepts which should improve the design

Regards

Brian Povey

Ph: +61 (08) 9442 3316
Mobile +61 (0)4 39110039

PROGRESS REPORT

ABN 64 064 105 488

To:	Michael Beer/Michael Hannington	Date:	3 rd August 2012
From:	Brian Povey	cc:	
Subject:	TNT – Metallurgical Consultancy Mintrex Progress Report 2– Period Ending Saturday 4 th August 2012		

Dear Mike/Mike,

The work as currently agreed is complete subject to final drafting of the report on the basis of a review with the client this coming week.

The draft report was submitted last week for review:

- The magnetite is low grade even at 80% passing 20micron – the composite chip sample went 58%Fe and the drill core sample went 63% Fe – the latter would be more interesting if widespread. There are issues about the Sn content of the concentrate and the content of alkaline metals – all of which exceed normal specifications. A marketing expert should be consulted on this issue.
- On the fluorite there are discrepancies with the F assays with values only being half those expected for a given CaO – especially since there doesn't appear to be a substantial carbonate content based on LOI and C assays. The resolution of this is continuing with the laboratory
- The value of the ore in terms of final products is twice the operating cost Jacobs have predicted to produce them in a sellable form.
- There would be value in redoing some variability work – unfortunately the laboratory enthusiastically crushed the previous samples as a single sample for the coarse testwork planned - my fault - I had left my detailed instructions with ALS Malaga where I do most DTR work but samples arrived at ALS AMMTEC where we were doing the metallurgical work.

Proposed work in the next weeks

No activity – work scope completed – subject to final report drafting

Schedule and S Curve

- N/A

Expenditure This (2 month) Period (ex GST)

- Total \$A 7,790.

The budget of \$A10,000 was exceeded by \$3,000 due to the additional time spent on trying to resolve F issues and inclusion of gravity and flotation work in the main report. .

Safety, Environment and Community

- None.

Issues of Concern

- Quality of the magnetite as an easily sellable commodity.
- Fluorine content of final product



Appendices

- Invoices
- TimePro reports.

I would like to thank you for the opportunity to be involved with this project and if a decision is made to continue that Mintrex can continue to be a part.

Regards

Brian Povey

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Mobile +61 (0)4 39110039

APPENDIX 3 – Moina Project Tasmania: Scoping Study August 2012. Jacobs.

TNT Mines (Moina) Pty Ltd

ACN 131 786 831
a wholly owned subsidiary of
TNT Mines Limited
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Moina Project, Tasmania
Scoping Study
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EXECUTIVE SUMMARY

TNT Mines Limited (TNT) is planning to undertake an Initial Public Offering (IPO) for a float on the Australian Stock Exchange (ASX) regarding a mineral tenement portfolio in Tasmania, Australia. One of these projects is known as the Moina Project and is the subject of this report. The tenement encompasses a poly-metallic deposit containing fluor spar, scheelite, magnetite and cassiterite.

The Moina Project (Project) is owned by TNT Mines (Moina) Pty Ltd, a wholly-owned subsidiary of TNT and is located in northwest Tasmania, 40 km south-west from Devonport.

Jacobs E&C Australia Pty Ltd (Jacobs) was commissioned by TNT in May 2012 to prepare a mining and ore treatment Scoping Study (Study) that would enable TNT to undertake a Preliminary Economic Analysis (PEA) of the Project.

The Study examined the production of saleable concentrates from the contained valuable minerals through crushing, heavy media, grinding, magnetic, gravity and flotation processes to produce a treatment rate of 800,000 tpa.

The investigation sought to establish the hierarchy of minerals to be produced based on quantity and value and hence, indicate the overall commercial viability of the Project.

Scope of Work

Jacobs' scope of work was to:

- Examine all available information to produce a conceptual ore process flow sheet
- Establish the Capital Expenditure (Capex) required to develop a mine to produce saleable concentrate.
- Establish the Operating Expenditure (Opex).

The Opex was to include:

- Mining costs
- Transport and logistics
- Power
- Water and associated costs, and any possible restrictions
- Potential customers.

In essence, the Study was to consider:

- Stockpile and feed systems
- Primary secondary and tertiary crushing circuit
- Cyclone heavy media pre-concentration
- Primary milling and fine screen classification

- Magnetic, gravity and flotation circuits including regrind mills
- Concentrate dewatering and filter plant
- Instrumentation and controls system
- Plant tailings disposal.

Note: TNT has commissioned more test work by ALS Burnie Research Laboratory Group (Burnie Research Laboratory); the results of which will not be available for this Study. However, an extension of this Study will involve revisiting the Burnie Research Laboratory report to accommodate its findings.

Limitations and Use of Report

This report or item (Report) was prepared by Jacobs E&C Australia Pty Ltd (Jacobs) for TNT Mines (Moina) Pty Ltd (the Client) pursuant and subject to the terms and conditions of an Engineering Services Agreement, dated 17 May 2012 (the Agreement).

This Report is provided solely for the purposes stated in the Agreement and for the exclusive benefit of the Client. This Report is to be read as a whole.

The recommendations and opinions contained in this Report assume that unknown, unforeseeable or unavoidable events which may adversely affect the cost, progress, scheduling or ultimate success of the relevant Project will not occur.

This Report is based, in whole or in part, on information provided by or on behalf of the Client. Except to the extent (if any) provided in the Agreement, the Client shall be responsible for verifying the accuracy and sufficiency of all information provided to Jacobs by or on behalf of the Client. Jacobs may rely on such information for the purposes of the Services (Reliable Information).

Jacobs disclaims responsibility for the accuracy or appropriateness of any recommendation or statement of opinion in the Report, to the extent that such recommendation or opinion is dependent on the Reliable Information.

For the avoidance of doubt, Jacobs makes the assumptions and other qualifications that are set out in the next section:

It must be noted that the quotations and costings contained in this Study report were received from TNT in May and June of 2012. Due to new Australian Government tax legislation as of 1 July 2012 and changing conditions in global economic circumstances, all information provided herein will need revising and updating subsequent to this report's issue for further assessment.

All Study information, findings and details provided in this document only meet a scoping level of accuracy.

Assumptions

The mining and ore treatment process was based on test work undertaken over a number of years by various organisations.

The nominal price of water from Lake Gairdner for mine operations was considered to be AU\$55/ML.

The Study parameters were based on the details provided in Table 0-1.

Table 0-1 Ore Production Targets

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

Note: the grades in the Table 0-1 are those of the 656,000t pre-concentrated ore.

CaF₂ = calcium fluoride

Fe₃O₄ = magnetite

WO₃ = wolframite tungsten trioxide

Fe = iron

The Study considered:

Mining Options: including Owner-operated, dry hire of equipment, contract and associated manning, all of which will depend on TNT's company strategy and policy.

An overall site manning chart was developed which will also need further review when TNT's company strategy and policy are considered.

Process Plant: incorporating the equipment that will liberate and concentrate magnetite, scheelite and fluorspar, based on test work carried out over recent years.

CAPEX AND OPEX ESTIMATES

General and Administration Cost

A cost of AU\$6/t was estimated and added to the Mining and Processing costs for General and Administration services.

Mining Capex

The TNT operators' Mining Capex was estimated to be AU\$30M by Mining One Consultants from Melbourne, Australia.

Process Plant Capex

The process plant Capex estimate is shown in Table 0-2.

Table 0-2 – Process Plant Capex Estimate

Item	AU\$M	%
Equipment and installation	50.365	
Engineering Procurement and Construction Management (EPCM)	9.07	18% Equip & Install
Contingency	14.86	25% Equip & EPCM
Subtotal	74.30	
Accuracy	22.29	30% of Subtotal
Total	96.59	

Capex for Mine Preparation

The estimated owner-operator Capex for mine preparation is included in Table 0-3.

Table 0-3 Mine Preparation Capex

Item	AU\$M
Clearing of 10 ha x AU\$4,500/ha	0.045
Stripping, levelling, dam building, internal roads	6
Total	~135

Mining Opex

Mining cost estimates were received from two Tasmanian operators, Mancala Pty Ltd and Shaw Contracting Pty Ltd (Shaw Contracting) (see Appendix A).

The Shaw team visited the Project site and produced a more considered quote than Mancala Pty Ltd, giving Jacobs an opportunity to assess the mining costs by a broader variety of ways.

This report has used the Shaw Contracting figures, however, a comparison of mining Opex estimates provided by Mining One, Shaw Contracting and Mancala Pty Ltd are shown in Table 0-4.

Table 0-4 Comparison of Mining Opex Estimates

Mining One Consultants	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	AU\$3.80	AU\$4.60	AU\$5.00
Manning	53		

Mancala Pty Ltd	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	-	-	AU\$4.25
Manning	-	-	31
Shaw Contracting	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	-	-	AU\$12.34
Manning	-	-	10
Total Mining Cost			AU\$12.34*

* This is regarded as high compared to other estimates and should be further investigated in later studies.

Process Plant Opex

The process plant Opex estimate is shown in Table 0-5.

Table 0-5 Process Plant Opex Estimates

Area	Unit Costs AU\$/t Run-of-Mine (ROM)	Total Costs Per Annum (AU\$)
Labour	6.35	5,082,000
Consumables	4.71	3,771,500
Power	3.50	2,797,704
Maintenance	2.0	1,600,000
Total Cost	16.56	13,251,204

Note: the battery limits and exclusions for the process plant given in this document include some infrastructure, such as accommodation, management, safety and fire control facilities which will need to be addressed specifically in later study stages.

Logistics

The Project logistics estimate for loading and transporting concentrate to Burnie in Tasmania was AU\$27/t. Kibble manufacture costs will be extra. This figure is high and will need further investigation and benchmarking in later studies.

Container transportation by ship from Burnie to Melbourne in Victoria was estimated to be AU\$1,051.88 plus a fuel charge of AU\$12.50 per 80-tonne container. As a result, the shipping cost was estimated to be approximately AU\$14/t.

Cost Comparison of Three Operating Methods

When comparing the costs between three operating methods, it was assumed that an additional mine-site civil preparation cost of AU\$6.045 was applicable to each method.

Logistics, transport of concentrate and port costs were also considered common between all the methods.

Method 1: Owner Mined and Concentrated

Capex estimate: AU\$125.2M

Opex estimate: AU\$16.26/t

Method 2: Contract Mining

Capex estimate: AU\$96.7M

OpEx estimate: AU\$12.34/t

Method 3: Dry Lease of Mining Equipment

Capex estimate: AU\$96.7M

Opex estimate: AU\$4.60/t + AU\$7.42/t labour = AU\$12.02/t

Method 1 is seen to be the least economical.

IDENTIFIED MARKETING OPPORTUNITIES

Investigations have revealed that the key product in today's market situation is fluorspar, and the best market negotiating opportunities were indicated in India.

NEXT STEPS AND RECOMMENDATION

In view of the mineral quality and concentration, the relative abundance of water, the closeness to a source of power; the good quality of the road system for transportation to shipping, the tentative indications of government support and indication of relative non-contentious environmental position as it is a past mining site, the potential mine at Moina warrants further investigation and consolidating investigative study work. The next phase (Phase 2) of the study work should be undertaken.

ABBREVIATIONS

Note: all units of measurement are metric while all references to currency are in Australian dollars, unless otherwise indicated.

Term	Definition
µm	micron
%	percent
±	plus or minus
Al ₂ O ₃	Alumina mineral
ASX	Australian Stock Exchange
AU\$	Australian dollars
Capex	capital expenditure
cm	centimetre
d	day
EPCM	engineering, procurement and construction management
Fe	iron mineral
Fe ₃ O ₄	magnetite mineral
GET	ground-engaging tools
H ₂ O	the chemical symbol for water
h	hour
ha	hectare
km	kilometre
kWh	kilowatt
kW/h	kilowatts per hour
L	litre
LOM	life-of-mine
M	million
ML	megalitre
mm	millimetre
Mtpa	million tonnes per annum
MW	Mega watts
OH&S	occupational health and safety
Opex	operating expenditure
OSA	online analyser
PEA	Preliminary Economic Analysis
RL	Retention Licence
ROM	run-of-mine
t	tonne
t/m ³	tonnes per cubic metre
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour

1. INTRODUCTION

TNT Mines Limited (TNT) is planning to undertake an Initial Public Offering (IPO) for a float on the Australian Stock Exchange (ASX) regarding a mineral tenement portfolio in Tasmania, Australia. One of these projects is known as the Moina Project and is the subject of this report.

1.1 Project Description

The Moina Project (Project) is owned by TNT Mines (Moina) Pty Ltd, a wholly-owned subsidiary of TNT and is located in northwest Tasmania, 40 km south-west from Devonport.

The Project consists of a single Retention Licence RL10/1988 and covers an area of 2 km². Access to the tenement is first via the sealed Cradle Mountain Road and then by unsealed farm tracks. The deposit lies on the Burnie 1:250,000 map sheet and the Arthur River 1:100,000 map sheet. The Cethana 1:25,000 geology map sheet covers geology in the Project area, however, more detailed geology of the mine area is presented on Map 9 (1:25,000), Geology of the Winterbrook – Moina Area, extracted from the Geological Survey of Tasmania's Mt Read Volcanics Project 1989.

The tenement is managed by TNT Mines (Moina) Pty Ltd, through an option agreement with Geotech International Pty Ltd.

The poly-metallic ore contains magnetite, scheelite, fluor spar, cassiterite, and other valuable minerals, however, magnetite, scheelite and fluor spar are the minerals deemed most likely suitable for commercial extraction.

Ore has been mined and processed at the mine site in the past by small underground vein mining, and simple gravity and magnetic separation methods to produce scheelite concentrates.

TNT is now planning to mine by open-pit methods and install modern processing facilities for treating the complex fine-grained Moina ore.

1.2 Scoping Study

In May 2012, TNT awarded Jacobs E&C Australia Pty Ltd (Jacobs) to undertake a mining and ore treatment Scoping Study (Study) of the Project that would enable TNT to then undertake a Preliminary Economic Analysis (PEA).

The Scope of Work required a study to a scoping level of accuracy for the design and construction management of a process plant and associated facilities to process 800,000 tpa of magnetite-scheelite-fluor spar-bearing ore from the Moina deposit.

The Study examined the production of saleable concentrates from the contained valuable minerals through crushing, heavy media, grinding, magnetic, gravity and flotation processes. The investigation sought to establish a hierarchy of minerals

to be produced, based on quantity and value for determining an indicative, overall commercial viability for the Project.

1.2.1 General Objectives

The Study's general objectives were to:

- Establish this Phase as being the initial study phase of possibly two Study phases.
- Obtain an order of magnitude cost estimate covering plant Capex, Opex and logistics costs, based on first-pass only and existing data and assumptions some of which have been gained from the existing past mining activities in the location.
- Look at marketing issues and possible potential customers, both in product offtakes and as possible investors.
- Prepare for more accurate next Phase (Phase 2) investigation and estimates.

1.2.2 Phase 1 Scope of Work

Phase 1 tasks have required Jacobs to undertaking the following:

- Reviewing existing lab and report data
- Preparing conceptual flow sheet
- Preparing a dry material balance
- Preparing a major equipment list
- Costing of major equipment
- Provide an estimate Capex
- Determine transport and logistics cost
- Providing an estimate Opex
- Providing a site layout drawing
- Delivering marketing research
- Producing a Study report.

1.2.3 Next Phase (Phase 2) Scope of Services

The Study's scope of design services should include:

- Developing a preliminary process flow diagram
- Developing a preliminary process material balance
- Developing a major equipment list
- Preliminary Power Study is based on installed power from major equipment list
- Development to a scoping study level the plant layout

- Development of a capital cost estimate
- Development of an operating cost estimate.

Table 1-1 shows the Capex and Opex estimates at scoping study level of accuracy for a conventional mineral processing plant using parameters of 800,000 tpa, and head grades and recoveries for producing saleable magnetite, scheelite and fluor spar concentrates.

Table 1-1 Ore Production Targets

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

Note: the grades in the Table 1-1 are those of the 656,000t pre-concentrated ore.

CaF₂ = calcium fluoride

Fe₃O₄ = magnetite

WO₃ = wolframite tungsten trioxide

Fe = iron

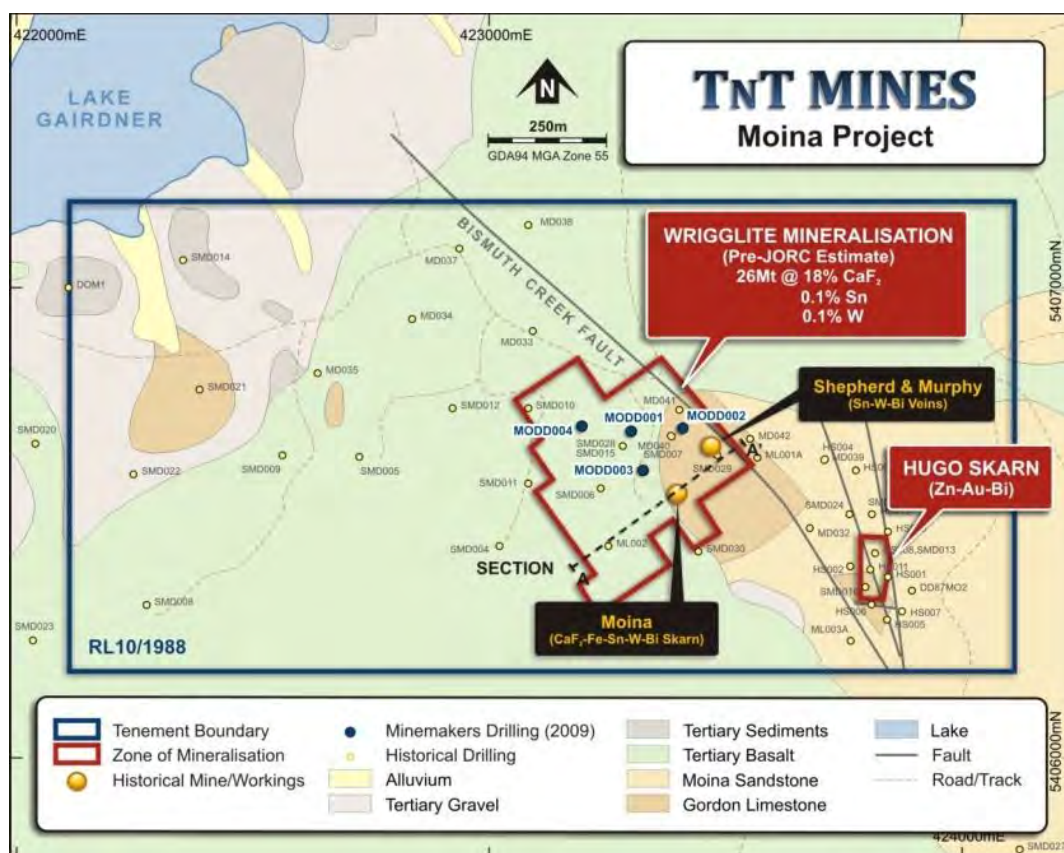
2. GEOLOGY AND MINING

2.1 Geology

Exploration work to date indicates that the Moina skarn deposit is a very significant deposit. The skarn is very fine-grained, which makes treatment of the ore to produce separate products difficult and potentially expensive. Currently, metallurgical studies are being conducted to determine a most effective method to process ore and produce saleable mineral concentrates.

Figure 2-1 shows a tenement and geology map of the Project site.

Figure 2-1 Tenement and Geology Map



2.1.1 Geological Structures

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

The main body of skarn is zoned and consists of:

A top zone: comprising a granular garnet-pyroxene-vesuvianite-fluorite skarn overlying the other units. This unit is relatively enriched in boron

The main skarn ('wrigglite'): comprising fluorite-magnetite-vesuvianite (cassiterite-scheelite-adularia) and having a characteristic fine-grained (less than 0.2 mm), rhythmic, finely layered and contorted structure

Within and near the base of the main skarn: comprising a granular, pale-green pyroxene skarn occurring as thin units (less than 5 cm) consisting of diopside-hedenbergite with very minor amounts of fluorite and garnet

A wollastonite-rich skarn: this may be present in places and can be a useful marker of more significant mineral concentrates. It was 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: comprising granular garnet-pyroxene-vesuvianite-fluorite skarn.

The skarn is essentially variable depending on local factors that controlled the metasomatism. A number of distinctly different skarn types can be found in limited quantities in other areas where metasomatic conditions varied. The two most notable are the pyrrhotite skarn and sphalerite skarn. The former consists of medium-to-fine-grained pyrrhotite, magnetite, fine-grained actinolite/chlorite and minor fluorite; the latter being 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-sulfide-amphibole alteration of the primary fluorine-tin-beryllium oxide skarn is related to the Bismuth Creek Fault. When the primary wrigglyite skarn is altered, tin is largely lost from that part of the skarn.

TNT considers that sufficient drilling has been undertaken at the Moina site to show that the skarn deposits are substantial. More drilling will be required for a undertaking feasibility study but at the present time, TNT has placed all its attention on metallurgical studies.

The mineralogy of the skarns pose problems in treating the fine-grained rock and until an economic treatment process can be established, work in finding such a treatment priority should take absolute priority; no further field work is justified at this point other than obtaining samples for metallurgical study.

3. MINING

3.1 Some Considerations

The mining method will be conventional open-pit mining, drill and blast using excavators and off-highway trucks combined with necessary support equipment.

Optimisation of the pit phasing and mining schedule before pre-stripping and mine production starts should be carried out to save costs in this area by refining material movements. This approach offers additional upside to reducing the Project's capital and operating costs.

Initial pit slope designs will be conservative, based on well-founded assumptions and appropriate engineering analysis. There could be a case to increase pit slope angles after two to three years of operations to confirm pit wall stability and this could reduce strip ratio.

Re-evaluating the pit slope angles will require ongoing geotechnical mapping and evaluation during the early phases of mining to refine the characterisation of the rock mass and revise the pit slope analysis. Such re-evaluation is a normal part of a continuous mining improvement program in any modern open-pit mining operation.

The mining options TNT can pursue are either to purchase or lease dry-hire, or contract the mining fleet. Both these options have been examined, including the size of the mining team required, which will depend on TNT's strategy and policy for Project development.

To keep the team small and the operational cost as low as possible, it is necessary for operator personnel to be multi-skilled and flexible about their roles. For example, the Mine Superintendent should be a mining engineer, the geologist will need to have mine grade-control experience, the surveyor will be required to mark all designed drill-hole positions with drill depths, the mine supervisor and a fitter/serviceman will be required to operate equipment. The Mining personnel will also need to assist contract drill and blast crew.

The mining operations should be able to mine sufficient material on a 5-day week based on 12-hour days to produce 1.6 Mtpa of ore and waste. Note, this strip ratio of 1:1 if life of mine; the early years will be lower waste:ore.

Figure 3-1 shows drill and blast equipment that will be needed to support the Project's development.

Figure 3-1 Drill and Blast Mining Equipment



Ore definition during mining will require further sampling, as well as constant supervision and investigation by the grade-control geologist. The blast hole sampling logging of the blast-hole chips and face sampling will only give rough estimates of the head grade, especially in the very early part of mining life.

Assistance from a senior geologist will be part of the initial grade-control program. The use of such a program will ensure that the ore-zone is not over-assayed, which will minimise the possibility of ore loss and maximise grade extraction. Parcels will need to be run through the mill to calibrate the grade control.

The open-pit design will require a formal assessment by a geotechnical engineer. To support such an assessment, the locations of any past workings will need to be plotted on a survey plan.

The wet climate in this part of Tasmania will result in significant water entering into the open pit and therefore, a need to manage groundwater within the pit area. Although the old underground mine has dewatered the upper parts of the pit, when the pit intersects the level of the underground workings, there may be more water to manage at that point.

3.2 Mining Operating Cost Estimate

As mentioned previously, both mining options being considered were included in the cost analysis.

With regards to manning, the Mining team may consist of a mixture of TNT personnel and contractors. The size of the Mining team, however, will depend on TNT's strategy and policy for Project development.

In regards to contract Mining cost, TNT requested that Shaw Contracting Pty Ltd (Shaw Contracting) and Mancala Pty Ltd (Mancala) in Tasmania be contacted to get a cost per tonne to mine and deliver to the run-of-mine (ROM) pad and waste dumps – assuming that the ROM pad and waste dump are close to the pit.

The number of people the contractor will use is important for estimating and providing for site facilities. TNT will also have a range of other people involved, including geologists, mining engineers, surveyors and the like.

3.2.1 Pit Physicals

The dimensions of the open pit are provided in Table 3-1.

Table 3-1 Pit Physicals

Item	Dimension	Unit of Measurement
Ore tonnes life-of-mine (LOM)	26,000,000	t
Total tonnes including waste	52,000,000	t
Strip ratio – LOM	1:1	-
Mine life	33	years

Mill throughput	800,000	tpa
Mining rate	1,600,000	tpa
Drill and blast	hard	-
Pit depth	100 – 150	m
Roster	5d, day-shift only, 12h	-
ROM and waste dump close to pit	-	m
Contractor labour numbers	-	-

3.2.2 Estimates and Budget Quotes

A comparison of the estimates and budget quotes between companies are presented in Table 3-2.

Table 3-2 Comparison of Mining Opex Estimates and Budget Quotes

Mining One Consultants	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	AU\$3.80	AU\$4.60	AU\$5.00
Manning	53	-	-
Mancala Pty Ltd	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	-	-	AU\$4.25
Manning	-	-	31
Shaw Contracting	Owner Operator	Lease – Dry Hire	Contract
Cost per tonne	-	-	AU\$12.34
Manning	-	-	10
Total Mining Cost	AU\$12.34		

Table 3-3 shows the Mining Opex estimate.

Table 3-3 Operating Cost Estimate

Breakdown of Mine Operating Costs (Est)	% of Total Operating Costs
Fixed	24
Labour	16
Tyres	8
Ground-engaging tools (GETs)	8
Parts	24
Fuel	20
Total	100

3.2.3 Mining Capital Equipment

Mining One Consultants Pty Ltd (Mining One) from Melbourne, Victoria estimated the fleet required and costs as shown in Table 3-4.

Table 3-4 Fleet Cost Estimate

Mining Fleet Item	Total Cost Estimate
Excavator	AU\$25,154,979
Trucks (40t to 90t capacity)	
Front-end loaders	
Bulldozer	
Grader	
Water cart	
Service truck	
Light vehicles	
Technical services vehicle	
Rock breaker	
Lighting plants	
Tool carrier	
Tyre handler	
Small loader with stemming bucket	
Flat tray bed truck	
Drill Rig for Blasting	AU\$3,349,916
Explosives Storage and Handling	AU\$631,000

3.3 Metallurgical Studies

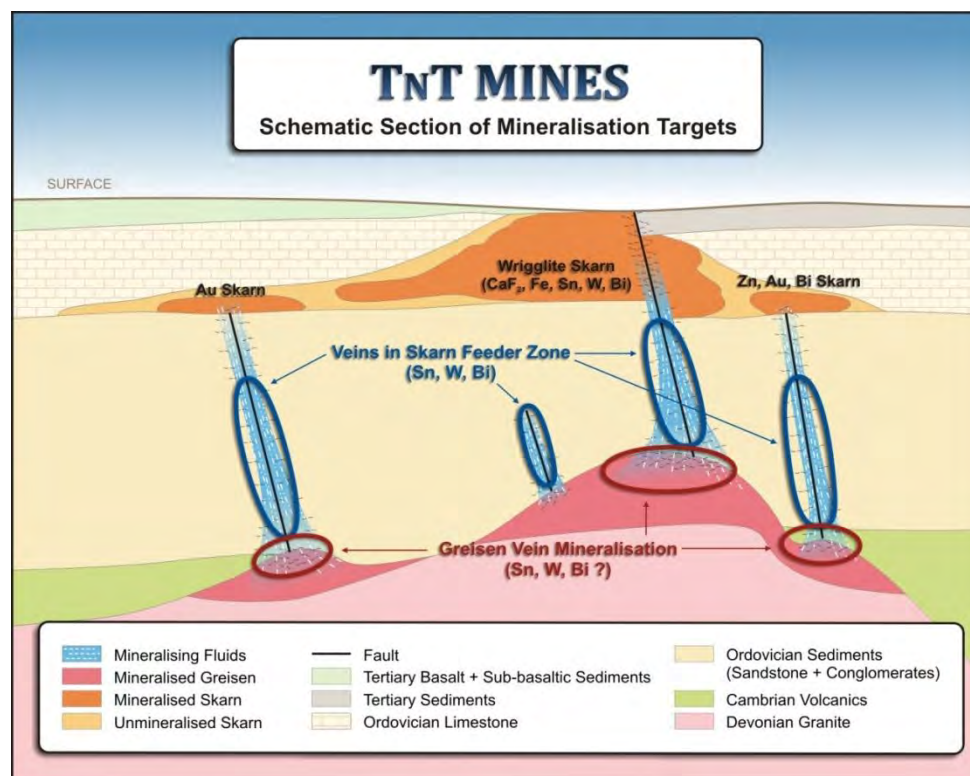
Metallurgical test work at the Moina site dates back to the late 1970s. Such work has found the skarn ore to be relatively refractory but when concentrating on fluorite, an acid-grade product was recoverable. In light of the considerable price increases in all components of the Moina skarn, Minemakers re-examined the test work and commissioned new work for determining the optimal circuit for maximising the net cash flow with respect to current metal prices, recoveries, and capital and operating costs.

While the mine in the past has been referred to as a 'fluorite' project because it is based on a very large fluorite resource, the tenement is located in a significant metallogenic province and contains several different styles of mineralisation comprising different metals and minerals.

The largest component of the mineral group in the deposit is actually magnetite. It is therefore more accurate to name the mine a 'magnetite' mine. TNT is not only going to investigate different avenues of treatment for the production of magnetite

and fluorite, but will also be looking at other mineral production opportunities that exist at Moina. These opportunities include vein mineralisation below the skarn, as illustrated in Figure 3-2.

Figure 3-2 Moina Project exploration targets beneath the Wrigglite Skarn

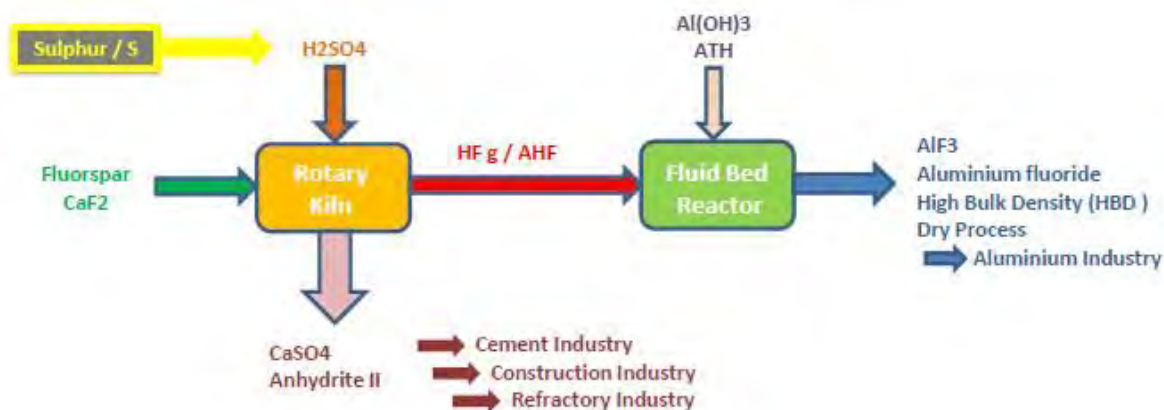


A Spanish company, Minersa undertook studies of the fluorite component of the wrigglite skarn with the aim to attempt to produce an acid-grade product. Minersa achieved a 12.5% increase in recovery for the same grade and are optimistic that following fine grinding, a marketable super-fine particle size acid-grade concentrate could be produced. Minersa was also optimistic that significant recovery of other commodity elements could be possible.

A total of 89 samples from holes SMD011 and SMD016 were crushed and composited into predominantly 5-metre intervals for sample preparation work at Burnie Research Laboratory. The sample preparation involved crushing, rod milling and sizing into three size fractions: +53 μm , -53 μm to +20 μm , and -20 μ .

All-sized samples from the composited intercepts were delivered to ALS Burnie Research Laboratory Group (Burnie Research Laboratory), where Davis Tube testing was performed on each sample. The Davis Tube testing indicated concentration into magnetic fractions in each sample tested, with the grade of magnetics increasing in iron (Fe) content with decreasing particle size, the main diluents in the coarser size fraction being silica (SiO₂) and alumina (Al₂O₃). There was a clear inverse relationship between iron-grade and iron-recovery with size.

Figure 3-3 Simple flow sheet showing process to convert fluorspar to aluminium fluoride



3.4 Exploration Potential and Development Strategy

A significant effort will be made to undertake extensive metallurgical test work to follow up on the initial metallurgical studies commenced by Minemakers.

It is clear from initial metallurgical studies that the very fine-grained mineralisation requires fine grinding and further treatment to liberate the fluorspar, tin and tungsten.

Magnetite may prove to be significant to the economics if a suitable metallurgical process is found to achieve saleable grade at high recovery. Significant funds will be used to identify a suitable metallurgical extraction process.

TNT intends to seek advice from a number of consultants with expertise in magnetite and fluorspar, as well as tungsten metallurgy and oxide flotation processes, magnetic separation and comminution, crushing pre-concentration and grinding methods.

Further, the economics of creating a number of products that include magnetite, fluorspar and tungsten concentrates will be assessed.

3.5 Process and Design Considerations

In the beneficiation of ores containing cassiterite (tin), wolframite and scheelite (tungsten), there are some general rules-of-thumb. For design guidance and in developing the process flow sheet, the following items should be particularly addressed in the metallurgical test program:

- Pre-concentrate at as-coarse-a-size-as-possible – equipment to be considered includes Heavy Media Separation, Conventional Jigging, Ore Sorting using some property of the minerals in the ore
- Provide for early removal of barren gangue or liberated mineral from size reduction circuitry, whether crushing or grinding

- Recover what can be recovered by conventional gravity methods. 'Conventional' in this sense means non-centrifugally-assisted 'g' forces – minimisation of over grinding is an important factor here.
- Provide for final concentrate dressing, taking into account any accessory minerals present in the ore which may result in penalties by the purchaser. The level of iron oxides or sulphides in the ore is an important consideration – high iron or high sulphide ores require removal, usually by magnets or flotation ahead of the gravity plant. 'High'-level iron or sulphur is a somewhat arbitrary statement but generally, feed contents over 5% iron or sulphur will interfere with gravity separation. High-level iron or sulphur occur in many mineral forms: magnetite, haematite, pyrite, pyrrhotite, arseno-pyrite, galena, sphalerite and so on.
- Regard fines recovery as a scavenger on the gravity plant and so provide future allowance for such; however, only add fines recovery circuits after the main plant is operating and when the economics can be re-assessed based on real tonnages, samples, sizing and assay information. Such information is almost impossible to accurately quantify in bench or even pilot-scale test work. A typical fines recovery system might involve, for example, scheelite flotation, but the flotation of scheelite is not easy, in that known collectors are not very selective and it is difficult to make good concentrate grades.

It can be seen by studying the process flow sheets of such existing or past mines such as Renison Tin, Cleveland Tin, King island Scheelite, Mt Carbine Wolfram, and Greenbushes tin-tantalum-niobium mine in Australia, San Rafael Tin in Peru, Huanuni and Colquiri tin mines in Bolivia, Rio Kemptville Tin in Canada, Wheal Jane, Geevor, South Crofty Tin in the United Kingdom (UK) and Neves Corvo Tin in Portugal, that the earlier mentioned general rules have been considered and applied.

The practical mineral dressing options to achieve a separation using the properties of minerals are size, specific gravity, magnetic susceptibility, conductivity, colour, floatability and solubility.

In the Moina case, the objective of producing saleable magnetite, scheelite and fluorspar concentrates presents a metallurgical challenge in that processes to recover one particular mineral can compete with processes to achieve the best result for other minerals. For example, fine grinding can be deleterious to the recovery of scheelite, and it is usual to operate the primary grinding section in closed-circuit with fine screens. This is an example of competing process requirements between magnetite and scheelite in that by ensuring liberation of the magnetite, the scheelite could be reduced to a size below a natural gravity recovery range.

Close-circuiting the ball mill with screens rather than cyclones was adopted, as hydro-cyclones provide ideal conditions for over-grinding of the scheelite, which would mis-report to the cyclone under-flow due to high specific gravity and then return to the grinding mill, only escaping the circuit when it is fine enough to exit via the cyclone overflow.

In scheelite-only plants, the ore is carefully stage-reduced, and closed circuit classification by screens ensures early removal of liberated scheelite from the grinding circuit, prior to delivery to downstream processes. Process streams from gravity separation with un-liberated scheelite are recycled as middlings to a regrind mill and then returned to the beneficiation processes to minimise over-grinding.

It is important at Moina to primary grind to reduce and liberate the magnetite mineral only to the point where it can be separated magnetically at an acceptable recovery into a bulk concentrate and then to saleable grade by regrinding and cleaning.

Of the aforementioned practical beneficiation options, use of certain combinations of specific gravity, magnets, hydraulic classifiers, and flotation, need to be employed to achieve saleable grades of magnetite, scheelite and fluorspar concentrates.

3.6 Process Logic and Design Philosophy

Based on sample testing, Moina ores may be described as complex and of high hardness.

Earlier test work indicated that pre-concentration was a possibility with Moina ore at a size that indicates pre-concentration using cyclone heavy media or conventional jigs between the crushing and grinding circuits could be incorporated into the process flow sheet.

To achieve a reasonable recovery of the valuable minerals, a process involving a 3-stage crushing to -12 mm followed by heavy media separation or conventional jigging, and rod-and-ball mill grinding to reduce the pre-concentrate to a size for magnetic separation and liberation of the scheelite from the rest of the gangue.

To minimise over-grinding of scheelite, the rod mill should be in open circuit. The ball mill needs to be closed circuit with fine screens to ensure early removal of liberated scheelite from the grinding circuit. Magnetic separation of the magnetite and concentration of the scheelite by a combination of staged cyclone classification, gravity separation (spirals and tables) and scheelite flotation, with inter-stage regrinding, could result in a saleable grade concentrate.

3.7 Process Plant

Based on the characterisation work and requirements mentioned in Section 3.6, the proposed process plant would comprise 3-stage crushing of ROM ore to a -12-millimetre particle size.

Heavy media plant based on cyclone separation and ferro-silicon media will be used to remove a barren siliceous fraction prior to grinding from a -12-millimetre to +1-millimetre fraction.

Rod and ball mill grinding to a 106-micron particle size which liberates the magnetite and scheelite mineral from the gangue to a size where a magnetic concentrate can be made and gravity separation for some scheelite is possible.

The process will provide for magnetic separation to first remove magnetite, which will interfere with downstream gravity and flotation processes for scheelite. Magnetic cleaning and re-cleaning with inter-stage regrind will be necessary to minimise fluorspar loss.

Hydraulic classification to separate the magnetic tails into feed for natural gravity devices (i.e. spiral and tables) is required, with fines reporting to scheelite flotation and enhanced gravity devices, plus an ultra-slimes tail (<6 µm), which is discarded as being too fine for gravity or scheelite flotation beneficiation methods.

Dewatering of the mineral concentrates in thickeners followed by filtration to within transportable moisture limits and storage in a concentrate shed will be necessary.

Tailings from the process plant will be thickened and delivered to a tailings storage facility (TSF) or filtered for dry stacking.

4. ASSUMPTIONS

For the purpose of this preliminary cost study, the following assumptions were made:

- The treatment plant will be located on a level greenfield site
- Access to the site to be via roads suitable for the use of heavy-lifting equipment
- The treatment rate is 800,000 tpa.

The design was based on practice at a number of other operations, therefore equipment sizing and selection is preliminary only.

The treatment plant will operate 24-hours-a-day for 365-days-a-year, with 91.5% availability.

Workforce numbers were determined to cover for four operating crews working 12-hour shifts on a suitable roster, with an appropriate number of management, supervisory and technical staff included according to a suggested site organisation.

5. EXCLUSIONS

To develop the capital and operating cost estimates, the following exclusions were made:

- Metallurgical test work and further process investigations
- Geotechnical test work. The cost of process equipment foundation was based on suitable ground conditions available for large equipment

- Allowance for the variation in head grade, ore type and hardness
- Mining, geology, administration, environment, occupational health and safety costs
- Royalties
- Personnel accommodation costs
- Refinery and treatment charges
- Marketing and warehousing
- **First Fill:** this is an allowance to acquire necessary supplies for the initial operation (e.g. grinding media for the rod and ball mill, reagents for flotation and flocculants for thickeners). Typical allowance of 4.5% of Capex
- **Start-Up and Commissioning:** this is an allowance for the retention of contractors during start up and commissioning. In addition, an allocation of funds is usually made for process and discipline commissioning engineers and vendor representatives for this period
- Any scope changes
- Owner's costs
- Interest during construction, Project financing costs and financing fees
- Import duties, corporation taxes and royalties
- Permitting costs
- Schedule delays
- Sunk costs escalation
- Preparation of a Development Proposal and Environmental Management Plan, as well as any other statutory legal requirements.

6. BATTERY LIMITS

The Project's battery limits included the following:

- ROM pad
- Tailings disposal dam or filtration
- Concentrate storage at the process plant and delivery to port
- Power supply to on-site substation
- Process water from the raw water dam.

7. DATA AND INFORMATION

The basis of the capital and operating cost estimate encompasses the following:

- Basic design criteria
- Current labour and power costs
- Operating consumables from test work and other similar operations

- Maintenance as a percentage of the installed equipment
- Process description from a conceptual flow sheet based on earlier test work and when available, results of current work at the Burnie Research Laboratory.

The major Mechanical Equipment List comprises:

- A 3-stage crushing plant including screens and conveyors (The design assumes a top size of 600 mm for the ROM ore and this is important, as sizing affects the primary jaw crusher selection. A rock breaker will be installed to deal with any oversize)
- Fine ore storage
- Cyclone heavy media plant
- Rod mill
- Ball mill
- Fine screens
- Hydro-cyclones/slurry pumps
- Magnetic separation
- Scheelite flotation
- Regrind mills including a vertical tower mill
- Spirals
- Shaking tables
- Fluorspar flotation
- Concentrate thickeners
- Concentrate filters
- Tailings thickener
- Tailings filter.

8. CONCEPTUAL PROCESS FLOW SHEET

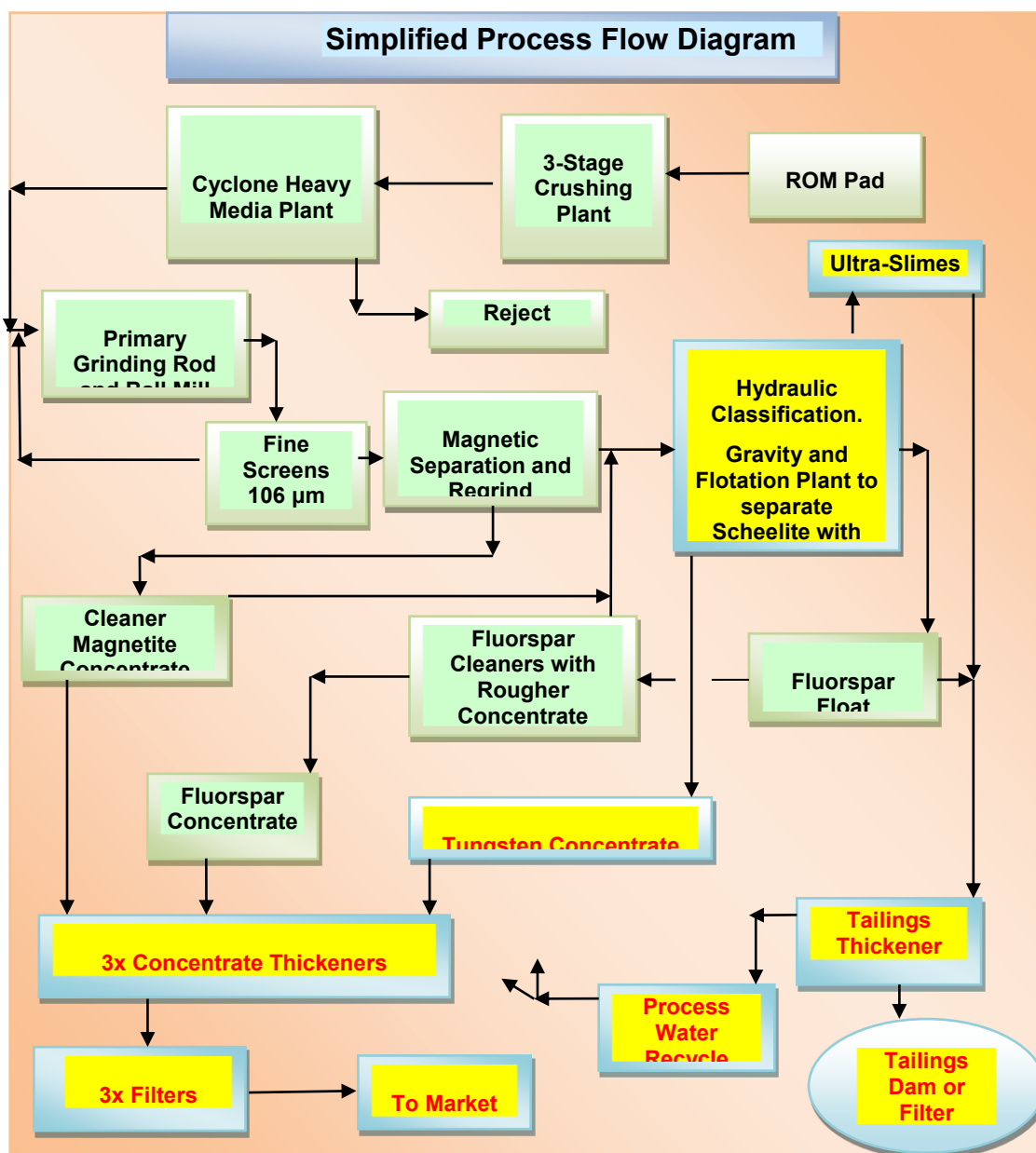
The conceptual process flow sheet in Figure 8-1 shows:

- Three-stage crushing to -12 mm, followed by:
- Cyclone heavy media plant to pre-concentrate the ore by removing hard barren siliceous material from a -12-millimetre to +1-millimetre fraction ahead of primary grinding
- Primary rod and ball mill grinding with fine screens to liberate the valuable minerals from the gangue, as well as magnetic separation to remove magnetite, with inter-stage regrind and magnetic cleaning to liberate fluorspar and make saleable magnetite concentrates
- Hydraulic classification to prepare feed to gravity and flotation circuits and remove ultra-slimes, as well as concentration of the scheelite by a

combination of gravity plant and flotation to produce saleable scheelite concentrates

- Flotation of the scheelite flotation tail to recover fluorspar incorporating fine grinding mills such as vertical tower mills and/or isa mills to reduce rougher fluorspar concentrate to 17 µm ahead of multi-stage cleaning.

Figure 8-1 Simplified Process Flow Diagram



9. DESIGN CRITERIA

The design criteria form the basis of the concentrator processing facilities and the required site services. Together with the process flow diagrams, this data allows definition of the mass balance, design and specification of equipment, development of schedules for operating requirement, and other items such as power, water and reagents in future studies. The data and criteria are preliminary and are subject to change as the Project progresses to development.

9.1 Ore Characteristics

Design criteria was based on the following ore characteristics in Table 9-1.

Table 9-1 Design Criteria Ore Characteristics

Design Criteria	Unit of Measurement	Characteristic
Specific Gravity	t/m ³	3.3
ROM moisture	% H ₂ O	5.0
Bulk density broken rock	t/m ³	1.8
Compressive strength	MPa	Assumed 200
Work index	kWh/t	Assumed 20.0
Abrasion index		n/a
ROM top size	mm	600 mm

9.2 Crushing Plant Operations (3-Stage Crushing)

The design criteria for the crushing plant operations are shown in Table 9-2.

Table 9-2 Design Criteria Crushing Plant Operations

Design Criteria	Unit of Measurement	Characteristic
Annual throughput target	tpa	800,000
Hours per shift	-	12
Shifts per day	-	1
Days per week	-	7
Weeks per year	-	52
Plant utilisation	%	40
Nominal throughput	tpa	250
ROM top size	mm	600
Crusher product size	P ₈₀ mm	12
ROM ore storage	t	10,000
Crushed ore storage	t	2000

9.3 Cyclone Heavy Media (HMS) Operations

The design criteria for cyclone heavy-media (HMS) operations is shown in Table 9-3.

Table 9-3 Design Criteria Cyclone Heavy Media Operations

Design Criteria	Unit of Measurement	Characteristic
Annual throughput target	tpa	800,000
Operations	Hours per shift	12
Operations	Shifts per day	2
Operations	Days per week	7
Operations	Weeks per year	52
Utilisation	%	91.5%
Treatment rate	tpd	2400
Treatment rate	tph	100
HMS Floats Reject	% Wt	18
HMS Floats Reject	tpa	144,000
HMS Sinks + Fines	tpa	656,000
HMS Valuable Mineral recovery	%	97

9.4 Concentrator Operations

The design criteria for the concentrator operations is shown in Table 9-4.

Table 9-4 Design Criteria Concentrator Operations

Design Criteria	Unit of Measurement	Characteristic
Annual throughput target	tpa	656,000
Operations	Hours per shift	12
Operations	Shifts per day	2
Operations	Days per week	7
Operations	Weeks per year	52
Utilisation	%	91.5%
Treatment rate	tpd	1800
Treatment rate	tph	75

9.5 Metallurgical Parameters

The metallurgical parameters for magnetite, fluor spar and tungsten are shown in Table 9-5, Table 9-6 and Table 9-7 respectively.

Table 9-5 Magnetite Metallurgical Parameters

Parameters	Chemical Designation	Characteristic
Ore grade	% Fe ₃ O ₄	18.5
HMS floats	% Fe ₃ O ₄	3.8
HMS sinks and fines	% Fe ₃ O ₄	21.9
Mill feed	tpa	656,000
Mill feed grade	% Fe ₃ O ₄	21.9
Overall Fe ₃ O ₄ recovery	%	70.0
Final concentrate grade	% Fe	67
Final concentrate	Dry tpa	150,100
Concentrate moisture	% H ₂ O	10
Final Magnetite Concentrate	Wet tpa	165,100

Table 9-6 Fluorspar Metallurgical Parameters

Parameters	Chemical Designation	Characteristic
Ore grade	% CaF ₂	15.4
HMS floats	% CaF ₂	2.57
HMS sinks and fines	% CaF ₂	18.2
Mill feed	tpa	656,000
Mill feed grade	% CaF ₂	18.2
Overall CaF ₂ recovery	%	68.0
Final Concentrate CaF ₂ grade	% CaF ₂	94
CaF ₂ concentrate	Dry tpa	86,369
CaF ₂ concentrate moisture	% H ₂ O	10
Fluorspar Concentrate	Wet tpa	95,006

Table 9-7 Tungsten Metallurgical Parameters

Parameters	Chemical Designation	Characteristic
Ore grade	% WO ₃	0.10
HMS floats	% WO ₃	0.024
HMS sinks and fines	% WO ₃	0.12
Mill feed	tpa	656,000
Mill feed grade	% WO ₃	0.12
Overall WO ₃ recovery	%	58
Final Concentrate WO ₃ grade	% WO ₃	65
Total WO ₃ in concentrate	tpa	457
Dry WO ₃ concentrate	tpa	703
WO ₃ concentrate moisture	% H ₂ O	10
Scheelite Concentrate	tpa	773

10. PROCESS PLANT CAPITAL COST ESTIMATE

10.1 Estimating Methodology

The method used in estimating the Capital cost of the plant at this level of accuracy was to obtain single vendor budget quotes from suppliers for the major equipment items, as shown in the conceptual process flow sheet (Figure 8-1).

Based on the result of vendor enquiries, engineering factors were applied to the major equipment costs to give an estimated Capital cost for a complete plant, including engineering design, procurement, construction and Project management.

Capital estimates for feasibility studies must be based on new equipment unless appropriate used equipment is already owned, or optioned. At Project implementation stage, any method to reduce costs including use of used equipment may then be employed.

Factoring at this stage of a Project is a standard method in Study work used by project engineers, and is based on inhouse data accumulated over many years of experience in designing and building similar types of process plant. Process project engineers may have slightly different factors, and so therefore, the factors for each area may vary.

For this Study, the major equipment estimates were based on price information obtained from requests to suppliers to give budget quotes on price and delivery.

Tasmanian-based Electrical Consultant Rob Hill & Associates supplied Capital costing for electrical supply and power cost based on estimated usage.

The process plant buildings will depend on the climate of the Project site and it is assumed that much of the equipment needs will be under substantial cover, which accounts for the buildings allowance in the following estimates (see Table 10-1, Table 10-2 and Table 10-3). Within the next phase of investigation this estimate should be subjected to value engineering assessment.

The EPCM costs are based on data from a number of projects similar in scope and scale.

Further investigation into the infrastructure costs such as a water supply pipeline should be undertaken during the next study phase. An estimate for an electrical substation and Infrastructure costs has been obtained within this study phase.

Table 10-1 Major Equipment Estimate

Items A Major Equipment	Capital Cost Estimate (AU\$M)
3-stage crushing plant mobile	4.0
Heavy media plant – cyclone-type	1.0
Rod mill	2.0
Primary ball mill	2.25
Derrick Fine Screening Unit (106 µm)	0.34
Low-intensity magnetic drum separators rougher and cleaners	0.72
Magnetic concentrate regrind tower mill	2.5
Cyclone classifiers	0.3
Spirals and tables	0.6
Gravity regrind ball mill	1.5
Scheelite flotation banks (4-stage)	0.5
Fluorspar flotation banks (roughing)	0.1
Fluorspar rougher concentrate regrind tower mill	1.6
Fluorspar cleaning flotation cells	0.4
3x concentrate thickeners	0.66
2x concentrate pressure filters	1.5
1x concentrate disc filter (magnetite)	0.5
On-stream analyser	0.5
Tailings thickener	0.225
Tailings filter	1.6
Power supply – main substation	0.75
Miscellaneous	0.5
Items A Subtotal	23.55

Table 10-2 Other Items Estimate

Items B Other Items by Factoring	A x factor	AU\$M
Instrumentation/samplers/reagent feeding	$A \times 0.08$	1.88
Plant services, compressors, water pumps (etc.)	$A \times 0.08$	1.88
Concentrate handling system	$A \times 0.08$	1.88
Main Process building	$A \times 0.25$	5.89
Auxiliary buildings	$A \times 0.11$	2.59
Electrical installation	$A \times 0.19$	4.47
Piping installation	$A \times 0.14$	3.30
Equipment installation	$A \times 0.21$	4.94
Items B Subtotal		26.84

Table 10-3 Installed Cost Estimate

Items C Total Installed Cost	Formula	AU\$M
Equipment and installation	$A + B$	50.39
EPCM	$A + B \times 0.18$	9.07
Contingency		14.86
Subtotal		59.46
Contingency	25%	14.86
Subtotal		74.32
Accuracy	30%	22.30
Items C Subtotal		96.62

The equipment estimate summarised in Table 10-4 was provided by Mining One to TNT for owner-operator mining.

Table 10-4 Equipment Estimate

Equipment Item	Cost Estimate (AU\$)
Mining fleet	25,154,979
Drill rig for blasting	3,349,916
Explosives storage and handling	631,000
Plant operating equipment	500,000
Total	29,635,895

The total Capex costs are summarised in Table 10-5.

Table 10-5 Total Capex Costs

Description	Capital Cost (AU\$M)
Mining One (as per Figure 10-4)	~30
Installed equipment cost	~97
Civil and miscellaneous	~10
Total	■ ~137

Note: if the Mining is contracted out, there is an AU\$30M Capex saving.

10.2 Process Equipment Inclusions

The process equipment estimate includes crushing, heavy media plant, grinding, fine screening, magnetic separation, cyclone classification, gravity, scheelite flotation, fluorspar flotation, fine grinding, thickening and filtration of concentrates, and thickening of plant tails.

Major process equipment costing was from recent EPCM vendor quotations for similar projects in scope and scale, and based on preliminary design parameters and test data.

The EPCM costs do not cover geotechnical testing, surveying, material testing, warehousing, the contractor's direct construction supervision or other Owner's costs.

Table 10-6 shows the major equipment selections used in this Study.

Table 10-6 Major Equipment Selections

Major Equipment	Sizing
3-stage crushing plant	1x Jaw 1.25W x 1.0L 2 x Cones HP 400 Model
Heavy media plant – cyclone-type	Bateman Supply – see Como website
Rod mill	2.9 Diameter x 5.7L
Primary ball mill	3.6 Diameter x 5.7L
Derrick Fine Screening Unit	Stack sizer
3x low-intensity magnetic drum separators (1,000 gauss) roughing and cleaning	3 Eriez Units 0.9 Diameter x 3.0W Single Drum 0.9 Diameter x 3.0W Double Drum 0.9 Diameter x 3.0W Double Drum
Magnetic concentrate regrind tower mill	VTM 1,000
Spirals	6x Rougher 5-turn triple-start (18 spirals) 6X Cleaner 5-turn triple-start (18 spirals)

Major Equipment	Sizing
Shaking tables	10 Holman tables
Gravity regrind ball mill	Ball Mill 3.05 Diameter X 2.4m
Scheelite flotation banks (3-stage) Roughing, cleaning and re-cleaning	6x OK16 6x OK8 5x OK5
Fluorspar flotation banks (roughing)	6x OK16
Fluorspar rougher concentrate tower mill	VTM 600
Fluorspar cleaning flotation cells	6x OK8 5x OK5
3x concentrate high rate thickeners	Scheelite 1.0m Magnetite 4.0m Fluorspar 7.0m
1x concentrate pressure filter (fluorspar)	64 m ² PF
1 x concentrate disc filter (magnetite)	32 m ² ceramic disc filter
On-stream analyser	Courier 6
Tailings thickener high rate	7.0m
Tailings filter	75 m ² ceramic disc filter

10.3 ROM Stockpile

Jacobs considered that a reasonable area from an operational point of view for the ROM stockpile would be 100m x 100m x 15m in height. This size would hold 100,000t, which is about one and a half month's production.

A ROM stockpile of 150m x 100m x 17 m in height would hold 307,183t or 170,657 m³.

However, if TNT decides to mine for 4 months of the year, than a ROM stockpile of 502,000t would be required with an area of 150m x 150 m x 17m in height.

11. PROCESS PLANT OPERATING COST ESTIMATE

11.1 Exclusions

The Study's preliminary Opex estimates exclude:

- Marketing, refinery and treatment charges for concentrates
- Effluent treatment
- Royalties
- Interest charges and tax
- Mining, geology, administration, environment, occupational health and safety costs

- Estimated Opex composition including:
 - Consumables
 - Heavy media ferro-silicon
 - Grinding mill liners
 - Grinding media
 - Flotation reagents
 - Filter cloths
 - Labour
 - Power (installed estimated at 5 MW)
 - Maintenance.

11.2 Labour

The list provided in Table 11-1 is an estimate of the management and labour costs required to operate the proposed concentrator, including wages, salary and on-costs which were based on current rates in Tasmania.

Table 11-1 Labour Estimate

Number	Position	Salary	On-Costs 40%	Annual Cost (AU\$)
1	Mill Manager	150,000	210,000	210,000
1	Planner/Met Clerk	60,000	84,000	84,000
1	Mill Foreman	100,000	140,000	140,000
4	Shift Manager	90,000	126,000	504,000
24	Shift Operators	80,000	112,000	2688,000
1	Maintenance Supt	120,000	168,000	168,000
1	Electrical Supt	120,000	168,000	168,000
2	Elect/Instrument Tradesmen	80,000	112,000	224,000
4	Mechanical Tradesmen	80,000	112,000	448,000
1	Chemist	80,000	112,000	112,000
2	Assay Technicians	60,000	84,000	168,000
2	Met. Technicians	60,000	84,000	168,000
44	Total			5,082,000
	Cost per Tonne (ROM)			6.35

11.3 Consumables

The reagent and grinding media costs were estimated to be AU\$5.90/t based on similar operations.

Table 11-2 Consumables Estimate

Item	Tonnes per Annum (tpa)	Cost per Tonne (AU\$/t)	Total Cost (AU\$)
Ferro-silicon	300	1,750	525,000
Fluorspar float collector	120	1,600	192,000
Sodium silicate	200	2,800	560,000
Frother	36	3,375	121,500
Scheelite float reagents	80	1,600	128,000
Lime	1,600	275	440,640
Flocculants	25	3,400	85,000
Grinding media	800	1,800	1,440,000
Grinding screen panels	-	-	180,000
Crusher liners	-	-	350,000
Crusher screens	-	-	250,000
Ball mill liners	-	-	250,000
Cyclone parts	-	-	100,000
Filter cloths	-	-	100,000
Total			4,722,140
Cost per Tonne (ROM)			5.90

11.4 Power

Power demand was based on an estimate of the installed equipment, as shown in Table 11-2. The estimate of the consumed power was made to determine the annual power costs based on equipment operating hours. Unit cost of power of 6 cents per kWh is typical for this region.

Rob Hill & Associates in Tasmania is the adviser on this aspect.

Table 11-3 Power Estimate

Item	MW	Cost AU\$ per kWh	Total Cost (AU\$/t)
Installed power	5.00		
Total consumption = Load Factor of 0.70 and plant running time 91.5% x installed power	3.5	0.06	1,683,234
Cost per Tonne (ROM)			2.10

11.5 Maintenance

The maintenance cost estimate is shown in Table 11-4.

Table 11-4 Maintenance Estimate

Item	Basis	Total Cost (AU\$)
Maintenance	3% of installed equipment	1,600,000
Cost per tonne (ROM)		2.0

Note: the nominal price of water from Lake Gairdner is AU\$55/ML.

11.6 Operating Cost Summary

Table 11-5 shows the Operating cost estimate.

Table 11-5 Operating Cost Estimate

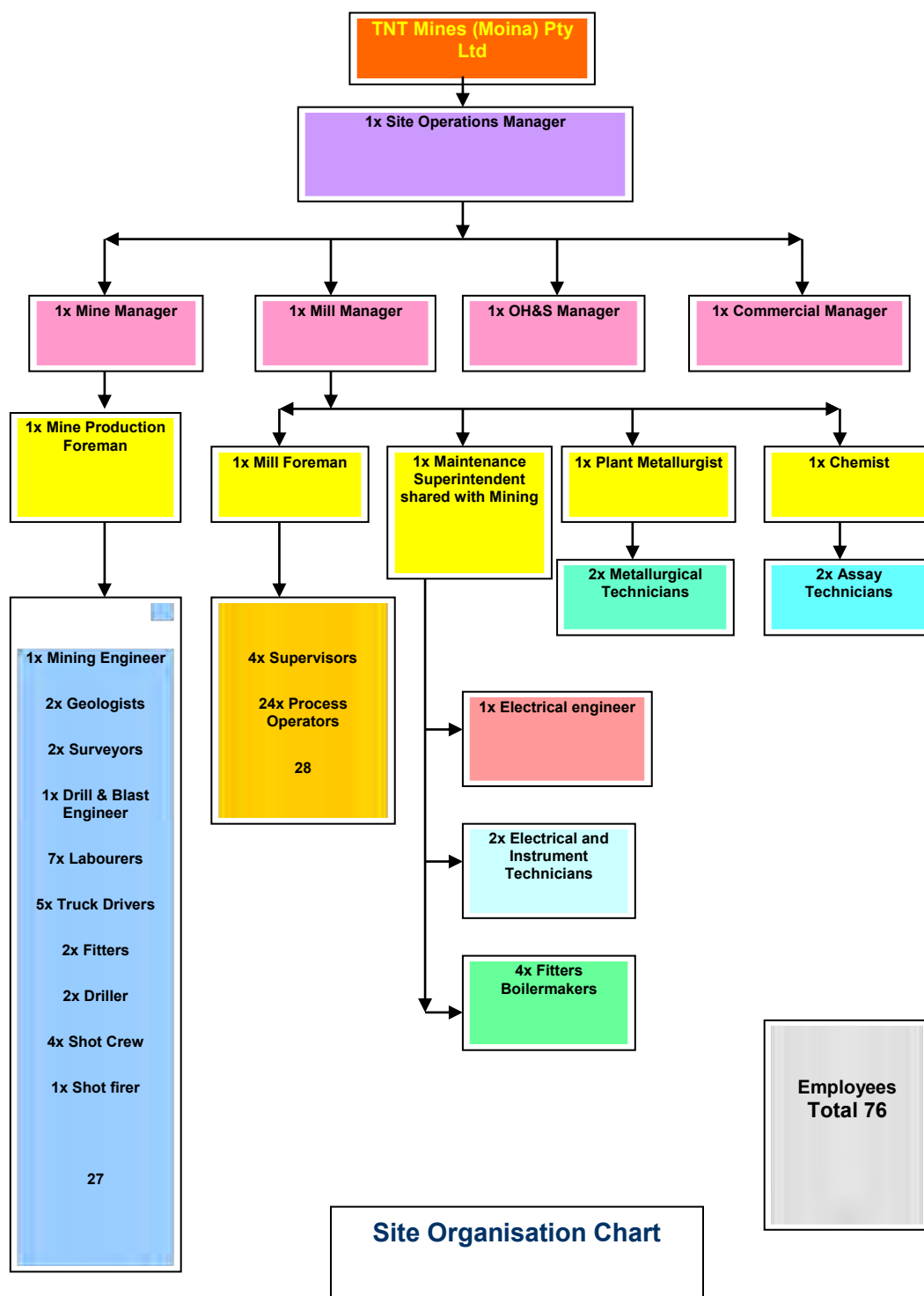
Area	Unit Costs (AU\$/t ROM)	Total Costs per Annum (AU\$)
Labour	6.35	5,080,000
Consumables	5.90	4,720,000
Power	2.01	1,608,000
Maintenance	2.0	1,600,000
Total Costs	16.26	13,008,000

12. PROPOSED SITE ORGANISATION CHART

Site Manning depends on TNT's strategy and policy for the Project's development.

A proposed Project site organisation chart is provided in Figure 21-1.

Figure 12-1 Site Organisation Chart



13. MINING COSTS**13.1 Results from Shaw Consulting Enquiry**

For this Study's purposes, the Shaw Consulting budget numbers were used.

Shaw Consulting's cost of AU\$12.34/t was considered high when compared to other contractors and should be investigated in further studies.

In addition, Shaw Consulting provided Jacobs with the following estimates:

Stripping rate: AU\$12.50 per m³ solid (no drilling and no blasting)

Clearing: AU\$4,500 per ha.

Transport to Burnie on a 130-kilometre distance basis was estimated to be AU\$25/t, plus a load cost of AU\$2.00/t (see Appendix A).

14. ANALYSIS OF METHODS OF OPERATION

An analysis of different Mining approaches resulted in a comparison of costs between the three operating methods. Jacobs assumed that an additional mine-site civil preparation cost of AU\$6.045 was applicable to each method.

Logistics, transport of concentrate and port costs were also considered common between all the methods.

Method 1: Owner Mined and Concentrated

Capex estimate: AU\$125.2M

Opex estimate: AU\$16.26/t

Method 2: Contract Mining

Capex estimate: AU\$96.7M

OpEx estimate: AU\$12.34/t

Method 3: Dry Lease of Mining Equipment

Capex estimate: AU\$96.7M

Opex estimate: AU\$4.60/t + AU\$7.42/t labour = AU\$12.02/t

14.1 Conclusions

There is seen to be very little difference between Method 2 (contract mining) and Method 3 (dry lease of mining equipment). Both these methods require less capital and produce a lower cost per tonne than Method 1. All operating costs, however, require logistics costs and minor miscellaneous costs to be added during the next phase.

15. DESCRIPTION OF FACILITIES

The process operating philosophy for Project Operations is 24-hours-per-day, 7-days-per-week and 52-weeks-per-year.

The process plant will be designed to treat 800,000 tpa of ore through a 3-stage crushing circuit that prepares feed for pre-concentration by heavy media separation and primary grinding mills.

Fine screens will be used for classification in the grinding circuit to minimise over-grinding of the scheelite mineral; the aperture of the screens will be specified to give a grinding product with a P_{80} of 106 μm , which has been indicated as the liberation size for scheelite.

The circulating load in the grinding circuit will be the feed to the fine screens; this circulating load stream pipeline will be fitted with mass-flow metering gauges and used as part of an automatic control system to ensure that the treatment rate is maintained at an optimum level when rod-mill-feed particle size or ore hardness changes.

The magnetic circuit will be configured as a rougher, with ball mill regrinding of the concentrate, which will then be delivered to cleaning.

Five cleaning stages will be required to produce saleable fluorspar concentrates. The concentrates will be sent to thickeners ahead of a filtration plant. The gravity-flotation tailings will then be pumped to the fluorspar flotation system.

The use of automatic pulp-level control and an on-stream multi-assay unit will be incorporated into the flotation circuit to optimise concentrate grades and mineral recoveries.

The final plant tailings will be thickened and pumped to a filtration plant for dry stacking or alternatively, to a tailings impoundment dam.

15.1 Instrumentation and Control Including On-Stream Analysis

Plant instrumentation and control to provide the best system for this Project has been considered and includes:

- Strategic installation of weightometers, metal detectors, magnets and bin-level indicators within the crushing circuit
- Maintaining optimum treatment rate where the grinding circuit will have a number of controls, with the circulating load being a key set point. Instrumentation will include flow and pulp-density measure gauges, high and low-level sensors on pump hoppers, variable speed motors on pumps and feed conveyors, controlled water addition through actuated flow valves, and mill power measurement.
- Flotation cells regulated by proprietary pulp-level controllers, with manual air-rate adjustment

- Speed of slurry pumps regulated by a variable speed drives on the basis of feed hopper levels
- A multi-stream, on-line analyser (OSA) with its selection based on using pressure samplers to feed the process streams for element measurement in the OSA system; with gravity return to the relevant circuit. Using this method, small sample pumps, which often require high maintenance requirements, will be avoided.

15.2 Data and Operating Information

The basis of the capital and operating cost estimate encompasses the following:

- Basic design criteria including tpa, feed grade, concentrate grade and concentrate recovery
- Process description from the conceptual flow sheet
- A major Mechanical Equipment List comprising a 3-stage crushing plant, cyclone heavy media plant, a Derrick screen classifier, magnetic separators, flotation banks, regrind mills, 3x concentrate thickeners and 3x concentrate filters, all sized to treat 800,000 tpa.

16. MARKETING AND LOGISTICS

16.1 Marketing

The magnetite market is currently exhibiting an increase in demand and price, but final receipt price will be determined by the concentrate size, grade and contaminants, which will be demonstrated in Davis Tube and grinding test work commissioned by TNT.

World production of fluorspar reached 5.5 Mt in 2008, nearing a peak of fluorspar production in 1989 before the adverse impact of the Montreal protocol regarding the use of fluorocarbons had worked its way through to the supply side.

In the years from 1989 to 1994, production declined to 3.5 Mt, yet in the following 8 years, production recovered slowly and then increased by almost 5% per annum from 2003, to reach 5.45 Mt in 2007. Most of this growth came from increased output of acid spar in China and Mexico.

Jacobs is not aware of any product off-take agreement that are yet in place. However, Jacobs is aware that Masan Resources Corporation's Nui Phao fluorspar project in Vietnam, currently engineered by Jacobs, has made arrangements for a local agent to buy the products and on-sell, possibly on the spot market. None of these details are known at this point by Jacobs.

Prior to firming up an agreement with a local agent, Masan was also negotiating with a company in India for a product off-take.

During Jacobs' inquiries, it made contact with Mr Rama Iyer, a Director of Gujarat Fluorochemicals Ltd, a publically-listed company in India and the largest consumer of fluorspar in India. Mr Iyer introduced Mr VK Soni, President of Projects at Gujarat Fluorochemicals Ltd, who supplied the details of the fluorspar market in India (see Annexure A).

India consumes approximately 145,000 tpa of acid-grade fluorspar. There appears to be some indications from Jacobs' contacts that there may be some interest in India for fluorspar from Australia. This potential market should be further investigated.

If Scheelite containing 65% WO₃ with low contaminant levels is recovered, there will be many outlets for sale in the United States of America (USA), Europe and China.

16.2 Logistics

Shaw Consulting has provided load costs at the mine plus road transport costs at AU\$27.00 per tonne from site to Burnie.

There is an additional cost of kibble manufacture.

In addition, Toll ANL has quoted AU\$1,051.88 per container from Burnie to Melbourne load-on-ship. Transport costs will be an indication of costs plus a possible AU\$12.50 charge per container for fuel after June 2012.

17. PROJECT SENSITIVITIES

TNT will carry out a Preliminary Economic Analysis (PEA) using data from this Study in a financial model and use sensitivity analysis to obtain an indication of the factors that influence the profitability of the Project.

These factors are likely to include Viz, metal prices, ore grade, metallurgical recovery, Capital cost and Operating cost to provide a measure of robustness of the Project to changes in key financial parameters. These costs should be further investigated and benchmarked against other concentrate despatch.

18. RISK ASSESSMENT

Table 18-1 presents a tabular summary of the findings and conclusions relative to the completeness of the Project's development plans based on:

- Work completed to date
- The potential impact of each information component provided
- The potential impact of each component.

This table is intended to provide insight into the risk associated with the Project. The format is intentionally brief to provide a quick appreciation of the Project's status. Definitions of the terms used in this table are as follows:

Component: identifies the information topic reviewed.

Completeness: expresses an opinion of the coverage of the topic to date.

High Completeness: indicates that the information provided exceeds the norm of similar projects

Medium Completeness: indicates that the information provided and is in-line with the norm of similar projects

Low Completeness: indicates that the data provided is somewhat less than the norm of similar projects. This level, combined with high-impact potential, suggests that additional analysis and engineering are required. This level when combined with low-impact potential suggests an acceptability for valuation but may require some additional information for adequate reporting

Impact Potential: identifies the degree of sensitivity re-elected in Project economics by the category. This allows rapid identification of areas that require substantial scrutiny versus those areas less likely to affect the overall Project performance.

High Impact: categories such as reserves and recovery have a high impact on cash flow. Relatively small variations in these areas could significantly alter the return on investment

Medium Impact: categories such as tailings, processing and power generation can undergo minor to possibly medium variations without significantly affecting the overall Project

Low Impact: categories such as the majority of the infrastructure can undergo medium to substantial variation without significantly affecting the overall Project

This Risk Assessment: combines Completeness and Impact Potential to subjectively assess the level of risk:

Low risk indicates sufficient work has been done for a particular component.

Medium risk suggests that additional work is required to adequately define analyse or engineer an item to ensure that unanticipated problems which could affect the Project do not occur as the Project proceeds.

High risk was applied if additional work is required where presently unknown aspects have a high probability of negatively affecting the overall Project (e.g. High Impact coupled with Low Completeness).

Table 18-1 Risk Assessment

Component	Completeness	Impact potential	Risk Assessment	Remarks
Geology/geologic setting	Medium	Low	Low	The Geology is sufficiently understood on a regional basis
Geotech – hydrology	Low	Medium	Medium	More work is needed on these aspects
Exploration data	Medium	Medium	Low	Data is considered sufficient for a scoping study
Resource estimate	Medium	High	Medium	Resource estimates were developed with accepted modelling tools. Basic checks should be made to verify the results
Reserves	Medium	High	Medium	Resources have been classified. Mine plans and cash flows should be developed to support the classification
Exploration potential	Medium	Low	Low	Existing deposits are fairly well defined
Mine Plan	High	Medium	Low	Mine plans and Mine design should be prepared for the pre-feasibility study. Information is adequate for this stage
Production Schedule	High	Medium	Low	A Production Schedule will be in the PEA
Mine equipment	Medium	Medium	Medium	The equipment specified appears to provide the means to mine the required tonnage
Process recoveries	Medium	Medium	Medium	Sufficient tests will need to be made to say that mineral grades and recoveries are reasonable
Power supply	Medium	Low	Medium	Option study in favour of connection to the power grid
General infrastructure and personnel services	Medium	Low	Low	Mine site infrastructure is partly in place. Good road and port access
Mine Operations	Medium	High	Medium	Mine operations are appropriate
Mine process and infrastructure Capital cost estimate	Medium	Medium	Medium	Capital costs appear reasonable and appropriate
Mine, process, G&A and Operating cost estimates	Medium	Medium	Medium	All categories of Operating costs appear reasonable
Water supply	Low	Medium	Low	Water supply source and system has been identified
Environmental management	Medium	Medium	Medium	Environmental Management Plan will be developed in pre-feasibility
Rehabilitation	Medium	Low	Low	Mine closure costs have been estimated
Permitting	Medium	High	Medium	Permitting in pre-feasibility stage
Socio-economic community relations	Medium	Low	Medium	Community Relations need support and encouragement from the local population

Appendix A - Mining Quotes, Cost Estimates and Background Information

APPENDIX A

MINING COSTS, CAPEX AND OPEX ESTIMATE, CONTRACTORS' QUOTES

- A1. Substation Quote – Rob Hill & Ass.
- A2. Cost of Water to Mine Estimate - Hydro Tasmania
- A3. Cost Estimate for Cr
- A4. Mining Operation Cost Estimate – Mancala
- A5. Mining Cost Estimate – Shaw
- A6. Minemakers Ltd Assay Results
- A7. Minemakers Ltd Davis Tube Results
- A8. Preparation and Sizing of Tin/Iron Skarn Ore Samples for Minemakers Ltd
- A9. Davis Tube Assay elements and sample procedure
- A10. Fluorspar market in India
- A11. Heavy Minerals Economic Importance
- A12. The Economics of Fluorspar
- A13. Geology and Genesis of the F-Sn-W(-Be-Zn) Skarn (Wrigglite) at Moina Tasmania
- A14. Metallurgical Appraisal for JORC Compliance

Company - Jacobs

Project: TNT Moina Main Substation

Capital Cost Estimate (ex GST)	Sub Total Cost	\$677,000.00
	Contingency 10%	\$67,700.00
	TOTAL BUDGET COST	\$744,700.00

Item	Description	Unit	No Off	Rate	Cost
1.0					
	22 kV Switchgear	ea	1	\$60,000.00	\$ 60,000.00
	Transformers 22 / 11 kV 3 MVA	ea	2	\$120,000.00	\$ 240,000.00
	Neutral Earthing Resistors	ea	2	\$30,000.00	\$ 60,000.00
	11 kV Switchgear	ea	1	\$50,000.00	\$ 50,000.00
	HV Cabling & terminations	lot	1	\$15,000.00	\$ 15,000.00
	Earthworks	lot	1	\$20,000.00	\$ 20,000.00
	Earth Mat	ea	1	\$15,000.00	\$ 15,000.00
	Substation Building	ea	1	\$25,000.00	\$ 25,000.00
	Foundations, Fencing, Signs etc.	lot	1	\$20,000.00	\$ 20,000.00
	Crane Hire	lot	1	\$5,000.00	\$ 5,000.00
	Minor materials -conduit cable ladder etc.	lot	1	\$10,000.00	\$ 10,000.00
	Installation (elect)	lot	1	\$150,000.00	\$ 150,000.00
	Testing & Commissioning	lot	1	\$7,000.00	\$ 7,000.00
	Sub Total				\$677,000.00

Notes:

- (a) Contractor Mob/Demob assumed to be included in Plant Electrical contract
- (b) Substation to be sited on level ground, no major excavation
- (c) Assume Batch Plant exists on site for other works

Olsen, Kobie

From: Greg Carson <Greg.Carson@hydro.com.au>
Sent: Tuesday, 5 June 2012 11:36 AM
To: Meija, George
Subject: RE: Cost of water

Hi

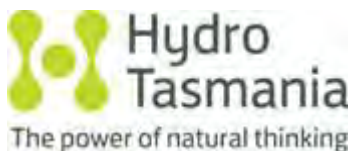
The nominal value of a megalitre of water in the 2011/12 financial year from Lake Gairdner is \$55/megalitre.

The fee is based on the expected value of the energy that could be generated if the water was used for hydro-generation purposes and is reviewed annually. The estimated fee for next year (2012/13) will be available in the next couple of weeks. It is expected to increase to reflect the influence of the 'carbon tax' and other factors.

Thank you

Greg

Greg Carson | Water Operations Advisor



p 0 3 6230 5442 | m 0 408 176 143
e greg.carson@hydro.com.au
w www.hydro.com.au
4 Elizabeth Street, Hobart TAS 7000

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No trees were killed in the sending of this message, but a large number of electrons were terribly inconvenienced.

From: Meija, George [mailto:George.Meija@jacobs.com]
Sent: Tuesday, 5 June 2012 11:24 AM
To: Greg Carson
Subject: Cost of water

Hello Greg,

As mentioned we would most probable not discharge any waste water into the lake.
Let us assume that for the moment.

The question is what would it cost to buy water from the authority.
Only ball park figure. And in the quantity of 800,000 M It per annum.

Regards

George

George Meija

Manager Studies

Mining and Minerals

Jacobs

Direct: +61 (0)7 3018 7261

Mob: 61 (0) 407 193 503

Fax +61 (0)7 3246 9130

Email: George.Meija@jacobs.com

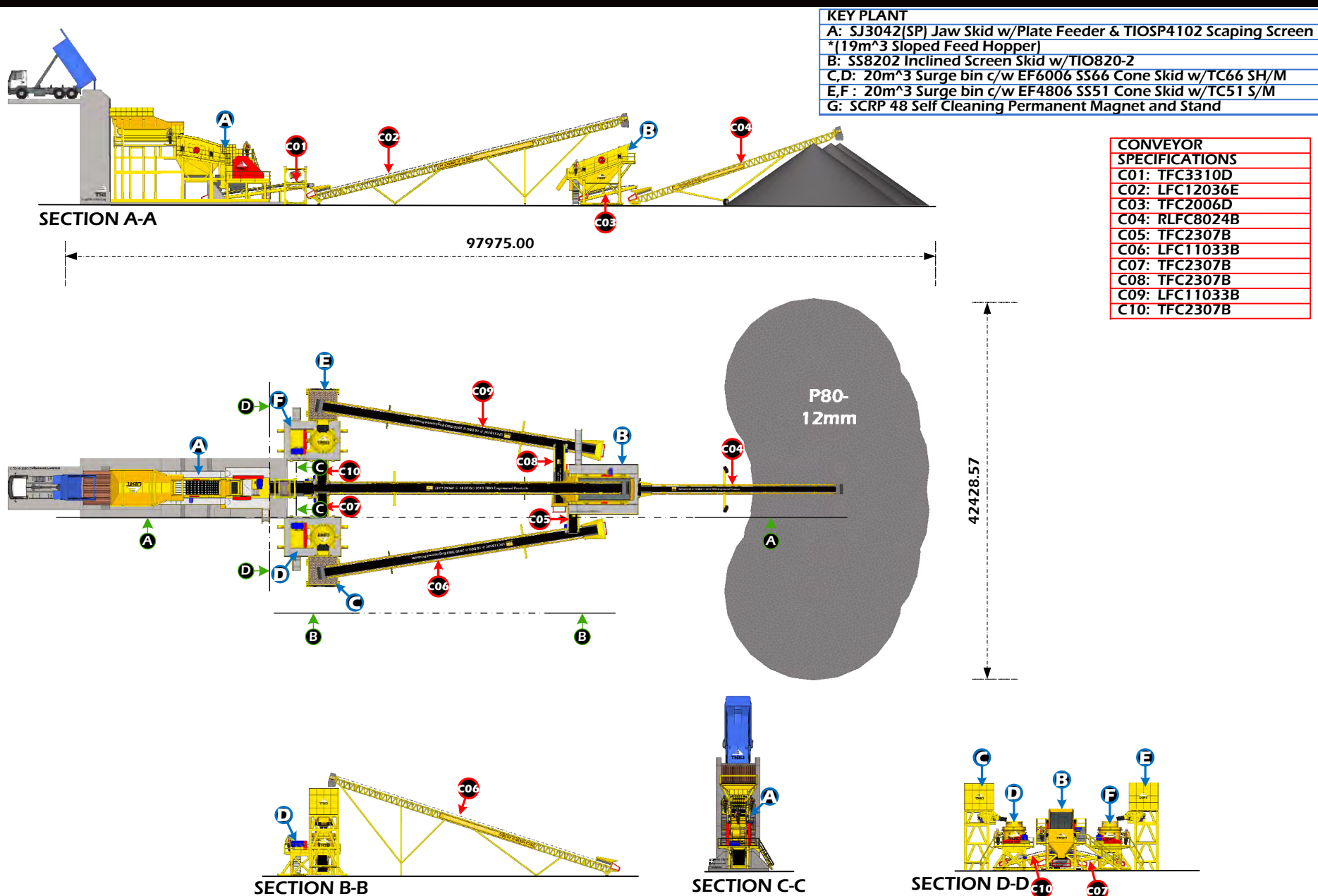
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Client:

INDEX (EMS)
Equipment & Machinery
Sales Pty Ltd.

Project:

JACOBS
250 mTPH
3-Stage Plant

Page: A4

Scale: 1/600

Issued: 12/06/12

Revision: -

Drawn By:
AMC
Drawing No.

ACoo21

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12823 Schabarum Ave,
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25/05/12

George Meija
Manager Studies
Mining and Minerals
Jacobs

Via e-mail to George.Meija@jacobs.com

Re: Moina Ope Cut operation

George,

Thank you for your interest in Mancala assisting you with your cost estimates for the Moina Project.

You may be aware that Mancala recently (Early 2011) completed the Que River Mine open cut where we were mining remnant base metal mineralisation requiring up to 80,000m³ of waste rock movement per month. Over the life of the project average waste movements were around 48,000/mth Que River is located in North West Tasmania and would have many similarities to the project proposed at Moina.

Given our recent operating history and cost base I would suggest the points tabulated below be taken into consideration.

Item	Comments
Workforce Perception	Treat the operation as a Quarry rather than a mine – a discount to mining labour rates of 30% may be realised. E.g. Truck driver in quarry \$25/hr, in a mine \$32/hr
Roster	Surface construction/quarry operators preference would be 10.5 hour shifts day and night. Week days only. Gives daily and weekend maintenance windows and better utilisation of equipment than day shift only
Transport	Bus employees to and from site from central location (say Devonport)
Equipment	Mine is small scale and should use small capacity equipment with a moderate number of each unit to allow for redundancy. Maintenance facility with day shift fitter, electrician, and TA. Maintenance supervisor on day shift. Good comms necessary
Diesel Supply	Western Tasmanian is serviced by two bulk fuel suppliers – Customers are price takers (we are currently consuming \$200,000/mth with no room to move on price). Deals however can be done with storage facilities provided FOC.
Explosive Supply	A major consumable cost – Orica only major supplier although a minor player (Forze) recently gained ability of bulk explosive supply. Mancala has commercial bulk discount arrangements with both.
Technical Labour	Engineering, geology, survey and environmental labour hard to attract and retain. Some local contractors/consultants available. East coast Aust. rates should be considered in any cost estimates.
Environmental	EPA has become very difficult to deal with under the Labour/Green accord. The situation may (hopefully) change shortly – consider delays in approval process and strict enforcement of operating permits.

In terms of operating methodology to determine a cost estimate I have assumed:

- Site infrastructure consisting of maintenance facility, crib/change room, muster area, office complex and explosive storage is present on site;
- Workforce is transported to site daily and two shifts per day of 10.5hrs duration;
- Daily maintenance carried out in operating window with planned maintenance on weekends;
- ROM and waste dump between 1 - 2,000m from operating face (latter as pit is deeper);
- Bench heights of 15 or up to 20m dependent on ground conditions;
- Ground conditions dictate that batters and berms do not need any form of artificial support;
- Material movements of 1,600kta, all of which require drill and blast;
- Typical GET, drill consumable consumption and truck body wear (i.e. not excessively abrasive);
- Drill, blast load and haul activities are unimpeded by geological or other delays;
- Water in pit is easily manageable, preferably by use of de-water bores beyond pit parameter;
- No restriction to blasting times and quantities.
- No restriction on night shift activities (ie drill, load, haul and dump all possible) – blasting day shift only

Given the above, the suggested equipment fleet would be:

Activity	Equipment	Quantity	Comments
Haulage	40 tonne ADT	5 to 7 depending on pit depth	Flexibility in not requiring as well-formed haul roads as ridged trucks. Weather independent Likely to be a number of cutbacks over the mine life.
Excavation	65 tonne excavator	1	Primary excavation unit
Excavation	45 tonne excavator	1	Secondary and backup
Excavation	25 tonne excavator	1	Site drainage, road maintenance general site machine
Drilling	Top Hammer Tracked Machine	2	Atlas Copco D9 or similar drilling 89mm holes (ore and waste)
Dump Maintenance	D10 Tracked Tractor	1	With ripper attachment for pre strip activities
Service	4WD Service Truck	1	Re-fuelling semi mobile equipment and daily service consumables
Road Maintenance	12' Grader	1	Haul road maintenance
Crew Site Transport	Twin Cab LV	4	Supervisor, Maint. Supervisor, Leading hand and Crew
Maint . Transport	Flat Tray 4WD	1	Fitter and/or electrician site transport
Night Shift Illumination	Light Towers	3	Each excavator and dump
Explosives Transport	Flat Tray 4WD	1	Detonator and booster transport from/to magazine
Bulk Explosives	Bomb Truck	1	Sub-contracted as needed
Pit De-watering	Bore Hole Pumps	?	Assume installed and maintained by others
Pit De Watering	Mobile Diesel	?	Assume installed and maintained by others

Given the above activities and equipment selection the anticipated manning levels on a two shift basis would be:

Classification	Number	Comments
Truck Driver	10	Up to 14 when pit depth and haul distance increases
Excavator Operator	5	General Site machine day shift only
Drill Operator	2	Day shift only
Dozer Operator	2	
Grader Operator	1	Day shift only
Shift Trades	3	
Shot Firer/Assistant	2	Day shift only
Maint. Supervisor	1	Day shift only
Supervisor	1	Day shift only
Total	27	

The maximum number of employees on site on a single day shift, early in the mine life would be 17.

In term of cost for a contractor to drill/blast load and haul ore and waste to the ROM or waste dump I would estimate in the region of \$3.00-3.50/tonne early in the mine life and up to 4.25/tonne when the pit reaches a 150m depth. Clearly the accuracy of the above estimate will be highly dependent upon site conditions and must be taken as having a potential error range of +/- 30%.

Attention must also be given to the need to delay excavation and haulage due to geological, engineering, survey or hydrological activities. This would add to the above unit cost estimate.

For a cost estimate to be conducted at a greater level of precision a mine plan and accompanying mine schedule would be required and the estimate based on first principles.

I hope the above assists with your deliberations.

Regards,

Tim Akerman
Business Development manager
Mancala Mining Pty Ltd.



Civil Contracting - Mining
Engineering - Land Developers
Heavy Plant Hire And Transport

TELEPHONE: (03) 6397 0222
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EMAIL: info@shawcontracting.com.au

HEAD OFFICE: 776 WHITEMORE ROAD
WHITEMORE TAS 7303

30 May 2012

George Meija
Jacobs Engineering
Address
Suburb State P/C

Dear George,

Reference: TNT Mines Limited – Moina Mine

Following our recent site visit with Michael Hannington and Russell Fulton, we have pleasure in providing budget figures as requested.

Shaw Contracting is in a unique position in that it has capability, experience and resources to provide mining operations as well as infrastructure installations for construction of roads, civil concreting, building works, tailings dam construction, environmental controls as well as providing high level OH&S, environmental and quality systems under its third party and FSC accreditation.

Shaw proposes to use the following equipment: -

- Drill rig
- PC450 Hitachi excavator to bulk load
- 4 x 40tonne articulated dump trucks
- D8 bulldozer
- 20tonne excavator for batters
- Mobile workshop – 2 x 40' containers with "dome" infill
- 4 x 4WD light vehicles for fitter, supervisor, key operators
- Bus to transport
- Bunded fuel storage area and bunded fuel delivery truck
- Crib facilities, fuel storage, chemical storage container, office.

Expected Manning Levels:

- Project Manager – 40% on site
- Site Supervisor
- 8 to 10 operators



- 1 fitter
- Support crews as required.

Assumptions, Inclusions and Exclusions:

In preparing our budget figures, we provide the following clarifications in relation to: -

1. Assumptions

- Existing tunnels to be mapped by others
- Survey will be provided
- Greenfields EBA
- SG 3.3t/m³

2. Exclusions

- No allowance has been made to construct internal access roads or upgrade the existing access road. We believe that road pavement materials can be gained on site by crushing of onsite basalt. We believe that the existing road will need to be widened from 3.5, to 7.0m. There may be land accusation issues. Possible cost in order of \$300,000.00 to upgrade existing access road and \$250,000.00 to build internal haul roads.
- Heritage issues management
- Cartage of ore by road trucks
- Loading road trucks out of stockpile (Ball park is \$2.00 per tonne based on 988 loader loading 1600t per 10 hour day)
- Crushing (Estimate cost to crush from 600mm minus to 10mm minus at \$8.00 per tonne)
- Processing
- Any preparation of waste dumps
- Any preparation of stockpile and truck loading areas
- Carbon tax
- Tailings dams
- Infrastructure for processing plant
- Environmental controls if acid drainage or other than silt
- Rehabilitation
- Site allowances

We can provide costing on some of the exclusions if required.

We can design and construct loading areas, tailings dams, processing plant civils and sort truck loading, heritage issues, road truck cartage if you want.

The more we do the better controls we can provide.





Civil Contracting - Mining
Engineering - Land Developers
Heavy Plant Hire And Transport

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WHITEMORE TAS 7303

3. Inclusions

- We have allowed to supply fuel for our machines
- Trimming batters
- Site facilities – crib room, office, toilets
- Fuel storage
- Mobile workshop
- Internal haul road maintenance.
- Living away from home allowance
- Environmental controls for silt mitigation and protection of watercourses from silt. (Allow silt fences, settlement ponds).

Budget Pricing

- Clearing \$4500.00 per Ha
- Stripping \$12.50 per m3 solid (no drill and blast allowed)
- Drill and blast \$7.94 per tonne
- Load and Haul \$4.40 per tonne (Based on 6600 tonnes per 10 hour day. Anything less than this becomes less efficient and would require a smaller excavator and becomes more expensive and is somewhat impractical in this application. I know this far exceeds you trucking rates but not sure how to deal with this at this stage. Happy to discuss and/or provide the higher rate to suit your trucking rates if you want.)
- ALL PRICES ARE GST EXCLUSIVE.

Our capability can be viewed on www.shawcontracting.com.au.

Please do not hesitate to call if we can be of further assistance. We look forward to working with you and TNT Mines Limited on this and future projects.

Yours sincerely,

Shaw Contracting Pty Ltd

Kevin Sturzaker
Contracts Manager



PO NUMBER : "per email dated 28/11/08"

[illegible]

Comp-14 +	21.9	6.38	0.002	0.002 >10.0	0.065	0.002	0.081	0.006	37.8	1.5	3.23	0.595	0.86	0.066	0.019	0.021	0.022	0.32 <0.001	0.22	0.006	0.067	0.006	-0.48	
Comp-15 +	25	5.91	0.001	0.009 >10.0	0.049	0.003	0.053	0.007	34.6	1.075	1.45	0.836	0.71	0.032	0.012	0.027	0.11	0.277	0.004	0.14	0.004	0.184	0.009	-0.38
Comp-16 +	22	8.74	0.015	0.01 >10.0	0.11	0.001	0.032	0.006	31.8	2.11	3.74	0.638	0.53	0.018	0.016	0.02	0.113	0.258	0.004	0.16	0.004	0.047	0.006	-0.14
Comp-17 +	21.3	8.61	0.02	0.021 >10.0	0.123	0.002	0.053	0.006	31.6	1.2	3.31	0.66	0.35	0.029	0.014	0.019	0.035	0.112	0.005	0.16	0.004	0.083	0.006	-0.28
Comp-18 +	20.8	7.71	0.014	0.004 >10.0	0.087	0.002	0.065	0.006	35.1	1.99	4.3	0.802	0.36	0.037	0.015	0.024	0.007	0.32	0.003	0.17	0.006	0.061	0.007	-0.4
Comp-19 +	7.39	2.21	<0.001	0.009 4.87	0.027	0.007	0.253	0.008	59.7	0.182	0.73	0.572	0.32	0.163	0.01	0.02	0.007	0.081	0.002	0.18	0.007	0.086	0.004	-1.9
Comp-20+5	19.65	6	0.001	0.002 >10.0	0.072	0.001	0.081	0.006	40.5	1.535	1.15	0.757	0.59	0.049	0.014	0.017	0.024	0.492	0.001	0.11	0.004	0.064	0.004	-0.76
Comp-15 -20DUP																								
Comp-15 +20DUP																								
Comp-15 +53DUP																								
STD DTR03	3.54	0.15	<0.001	0.007	0.09	0.008	0.001	0.048	0.006	67	0.084	0.19	0.009 <0.01	<0.001	0.009	0.011	0.007 <0.001	<0.001	0.02	0.004	0.004	<0.001		0

Minemakers

SEQ NO	Number	Wght Feed Grams	DTR Wash Time	Wght F/Paper Grams	Wght Conc+ Paper Grams	Wght Conc Grams	Rec Perc %
1	Comp-1 -20	20.132	20	78.913	82.457	3.544	17.60
2	Comp-2 -20	20.390	20	93.755	98.894	5.139	25.20
3	Comp-3 -20	20.235	20	88.823	93.090	4.267	21.09
4	Comp-4 -20	20.110	20	73.969	78.790	4.821	23.97
5	Comp-5 -20	20.460	20	89.992	95.434	5.442	26.60
6	Comp-6 -20	20.468	20	96.770	101.473	4.703	22.98
7	Comp-7 -20	20.230	20	94.075	95.714	1.639	8.10
8	Comp-8 -20	20.272	20	71.729	76.256	4.527	22.33
9	Comp-9 -20	20.295	20	68.339	69.180	0.841	4.14
10	Comp-10 -20	20.382	20	97.416	98.553	1.137	5.58
11	Comp-11 -20	20.415	20	98.339	99.337	0.998	4.89
12	Comp-12 -20	20.478	20	87.349	90.878	3.529	17.23
13	Comp-13 -20	20.060	20	94.315	97.547	3.232	16.11
14	Comp-14 -20	20.464	20	95.168	98.170	3.002	14.67
15	Comp-15 -20	20.390	20	70.348	74.779	4.431	21.73
16	Comp-16 -20	20.092	20	64.997	70.285	5.288	26.32
17	Comp-17 -20	20.199	20	92.624	97.687	5.063	25.07
18	Comp-18 -20	20.071	20	91.849	95.919	4.070	20.28
19	Comp-19 -20	20.453	20	88.660	90.834	2.174	10.63
20	Comp-20 -20	20.699	20	64.863	69.761	4.898	23.66
21	Comp-1 +20	20.155	20	69.032	73.714	4.682	23.23
22	Comp-2 +20	20.221	20	73.036	79.697	6.661	32.94
23	Comp-3 +20	20.338	20	70.406	77.620	7.214	35.47
24	Comp-4 +20	20.127	20	79.111	87.728	8.617	42.81
25	Comp-5 +20	20.732	20	97.355	105.249	7.894	38.08
26	Comp-6 +20	20.075	20	72.467	78.231	5.764	28.71
27	Comp-7 +20	20.875	20	98.258	100.697	2.439	11.68
28	Comp-8 +20	20.611	20	88.290	94.312	6.022	29.22
29	Comp-9 +20	20.147	20	85.829	86.604	0.775	3.85
30	Comp-10 +20	20.074	20	78.317	79.509	1.192	5.94
31	Comp-11 +20	20.655	20	82.880	83.975	1.095	5.30
32	Comp-12 +20	20.113	20	102.070	107.230	5.160	25.66
33	Comp-13 +20	20.044	20	84.168	88.161	3.993	19.92
34	Comp-14 +20	20.789	20	95.711	100.098	4.387	21.10
35	Comp-15 +20	20.339	20	79.767	85.562	5.795	28.49
36	Comp-16 +20	20.426	20	89.379	96.857	7.478	36.61
37	Comp-17 +20	20.971	20	104.643	111.380	6.737	32.13
38	Comp-18 +20	20.949	20	96.817	103.037	6.220	29.69
39	Comp-19 +20	20.545	20	95.000	98.431	3.431	16.70
40	Comp+20 +20	20.287	20	85.208	88.614	3.406	16.79
41	Comp-1 +53	20.187	20	97.416	103.516	6.100	30.22
42	Comp-2 +53	20.517	20	98.337	106.430	8.093	39.45
43	Comp-3 +53	20.442	20	87.350	96.442	9.092	44.48
44	Comp-4 +53	20.125	20	94.313	104.640	10.327	51.31
45	Comp-5 +53	20.211	20	95.166	104.306	9.140	45.22
46	Comp-6 +53	20.150	20	70.349	77.340	6.991	34.69
47	Comp-7 +53	20.456	20	64.999	65.824	0.825	4.03
48	Comp-8 +53	20.259	20	91.848	99.590	7.742	38.22
49	Comp-9 +53	20.370	20	88.663	90.183	1.520	7.46
50	Comp-10 +53	20.019	20	64.864	66.338	1.474	7.36
51	Comp-11 +53	20.077	20	69.036	70.193	1.157	5.76
52	Comp-12 +53	20.090	20	73.037	77.899	4.862	24.20
53	Comp-13 +53	20.190	20	70.408	74.659	4.251	21.05
54	Comp-14 +53	20.052	20	79.114	82.632	3.518	17.54
55	Comp-15 +53	20.045	20	92.625	97.409	4.784	23.87
56	Comp-16 +53	20.047	20	85.831	94.776	8.945	44.62
57	Comp-17 +53	20.272	20	95.713	103.886	8.173	40.32
58	Comp-18 +53	20.194	20	79.772	86.859	7.087	35.09
59	Comp-19 +53	20.075	20	102.074	104.278	2.204	10.98
60	Comp-20+53	20.090	20	82.884	89.151	6.267	31.19
61	Comp-15 -20DUP	NSS	NSS	NSS	NSS	NSS	NSS
62	Comp-15 +20DUP	NSS	NSS	NSS	NSS	NSS	NSS
63	Comp-15 +53DUP	NSS	NSS	NSS	NSS	NSS	NSS
64	STD DTR031008	20.130	20	88.291	92.634	4.343	21.57



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PREPARATION AND SIZING
OF
TIN/IRON SKARN ORE SAMPLES
FOR
MINEMAKERS LTD

JAMES DEBERNER
DATA REPORT NO T0442
DECEMBER 2008

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2. SAMPLES RECEIVED.....	4
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4. DATA	Error! Bookmark not defined.

1. SUMMARY

Eighty-nine samples of a tin/iron skarn ores were received for preparation and sizing to client request.

Tasks performed and sizing data are presented below.

2. SAMPLES RECEIVED

Some 89 samples of dry $\frac{1}{4}$ core pieces were received for preparation then equal weight compositing to twenty samples for further work. Composited intervals are presented in Table 1 below.

TABLE 1:

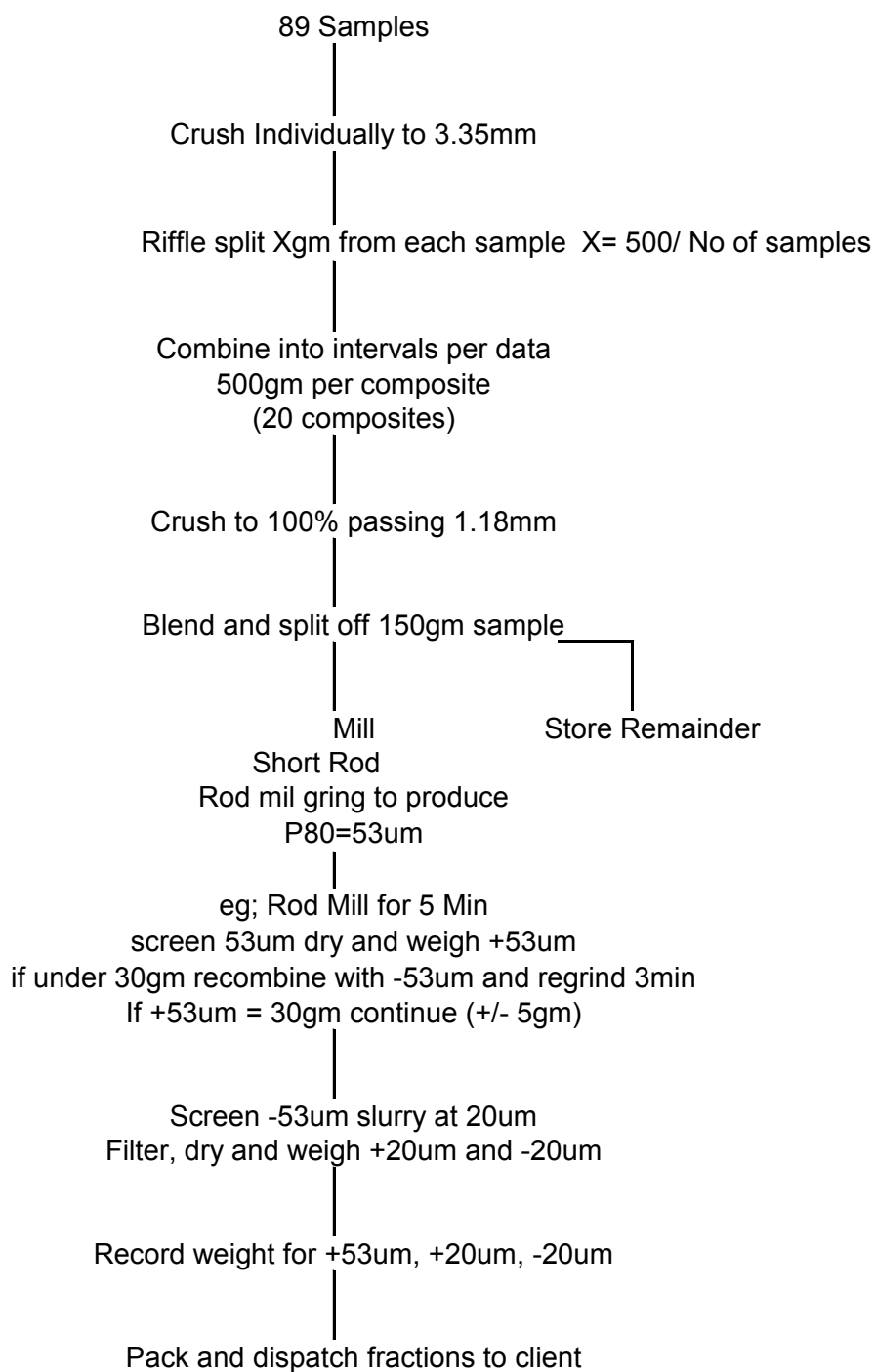
HOLE	INTERSECTION	BRL COMPOSITE NO
SMD11	31.0-37.0	1
SMD11	37.0-42.0	2
SMD11	42.0-47.0	3
SMD11	47.0-52.0	4
SMD11	52.0-57.0	5
SMD11	57.0-60.7	6
SMD11	60.7-63.1	7
SMD11	63.1-68.0	8
SMD16	30.0-34.0	9
SMD16	34.0-38.0	10
SMD16	38.0-42.0	11
SMD16	42.0-45.0	12
SMD16	45.0-49.0	13
SMD16	49.0-61.0	14
SMD16	61.0-105.0	15
SMD16	105.0-110.0	16
SMD16	110.0-115.0	17
SMD16	115.0-120.0	18
SMD16	120.0-125.0	19
SMD16	125.0-129.0	20

3. PROCEDURE

Task sheet below details procedure used to perform sizing request.

TESTWORK TASK SHEET

Preparation and Sizing	6/11/2008	T0 442
------------------------	-----------	--------



RESULTS

LAB GENERAL TEST DATA

SAMPLE	Minemakers Limited
TEST TYPE	+53,+20um,-20um
	Staged Rod Mill Grinds

PROJECT	T0442
TEST NO	
DATE	28/11/2008
TECHNICIAN	ID

COMP	+53um (gm)	Total Grind Time	+20um (gm)	-20um (gm)	End Mass
1	36.49	4.50	48.08	62.59	147.16
2	28.28	5.50	54.22	66.57	149.07
3	27.67	6.00	45.92	74.44	148.03
4	28.11	6.00	44.22	76.35	148.68
5	35.41	6.50	42.72	68.98	147.11
6	29.92	6.00	48.28	71.46	149.66
7	31.62	8.00	49.68	67.48	148.78
8	38.53	5.50	42.74	69.14	150.41
9	38.03	5.00	57.61	52.17	147.81
10	30.9	5.50	67.73	47.76	146.39
11	24.73	5.50	67.08	52.8	144.61
12	33.16	4.00	55.31	57.47	145.94
13	25.72	5.00	50.19	72.26	148.17
14	31.17	3.50	50.3	64.38	145.85
15	32.46	3.50	56.99	58.53	147.98
16	26.9	4.00	52.39	69.49	148.78
17	38.46	4.00	45.94	64.36	148.93
18	28.85	4.00	51.56	68.52	148.93
19	30.75	5.50	64.55	51.8	147.1
20	31.39	4.00	53.38	63.74	148.51

SAMPLE PREPARATION PROCEDURE

Procedure 1 – Crushing

1. Dry & Crush entire sample to 100% less than **client to specify mm**
2. Split to ~ 2kg using Jones Riffle Splitter
3. Retain split in vac seal bag (if required)
4. Homogenise via mat rolling then selectively sub sample to produce a 150g sample

Procedure 2 – Pulverising

1. Pulverise the 150g sample for (**client to specify seconds**) in a ring mill pulveriser (**nominated by client**).
2. Wet screen the sample (**client to specify micron size**) and record oversize weights. If less than 5g of oversize is produced then a 150g sample must be re-split and pulverised for a shorter time.
3. Dry and regrind the oversize for 1 second for every 1g of oversize (**client to specify times**).
4. Repeat the screening, until plus fraction (**client to specify fraction**) is less than 5g.
5. Filter press total sample, dry and homogenise.
6. Using a 3 decimal place balance, sample the pulverised product to give a 20g sample for DTR test work.
7. The remaining pulverised material to be used for head grade assay.

DAVIS TUBE RECOVERY PROCEDURE

Procedure 3 – Davis Tube Recovery

The following procedure needs to be confirmed by client

- | | |
|----------------------------|------------------------------------|
| 1. Stroke Frequency | 60/minute |
| 2. Stroke Length | 38 mm |
| 3. Magnetic Field Strength | 3000 gauss |
| 4. Tube Angle | 45 degrees |
| 5. Tube Diameter | 25 mm |
| 6. Washing Time | 15 minutes or until water is clear |

The concentrate sample is collected in a small container after washing is complete. The concentrate is then vacuum filtered, washed, dried and weighed.

ANALYTICAL PROCEDURE

FEED ASSAY

*Fused Disk Method ME-XRF11s Iron Ore Suite
(Lithium Borate Fusion, Lithium Nitrate Oxidant)*

The listed elements needs to be confirmed by client

Elements and Ranges (%)

Al ₂ O ₃ (0.01)	K ₂ O (0.01)	Na ₂ O (0.01)	SiO ₂ (0.01)
CaO (0.01)	MgO (0.01)	P (0.001)	TiO ₂ (0.01)
Fe (0.01)	Mn (0.001)	S (0.001)	

Fusion, **package** plus LOI 1000°C

CONCENTRATE ASSAY

*Fused Disk Method ME-XRF11s Iron Ore Suite
(Lithium Borate Fusion, Lithium Nitrate Oxidant)*

The listed elements needs to be confirmed by client

Elements and Ranges (%)

Al ₂ O ₃ (0.01)	K ₂ O (0.01)	Na ₂ O (0.01)	SiO ₂ (0.01)
CaO (0.01)	MgO (0.01)	P (0.001)	TiO ₂ (0.01)
Fe (0.01)	Mn (0.001)	S (0.001)	

Fusion, **package** plus LOI 1000°C

FERROUS IRON ASSAY

Method code	Description	Range %
VOL-05	Ferrous iron by HCL-HF acid digestion and titrimetric finish.	0.01 – 100

Method VOL-05.....

LASER SIZING

Mastersizer 2000 Dual laser automated particle counter.

Per sample.....

Fluorspar market in India

There are three principal market grades of beneficiated fluorspar

- Acid grade fluorite (97% or more CaF₂): is the primary industrial source of fluorine. Acid grade fluorite is used for making hydrogen fluoride (HF) and hydrofluoric acid.
 - It is an important raw material for most fluorine-containing fine chemicals, including fluorocarbons, aluminum fluoride, and synthetic cryolite.
 - Acid grade production is 68% of fluorspar production worldwide
- Inferior grade fluorite (85-96% CaF₂) further divided into
 - Ceramic (intermediate) grade fluorite (85-95% CaF₂): is used in the manufacture of opalescent glass, enamels and cooking utensils.
 - Metallurgical grade fluorite (60-95% CaF₂): is used as a flux to lower the melting point of raw materials in steel production for removal of impurities

Consumption of fluorspar in India

- India mainly consumed Acid grade fluorspar for production of fluorochemicals, aluminum fluoride- Acid grade constitutes more than 85% of total fluorspar consumption totaling approx 145,000 tpa

	HF Capacity	HF production in 2010-11	Fluorspar demand @2.4t/t	Import of HF	Net demand of fluorspar	
SRF	12000	12000	28800		28800	Not reported, assumed
NFIL	24000	23400	56160		56160	
GFL	12000	12600	30240		30240	
Chemplast	0	0	0		0	
Tanfac prdn	15600	10895	26148		26148	
	63600	58895	141348	1240	144324	Derived

- Metallurgical grade consumption is approx 23,000 tpa.
- GMDC, is the only producer of Met grade fluorspar in India- the production at GMDC has remain very erratic during last 5 years.

Import of Fluorspar into India as reported by Indian Customs is given below:

Qty Imported -India- MT									MT/Year
S.No.	HSCode		2011-12 (6 months)	2010-2011	2009-2010	2008-2009	2007-2008	2006-2007	2005-2006
1	25292100	Acid grade	11,203	18,425	36,968	9,714	12,302	25,600	17,964
2	25292200	Met grade	99,381	143,202	109,827	143,794	149,732	105,313	87,872
Total			110,584	161,627	146,795	153,508	162,034	130,913	105,836
Local Production - GMDC - MET				4,801	8,515	6,830	4,195	0	2,297
Apparent Consumption of Metgrade fluorspar				23,226	45,483	16,544	16,497	25,600	20,261



Assessment Manual



**Heavy Minerals of
Economic Importance**

Heavy Minerals of Economic Importance

(Ilmenite and Leucoxene, Rutile, Zircon, Monazite and Xenotime, Kyanite, Sillimanite and Andalusite, Staurolite, Garnet, Chromite, Magnetite, Cassiterite, Columbite-Tantalite, Wolframite and Scheelite)

HARALD ELSNER

[218 p., 31 fig., 125 tab.]



Heavy minerals (ilmenite, leucoxene, rutile, zircon, monazite, xenotime, kyanite, sillimanite, andalusite, staurolite, garnet, chromite, magnetite, cassiterite, columbite-tantalite, wolframite, scheelite), genesis, deposits, quality, use, parameters, technological properties, occurrences, production, reserves

[Heavy minerals of economic importance]

Abstract: Usually, heavy minerals are not the first thing on the mind of working geologists in Germany. However, on a global scale, they are of crucial economic importance as non-metallic and metallic raw materials alike. Economic heavy mineral deposits are mainly located in tropical and sub-tropical countries. Heavy minerals are traded globally as bulk and bagged minerals. The economics of the various heavy minerals is the subject of this monograph. The group of heavy minerals described includes ilmenite and leucoxene, rutile, zircon, monazite and xenotime, kyanite, sillimanite and andalusite, staurolite, garnet, chromite, magnetite, cassiterite, columbite-tantalite, as well as wolframite and scheelite.

The report starts with an outline of the ore genesis of heavy mineral deposits and how they are mined. It is followed by data on the mineralogy, petrography, chemistry and ore genesis of each of the heavy minerals, plus extensive information on their application, their uses in various products, and their potential substitutions by other materials. Background information, reserves and production data, as well as a selection of basic references, conclude each chapter.

[Schwerminerale von wirtschaftlicher Bedeutung]

Kurzfassung: Schwerminerale stehen nicht so sehr im Bewusstsein des in Deutschland tätigen Geologen. Weltweit sind sie als nichtmetallische aber auch metallische Mineralrohstoffe jedoch von großer wirtschaftlicher Bedeutung. Vor allem in Ländern der tropischen und subtropischen Zone finden sich bauwürdige Anreicherungen von Schwermineralen. Fast alle Schwerminerale werden weltweit und in großen Mengen gehandelt. Die wirtschaftliche Bedeutung der verschiedenen Schwerminerale ist Gegenstand dieser Monographie. Die Gruppe der hier beschriebenen Schwerminerale umfasst Ilmenit und Leukoxen, Rutil, Zirkon, Monazit und Xenotim, Disthen, Sillimanit und Andalusit, Staurolith, Granat, Chromit, Magnetit, Cassiterit, Columbit-Tantalit sowie Wolframit und Scheelit.

Zu Beginn der Abhandlung steht ein Überblick über die Genese von Schwerminerallagerstätten und deren Abbau. Die einzelnen Schwerminerale werden danach getrennt mineralogisch, petrographisch, chemisch sowie lagerstättengenetisch beschrieben. Es folgen umfangreiche Informationen zur Verwendung der Minerale und der aus ihnen gewonnenen Produkte einschließlich der Möglichkeiten ihrer Substitution. Hintergrundinformationen und Zahlen zur Vorratssituation und Produktion sowie eine Auswahl grundlegender Schriften schließen jedes Kapitel ab.

[Minerales pesados de importancia económica]

Resumen: Minerales pesados no tienen un gran papel en la consciencia de los Geólogos en Alemania. Sin embargo los minerales pesados metálicos y no-metálicos tienen una gran importancia económica a escala mundial. Especialmente en los países de la zona tropical y subtropical se encuentran enriquecimientos de minerales pesados que son dignos de explotar. Casi todos los minerales pesados se venden y además en gran cantidades a escala mundial. La

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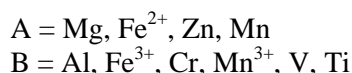
10 Magnetite

[German: *magnetite*, Spanish: *magnetita*, French: *magnétite*]

10.1 Mineralogy, petrography, chemistry

The mineral magnetite, colloquially magnetic iron ore, is a member of the spinel group with the general formula is AB_2O_4 – just as chromite described in Chapter 9.

The natural order of frequency is:



In accordance with the most common occupation of the B-position, the spinels are subdivided into aluminum spinels, iron (III) spinels and chrome spinels. Manganese(III), vanadium and titanium spinels are rarer. Magnetite is the most common iron (III)-spinel and at the same time the most frequent natural spinel of all.

Chemically pure **magnetite** has the following properties:

Formula	Fe_3O_4
Chemistry	31.03 % by mass FeO, 68.97 % by mass Fe_2O_3 and 72.36 % by mass Fe, 27.64 % by mass O
Density	5.16 (5.18) g/cm^3
MOHS hardness	5 ½
Color	iron black
Melting point	1,594 °C
Magnetic properties	ferromagnetic
Electrostatic properties	conductive

Due to the preferred incorporation of the elements Mg, Mn, Zn, Ni, Cr, Ti, Al, V in the spinel lattice, naturally occurring magnetite nearly always has varying percentages of these elements. Co is rarely found.

Magnetite is an important mineral in the mixing triangle FeO- Fe_2O_3 - TiO_2 (cf. Figure 2.1). This explains why magnetite frequently occurs in magmatic and metamorphic rocks in conjunction with ulvite (Fe_2TiO_4) and titanomagnetite (solid solution between magnetite and ulvite) as well as ilmenite (FeTiO_3) and goethite ($\alpha\text{-FeOOH}$). Depending on the crystallization temperature, the cooling rate, availability of oxygen, magma composition and rock pressure, these minerals occur either separately from each other or in homogenous fixed compounds or as inclusions within each other (MÜCKE 2003).

Titanomagnetite contains up to 13 % TiO_2 (cf. Table 10.1), vanadium magnetite (coulsonite) up to 5 % V_2O_5 , and also Cr, Al and Mg.

Table 10.1: TiO₂ contents (%) of (titano-) magnetites in placers in Oceania (NOAKES 1977).

Country or region	Stratigraphy/host rock	TiO ₂ content
Philippines, Taiwan, Japan	Tertiary volcanites	~7
Indonesia-Java	Quaternary volcanites	≥10
New Zealand, Mount Egmont	Quaternary andesitic lava	8-9
Papua New Guinea	Quaternary volcanites	~10
	Mesozoic- Tertiary rocks	~10
Philippines Manila region	Quaternary volcanites	~11
	“older rocks”	3.4-6.8

Magnetite has the strongest magnetism of all minerals. Magnetic iron ore was thus used as a “compass needle” in the middle ages. At temperatures >578 °C (the Curie temperature) magnetite converts to hematite and loses its magnetic properties. Below 578 °C, it can be re-magnetized.

Magnetite easily weathers to Fe₂O₃, in particular to α-Fe₂O₃ (hematite). The shape of the magnetite crystals frequently remains intact during this conversion. Aggregates of hematite or pseudomorphs of magnetite are called martite. Occasionally, the magnetite lattice remains intact during oxidation and γ-Fe₂O₃ (maghemite) forms. These two minerals, martite and maghemite, are also magnetic.

10.2 Formation of deposits and occurrences

Magnetite can form under magmatic, metamorphic, sedimentary and even biogenic conditions. The most important conditions for these formations and occurrences are:

- as early crystallites from basic magmas in plutonites (diorite, gabbro etc.) as well as volcanites (the dark appearance of basalt is due to finely dispersed magnetite)
- due to crystallization differentiation, i.e. gravitative precipitation of the heavy elements Fe and V in ultrabasic and basic magmas. Globally this process resulted in the formation of important intramagmatic magnetite and titanomagnetite deposits. The paragenetic ores mainly contain magnetite, titanomagnetite, coulsonite, ilmenite, spinel and corundum. Examples of important deposits are: Taberg/Sweden, Otanmäki/Finland and Bushveld/Rep. South Africa
- also in huge amounts in gravitatively enriched melts, no longer “in-situ”, but as “ore magma” intruded into adjacent geological structures. These occurrences (paragenetic ore magnetite, apatite) are amongst the largest concentrations of iron in the earth's crust. Examples of important deposits are: Kirunavaara, Gellivaara, Grängesberg and Luossavaara/Sweden as well as Lake Superior/Canada.
- contact pneumatolytic titanium-free and phosphorous-free paragenetic rocks with skarn minerals and sulfides. Examples of large occurrences are: Magnitogorsk (Name!) in the Ural mountains/Russia, Dannemora and Persberg/Sweden.
- in pegmatites occasionally in larger amounts, e.g. in the gabbro-pegmatites of the Kola Peninsula/Russia.

Magnetite is also present in:

- the products of volcanic exhalations, e.g. on the Aleutians/Alaska,
- marine-exhalative-hydrothermal-sedimentary rocks in iron ore deposits of Lahn-Dill type,
- hydrothermal-metasomatic rocks at the point of contact to alkaline rock,
- sedimentary rocks in the Precambrian banded iron formations (itabirites) of ancient shields,

- contact metamorphic rocks as a result of the metamorphosis of siderite, chamosite or thuringite,
- regional metamorphic rocks due to the reduction of hematite,
- alpine fissures,
- heavy mineral placers (cf. below)

as well as non deposit-forming, but still important

- biogenic form in bacteria (e.g. *Aquaspirillum magnetotacticum*) as well as in the brains of bees and some birds (and humans). In this case, magnetite serves as a magnetoreceptor, i.e. to sense the earth's magnetic field to help navigation.

(Titanom)agnetite is a typical heavy mineral, whose enrichment in placers is linked to the proximity of large basalt bodies or young volcanites. Economically exploitable magnetite and titanomagnetite placers, frequently called “iron sand”, occur in:

- **New Zealand, North Island:** New Zealand is by far the largest producer in the world of titanomagnetite from placers. On North Island there are 24 individual deposits with total resources of >850 million tonnes of concentrate with an average content of 55-56 % Fe, 7-9 % TiO₂ and 0.3-0.4 % V₂O₅. Mining is active in the Waikato North Head deposit, and in the Taharoa deposit. The Waipipi deposit was mined between 1971 and 1987 producing a total of 15.7 million tonnes of concentrate with an average content of 56 % Fe and 7.8 % TiO₂ from 60 million tonnes of ore sand. All occurrences are Holocene placers as well as Pleistocene beach placers and above all dune placers. Host rocks are the very common andesite and rhyolite of the North Island. The data on the two deposits still being mined (both by BlueScope Steel Ltd.) are:
 - Waikato North Head: Since 1969, dune sands up to 120 m thickness are mined by bucket wheel excavators, to a lesser degree also beach sands, tephra and silts. The resources (as of 1989) were 780 million tonnes of raw sand at an average content of 18 % titanomagnetite, corresponding to 140 million tonnes of concentrate. The proven reserves (as of 1997) were 74 million tonnes of ore sand with an average content of 33.8 % by mass titanomagnetite, corresponding to 19.4 million tonnes of concentrate with an average content of 59 % Fe. The processed concentrate is used for the domestic manufacture of steel.
 - Taharoa: More than 120 m thick dune sands in a basin measuring 8 x 2 km have been mined since 1972 using cutting suction dredgers, i.e. wet (dredge) mining. The estimated total resources were (as of 1979) 2,560 million tonnes of ore sand, corresponding to 565 million tonnes of concentrate, of which only 593 million tonnes of ore sand with an average content of 35 % by mass titanomagnetite, corresponding to 208 million tonnes of concentrate were extractable, (as of: 1969). The proven reserves (as of 1997) were 11 million tonnes of ore sand with an average content of 55 % by mass titanomagnetite, corresponding to 5.3 million tonnes of concentrate with an average content of 57.5 % Fe and 8 % TiO₂. All mineral concentrates from Taharoa are exported for manufacturing steel, primarily to Japan, secondarily to South Korea and China.
- **Indonesia:** On the islands of Sumatra, Bali, Flores, and most of all in the south of Java, numerous occurrences of economically extractable (titano)magnetite deposits are known and also in some cases these have been mined for some time. The mid-Tertiary andesites of the islands are the host rocks. The most important occurrence (all in Java) are:
 - Cilacap: on average 3 m thick beach and dune placers with 20-50 % by mass (titano)magnetite, average contents of the concentrate: 57-59 (53-58) % Fe, 11 (6-8) % TiO₂, 4.5 % SiO₂, 5 % Al₂O₃, 0.05 % P and 0.03 % S, extraction of <400 kilotonnes concentrate/year from 1971 to 2003, deposit depleted, original content of the deposit: 6 million tonnes at a *cut-off grade* of 37 % Fe,

- Kutoarjo: reserves of 11.6 million m³ ore sand with 2.55 million tonnes of magnetite (average contents of the concentrate: 45.7-46.2 % Fe) at a *cut-off grade* of >45 % Fe (as of: 31.12.2003), mined, used in the local cement industry and for export to Japan,
- Lumajang: reserves of 1.785 million tonnes of ore sand with 700,000 tonnes of magnetite (average contents of the concentrate: 48.6-50.2 % Fe) at a *cut-off grade* of >45 % Fe (as of: 31.12.2003), mined, used in the local cement industry and for export to Japan,
- Yogyakarta: on average 2 m thick, elevated beach and dune placers (JACOB et al. 1979) with an average content of 12.25 % Fe, average contents of the concentrate: 59 % Fe, 7-10 % TiO₂, resources: 230 million tonnes of sand or 28 million tonnes of extractable concentrate,
- Djombang Kulon: a) fossil (beach?) placer, average contents: 37 % Fe, 12 % TiO₂, content in the deposit 6.7 million tonnes at a *cut-off grade* of 37 % Fe, b) recent dune placers, average contents: 20 % Fe, 4 % TiO₂,
- Cipatuja: beach and dune placers with an average of 19.8 (-31.5) % M. (titano)magnetite, average contents of the concentrate: 57 % Fe, 12.8 % TiO₂, 0.25-0.4 % V and 0.05-0.1 % P, contents of the deposit: 4.21 million tonnes at a *cut-off grade* of 7 % by mass (titano)magnetite,
- Cadman: beach and dune placers with an average of 23.2 % by mass (titano)magnetite, average contents of the concentrate: 57.39 % Fe, 12.75 % TiO₂, contents in the deposit: 3.22 million tonnes,
- Ciaos: beach and dune placers between the Ciaos and Cabaña Rivers with an average of 18.5 % by mass (titano)magnetite, average contents of the concentrate: 57.18 % Fe, 12.32 % TiO₂, resources: 25.0 million tonnes of sand or 4.65 million tonnes of extractable concentrate, in addition 0.46 million tonnes of concentrate offshore, completely protected as national iron ore reserve,
- **Philippines:** On the Philippines, and on the main island of Luzon in particular, (titano)magnetite is the main valuable mineral in numerous beach placers. The host rocks are andesites and other volcanites in the interior of the island. Mining of this mineral started in 1964 and was continued on a large scale until 1976. Approx. 8.5 million tonnes of ore sands were produced, whose contents were sold mainly to Japanese steel companies. A national mining prohibition because of ecological concerns was only eased in 1981. Currently no mining on a large scale takes place. The following is known about the individual islands:
 - Luzon: 22 individual deposits of titanomagnetite and/or magnetite with >1 million tonnes in each deposit, total reserves and resources approx. 45 million tonnes of (titano)magnetite, contents in the concentrate: 57-62 % Fe, 4.8-12.67 % TiO₂, 2.5-3.7 % SiO₂, 1.2-1.5 % Al₂O₃, 0.02-0.04 % P, 0.03 % S
 - Mindanao: 4 individual deposits of titanomagnetite with only small total reserves and resources of approx. 100,000 tonnes; chromium bearing, also chromite placers,
 - Leyte Island: 2 individual deposits of magnetite with total reserves and resources of approx. 5 million tonnes and contents of 4-55 % by mass in the sediment
 - Negros Island: 2 individual deposits of magnetite with total reserves and resources of approx. 1 million tonnes
- **Japan:** In Japan, magnetite sands were mined on all islands from ancient times until 1979. The iron used to be used for the manufacture of Samurai swords, and in World War II as a source of raw materials for the Japanese arms industry. Currently, only the severely restricted extraction of a few tonnes of iron sands for the manufacture of hand-forged kitchen knives is conducted. At first, fine-grained beach placers with an average content of 25.6 % by mass were mined, later only 10 % Fe in the ore sand. After these deposits had been depleted, the extraction was moved to older beach terraces, residual placers (TSUSUE & ISHIAHRA 1975) and coarse-grained fluvial placers with an average content of 20-30 % Fe and/or 30 – 40 % by mass magnetite in the sediment. In 1956, the production of offshore deposits started. A comparison of the chemical compositions of Japanese titanomagnetite concentrates is listed in Table 10.2. The remaining total geological resources in Japan have been estimated at 1,306 million tonnes of ore sand at an average content of 6.3 % Fe,

corresponding to 82.3 million tonnes of Fe. The economically extractable reserves and resources are estimated to be 67.5 million tonnes of ore sand with an average content of 14.5 % Fe, corresponding to 9.8 million tonnes of Fe. (WARD & TOWNER 1985b).

Table 10.2: Chemical composition (% by mass) of Japanese titanomagnetite concentrates (WARD & TOWNER 1985b).

	Onshore		Offshore
	recent beach	older terraces	
Fe	58.76	58.40	56.77
TiO ₂	10.92	7.54	10.44
SiO ₂	2.16	4.30	8.50
Al ₂ O ₃	2.69	2.95	2.95
P	0.009	0.133	0.170

- **Canada, Quebec**, Natashquan River: resources of 22 million tonnes of magnetite in 1,659 million tonnes of sediment with an average heavy mineral content of 8.03 % by mass. Accompanying valuable minerals are ilmenite, garnet and zircon,
- **Chile**, La Barca and Duna Choapa Norte: magnetite, gold and garnierite [(Ni,Mg)₆Si₄O₁₀(OH)₈] are gangue valuable minerals in the local dune placers. Ilmenite is the main valuable mineral. La Barca: resources of 100.56 million tonnes of ore sand with an average content of 9.71 % by mass of magnetite. Duna Choapa Norte: resources of 45.77 million tonnes of ore sand with an average content of 10.70 % by mass of magnetite.
- **India**: magnetite is enriched in the coastal placers of the Indian states of Andhra Pradesh and Maharashtra. Table 10.3 lists the known pertinent data.

Table 10.3: Magnetite bearing beach placers in India (according to RAJU et al. 2001).

State	Coastal region	Content (% by mass)		Resources (million tonnes)
		in the ore sand	in the HM-fraction	
Andhra Pradesh	northern		0.20	0.56
	central	11.66-36.68 (av. 16.50)	5.03	10.28
	southern	4.65-25.48	7.44	n.a.
Maharashtra	Ratnagiri			0.39

- **South Korea**: In South Korea, four smaller deposits with total resources of 285,000 tonnes of magnetite are known, accompanying valuable minerals are: ilmenite, monazite, zircon and in part gold,
- **Rep. of South Africa, KwaZulu Natal**, Richards Bay: magnetite and garnet are valuable gangue minerals in the local dune placers. Main valuable minerals are ilmenite, rutile and zircon,
- **Ecuador, Guayana**: small-scale mining of beach placers with titanomagnetite for the domestic cement industry, accompanying heavy minerals are ilmenite, pyroxene, amphibole, epidote, staurolite, as well as to a very small degree zircon and garnet,
- **Costa Rica, Guatemala**: Because of the predominance of volcanic source rocks, rock magnetite placers can be found along most of the pacific coast of Central America.

10.3 Applications

Magnetite is an important iron ore. The theoretical Fe-content of 72.4 % (the highest of all Fe-minerals) is only rarely attained because of the presence of isomorphous additions of Al, Ti, Mg and V. In particular Ti and V lower the value of the mineral as an iron ore, possibly turning Ti-V magnetites into ores of these two elements, however.

Because of its properties:

- high bulk density
- high physical and chemical resistance
- good thermal conductivity
- high magnetic sensitivity
- non-toxicity

Magnetite is used for the following purposes outside of the metallurgical sphere:

- as the raw material for the manufacture of high-purity iron oxide for audio/video tapes (decreasing use),
- as additive in toners for laser printers and copiers. The percentage of magnetite in toners differs significantly depending on the manufacturer and the exact application, and amounts to a few to 70 % by mass. Further important components of toners are synthetic resins, various polymers, polyolefin waxes and carbon,
- as an iron additive for the manufacture of cement. For producing cement in Indonesia a raw mix of ~82 % limestone, ~16.5 % sandy clay and ~1.5 % "iron sand" (= magnetite concentrate) is ground, homogenized and then baked in a rotary furnace.
- as a functional filling material in chemistry (e.g. for heavy polypropylenes for sound insulation in automotive applications and building construction),
- in physical water treatment for the removal of turbidity, algae and other impurities,
- as a dense medium for coal flotation. Finely ground magnetite (<45 µm) mixed with water produces a defined dense medium for the gravity separation of impurities in the introduced coal dust which sink together with the magnetite. The sunken magnetite is then siphoned off at the bottom with the impurities and can easily be recycled because of its magnetic properties. The clean coal dust floats and is collected. After drying it is ready for use.
- as a raw material in the manufacture of pigments for special colors. For this purpose, Fe₂O₃ manufactured by oxidation from Fe₃O₄ is considered to be of higher value than natural hematite.
- as a dense medium for radiation protection construction materials, heavy concrete (underwater concrete, coastal protection) and ballast (e. g. on oil platforms),
- as ballast (counterweights) for bridges, in excavators, ship loading devices, elevators and washing machines,
- as a blasting abrasive,
- as a molding sand for foundry purposes
- as an additive in fertilizers.

10.4 Substitutes

Depending on the usage, magnetite can easily be replaced by other mineral resources; some options are listed below:

- iron ore: all other, but usually lower grade Fe ore,
- cement industry: other iron bearing minerals, copper slag, used foundry sands and dusts,
- dense medium: barite, galenite, clay, quartz powder,

- ballast: barite, hematite, ilmenite, iron granulate,
- blasting abrasives: garnet, staurolite, emery, nepheline syenite, olivine, perlite, pumice, quartz sand, different slags, steel shot, silicon carbide, ilmenite.

10.5 Specifications for the raw materials (reference values)

The minimum content of (titano)magnetite in placers mined mainly for this mineral is 10 % by mass.

(Titano)magnetite for metallurgical purposes should have a low TiO_2 -content and a high Fe-content. Contents of >65 % Fe and <7 % TiO_2 are best. The more unfavorable these values, the more high-quality iron ore has to be added.

Dense magnetite media for coal floatation must have a grain size distribution of 80 % by mass, optimally 90 % by mass <325 mesh (45 μm). The quartz content and the super fine contents should be low and the Fe-content should be high. These specifications ensure the good recyclability of the magnetite.

Tables 10.4 and 10.5 contain data on chemical compositions, grain size distributions as well as the physical parameters of natural and commercial (titano)magnetite concentrates.

Table 10.4: Chemical composition (%) of natural titanomagnetite and magnetite concentrates.

	Titanomagnetite Filmag Inc. La Union Province Luzon, Philippines ¹⁾	Titanomagnetite Yogyakarta Java, Indonesia ²⁾	Titanomagnetite BlueScope Steel Ltd. Waikato North Head North Island, New Zealand ³⁾	Titanomagnetite BlueScope Steel Ltd. Taharoa North Island, New Zealand ³⁾	magnetite "masa" black concentrate Hamidani Mine Yokota, Shimane prefecture, Japan ⁴⁾	Magnetite "akome" reddish concentrate Zakka Mine Nita, Shimane prefecture, Japan ⁴⁾
Fe total	61.61		60.43	60.23	64.31	62.50
FeO	25.21	32.86	30.57	31.1	16.40	22.35
Fe ₂ O ₃	59.44	54.34	52.47	51.6	73.72	64.53
TiO ₂	6.17	6.07	8.00	7.74	1.95	5.20
SiO ₂	2.34		1.00	0.01	3.84	3.14
Al ₂ O ₃	3.03	2.83	3.51	3.1	0.72	1.42
Cr ₂ O ₃	0.031		-	0.023	-	-
MgO	2.40	2.35	3.30	2.8	0.23	0.43
CaO	0.66		0.10	0.52	0.10	0.98
V ₂ O ₅		0.56	0.36	0.57	0.11	0.35
P ₂ O ₅			0.15	-	0.16	0.11
MnO	0.75	0.60	0.59	0.71	-	-
Zn	0.081					
Ni	0.006					
Cu	0.011					
Bi	0.004					
Pb	+					
S	0.098				0.019	0.008

¹⁾ WARD & TOWNER (1985c) ²⁾ MÜCKE (2003) ³⁾ WILLIAMS (1974) ⁴⁾ TSUSUE & ISHIHARA (1975)

Table 10.5: Chemical composition (%) and physical parameters of commercial magnetite-concentrates according to company information.

	65 Grade Magnetite Universal Minerals	UMI 90 Grade Magnetite Universal Minerals	7 Mesh Grade Natural Iron Oxide Universal Minerals	UMI Superfine Magnetite Universal Minerals	UMI Magnetite 96 Grade (A) Universal Minerals	Magnetite Dal Logistics	Magnetite Cleveland-Cliffs Inc.
Chemical composition							
Fe total	70.00	68.2	67.0	68.8	68.0	69.0	67.0
FeO	29.4		26.63				
TiO ₂				0.38	0.38		
SiO ₂	1.65	2.26	3.03	1.844	2.50	2.3	4.5
Al ₂ O ₃	0.31	0.87	0.62	0.40	0.60	0.78	0.3
MgO	0.58	0.42	1.07	0.565		0.55	0.5
CaO	0.20	0.61	0.57	0.637		0.65	0.9
Na ₂ O	0.018		0.132			0.04	
K ₂ O	0.049		0.143	0.080		0.08	
P	0.01		0.020	0.076	0.09		
Mn	0.02		0.019	0.83		0.12	
Zn	0.003					0.02	
Ni						0.02	
Cu	0.006		0.019	0.015	0.03	0.02	
As			+				
Pb	+					0.01	
S	0.138	0.29	0.315		0.05	0.3	
Typical grain size distribution							
<100 mesh (150 µm)	95.0		40.9				
<325 mesh (45 µm)	67.0		7.0			78-85	
>10 µm							
>6.35 µm							
>1.00 µm							
>0.15 µm							
<0.15 µm							
Guaranteed grain size distribution							
<100 mesh (150 µm)	90.0						
<325 mesh (45 µm)	63.0	90.0			96.0		90.0
<400 mesh (38 µm)		86.0					
<10 µm		21.0					
<5 µm				61.38			
Physical parameters							
Magnetic proportion	97.0	97.0		92.4	96.0	92.92	96.0
Humidity	8.0	9.0	4.9	1.0	0.04	9-10	9.5
Apparent density(g/cm ³)	5.07	4.95	4.85	5.0	4.95		5.0
Bulk density			2,900			2,770-2,810	
Spec. surface (cm ² /g)						1,500-1,700	

10.6 Size of deposits and resources

The exact proportion of magnetite in the total iron ore resources and reserves of approx. 330 billion tonnes is not known. The following standard values apply to the size of placer deposits, where solely (titano)magnetite is mined.

	Placers
	mineral content
Small	100,000 – 500,000 tonnes
Medium	500,000 – 1 million tonnes
Large	1 million – 5 million tonnes
Very large	>5 million tonnes

The distribution of the global deposits (reserves and resources) of (titano)magnetite in placers – defined as known and published amounts of currently workable, mineable heavy minerals, not blocked by competitive usages – is shown in Figure 10.2.

10.7 Important producing countries

Titanomagnetite from placers is currently only produced in New Zealand, Indonesia and Japan.

Figures 10.1-10.3 show the percentage distribution of global production (2007) and the currently known global resources of (titano)magnetite in direct comparison.

Global production of titano-magnetite from placers in 2007

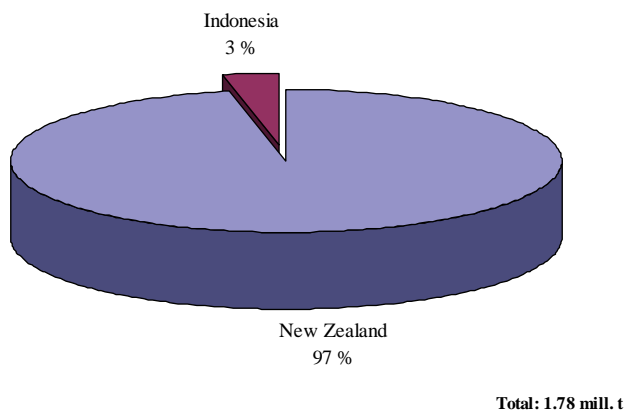


Figure 10.1: Percentage distribution of the global production (2007) of (titano)magnetite from placers.

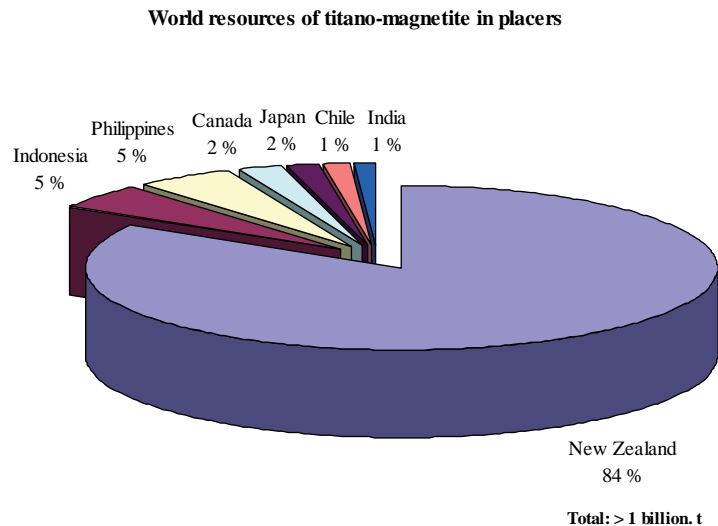


Figure 10.2: Percentage distribution of the currently known global reserves and resources of (titano)magnetite from placers.

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11 Cassiterite

[German: *Cassiterit*, Spanish: *cassiterita*, French: *cassiterite*]

11.1 Mineralogy, petrography, chemistry

Cassiterite, also called tinstone, is a heavy mineral of great economic importance. Pure cassiterite has the following properties:

Formula	SnO ₂
Chemistry	78.8 % by mass Sn, 21.2 % by mass O ₂
Density	7.02 g/cm ³
MOHS hardness	6-7
Color	black, brown-black, yellowish brown, reddish brown
Magnetic properties	nonmagnetic
Electrostatic properties	conductive

In cassiterite, Sn can be replaced by Fe at a ratio of 1:6 as well as by Nb and Ta to a ratio 1:30. Other frequently occurring trace elements are Ti, Li, Sc, Zn, Fe, W and Mn. Up to 1 % of oxygen can be substituted by (OH) groups. Correspondingly, rutile, lepidolite, columbite and tantalite are frequent accompanying minerals of cassiterite. Inclusions in cassiterite include rutile, wolframite, chalcopyrite and pyrite, to a lesser degree also ilmenite, zircon, columbite, sphalerite, pyrrhotine, magnetite and hematite depending on the genesis.

Submicroscopic inclusions of wolframite cause the dark color of the intrinsically transparent cassiterite. Inclusions of columbite in turn cause the strong pleochroism of some pegmatitic cassiterites (BINDE 1986).

11.2 Formation of deposits and occurrences

The element tin is mainly lithophile and is enriched in the residual melt of acid magmas. Many types of granite thus contain tin in the form of the mineral cassiterite. Cassiterite is mainly enriched in the “tin granites”, which contain approx. 15 – 50 g Sn/tonne. Cassiterite also occurs as “mine tin”:

- intra magmatic: in domes of granite intrusions (Saxony Erzgebirge), in feldspar topaz granites (Czech Republic, Saxony Erzgebirge), in rhyolites (Sierra Madre Occidental/Mexico) and titanomagnetite deposits (e.g. Grängesberg/Sweden with ~100 g SnO₂/ton). The more acidic the rock, the higher its Sn-content.
- pegmatitic: Besides cassiterite, Sn-W pegmatites contain the minerals wolframite, columbite, tantalite and lepidolite. Deposits of this type are Bikita/Zimbabwe, Manono-Kitotolo/DR Congo, Greenbushes/Western Australia and Phuket/Thailand.
- pneumatolytic: “greisenization” of granites and their adjacent rock. Known deposits of this type are Altenberg/Saxony Erzgebirge, East Kemptville/Nova Scotia/ Canada, Haad Som Pan/Thailand and Talushan and Chiu Lung Nao/South East China, Blue Tier/Tasmania as well as Zaaipants and Rooiberg/Bushveld complex/Rep. South Africa. Contact metasomatic cassiterite mineralizations are known from the Erzgebirge and Namibia. One of the largest tin deposits in the world developed metasomatically – in dolomites: Renison Bell/Tasmania. Further examples are Cleveland and Mt. Bischoff/Tasmania as well as Dachang and Geiju/China.

- hydrothermally: Highly thermal veins are mineable in numerous cassiterite deposits. Primarily these are the tin ore veins in the Cornish, Saxony and Bohemian deposits (Zinnwald, Geyer, Ehrenfriedersdorf, Schlaggenwald etc.), Herberton/North Queensland/Australia, Aberfoyle/Tasmania/Australia, Sungei Lembing/Pahang/Malaysia as well as Billiton/Indonesia. The individual deposits are mostly small (<4.5 tonnes of ore), but high-grade (0.7-2.3 % Sn). In the Bolivian subvolcanic deposit province, mainly sulfidic tin lodes with the valuable minerals cylindrite ($\text{Pb}_3\text{Sn}_4\text{FeSb}_2\text{S}_{14}$), frankeite ($\text{Pb}_5\text{Sn}_3\text{Sb}_2\text{S}_{11}$), stannite ($\text{Cu}_2\text{FeSnS}_4$) and teallite (PbSnS_2) occur.

Because cassiterite is very resistant to all weathering influences, is it a typical placer mineral. “Placer tin” was produced a long time before “mine tin” and traded long distances. Important placer tin deposits include:

- **Bolivia:** Since 1924, mining of alluvial placers of the Lipez Huayco and Antequera for cassiterite, accompanying minerals are hematite, magnetite, ilmenite and tourmaline, average depths of the mineable areas are 27.5 m, and the overburden 3.5 m,
- **Brazil:** Small-scale mining of alluvial placers of the Rio Abaixo in Minas Gerais; fossil alluvial cassiterite placers in Rodonia, mineable thickness of the gravel sands 1-6 m, average content 0.3-1.2 kg Sn/m³, Holocene fluvial placers have significantly smaller thicknesses, but higher contents,
- **Burma:** Fluvial placers of the Tenasserim range with cassiterite, wolframite, bismuthite, magnetite and pyrite,
- **China:** Province of Hunan, Changning region: placers near Baisha, Xiling and Zoujiaqiao with an average content of 650 g cassiterite /m³; Province Guangxi, Hexian region: Shuiyanba placer currently being mined with 0.053 % Sn,
- **Mongolia:** Eluvial and in particular fluvial placers with cassiterite, wolframite and other heavy minerals (mainly garnet and zircon) in the Modot Valley and at Khujkhaan near Tsernhermandel, Khentii Aimag, in Eastern Mongolia. Production with interruptions since 1949. Up to now cassiterite concentrate with approx. 10,000 tonnes Sn capacity has been mined. The tin content of the placers varies between 170 g/m³ and more than 700 g/m³ (frequently up to 450 g/m³),
- **Nigeria:** Tertiary to Holocene alluvial and eluvial placers on the Jos Plateau in the provinces of Bauchi, Benue, Kano and Zaria with cassiterite and columbite, secondary gangue mineral wolframite, tantalite and thorite (ANONYMOUS 1981), extractable reserves 1977: 140 kilotonnes cassiterite,
- **Russia:** North-eastern Yakutia and Chukotka, e.g. eluvial-alluvial placers on the Pyrkakaivaam river (contents up to 6 kg cassiterite /m³) and Krutoi brook (up to 17.3 kg cassiterite /m³). Accompanying minerals are wolframite, gold, magnetite, pyrite, arsenopyrite, hematite, garnet, topaz, zircon, fluorite and anatase. Only the cassiterite and the gold are economic.
- **South-East Asian tin belt:** nearly 3,000 km long from Moulmein District in Northern Burma via Thailand and Malaysia to the Indonesian island of Belitung. Mine tin in coarse-grained biotite granites and secondary fine to medium grained two-mica granite. Age: Carboniferous to Lower Tertiary. Tin placers of colluvial, eluvial origin (average thickness 1-2 m) and most of all alluvial placers (thickness dm->100 m) onshore (2/3), as well as offshore (1/3) in Western Malaysia, Thailand and Indonesia (cf. Table 11.1). Mining by gravel pump “dry mining” using high-pressure water jets), floating dredge (ladder dredger or suction dredger) and quarrying (e.g. Hong Fat mine in the Province of Selangor/Malaysia, 152 m depth) as well as secondary underground mining and artisanal mining. Estimated resources, 6 million tonnes Sn – more than the total resources of the rest of the world. The heavy-mineral concentrate from which cassiterite has already been separated is called “amang”, and is processed further to extract additional valuable minerals (cf. Chapter 11.6, Tables 5.6, 11.2-11.4).

Table 11.1: Tin contents of placers mined in Indonesia and Malaysia according to SCHMIDT (1976) and LAHNER (1982).

Country	Mining region	Range (g Sn/m ³ Sediment)	Average (g Sn/m ³ Sediment)	Comment
Indonesia	Bangka	400-1,400	750	
	Belitung	190-900	400	
	Singkep	-	250	cassiterite contains 1 % Nb/Ta
	Sumatra	-	260	
Malaysia	Kinta near Ipoh	150-350	-	
	Berjuntai	130-160	-	
	Sungei Besi	76 – 15,310	-	

Table 11.2: Distribution of valuable minerals (% by mass) in the raw sand in tin placers of Thailand according to PRADITWAN (1985, 1988b, c, 1989a).

Mineral	Phuket Island		Central Thailand		Southern Thailand		Northern Thailand	
	range	mean value	range	mean value	range	mean value	range	mean value
Cassiterite	15.24-89.15	62.16	1.4-93.7	66.7	6.1-90.8	52.7	24.6-81.9	71.52
Columbite-tantalite	0.08-0.83	0.24	0.1-3.4	0.9	0.2-2.5	1.1	-	
N-Ta rutile	0.01-6.09	0.34	+ -3.3	0.4	0.1-5.0	1.0	-	
Wolframite	0.01-0.52	0.18	0.1-0.3	0.2	0.1-1.1	0.3	0.7-36.8	0.8
Scheelite	-		+ -1.6	0.3	-		0.1	0.1
Monazite	0.32-5.01	1.5	+ -11.8	0.9	0.1-3.4	0.6	0.9-20.4	9.5
Ilmenite	0.37-51.14	10.7	0.5-49.6	6.5	0.1-38.6	4.0	0.3-3.5	2.5
Leucoxene	-		+ -41.0	0.1	-		-	
Rutile	-		+ -5.6	0.3	-		0.1-0.6	0.3
Anatase	-		-		-		13.5	13.5
Zircon	-		-		0.1-3.5	0.7	0.1-3.2	1.0
Sphalerite	-		-		0.1	0.1	-	
Galenite	-		-		0.1	0.1	-	

+ = traces

Table 11.3: Mineralogical composition (% by mass) of heavy mineral pre-concentrates from the tin belt of South-East Asia.

	Indonesia				Thailand ²⁾				
	Banka, Muntok District ¹⁾	Belitung ³⁾	Chiang Mai	Phangnga	Phuket	Trang	Kanchanaburi		
	Mine # 4	Mine # 2	(avg. of n=4)	Omkoi (avg. of n=6)	Chaiyayuth Dredge	Tantikovit Mine	Loom Seng Mine	Patana Mine	
Cassiterite	36	60	67.1	62.2	34.3	49.6	34.3	24.2	
Columbite-tantalite				3.9	1.4	0.3	1.8	0.3	
Struverite					0.9	+	1.8	0.1	
Samarskite				0.1			0.1		
Nb-Ta rutile				0.3					
Rutile	+	+	0.4	0.3	0.4	+	0.8	0.7	
Anatase	1	+		2.2	+		0.5	0.3	
Brookite	+	1							
Ilmenite(+leucoxene)	31	11	0.6	1.8	23.0	0.4	9.2	23.3	
Wolframite				0.3					
Scheelite				0.4	+			+	
Zircon	3	3	4.9	1.0	1.6		0.7	2.2 ⁵⁾	
Monazite	19	22	0.6	8.5	7.7	0.1	0.7	0.8	
Xenotime	5	1		+					
Magnetite			0.1		0.1			+	
Martite			0.1						
Pyrite (+marcasite)			7.0	0.1	2.2	0.1	+		
Arsenopyrite						0.9	+	+	
Fe-(Hydr)oxide ⁴⁾			4.6	0.9	8.8	1.0	2.3	13.0	
Mn-oxides			+	0.4	0.1	13.1		0.5	
Garnet			0.1	4.4	13.7	19.6	11.0	26.9	
Tourmaline	4	2	0.1	1.1	0.3	1.9	20.6	3.6	
Topaz			0.8		2.4				
Spinel					0.1		1.6	0.4	
Beryl				0.1					
Corundum	+	-							
Ankerite	+	+							
Quartz	1	+	13.1	12.3	3.6	13.0	14.7	3.6	

¹⁾ BODENHAUSEN (1954) ²⁾ PRADITWAN (1988a) ³⁾ PRADITWAN (1989b) ⁴⁾ lepidocrocite, goethite, hematite, limonite ⁵⁾ plus 0.2 % allanite + = traces

11.3 Applications

Cassiterite is mainly used for extracting the metal **tin** and is almost the only tin mineral of economic importance. Tin is silvery white, very inert because of the oxidation layer, and thus singularly suitable for protection against corrosion (tin coating of sheet metal = tinfoil). Tin is also used for:

- alloys: bronze, bearing metals, type metal, German silver,
- tin casting (tin figures) consisting of 60-70 % Sn and 30 – 40 % Pb,
- soldering tin,
- opaque white glazes,
- organotin compounds (tributyltin) as toxic protective paints, e. g. in shipbuilding,
- catalytic converters,
- as a stabilizer in PVC,
- together with Ca for strengthening the Pb-electrodes in batteries,
- window pane production by casting on liquid tin,
- tin halogenides as stabilizers in perfumes and soap (SnCl_4) or as a main component in toothpaste (SnF_2),

11.4 Substitutes

Cassiterite is the most frequent and economically most important Sn mineral and can only be substituted to a limited extent by sulfidic Sn minerals. In all of its metallic applications, it is very difficult to substitute tin, and then only with a deterioration in quality. Depending on the usage, tin can be replaced by other mineral resources or elements; some options are listed below:

- glazes: marble powder, white lead, zinc white, lithopone, titanium dioxide,
- toxic protective paint: cuprous or silicon paint,
- stabilizers in PVC: Ca-Zn compounds.

11.5 Specifications for the raw materials (reference values)

The references give very different minimum contents (cut-off grades) for the mineability of cassiterite deposits:

- deep mining: >1.0 % Sn
- mine tin: >0.5 % Sn or Sn+W
- placer tin: >0.01 % Sn (typical: 0.05 % Sn = 500 g Sn/ton)
- placer mining with floating dredge: 60 g cassiterite/m³
- placer mining using gravel pumps: 120 g cassiterite/m³
- large-scale industrial mining: 240 g cassiterite/m³
- small-scale mining: 300 g cassiterite + columbotantalite/m³

There are no specifications for commercial concentrates of cassiterite because cassiterite is always smelted directly – depending on the concentration – without further treatment.

Table 11.4: Chemical and mineralogical compositions (% by mass) of different concentrates from Malaysia accruing in the course of cassiterite production and processing.

	Excavator pre-concentrate Berjuntai, Selangor ¹⁾	Smelting pre-concentrate average ¹⁾	Cassiterite-concentrate ²⁾	Amang Ipoh, Perak ²⁾
Chemical composition				
SnO ₂		95.54	81.37	0.58
(Zr,Hf)O ₂			0.03	0.51
TiO ₂		0.31	3.08	40.57
SiO ₂			2.99	18.97
Al ₂ O ₃			1.58	0.90
Fe ₂ O ₃		0.74	8.10	35.93
MnO			0.21	2.30
MgO			0.12	0.05
CaO			0.05	0.02
V ₂ O ₅			0.02	0.06
Nb ₂ O ₅			0.31	0.16
P ₂ O ₅			0.02	0.10
SO ₃		0.15	<0.01	0.03
As		200 ppm	85 ppm	26 ppm
Bi		40 ppm	52 ppm	13 ppm
Ce			49 ppm	459 ppm
Cr			110 ppm	242 ppm
Cu		60 ppm	222 ppm	22 ppm
La			<20 ppm	252 ppm
Nd			<50 ppm	1111 ppm
Pb		240 ppm	84 ppm	110 ppm
Ta		<1,000 ppm	778 ppm	143 ppm
Th			<5 ppm	104 ppm
U			440 ppm	39 ppm
W			4694 ppm	118 ppm
Zn			122 ppm	677 ppm
LOI			0.36	-0.96
Mineralogical composition				
Cassiterite	9.7		85.0	
Ilmenite	41.5		9.5	80.7
Rutile	0.5			0.7
Zircon	2.1			12.5
Monazite	2.0			
Staurolite			+	
Garnet			+	
Tourmaline	0.8			+
Hornblende			+	
Quartz	43.4		5.4	6.1

¹⁾ LAHNER (1982) ²⁾ BGR-analysis

11.6 Size of deposits and resources

These reference values apply to the evaluation of the size of cassiterite deposits according to SUTPHIN et al. (1990).

	All deposits Sn-content
Small	<10,000 tonnes
Medium	10,000 – 100,000 tonnes
Large	100,000 – 500,000 tonnes
Very large	>500,000 tonnes

The estimated global reserves and resources of tin are shown in Figure 11.1.

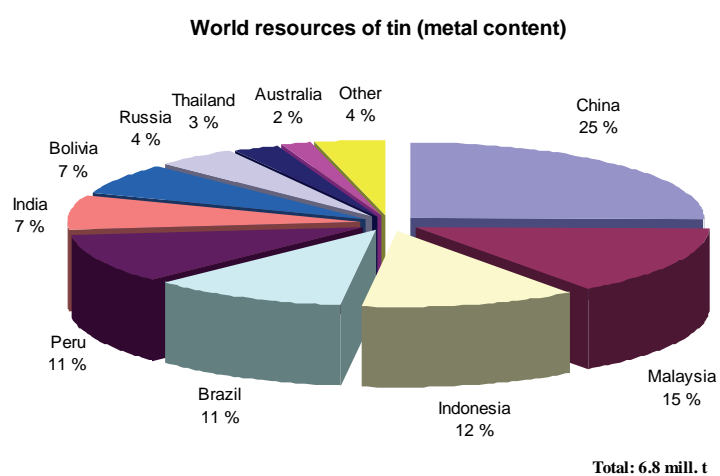


Figure 11.1: Percentage distribution of the known global reserves and resources of tin (metal contents).

11.7 Important producing countries

Figure 11.2 shows the distribution of the global mine production of tin (metal content) from cassiterite and sulfidic Sn-ore in 2007.

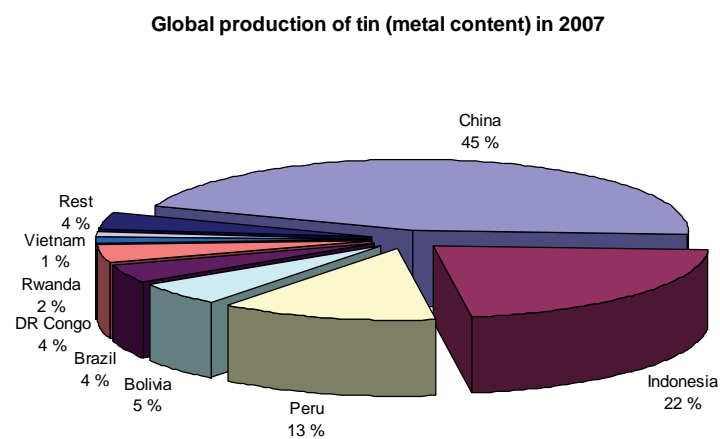


Figure 11.2: Percentage distribution of the global production of tin (metal contents) in 2007.

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Industrial Minerals / The Economics of Fluorspar, 10th edition, 2009

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The Economics of Fluorspar, 10th edition, 2009

World production of fluorspar reached 5.5Mt in 2008, thus nearing the peak of fluorspar production of 1989, before the adverse impact of the Montreal protocol on the use of fluorocarbons had worked its way through to the supply side. In the years from 1989 to 1994, production declined to 3.5Mt, in the following eight years production recovered, but slowly, and then increased by almost 5%py from 2003 to reach 5.45Mt in 2007. Most of this growth came from increased output of acidspar in China and Mexico.

Report highlights

- Five countries, China, Mexico, Mongolia, South Africa and Russia, accounted for 86% of world production in 2007. The role of the leading producer, China, has changed significantly over the last 20 years. Chinese exports of fluorspar began to increase from the mid 1980s and by the early 1990s, China was established as the major source of supply for many countries. Mine production elsewhere declined in response to Chinese low prices and diminishing consumption in fluorochemicals in the main industrialised countries. Following a period in the mid to late 1990s when fluorspar producers elsewhere sought to protect their markets by means of anti-dumping measures, the Chinese government introduced a number of actions, including export quotas and taxes, which significantly reduced supply of fluorspar to the international market.
- The diminishing availability of fluorspar from China and strong growth in demand from 2002 to mid 2008 has led to a number of proposed new mining operations. Of the total identified potential output of 1.4Mt, about 0.7Mt has acquired significant funding. The remaining projects are either at a pre-feasibility or bankable feasibility stage.

- Over the past decade as a whole, demand for fluorspar has grown by just over 2.2%py but in the years from 2003 to 2007, growth rates exceeded 4%py as the industry started to recover from the effects of restrictions on the use of some fluorocarbons. Of the 5.5Mt of fluorspar consumed in 2007, an estimated 50% was used in the manufacture of chemicals, 24% in iron and steel and 17% in the manufacture of aluminium. Growth has been associated with production of chemicals and aluminium fluoride, both of which are derived from acidspar. This has led to some tightness in supply of acidspar, as production has grown by 6.6%py since 2002 while consumption has grown by 7.4%py over the same period.

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Geology and Genesis of the F-Sn-W(-Be-Zn) Skarn (Wrigglite) at Moina, Tasmania

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Abstract

The Moina skarn deposit, with its associated Sn-W-F veins and greisen, occurs at the margin of the Dolcoath leucogranite. The skarn occurs as a thick horizontal plate approximately one km in its longest dimension and up to 100 m thick and is separated from the granite's upper near-horizontal contact by approximately 200 m of the Moina sandstone. The necessary plumbing system for access of mineralizing fluids is probably a series of east-west-trending tension fractures, now Sn-W quartz veins, associated with a major northwest-southeast-trending fault known as the Bismuth Creek fault. Emplacement of the granite was at shallow depths (<3 km?).

The skarn unit section consists of: (a) a granular garnet-pyroxene-vesuvianite-fluorite skarn; (b) the main skarn ("wrigglite") consisting of fluorite-magnetite-vesuvianite (cassiterite-scheelite-adularia) and having a characteristic fine-grained, rhythmic, finely layered contorted structure; (c) a granular, pale green pyroxene skarn which occurs as thin units (<5 cm) within and near the base of unit (b) above; (d) a wollastonite-rich skarn (>80 vol % wollastonite); and (e) a granular garnet-pyroxene-vesuvianite-fluorite skarn overlying the other units. Unit (e) is relatively enriched in boron (≈ 600 ppm).

The skarn unit carries up to 25 weight percent F; 0.6 percent Sn, 0.5 percent W, 0.2 percent Be, 27.5 percent Zn, and 4.5 ppm Au. Sn, Be, and Fe values increase toward the upper part of the skarn sequence whereas Zn, Cu, and Mo values are erratic. Secondary Zn-Cu-In-Cd-Au sulfide-amphibole alteration of the primary F-Sn-Be oxide skarn is related to the Bismuth Creek fault. When the primary wrigglite skarn is altered, Sn is largely lost from that part of the skarn.

Introduction

THE Moina skarn deposit is located 40 km southwest of Devonport, Tasmania, near the margin of the Devonian Dolcoath granite. Quartz lodes (veins) which cut the skarn and underlying quartzite were mined for Sn and W until 1956 (Jennings, 1965), but only recently has an investigation of the Fe-F-Sn-W-Be-Zn-Au skarns been reported (Askings, 1978). Apart from studies by Sainsbury (1964, 1969) and Jahns (1944a, b), little has been published on this type of skarn in the western literature, although much has been published in the USSR (e.g., Govorov, 1958, Getmanskaya, 1972; Miroshnichenko and Gulyayev, 1978).

In addition to being of obvious potential economic interest, the relationship of these Sn-bearing oxide skarns to Sn-bearing sulfide replacement deposits such as the Renison Bell deposit (Patterson, 1976) needs more study. The purpose of the present study has

been to determine the mineralogy, paragenesis, and bulk chemical relationships of the skarn and related rocks; the nature of the mineralizing fluids; and the distribution of economic elements in the skarn. With this information, a genetic model has been suggested and relations to Sn-sulfide replacement deposits inferred.

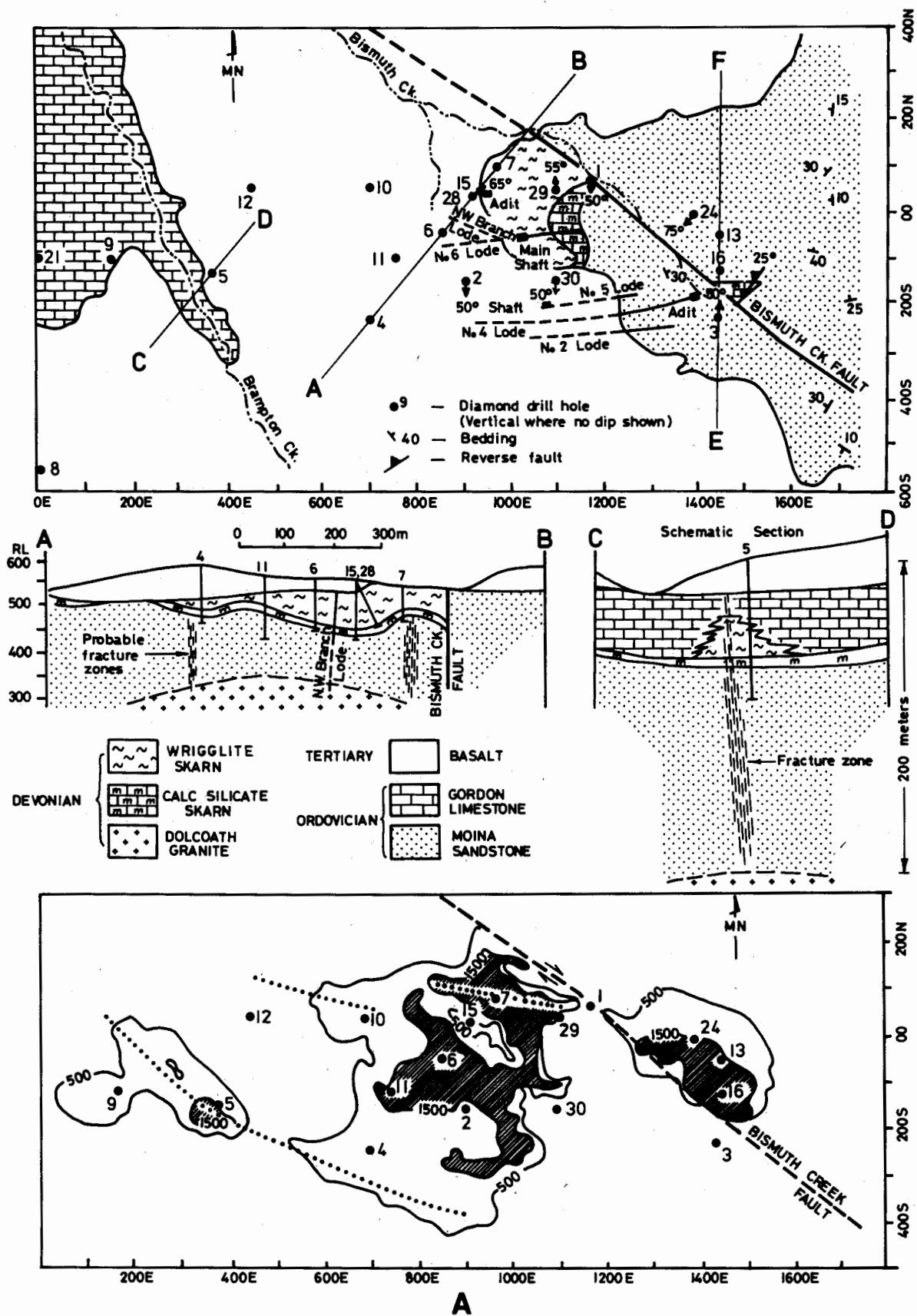
Geology, Petrology, and Mineralogy

General

At Moina F-Sn-W-Be(-Zn) mineralization occurs as skarn, in veins, and in greisen (Fig. 1A). The skarns are a replacement of essentially flat-lying Ordovician limestone and calcareous siltstone (the Gordon Limestone) which conformably overlies quartz sandstone and siltstone (the Moina sandstone).

Mineralization is associated with a Devonian leucocratic granite (Dolcoath granite) which is never in contact with the limestone or skarn but is separated by about 200 m of Moina sandstone. The most intense mineralization occurs in a folded and fractured zone

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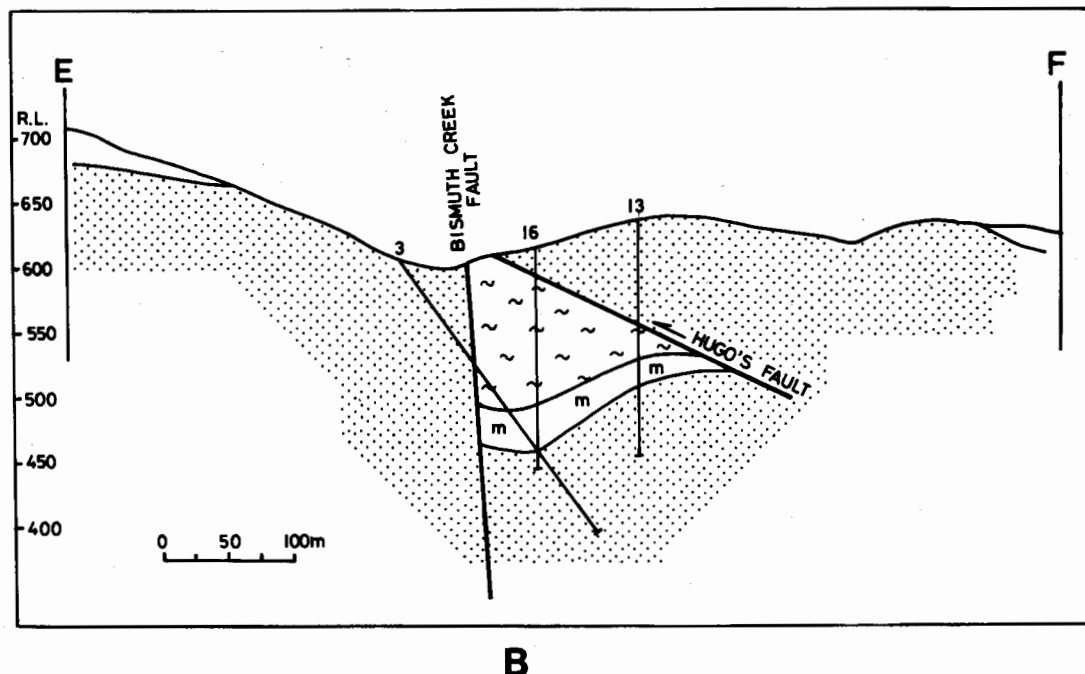


FIG. 1. B. Section E-F is an enlarged section showing skarn occurring beneath a reverse fault.

associated with a major fault. Tertiary basalt covers much of the area.

Granite

Granite has been intersected at a depth of approximately 200 m in diamond drill holes ML-1 and ML-2. Greisenization in the granite in ML-1 decreases downward from its contact at 279 to 317 m (distances down a hole of 50° dip) and consists of a grayish-white-colored zone nearest the contact (zone 1: 279 to 280.8 m), a gray zone (zone 2: 281.8 to 281.8 m), a green zone (zone 3: 281.8 to 292.7 m) and a pinkish-yellow zone (zone 4: 292.7 to 317 m) farthest from the contact. On the basis of color differences, the contacts between zones 1 and 2, and 2 and 3 are abrupt—over 5 cm—whereas that from zone 3 to 4 is less abrupt. Small areas of pinkish (zone 4) greisen appear in zone 3 and greenish (zone 3) greisen in zone 4. This indicates that irregularities of permeability may have existed and may account for minor chemical variations.

In zone 4, the least-altered samples consist of approximately 60 percent quartz, 20 percent white mica, <1 percent brown biotite, 20 percent orthoclase, 8 percent oligoclase, and accessory fluorite, py-

rite, rutile, and hematite. By comparison, unaltered Dolcoath granite cropping out in the Forth River Valley 4 km east has an average composition of 35 percent quartz, 5 percent biotite, 40 percent K-feldspar, and 20 percent plagioclase (Gee, 1966), with accessory fluorite and zircon (Jennings, 1963). In zone 4, primary dark brown biotite occurs only as armored relicts in quartz. Fe calculated as Fe^{+2} is 4.451 atoms per molecule (24 oxygens per unit cell), $F = 1.126$, $\text{Ti} = 0.431$, and $\text{Cl} = 0.106$ with $\text{Fe}^{+2}/\text{Fe}^{+2} + \text{Mg} = 91.5$ (analysis 8, Table 1).

In zones 3 and 4, two types of white mica coexist having distinctly different compositions. Type 1 is coarse grained with high values of F (0.807 atoms/molecule), $\text{Fe}(0.846)$, and $\text{Mg}(0.123)$ and with relatively low Si and high Al for muscovite (analysis 4, Table 1). Type 2 in zone 4 occurs as fine-grained masses with low values of F (0.215), $\text{Fe}(0.415)$, $\text{Mg}(0.075)$, and high Si with lower Al than type 1 (analysis 6, Table 1). The $\text{Fe}^{+2}/\text{Fe}^{+2} + \text{Mg}$ ratio of type 1 is higher than that of type 2 (87.3 vs 84.7). Texturally, type 2 commonly occurs between type 1 mica and unreplaced feldspar relicts. The modal ratio of type 1/type 2 decreases upward until no type 2 occurs in zones 1 and 2. This and the relatively great

FIG. 1. A. Geologic map, cross section, and magnetic anomaly map of the Moina Laminar skarn area. The geologic map shows the location of diamond drill holes. The position of shafts and cassiterite-wolframite-quartz lodes of the Murphy and Shepherd mine are shown. Section A-B is normal to fold trends and to the Bismuth Creek fault and is drawn to scale. Section C-D is schematic showing the probable nature of replacement of the limestone in fracture zones. The magnetic anomaly map is simplified fluxgate data (vertical component of the field) in nanoteslas and shows northwest-southeast and east-west trends.

TABLE 1. Electron Microprobe Analysis of Minerals Present in Greisenized Granite and Quartz Veins

	1	2	3	4	5	6	7	8	9	10	11
SiO ₂	44.57	45.25	45.27	44.91	47.92	47.97	50.07	32.60	0.35	1.39	31.68
TiO ₂	0.37	0.05	0.04	0.21	0.05	0.03	0.13	3.20	0.50	90.92	0.01
Al ₂ O ₃	28.64	29.99	30.44	29.74	31.96	31.37	28.96	12.87	0.32	1.33	53.18
FeO	6.85	6.63	6.31	6.73	2.73	3.36	1.14	29.73	64.54	4.49	0.03
MnO	0.46	0.22	0.23	0.21	0.09	0.07	0.06	1.65	3.49	0.07	0.03
MgO	0.50	0.22	0.10	0.55	0.28	0.34	1.91	1.56	0.43	0.02	0.02
CaO	0.00	0.00	0.00	0.00	0.27	0.05	0.00	0.00	0.40	0.09	0.00
Na ₂ O	0.35	0.26	0.22	0.30	0.09	0.16	0.04	0.01	0.13	0.15	0.00
K ₂ O	10.22	10.19	10.01	10.21	9.24	9.22	10.58	9.11	0.01	0.29	0.02
F	1.90	1.55	1.02	1.70	0.30	0.46	1.38	1.99	0.05	0.09	4.07
Cl	0.02	0.05	0.05	0.02	0.11	0.04	0.01	0.35	0.04	0.07	0.01
Anhydrous total	93.88	94.41	93.69	93.58	93.04	93.07	94.40	93.15	69.76	98.94	96.05
Number of O ions	24	24	24	24	24	24	24	24	9	4	24
Si	6.680	6.742	6.793	6.745	7.072	7.087	7.214	5.836	0.040	0.038	3.949
Ti	0.043	0.006	0.005	0.024	0.006	0.004	0.014	0.431	0.000	1.866	0.001
Al ^{iv}	1.120	1.258	1.207	1.255	0.928	0.913	0.786	2.164	0.044	0.064	0.051
Al ^{vi}	3.941	4.010	4.178	3.833	4.623	4.551	4.133	0.552			7.764
Fe ⁺²	0.859	0.826	0.792	0.846	0.337	0.415	0.137	4.451	5.539 ¹	0.102	0.003
Mn	0.059	0.028	0.029	0.027	0.012	0.009	0.007	0.251	0.343	0.002	0.003
Mg	0.112	0.030	0.023	0.123	0.061	0.075	0.410	0.416	0.008	0.001	0.004
Ca	0.000	0.000	0.000	0.000	0.043	0.008	0.000	0.000	0.005	0.003	0.000
Na	0.102	1.937	0.063	0.087	0.027	0.046	0.011	0.000	0.029	0.008	0.000
K	1.954	1.937	1.916	1.956	1.740	1.738	1.947	2.081	0.001	0.005	0.004
F	0.901	0.731	0.484	0.807	0.140	0.215	0.629	1.126	0.018	0.008	4.364
Cl	0.005	0.013	0.013	0.005	0.027	0.010	0.002	0.106	0.007	0.003	0.002

¹ Fe⁺³ not Fe⁺²

1, type 1: white mica, average of 2, zone 1; 2, type 2: white mica, average of 2, zone 2; 3, type 1: white mica, average of 5, zone 3; 4, type 1: white mica, average of 8, zone 4; 5, type 2: white mica, average of 5, zone 3; 6, type 2: white mica, average of 8, zone 4; 7, white mica, alteration of 11; 8, Fe biotite, average of 3, zone 4; 9, Fe oxide (clot), zone 4 (rock 815, Table 2); 10, rutile, zone 4 (rock 862, Table 2); 11, topaz, in cassiterite-wolframite quartz veinlet. Chemical compositions were determined by means of a Joel JXA-5A electron microprobe with computer control, located at the Department of Geology, University of Melbourne, using a beam current of 0.1 mA and an accelerating potential of 15 kV. The following standards were used: SiO₂, wollastonite and quartz; Al₂O₃, corundum; TiO₂, rutile; Fe total and Ba, anandite; K₂O, potassium tantalite; Mn, Mn metal; Ni, Ni metal; F, fluorite; Cl, halite; and Na, jadeite. The computer program for the reduction of electron microprobe data was written by Mason et al. (1969) with modifications by A. K. Ferguson.

depth suggest that type 2 is not related to any surface alteration process. The occurrence of two coexisting micas may be due to the solid solution of another component such as Li. The Li content of type 1 (sample 799) is 770 ppm, but coexisting types 1 and 2 could not be separated. The Li content of the rocks is, however, relatively low (<700 ppm) (Table 2). The lower F content of type 2 may indicate that this mica formed late in the paragenesis at the expense of unreacted feldspar. Related Mn-rich hematite "clots" (analysis 10, Table 1) and fluorite may represent Fe-biotite relicts.

In zone 3, hematite clots are largely absent and only minor relicts of feldspar remain associated with type 2 mica clots. Also, Fe-biotite armored relicts are absent.

Zone 2 is probably a greisen vein which may originally have been a fault or shear zone prior to and/or during greisenization. It consists largely of coarse-grained type 1 mica (see analysis 2, Table 1), fluorite,

quartz, pyrite, minor amounts of rutile (analysis 10, Table 1), and minute cassiterite grains. Throughout the zone there are quartz crystal-filled vugs suggesting high porosity (and permeability?) in the zone.

Zone 1 consists largely of quartz with type 1 mica (analysis 1, Table 1), fluorite, pyrite, and very minor amounts of sphalerite.

Above the granite, quartzite has been greisenized and some greisenized granitic dikes also occur.

The bulk chemistry and densities of 16 samples of the greisenized granite collected at regular intervals are shown in Table 2 with the average composition for each zone. For true comparisons to be made, these rocks should have the same densities. The densities of the samples in Table 2 do vary (e.g., 784 = 2.544 g/cm³; 780 = 2.826 g/cm³) and, accordingly, for comparison purposes, molar quantities per 100 cm³ volume of rock were calculated (Fig. 2). As can be seen, Na₂O and Sr decrease systematically toward the contact representing, in part, the progressive break-

down of plagioclase. K_2O , CaO , Al_2O_3 , Fe_2O_3 , FeO , TiO_2 , H_2O^+ , Zn , Cl , Rb , Sn , Ga , Li , Yb , Cu , and to a lesser extent B are all enriched in zone 2. These enrichments reflect the large abundances of muscovite, fluorite, and cassiterite with some base metal sulfides. SiO_2 , Zr , Th , U , Y , and Pb decrease in zone 2. MnO increases toward the contact. The inverse trends of Ti and Zr probably indicate that one ($Ti?$) or both of these elements are mobile in this environment.

Sedimentary rocks

The (meta-) sedimentary rocks of the area consist of the Ordovician Moina Sandstone (intertidal facies?), conformably overlain by Ordovician Gordon Limestone (intertidal to subtidal facies).

The Moina Sandstone is a well-bedded sequence of pale quartz sandstones and quartz siltstones with rare pebbly beds and rare shale. At the top of the sequence more finely grained silty beds predominate over sandy beds. Close to the granite, the sandstone has been recrystallized to a dense compact coarse-grained quartzite. For about 10 m below the skarn the unit consists dominantly of highly fractured dense gray fine-grained quartzite consisting of quartz \pm biotite \pm chlorite \pm magnetite \pm clinopyroxene \pm amphibole \pm fluorite \pm calcite. The clinopyroxene is commonly partly altered to amphibole. Unaltered Gordon Limestone is dominantly a fine-grained, compact, pale to dark gray mudstone with well-defined bedding which was slightly disturbed, apparently by burrowing organisms. Particularly toward the base there are thin beds of silty or dolomitic limestone. When unaltered, the bottom transition to the Moina sandstone consists of about 10 m of limestone containing 0.5- to 10-cm-wide interbeds of siltstone, calcareous siltstone, and limestone. The bulk of the limestone is nondolomitic and contains less than 3 percent MgO .

Fracturing and veining

Fracturing and veining of skarn and quartzite is most intense near the contact between quartzite and skarn (Fig. 4A). Alteration of the massive quartzite has occurred out from fractures which are now filled with vein material. Early formed veins were intersected by later ones having different mineralogies and/or abundances of minerals. Thus, for a sample from drill hole SMD 12 at 62 m (see Fig. 1A for location), the general sequence of vein types is amphibole-biotite-muscovite, fluorite-adularia-scheelite, adularia-fluorite-biotite, and calcite-amphibole (Fig. 3B). Only the fluorite-rich fractures, which constitute the largest number, invariably have a magnetite-diopside selvage separated from the vein by granular diopside (see Fig. 4A). Analyses 8 and 9 (Table 4) are

of fractured and veined quartzite. On a 100 cm^3 basis it can be shown that relative to unveined quartzite (see 5 and 6, Table 5 and Fig. 7) the rocks are enriched in Be , Cl , Li , Cu , Zn , Sn , Sr , Rb , Th , W , Ca , Fe^{+2} , Fe^{+3} , K , F , Al , and Mn , with a loss of Si and Y . The fact that the most intense fracturing occurs nearest the lower contact and throughout the skarn but not nearer the granite suggests that the fracturing is produced by a skarn-forming process and is not directly related to the intrusion of the granite. The sequence of vein fillings reflects changing conditions (composition of the fluids, temperature, etc.) during skarn genesis as will be discussed later.

Quartz veins ("lodes" in Fig. 1A) from 0.5 mm to 1 m wide, generally striking east-west and dipping steeply north, are found throughout the quartzite and the lower part of the skarn unit. The widest and greatest abundance of these occur near the skarn-quartzite boundary (see Fig. 1A), although they are reported to have been mined to a maximum depth of 145 m into the underlying quartzite (Jennings, 1965). They were mined for Sn and lesser W until 1956 and consist of quartz containing wolframite, cassiterite, bismuthinite, native bismuth, sphalerite, molybdenite, chalcopyrite, pyrite, scheelite, arsenopyrite, galena, fluorite, topaz, beryl, phlogopite, muscovite, chlorite, and laumontite. Topaz from a cassiterite-wolframite-quartz vein (coordinates 1700E 100S, Fig. 1A) has a high F content (analysis 11, Table 1) and contains high-temperature (430°C , uncorrected for pressure), saline fluid inclusions as will be discussed. In the veins, the $Sn:W$ ratio decreases from 20:12 near the surface (Williams, 1958) to 10:40 at depth (Reid, 1971).

Skarns

Skarn here refers to all calc-silicate-bearing rocks derived from Ca -rich sedimentary rocks. The generalized section consists of a lower calc-silicate skarn overlain by a wrigglite (magnetite-fluorite) skarn. The total skarn unit extends over more than 1 km in its longest dimension and is up to 100 m thick (Fig. 1A). The major magnetic anomalies shown in Figure 1A reflect the major skarn areas, but skarn thicknesses of up to at least 20 m exist below some areas of low magnetic relief (e.g., in drill hole SMD 12). Fortunately the overlying Tertiary basalt is relatively non-magnetic, but there are pyrrhotite-rich and sphalerite-rich areas of as yet unknown extent which could complicate a simple interpretation of the magnetic picture.

The magnetic picture shows clearly a 300-m lateral displacement of the skarn along the Bismuth Creek fault, though some of this may actually be an apparent lateral movement caused by vertical displacement of shallowly dipping beds. Elsewhere in the

TABLE 2. Compositions of Greisenized Granite and Average of Zones in the Greisen

Sample no.	774	776	779	780	784	788	795	809	812
SiO ₂	76.16	73.51	62.64	44.23	74.15	76.96	76.12	76.94	75.52
TiO ₂	0.04	0.03	0.05	0.10	0.04	0.04	0.04	0.04	0.04
Al ₂ O ₃	11.21	11.43	18.99	27.86	10.98	12.70	12.86	12.37	12.28
Fe ₂ O ₃	0.00	0.19	1.36	2.15	1.10	0.42	0.70	0.91	0.49
FeO	4.63	4.37	3.78	5.08	4.35	1.12	1.09	0.77	1.54
MnO	0.10	0.17	0.19	0.18	0.26	0.04	0.03	0.03	0.06
MgO	0.10	0.15	0.10	0.35	0.16	0.12	0.14	0.14	0.13
CaO	0.01	1.29	1.68	2.22	0.51	0.41	0.62	0.30	0.46
Na ₂ O	0.23	0.22	0.30	0.45	0.20	1.38	0.48	0.56	2.03
K ₂ O	3.68	3.98	6.50	9.61	3.67	4.95	4.52	4.90	5.10
P ₂ O ₅	0.00	0.00	0.00	0.02	0.00	0.05	0.01	0.00	0.00
S	1.17	0.56	0.00	0.54	0.02	0.05	0.21	0.09	0.06
H ₂ O ⁺	1.62	1.34	1.56	2.40	0.34	1.76	1.84	1.30	1.08
H ₂ O ⁻	0.26	0.15	0.11	0.25	0.35	0.38	0.46	0.93	0.35
CO ₂	2.53	3.17	1.58	0.99	3.11	0.37	0.39	0.17	0.93
Total	101.74	100.54	98.84	96.70	99.24	100.76	99.51	99.45	100.07
-S = 0	0.59	0.28	0.00	0.27	0.01	0.03	0.10	0.04	0.03
Total	101.16	100.26	98.94	96.42	99.23	100.73	99.41	99.41	100.04
+F	0.34	1.36	2.27	3.43	0.51	0.46	0.60	0.29	0.41
Total	101.50	101.61	101.11	99.85	99.74	101.19	100.01	99.70	100.45
Sr	3.4	5.1	5.5	8.4	4.1	6.2	6.7	7.5	8.7
Y	78.4	97.3	0	0	95.9	785.7	212.0	156.7	84.0
Pb	200.0	25.2	12.2	14.9	214.4	114.5	45.6	57.9	38.8
Th	48.5	52.1	14.9	3.0	45.1	66.5	61.0	58.5	57.9
U	18.8	7.7	3.4	2.1	23.3	18.9	39.2	32.7	32.4
Rb	866.0	1,003.2	2,480.2	2,816.0	563.1	709.2	698.9	672.8	661.4
Zr	107.4	104.0	34.6	21.0	111.4	112.5	124.5	121.1	118.0
Nb	34.4	22.7	35.0	86.8	36.0	49.9	53.1	53.1	38.3
Ga	25	27	69	126	23	25	25	23	24
Be	25	30	40	60	50	30	40	30	50
Sc	0	0	0	30	3	0	0	3	3
Mo	0	0	3	3	15	0	0	0	10
Yb	10	25	25	40	10	80	25	30	20
Sn	85	360	330	560	36	50	55	46	50
Li	250	250	300	700	200	150	150	100	150
Gl	48	51	77	82	34	<20	39	38	<20
Ni	48	52	92	136	42	128	56	47	38
Cu	82	111	2	51	72	51	42	24	31
Zn	56	37	36	35	502	51	21	8	5
W	ND	20	ND	ND	ND	ND	ND	ND	ND
B	0	0	0	0	0	0	0	0	0
Density	2.6612	2.7268	2.7238	2.8265	2.5443	2.6180	2.6409	2.5989	2.6369

The major oxides SiO₂, TiO₂, Al₂O₃, Fe_{total}, MnO, MgO, CaO, P₂O₅, and SO₃ were analyzed by X-ray spectroscopy; K₂O and Na₂O by flame photometry; H₂O and CO₂ by weight differences on absorption of phosphorus pentoxide and "carbosorb," respectively; FeO by colorimetry at La Trobe University. F was determined by specific ion electrode analyses at Comalco Laboratories (Thomastown, Victoria, Australia). B and Li were determined by wet chemical means at the Australian Mineral Development Laboratories (Amdel). Be, Sc, B, and Mo were done spectrophotographically; Sn and W were done by X-ray fluorescence at Amdel. Sr, Y, Pb, Th, U, Rb, Zr, Nb, Ga, Cl, Ni, Cu, and Zn were done by X-ray fluorescence at La Trobe University.

Assigning limits to the possible analytical errors for these analyses is problematical. For the trace elements the following detection

Moina area there is evidence of the opposite sense of displacement on the fault, and so it is possible that there has been more than one episode of movement. The magnetic picture also shows northwest trends and east-west trends. The northwest trends seem to correlate with known folding and/or faulting with northwest axes in an area close to the Bismuth Creek fault and the east-west trends seem to correlate with zones

of tension fractures, which would have acted as the main part of the hydrothermal plumbing system. The major quartz lodes are part of this east-west system (Fig. 1A). The skarn east of Bismuth Creek is compositionally variable and lies beneath a reverse fault (Fig. 1B).

Three general types of calc-silicate skarn occur, namely, (a) a distinctive pale green pyroxene skarn,

from Diamond Drill Hole ML-1 (The sample positions are those shown in Figure 2.)

815	825	830	839	856	862	870	Avg. zone 1	Avg. zone 2	Avg. zone 3	Avg. zone 4
77.69	76.72	78.02	75.22	75.54	76.11	76.64	74.84	53.44	76.04	76.43
0.04	0.03	0.04	0.03	0.04	0.03	0.03	0.03	0.07	0.04	0.04
10.52	11.54	11.19	12.09	12.17	11.94	11.84	11.32	23.43	12.33	11.70
0.92	0.45	0.48	0.41	0.29	0.09	0.00	0.10	1.76	0.78	0.26
1.45	1.84	1.05	1.57	1.36	1.35	1.32	4.50	4.43	1.83	1.44
0.08	0.04	0.03	0.07	0.05	0.03	0.03	0.14	0.19	0.09	0.05
0.08	0.04	0.06	0.15	0.13	0.04	0.07	0.13	0.23	0.14	0.09
0.52	0.60	0.54	0.55	0.53	0.48	0.51	0.65	1.95	0.46	0.52
1.83	1.86	2.47	2.40	2.76	2.54	2.93	0.23	0.38	0.66	2.35
4.19	4.83	4.24	4.81	4.87	5.09	4.99	3.83	8.06	4.51	4.77
0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.02	0.00
0.07	0.02	0.03	0.04	0.00	0.04	0.00	0.87	0.27	0.09	0.03
0.78	0.99	1.08	0.66	0.71	0.50	0.99	1.48	1.98	1.31	0.85
0.18	0.29	0.19	0.27	0.21	0.27	0.31	0.21	0.18	0.53	0.26
1.07	0.83	0.72	0.74	0.61	0.56	0.66	2.85	1.29	1.01	0.77
99.42	100.09	100.13	99.02	99.27	99.08	100.34	101.14	97.77	99.74	99.68
0.03	0.01	0.01	0.02	0.00	0.02	0.00	0.44	0.14	0.15	0.02
99.39	100.03	100.12	99.00	99.27	99.06	100.34	100.70	97.63	99.69	99.66
0.38	0.47	0.38	0.37	0.38	0.33	0.70	0.35	2.85	0.37	0.43
99.77	100.55	100.40	99.37	99.65	99.39	101.04	101.05	100.48	100.16	100.09
8.7	8.5	9.1	9.0	10.0	9.7	9.5	4.3	7.0	6.1	9.2
86.9	77.6	126.6	163.9	118.1	142.2	111.9	87.9	0	312.6	113.9
53.2	20.7	23.2	26.1	36.8	32.1	31.4	112.8	13.6	108.1	32.8
59.0	39.5	50.0	48.0	52.2	52.5	44.9	50.3	13.6	57.8	50.6
51.2	14.7	13.3	16.9	11.6	52.9	15.6	13.3	2.8	28.5	26.1
532.8	542.3	557.0	617.0	628.1	636.0	613.1	934.6	2,648.1	661.0	598.5
109.3	92.1	105.4	119.8	105.9	123.8	97.3	105.7	27.8	117.4	109.0
55.5	61.8	53.1	51.9	43.5	58.8	31.6	28.6	60.1	48.0	49.3
19	25	21	20	23	21	22	27	97.5	24.0	21.9
30	50	30	30	30	50	40	28	50	37.5	38.8
3	0	0	0	0	0	0	0	15	2	1
59	0	30	0	0	7	0	0	3	4	12
20	20	25	25	15	30	15	18	33	36	21.3
40	4	44	50	40	26	30	222.5	445.0	47	40.5
150	100	100	100	100	70	80	250	500	137.5	106.3
<20	20	20	20	20	39	21	50	79.5	27.8	7.5
30	27	38	43	40	40	33	50	114.0	68.3	36.1
181	3	8	9	16	15	15	96.5	27	47.3	34.8
7	11	5	4	6	4	3	46.5	36	146	6
				50		30	20	110		40 ¹
0	0	0	0	0	0	0	0	0	0	0
2.6536	2.6155	2.6448	2.6183	2.6259	2.6074	2.6034	2.6940	2.7752	2.6005	2.6257

levels in ppm are suggested (R. Price, pers. commun.): Cu (0.5), Zn (0.5), Rb (0.4), Sr (0.3), Y (0.3), Zr (0.8), Nb (0.9), Pb (1.2), V (1.0), Cr (0.9), Ba (3.7), La (1.3), and Ce (4.0). The values for Mo, W, Th, and U are unknown but most likely are of the same order as the other elements (1.0 ppm or so). Analytical errors of major elements as suggested by Norrish and Hutton (1969) for X-ray spectroscopy-derived values are as follows (coefficient of variation of mean values percent) SiO₂ (0.30), TiO₂ (1.1), Al₂O₃ (0.63), Fe₂O₃ (0.71), MnO (2.4), MgO (0.92), CaO (0.81), and P₂O₅ (1.6). Values for Na₂O, K₂O, and SO₃ are (again) probably of the same order. Detection levels for the other trace elements in ppm are B (3), Yb (1), Mo (3), Sc (3), Y (10), Be (1), Sn (20), W (20), and Li (5). The samples were crushed in an Mo-W-free Mn-Cr Sieb mill, thus introducing the possibilities of Mn and Cr contamination.

(b) a garnet-vesuvianite-fluorite \pm wollastonite \pm amphibole \pm epidote \pm magnetite skarn, and (c) a wollastonite skarn (see Fig. 4B for types a (GS) and b (PS)). Type a consists of thin beds up to 3 cm wide having irregularities along strike but generally having a sharp contact with type b. Thin pyroxene skarn units are common nearest the lower contact of the overall skarn unit, becoming less common higher in

the column. These represent replacement of calcareous siltstone interbeds which occur unaltered about 100 m west of SMD 21 (Fig. 1A) in the adjacent unreplaced Gordon Limestone and are bioturbated, with irregular bedding. This feature is retained in the skarn (Fig. 4C) which suggests that, apart from fracturing, the replacement of the units was a relatively passive event. In all of the skarn units, bimetasomatic zon-

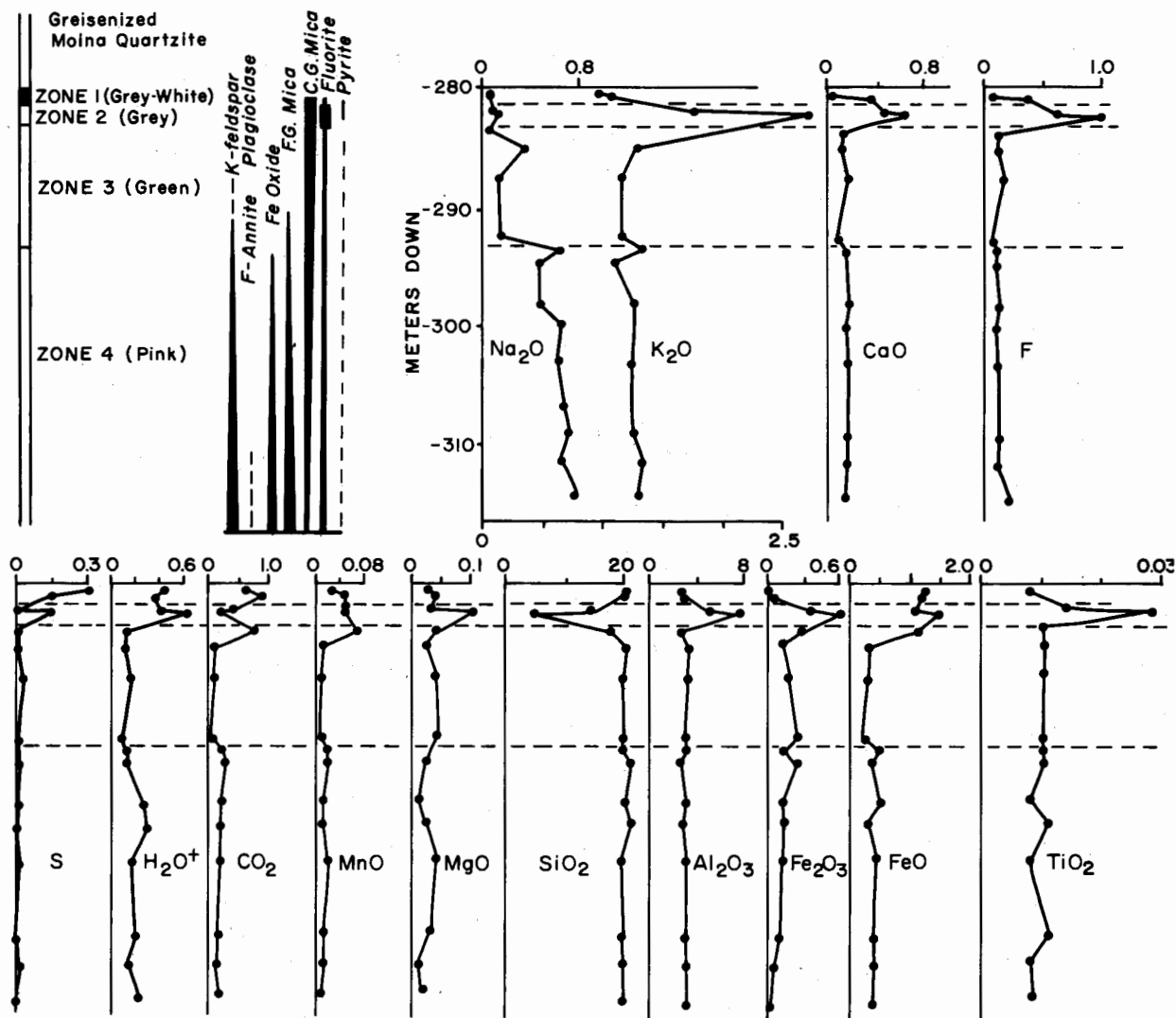


FIG. 2. Chemical variations of major and minor elements from the granite (greisen)-quartzite contact from 280 m down to 316 m in drill hole ML-1 (see Fig. 1 for location). The zones and mineralogy are explained in the text. The values are in moles per 100 cm³ of rock (thus corrected for density differences). The values were calculated from those in Table 2.

ation is generally absent, reflecting the lack of major chemical or permeability gradients.

The pyroxene skarn consists mainly of fine-grained (<0.2 mm) diopside-hedenbergite pyroxene (ratio 40:60) with very minor amounts of fluorite and garnet. By comparing pyroxene skarn from SMD 5 at 56 m with similar interbeds beyond SMD 21 in unreplaced Gordon Limestone, (Table 4), it can be seen that Na, Sn, F, W, Zn, and Fe were added to the former (7, Table 5). This change could not have been produced solely by thermal metamorphism and indicates F metasomatism.

The second skarn, type b, occurs mainly near the

base and in minor amounts interbedded with recrystallized limestone directly above the wrigglyte skarn. All gradations exist but garnet is invariably present in large proportions. The garnet is andradite-grossularite (analyses 8 and 9, Table 3) and has concentric color zoning accentuated by anomalous anisotropy in polarized light. Values of Sn of up to 0.70 weight percent were found in garnet. The pyroxene interstitial to and occurring as inclusions in garnet is diopside-hedenbergite (analysis 4, Table 3), whereas the accompanying vesuvianite is relatively Fe and F rich (analysis 1, Table 3). Fe-rich hornblende amphibole (analysis 12, Table 3) and epidote occur replacing

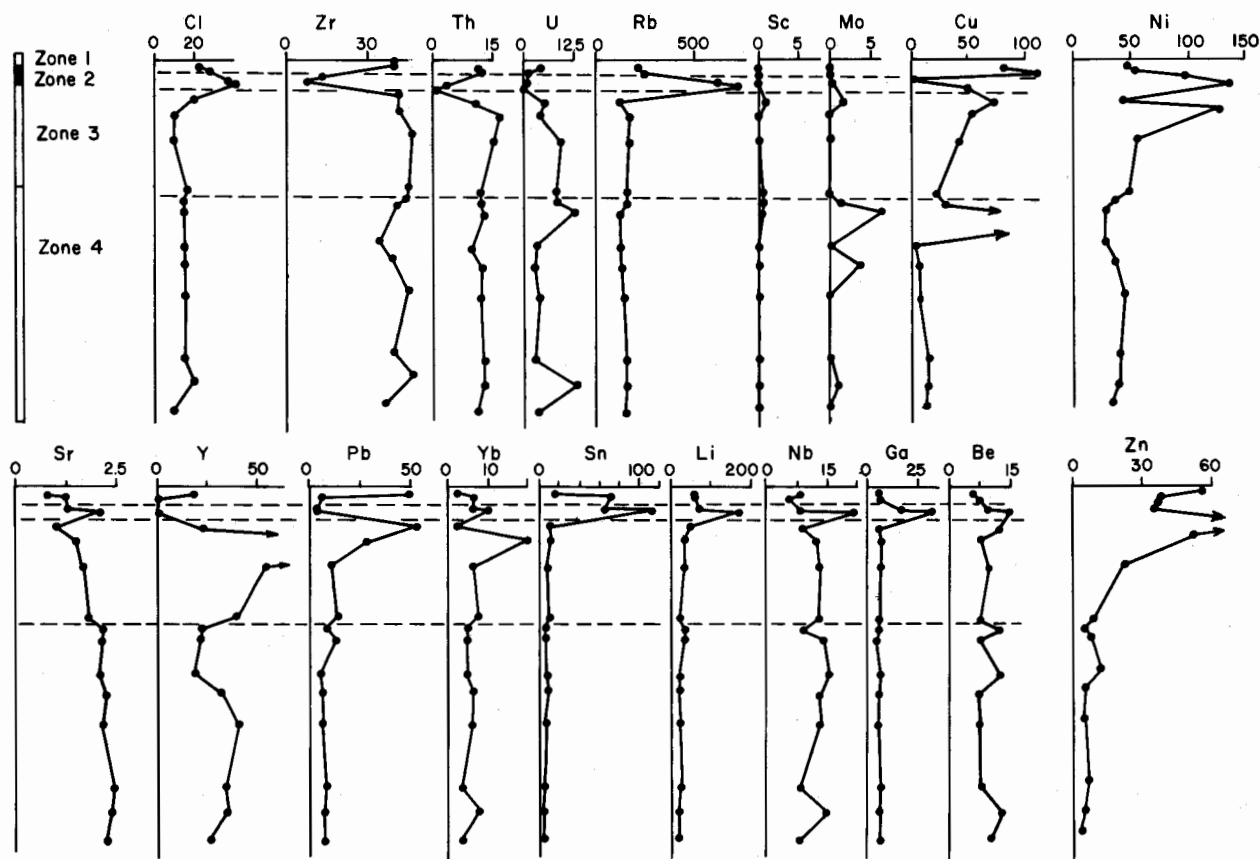


FIG. 2 (cont.)

pyroxene, vesuvianite, and garnet. Interstitial fluorite and adularia occur, occasionally with epidote and calcite in pods up to 2 cm long. Magnetite, scheelite, and pyrite occur as minute grains (to 0.1 mm across), but no cassiterite was observed.

Evidence that this skarn has formed from Gordon Limestone (marble) occurs in some cases (Fig. 4E). Bulk chemical analyses of garnet skarn (6, 11, and 18, Table 4) recalculated on a molar basis per 100 cm³ volume (8, Table 5, and Fig. 7) show that relative to limestone (17 and 19, Table 4), Sn, H, U, Y, Zn, Li, Cl, Be, Si, Fe⁺², Fe⁺³, F, Al, Mn, and B were added whereas Sr, C, and Ca were removed. The light elements Li, Be, B, and Cl are particularly enriched in garnet skarn overlying the wrigglyte skarn (see Fig. 4E). Specific boron minerals were not found.

As in the quartzite, veinlet-filled fractures in calc-silicate skarn show a sequential deposition so that for SMD 5, 56 m down, the sequence is veins rich in: (1) muscovite, (2) potassium feldspar (adularia)-fluorite, and (3) calcite (in Fig. 4B and C). The sequence may be only partly shown in any one sample so that in Figure 3A only (2) above is present. The margins

of fluorite-bearing veins commonly show marginal alteration to magnetite \pm amphibole of garnet and pyroxene skarn (i.e., Fig. 4B).

Calc-silicate skarn type c consists mainly of wolastonite (>80 vol %) with lesser amounts of garnet, pyroxene, vesuvianite, and fluorite. It is probably a variation of type b in representing the metamorphism of extremely quartz-rich impure limestone interbeds, which occur up to 3 m thick. Like the other calc-silicate skarns, it is enriched in F, Sn, Fe, W, and Cl (10 and 14, Table 4).

The main skarn type is a dark, heavy, fine-grained rock showing chaotic laminar patterns of alternating light and dark lamellae which are up to 0.5 mm wide. The rock type has been called "ribbon rock" skarn (Jahns, 1944a), "rhythmically banded rock" (Shabynin, 1977), "wrigglyte" skarn (Askins, 1975), and "apocarbonate greisen" (Govorov, 1958).

The skarn here called wrigglyte skarn occurs first as thin interbeds within the upper part of the lower calc-silicate skarn unit (see Fig. 5A). Higher in the column wrigglyte skarn occurs interbedded with pyroxene skarn (Figs. 5B and 4C) but generally not with

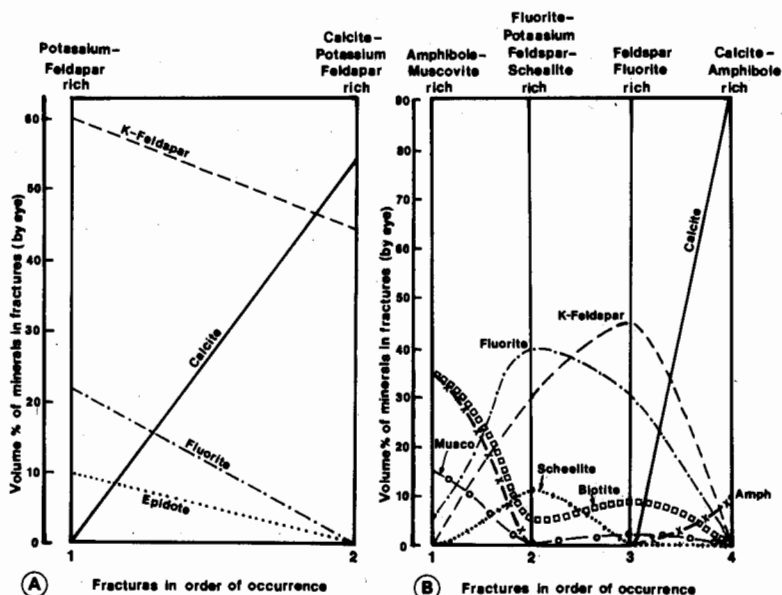


FIG. 3. Graphs showing the mineral contents of sequentially deposited veins in quartzite (B, SMD 5, -65 m), calc-silicate skarn (C, SMD 12, -65 m) and laminar skarn (A and D, SMD 5, -46.2 m). The vein contents were estimated by eye. In most cases between three to ten veins of the same period were averaged to produce the graphs.

garnet skarn. The wrigglyite skarn terminates upward against unreplaced marble (where it is not removed by erosion) and septa of skarn occur along fractures extending into the marble (Figs. 5C and D).

The lamination is closely related, and parallel, to fractures (Figs. 5C and D), not bedding (Fig. 5B). In some cases the skarn-marble contact is irregular enough to produce embayments (Fig. 5D) or even pockets of unaltered material. These latter are interpreted eventually to form augen-type structures ("1" in Fig. 5C and "3" in Fig. 5B) or foldlike areas as in Figure 6A. The augen often occur where two or more fractures intersect (Figs. 5C, 4C, and 4D).

The normal primary mineralogy of the wrigglyite skarn is magnetite in the dark lamellae and vesuvianite + fluorite in the light-colored lamellae. More rarely, adularia + fluorite and pyroxene \pm garnet + fluorite and pyroxene \pm garnet + fluorite from the light-colored lamellae. Minor cassiterite occurs, particularly in the magnetite layers, and minor Sn-rich sphene, scheelite, and bismuthinite crystals occur scattered throughout. Molybdenite, sphalerite, pyrite, scheelite, cassiterite, amphibole, bismuthinite, adularia, topaz, unusual K-Ca silicates (on which study is still in progress), and laumontite occur in veinlets. As seen in Figure 6A, the dark layers need not be continuous but pinch and swell making the exact classification of which minerals belong to which lamellae difficult.

Vesuvianite has $\text{Fe} > \text{Mg}$ (analysis 3, Table 3) and high F (2.369 atoms per formula of 78 oxygens). Pyroxene is rarely found in the light layers, but when it is present it is hedenbergitic (i.e., $\text{Fe}/\text{Fe} + \text{Mg} = 0.619$, analysis 5, Table 3). This is in sharp contrast to diopsidic pyroxene in accompanying garnet skarn (analysis 4, Table 3). Adularia + fluorite wrigglyite probably occur only where high Sn values are found ($>0.4\%$ Sn?). The adularia has few impurities (analysis 16, Table 3). Tin values of 6.31 weight percent are found in rare Sn-rich sphene. Magnetite contains little Sn or W (analysis 15, Table 3) whereas cassiterite, which forms irregular patches up to $15\ \mu\text{m}$ in diameter, contains less than 2.00 weight percent FeO.

As in the quartzite and lower calc-silicate unit, the wrigglyite skarn has numerous intersecting fractures and veinlets. For example, in a sample from SMD 5 at 46.5 m (Fig. 3D) the general sequence is veins rich in (1) garnet + fluorite, (2) adularia + fluorite (+ to 2% topaz and 8% scheelite), and (3) fluorite + adularia.

The alteration of the primary wrigglyite skarn to form amphibole and/or sulfide-rich equivalents occurs at Moina, as in many skarns throughout the world. The distribution of alteration in the skarn is difficult to define because of the limited number of drill holes and poor surface exposure. Variable amounts of amphibole (\pm sulfide) alteration occur throughout. In some cases the wrigglyite skarn may

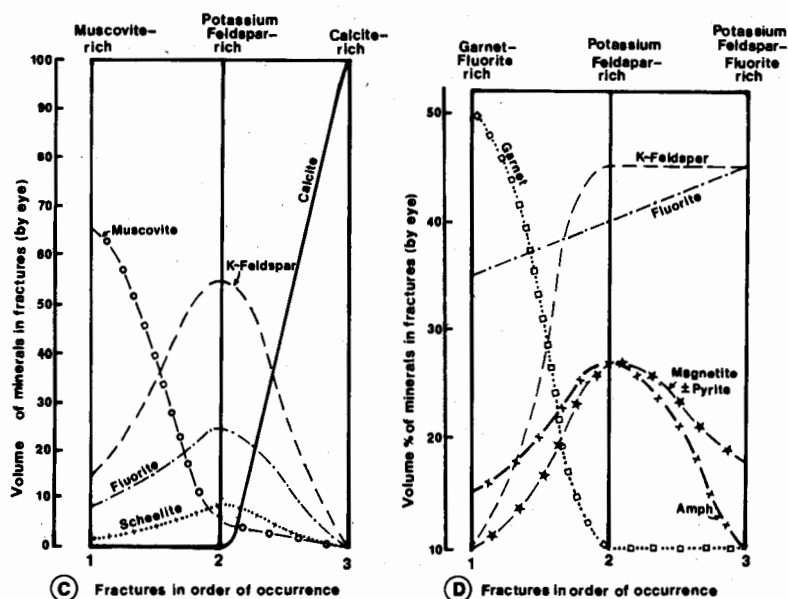


FIG. 3. (cont.).

be altered producing an amphibole-rich area having sharp contact with unaltered skarn (e.g., at "2", Fig. 5D); in others magnetite \pm cassiterite relict layers are retained in an amphibole-Fe sulfide matrix. Apart from intermittent alteration throughout the skarn, major areas are known to occur in SMD 9 at the skarn extremity where there is a pyrrhotite-rich skarn and in SMD 13 near the Bismuth Creek fault, beneath a reverse fault, where there is a coarse-grained, sphalerite-rich rock. In both examples, but particularly in the former, relict wrigglyte structure occurs with a superposition of a later structure consisting of plates of hematite ($\text{SiO}_2 = 0.021$ moles, $\text{CaO} = 0.009$ moles per 3 oxygens) with marginal Fe-F biotite (analysis 13, Table 3 and Fig. 6D). Amphibole is Fe rich with considerable K + Na (analysis 10, Table 3). Pyrrhotite has the composition $\text{Fe}_{0.955}\text{S}$ in sample SMD 9 at 97.2 m (Figs. 5F and 6D) whereas hematite (0.027 moles SiO_2 per 3 oxygens) occurs with chalcopyrite.

Near the Bismuth Creek fault the replacement of wrigglyte skarn by base metal sulfides, Fe sulfides, and hematite is common (Fig. 5E). Sphalerite in sample SMD 16 at 65 m, has the composition $\text{Zn}_{0.863}\text{Fe}_{0.164}\text{Mn}_{0.020}\text{S}_{1.067}$ whereas coexisting pyrrhotite has the composition $\text{Fe}_{0.936}\text{S}$ and hematite has 0.008 moles SiO_2 (3 oxygens per formula) as impurities. Garnet (and vesuvianite?) alter to hematite + quartz (Fig. 6B); pyroxene, to amphibole (Fig. 6C) and magnetite to sulfides. The amphibole in the alteration is similar to that found in less-altered wrigglyte skarn (analyses 11 and 12, Table 3).

Mass Balance Relationships

Mass balance calculations were done to gain some insight into the amount of materials added by the mineralizing solution on traversing the rock units and to see if the constituents were produced by either redistribution in the skarn column, derivation from the quartzite column, or derivation from the greisenized granite. Most of the trace element contents of the skarn are so anomalously high that these could not have been produced by redistribution within the columns and rock types analyzed. To account for density differences, the mass balance calculations for major constituents are shown in Table 5 on the basis of molar changes per 100 cm^3 of rock. Columns 1 to 3 and 5 to 11 are for changes of rock compositions caused by replacement from a previous rock type whereas columns 4 and 12 represent the gains and losses to the fluid from the greisenized granite (column 4) and the skarn (column 12). As can be seen in Figures 2 and 7, when the fluid traversed and reacted with the column of granite, the solution gained Si, Na, and K whereas all other major constituents were lost. For the skarn column, including some quartzite, the solution gained only Ca, a minute amount of P, and large amounts of CO_2 . The major constituents added to the greisenized granite column are Al, Fe^{+2} , Fe^{+3} , S, and F; Si, Al Fe^{+2} , Fe^{+3} , Mg, Na, K, S, H_2O , and F were added to the skarn column. Much Si could have been added when the solutions traversed the quartzite (column 5), but clearly most

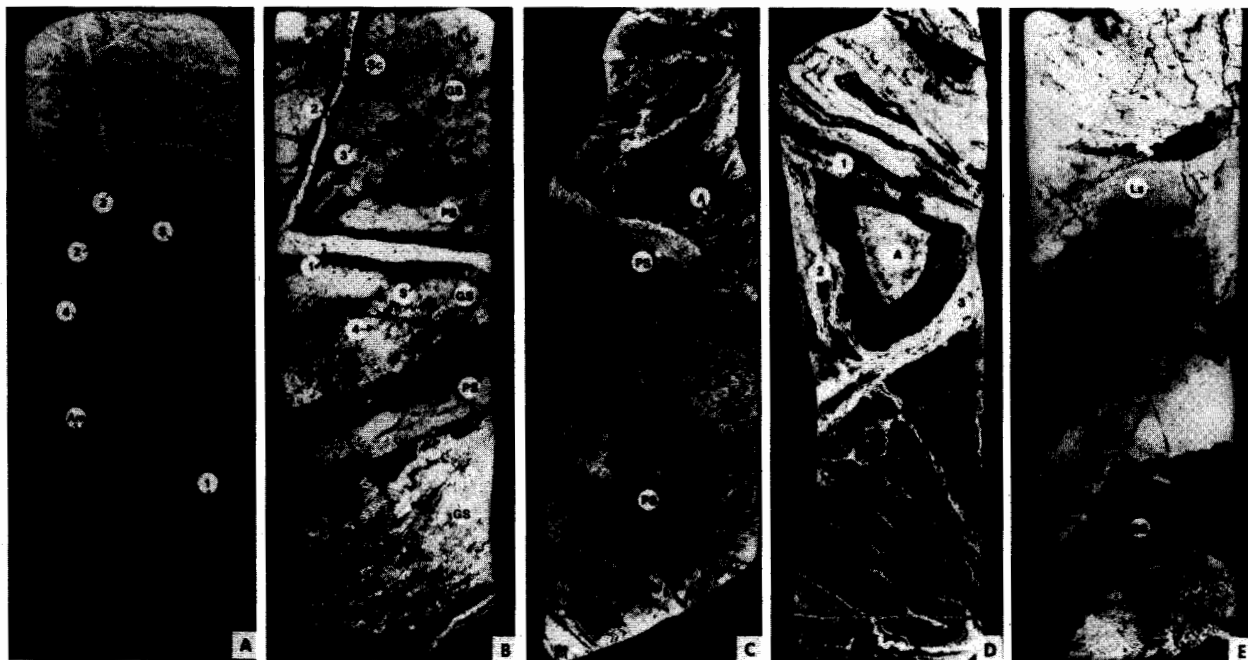


FIG. 4. Examples of drill core from an idealized section through the skarn layer at Moína. The core is approximately 4 cm wide.

A. Photograph of intensely fractured and veined quartzite immediately beneath the skarn units (SMD 5, -62 m). Numbers 1 to 4 = the sequence of vein fills described in the text; S = the magnetite-diopside selvage found at fluorite-rich vein margins described in the text; Am = to amphibole (-epidote)-rich pods.

B. Transmitted light photo of basal skarn showing alternating garnet (GS) and pyroxene skarn (PS) units (SMD 5, -56 m). Numbers 1 to 4 = the sequential nature of intersecting veins; Sc = scheelite grains observable in vein 2; S = the magnetite (-diopside) selvage peripheral to the fluorite-bearing veins (1 and 2).

C. Transmitted light photo of alternating layers of wrigglyte skarn and pyroxene skarn (PS) common near the base of the main wrigglyte skarn unit. A = a concentrically zoned area of wrigglyte skarn. An enlargement of this wrigglyte skarn-pyroxene skarn contact is shown in Figure 5B (SMD 12, -101.28 m).

D. Transmitted light photo of wrigglyte skarn showing sequential vein fillings (1 to 3) and granular skarn area A (SMD 5, -46.5 m). The latter consists of garnet + fluorite + pyrite.

E. Transmitted light photo of garnet (+ vesuvianite + fluorite) skarn-marble. The skarn is enriched in B, Li, etc., but is not wrigglyte. It is a replacement of undisturbed limestone with dolomitic siltstone interbeds.

of the constituents that must have been added to produce the relatively thin skarn column of SMD 12 could not have been derived solely by alteration of quartzite or greisenization of an amount of granite equivalent to the column studied in drill hole ML-1. Also, the skarn overlying adjacent areas of ML-1 is much thicker than that found in SMD 12.

Distribution of Economic Constituents

In the course of exploration by Comalco Limited, approximately 5,000 analyses of Pb, W, Bi, Sn, Mo, Cu, Zn, Be, Sc, Y, Cd, In, Ge, Au, and Ag were done of drill cores SMD 4, 11, 6, 15, and 7 (traverse 1, section line A-B in Fig. 1A) and ML-3, -16, and -13 (traverse 2, section line E-F in Fig. 1A) by the Australian Mineral and Development Laboratories

(AMDEL). These are traverses both out from and across the Bismuth Creek fault. The data were analyzed by computer to see (a) the degree of correlation between all possible element pairs, (b) whether there is a relation between the concentration of an element and depth, and (c) whether there is a lateral relation of concentration and proximity to the Bismuth Creek fault. Correlation coefficients were calculated relative to a linear model. Scatter can result from the data not filling a linear model, analytical variation, and the fact that the analyses were done on different skarn rocks and quartzite where density differences were not accounted for. The correlation coefficient data of (a) above shows (Table 6) that elements fall approximately into two groups, namely, (1) Be, F, Sn, and Ag and (2) Zn, Cd, In, Ge, Au, and possibly Bi. The

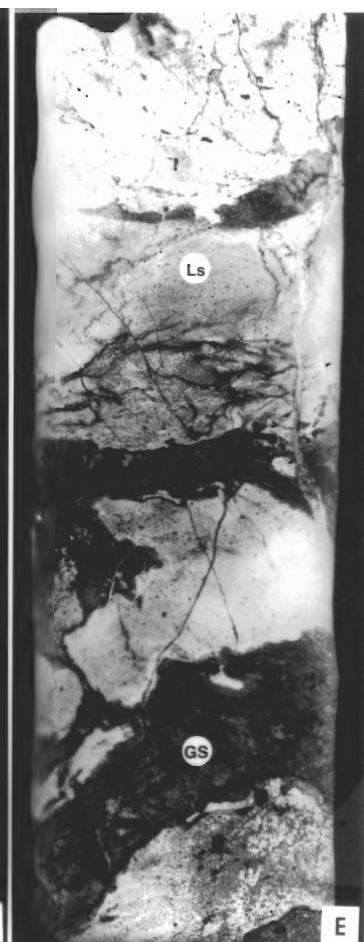
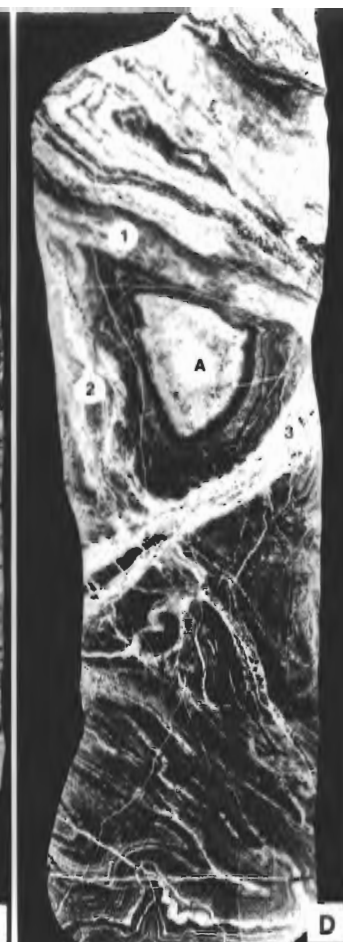
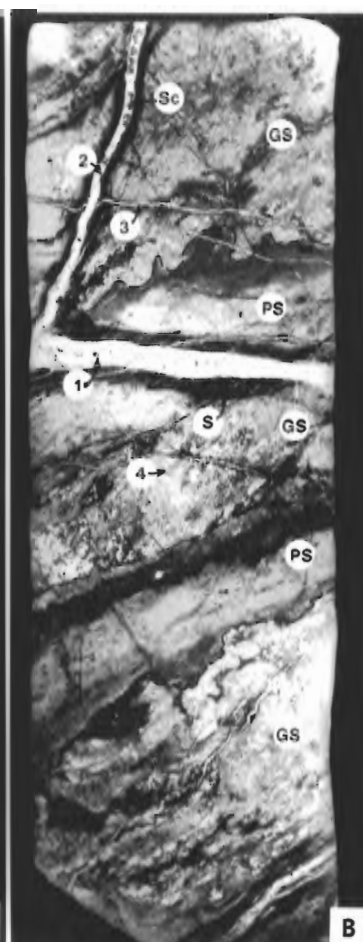
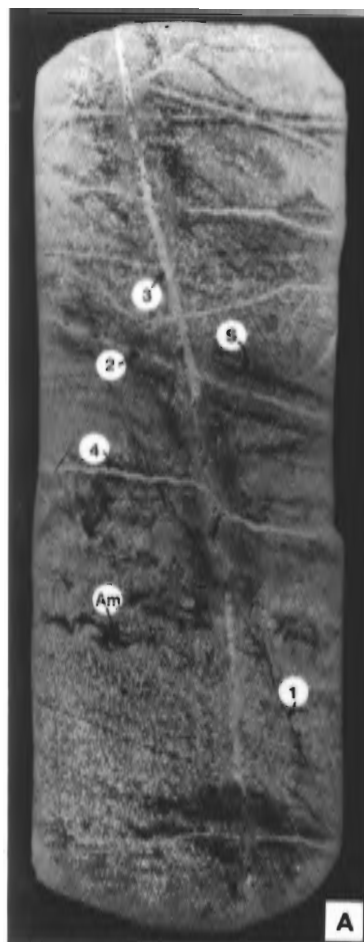


TABLE 3. Composition of Representative Skarn Minerals

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
SiO ₂	36.87	35.19	34.21	53.07	48.94	37.62	37.21	36.89	37.01	36.80	36.55	39.14	34.90	45.61	0.81	66.42
TiO ₂	0.07	0.08	0.20	0.00	0.07	0.17	0.20	0.22	0.56	0.20	0.25	0.34	0.00	0.00	0.06	0.02
Al ₂ O ₃	13.70	15.64	13.65	0.19	0.37	5.97	6.09	6.97	8.29	11.54	13.23	14.44	10.90	30.59	51.14 ¹	18.73
FeO	5.43	3.98	6.62	4.97	18.33	22.02 ¹	21.12 ¹	22.29 ¹	20.64 ¹	27.14	26.16	23.83	36.19	6.76	46.02	0.32
MnO	0.75	0.83	1.20	0.27	2.83	1.79	1.00	1.86	2.41	0.83	1.42	1.26	1.77	0.00	0.34	0.00
MgO	3.00	2.28	2.15	15.01	4.74	0.09	0.08	0.00	0.01	3.39	2.62	2.00	3.40	0.53	0.00	0.07
CaO	35.19	35.11	35.84	24.86	22.68	31.78	32.51	31.35	31.25	11.22	9.64	11.05	0.15	0.00	0.25	0.00
Na ₂ O	0.01	0.04	0.02	0.04	0.15	0.05	0.09	0.00	0.01	1.53	1.49	0.28	0.00	0.28	0.01	0.01
K ₂ O	0.00	0.01	0.04	0.01	0.00	0.02	0.02	0.00	0.02	2.22	2.93	2.25	8.10	10.77	0.00	13.01
F	1.70	1.35	1.40	0.11	0.22	0.23	0.31	0.31	0.32	0.60	0.45	0.29	2.22	2.89	0.00	0.00
Cl	0.24	0.23	0.17	0.02	0.00	0.02	0.06	0.00	0.05	0.52	0.61	0.62	0.24	0.00	0.00	0.00
Anhydrous total	97.05	94.84	95.62	98.56	98.34	99.76	98.69	99.89	100.57	95.99	93.35	95.50	97.78	97.43	98.83	98.58
Number of O ions	78	78	78	6	6	12	12	12	12	24	24	24	24	24	4	4
Si	19.014	18.558	18.304	1.986	1.976	5.978	6.104	5.992	5.943	6.288	6.268	6.608	6.044	6.439	0.033	12.168
Ti	0.056	0.063	0.081	0.000	0.002	0.020	0.025	0.027	0.068	0.026	0.032	0.026	0.000	0.000	0.002	0.003
Al ^{IV}	8.330	9.724	8.611	0.002	0.018	0.022	0.00	0.008	0.057	1.712	1.732	1.392	1.956	1.561	0.010	4.045
Al ^{VI}	2.342	1.755	2.962	0.156	0.619	0.000	0.000	0.000	0.000	3.878	3.752	3.878	5.228	0.000	1.557	0.050
Fe ⁺²	0.000	0.000	0.000	0.000	0.000	3.015	2.607	2.725	2.491	0.000	0.000	0.000	0.000	0.718	1.557	0.000
Fe ⁺³	0.328	0.371	0.544	0.009	0.097	0.241	0.139	0.256	0.312	0.120	0.206	0.120	0.260	0.000	0.012	0.000
Mg	2.306	1.792	1.714	0.837	0.285	0.021	0.020	0.000	0.002	0.863	0.670	0.863	0.877	0.111	0.000	0.019
Ca	19.446	19.840	20.548	9.997	9.981	5.411	5.714	5.457	5.377	2.054	1.771	2.054	0.028	0.000	0.011	0.000
Na	0.010	0.041	0.021	0.003	0.006	0.015	0.014	0.000	0.003	0.507	0.495	0.507	0.000	0.076	0.000	0.002
K	0.000	0.006	0.013	0.000	0.000	0.004	0.002	0.000	0.002	0.484	0.641	0.484	1.788	1.939	0.000	3.041
F	2.772	2.251	2.369	0.011	0.028	0.031	0.161	0.159	0.163	0.324	0.244	0.324	1.215	1.290	0.000	0.000
Cl	0.210	0.205	0.154	0.002	0.000	0.005	0.017	0.000	0.014	0.150	0.177	0.150	0.070	0.000	0.000	0.000

¹ = Fe₂O₃ instead of FeO

Analyses were done as explained in caption of Table 1. 1, vesuvianite: SMD 5, -56 m, in calc-silicate skarn; 2, vesuvianite: SMD 12, -104.18 to -104.29 m, lower calc-silicate skarn; 3, vesuvianite: SMD 12, -93.00 to -93.09 m, wriggilite skarn; 4, pyroxene: SMD 5, -56 m, in calc-silicate skarn; 5, pyroxene: SMD 12, -93.00 to -93.09 m, wriggilite skarn; 6, garnet edge: SMD 12, -93.00 to -93.09 m, wriggilite skarn; 7, garnet core: as 6 above; 8, garnet: SMD 5, -56 m, in calc-silicate skarn; 9, garnet: SMD 12, -92.00 to 92.12 m, upper calc-silicate skarn; 10, amphibole: SMD 12, -93.00 to 93.09 m, wriggilite skarn; 11, amphibole: SMD 12, -104.18 to -104.29 m, lower calc-silicate skarn; 12, amphibole: SMD 12, -92.00 to -92.12 m, upper calc-silicate skarn; 13, annite: SMD 9, -97.2 m, sulfide-wriggilite skarn; 14, muscovite: SMD 16, -126.557 m, altered wriggilite skarn; 15, magnetite: SMD 12, -93.00 to -93.09 m, wriggilite skarn; 16, adularia: SMD 12, -39.00 to 93.09 m, wriggilite skarn.

TABLE 4. Composition of

The analyses were determined as described in Table 2 except that instead of being calculated as oxides they were calculated in terms

	1	2	3	4	5	6	7	8	9	10
Si	4.39	21.53	31.69	10.06	23.80	12.27	18.41	37.97	22.36	17.87
Ti	0.06	0.34	0.13	0.13	0.49	0.13	0.30	0.21	0.28	0.20
Al	1.08	5.48	2.38	4.68	7.56	4.73	6.46	1.58	4.25	4.08
Fe ⁺³	0.00	0.00	0.95	4.33	1.89	9.83	4.75	0.00	1.47	9.88
Fe ⁺²	0.63	1.95	2.88	14.55	3.68	6.34	2.64	1.76	6.23	0.06
Mn	0.05	0.02	0.12	0.12	0.62	0.59	0.85	0.10	0.30	0.35
Mg	0.95	1.39	2.61	1.14	1.75	1.15	1.39	2.05	5.09	1.88
Ca	36.83	13.41	7.00	18.20	11.70	22.95	20.80	4.34	13.51	21.68
Na	0.14	0.42	0.35	0.59	2.66	0.46	0.42	0.27	0.33	0.08
K	0.62	3.60	1.75	2.48	0.51	0.35	0.27	0.97	1.42	0.01
P	0.02	0.04	0.12	0.02	0.05	0.06	0.05	0.23	0.13	0.10
S	0.21	0.81	0.02	2.63	0.01	0.18	0.02	0.03	0.22	0.01
H ⁺	0.69	0.17	0.17	0.13	0.17	0.12	0.06	0.04	0.10	0.05
H ⁻	0.01	0.04	0.02	0.03	0.02	0.01	0.02	0.01	0.02	0.02
C	9.14	3.75	0.08	1.26	0.23	0.28	0.24	0.17	0.36	0.30
F	0	0.84 (?)	10.03	24.86	3.01	23.93	8.09	0.09	4.26	3.54
Total	54.82	53.79	60.30	85.21	58.05	83.38	64.77	49.82	68.33	61.11
wt. oxygen inferred (approx.)	45.18	46.21	39.70	14.79	41.95	16.62	35.23	50.18	32.67	48.89
Sr	344	272	38	97	188	36	34	50	100	11
Y	13 (40)	8 (60)	32 (100)	0 (60)	11 (60)	4 (70)	22 (50)	61 (100)	28 (70)	31 (60)
Pb	1	13	18	4	4	51	2	1	24	14
Th	4	15	8	123	15	148	15	10	67	16
U	0	1	1	5	1	5	1	1	1	2
Rb	20	102	591	791	217	42	60	158	233	6
Ga	3	14	10	44	19	24	20	4	17	13
Be	5	20	400	300	100	400	80	40	70	60
Sc	5	15	5	7	20	7	6	3	7	5
Mo	0	0	20	300	15	70	0	0	0	0
Yb	4	3	15	10	5	8	8	10	5	5
Sn	<20	<20	280	1,000	180	940	220	20	130	900
Li	5	25	100	65	25	30	15	20	50	5
Cl	0	0	99	35	0	1,799	230	32	116	750
Ni	28	45	77	112	111	62	65	42	100	79
Cu	20	27	7	869	36	289	25	16	66	26
Zn	0	28	89	67	131	173	82	45	334	129
W	<50	<50	2,400	2,100	130	500	190	<50	160	<50
B	0	50	3	0	3	300	0	3	5	150
Density	2.7382	2.6773	2.8479	3.6349	3.0090	3.5914	3.4473	2.8035	2.9698	3.5498

1, Gordon Limestone, field specimen from Iris River, CO₂ too high to do by the methods available; 2, siltstone interbed in Gordon Limestone (analysis 1); 3, SMD 5, -62 m, whole rock, fractured quartzite; 4, SMD 5, -46.5 m, whole rock, wrigglyte skarn; 5, SMD 5, -56 m, No. 1, green pyroxene-rich layer in calc-silicate rock, silty interbed in original limestone; 6, SMD 5, -56 m, No. 2, whole rock, calc-silicate, highly fractured + veins + green pyroxene-rich layer; 7, SMD 5, calc-silicate next to analyses 5 (free of veins and green pyroxene layer); 8, SMD 12, -111.36 to -111.46 m, pure quartzite whole rock; 9, SMD 12, -110.08 to -110.18 m, quartzite with vein-filled fractures, whole rock; 10, SMD 12, -106.02 to 106.13 m, calc-silicate rock, whole rock; 11, SMD 12, -104.18 to -104.29 m, calc-

correlation coefficients between the individual members from the two groups are usually negative (i.e., Zn:F = -0.13; Sn:Zn = -0.16). The two groups are believed to relate to the composition of the primary wrigglyte skarn (group 1) and the later sulfide-rich alteration (group 2). These data imply that group 1 is removed by solution when sulfide (group 2) alteration occurs, as will be discussed. The distribution of W is erratic except for its correlation with Ge which may be accidental as only 13 pairs were analyzed.

The W distribution is probably related to its irregular occurrence in veinlets (e.g., Fig. 4B). The Sc and Y correlations are erratic which may in part reflect the possible inaccuracy in the analysis (semiquantitative). Bi may have been redistributed during sulfide-amphibole alteration but not lost to solution as were the group 1 elements.

Table 7 shows the correlation coefficients of elemental concentrations as a function of depth in each drill hole. Those from group 1, namely Sn, F, and to

Quartzite, Skarn, and Limestone

of elemental abundances. The numbers in brackets are semiquantitative analyses by AMDEL.

11	12	13	14	15	16	17	18	19	20	21
12.63	8.68	10.40	16.67	5.13	12.19	3.06	16.78	7.67	10.79	5.06
0.16	0.06	0.08	0.28	0.07	0.13	0.05	0.37	0.14	0.11	0.08
4.55	5.49	5.21	5.18	1.34	5.10	0.64	5.45	2.56	5.01	1.41
9.50	2.48	2.85	5.33	0.17	1.65	0.18	3.57	0.00	4.16	0.09
5.87	18.26	16.80	1.53	2.08	11.75	1.36	2.14	1.30	13.45	1.34
0.88	0.35	0.54	1.48	0.23	0.60	0.21	0.98	0.02	0.50	0.14
1.13	0.86	0.92	1.74	1.42	1.80	1.40	2.92	1.47	1.17	1.31
22.79	18.26	18.66	23.46	36.58	21.88	36.90	21.66	33.11	19.96	35.86
0.30	0.30	0.18	0.01	0.10	0.38	0.01	0.04	0.15	0.35	0.10
0.46	1.31	1.57	0.02	0.22	0.76	0.12	0.03	1.26	1.32	0.56
0.06	0.02	0.03	0.04	0.02	0.03	0.03	0.06	0.03	0.03	0.03
0.01	1.96	2.48	0.01	0.15	1.95	0.10	0.04	0.37	1.81	0.21
0.12	0.35	0.22	0.06	0.00	0.22	0.02	0.13	0.02	0.21	0.18
0.02	0.03	0.03	0.01	0.04	0.03	0.06	0.04	0.04	0.03	0.03
0.15	0.64	0.66	0.38	9.53	1.23	10.18	1.06	8.44	0.79	9.32
24.05	23.25	22.32	13.78	0	21.84	0	3.31	0	23.26	0
82.68	81.20	81.94	69.98	47.08	81.54	54.32	58.58	56.56	82.95	55.72
17.32	17.70	17.05	30.02	42.92	18.46	45.68	41.42	43.44	17.05	44.28
21	61	57	21	279	109	225	16	361	69	302.3
13 (60)	0 (50)	0 (40)	20 (60)	9 (40)	0 (40)	13 (40)	16 (60)	12 (50)	3	12
20	0	0	6	2	0	0	6	0	5	1
64	238	298	37	4	190	7	27	7	183	5
3	8	2	2	1	4	1	2	1	4	1
32	605	699	4	37	84	28	7	58	442	31
20	33	32	14	4	26	4	16	7	31	4
300	400	300	200	70	400	30	100	5	340	28 (?)
7	4	3	30	3	4	3	15	7	5	4
0	70	5	0	0	60	0	0	0	87	0
5	5	3	4	2	2	3	4	3	5	3
950	660	780	200	100 (?)	620	30 (?)	310	<20	802	33 (?)
10	130	75	10	10	50	5	30	15	66	1
2,001	924	904	649	0	1,732	0	1,403	0	1,119	0
88	54	69	58	20	35	31	56	30	72	27
59	738	1,140	11	45	523	100	25	17	666	46
183	165	189	170	82	258	140	264	8	172	58
100	530	480	100	<50	240	<50	<50	<50	690	<50
200	40	50	150	3	200	0	600	3	98	1
3.5117	3.2988	3.4620	3.5368	2.8973	3.4285	2.8491	2.9431	2.7586	3.4672	2.8108

silicate rocks + wrigglite skarn, whole rock; 12, SMD 12, -99.13 to -99.20 m, wrigglite skarn, whole rock; 13, SMD 12, -97.5 to -97.59 m, wrigglite skarn, whole rock; 14, SMD 12, -96.19 to -96.28 m, calc-silicate rock, whole rock; 15, SMD 12, -93.00 to -93.09 m, unreplaced marble, whole rock; 16, SMD 12, -93.00 to -93.09 m, wrigglite skarn replacement of (15), whole rock; 17, SMD 12, -92.00 to -92.12 m, unreplaced marble, whole rock; 18, SMD 12, -92.00 to -92.12 m, calc-silicate replacement of (17), whole rock; 19, SMD 12, -83.32 to -83.48 m marble, whole rock; 20, average wrigglite skarn (this table); 21, average marble (this table).

a lesser extent Be, show a decreasing concentration trend with depth in traverse 1 but not traverse 2. The values from group 2, namely Mo, Cu, and Zn, are erratic throughout. This is interpreted to show that the sulfide-rich replacement, particularly common near the Bismuth Creek fault, is somewhat erratic and selective, relating to permeability present after the primary skarn crystallization.

In traverse 1, the average value of the concentration for each drill hole of the group 1 elements (F,

Sn, and Be) as well as W, Mo, and Bi is approximately constant relative to their proximity to the Bismuth Creek fault. Cu shows a decrease while Zn shows an almost three-fold increase toward the fault. In traverse 2, where there are only three values per element, Zn increases to the west while Sn, F, Be, and Mo decrease. The data suggests that Zn is derived from the Bismuth Creek fault whereas the group 1 elements were derived from a more extensive plumbing system (east-west fracture systems incorporating

TABLE 5. Differences of Composition between Rocks and Their Unreplaced Equivalents

The values are in terms of molar differences per 100 cm³ of rock analyzed. The molar values were calculated from Tables 2 and 4 using the density data.

	1	2	3	4	5	6	7	8	9	10	11	12
Si	0.0073	-0.4074	-0.0229	0.6639	-1.4261	-0.5768	2.7273	0.4974	0.8256	0.9587	1.4479	-7.5205
Ti	-0.0002	0.0007	0	-0.0039	0.0049	-0.0065	0.0170	0.0117	0.0032	0.0050	0.0199	-0.1105
Al	-0.0006	0.0891	0.0028	-0.1230	0.3034	0.0869	0.6806	0.2993	0.4969	0.5040	0.5270	-6.8172
Fe ⁺³	-0.0007	0.0089	0.0033	-0.0440	0.0077	0.0048	0.2924	0.1017	0.2577	0.0924	0.1790	-2.7357
Fe ⁺²	0.0903	0.0921	0.0106	-0.3699	-0.0102	0.0586	0.0967	0.1049	0.7674	0.6136	0.0435	-6.0764
Fe ^{total}	0.0910	0.1010	0.0139	-0.4139	-0.0025	0.0634	0.3891	0.2066	1.0251	0.7060	0.2225	-8.8121
Mn	0.0027	0.0044	0.0012	-0.0223	0.0113	0.0013	0.0465	0.0329	0.0246	0.0251	0.0416	-0.4025
Mg	0.0016	0.0060	0.0019	-0.0298	0.3856	-0.2223	0.0480	0.0636	0.0153	0.0845	0.1894	-1.7957
Ca	0.0049	0.0516	-0.0022	-0.0867	0.6975	0.1939	-0.6870	-0.0175	-0.7881	-0.7728	-1.0327	5.4418
Na	-0.0650	-0.0599	-0.0522	1.7911	0.0100	0.0105	0.0512	0.2992	0.0406	0.0421	0.0039	-1.1623
K	-0.0097	0.0434	-0.0034	0.0083	1.0091	0.0581	-0.0158	0.2075	0.0770	0.0503	-0.0064	-4.4106
P	0	0	0.0001	-0.0011	-0.0082	-0.0096	0.0027	0.0013	0.0007	0.0017	0.0028	0.0125
S	0.0706	0.0209	0.0048	-0.1977	0.0018	-0.0008	-0.0159	-0.0668	0.1777	0.1948	-0.0053	-1.0659
H ⁺	0.0054	0.0101	0.0036	-0.0191	0.1849	0.3720	-0.2914	0.0564	0.2222	0.7543	0.3257	-1.5751
C	0.0351	0.0097	0.0038	-0.0592	0.0493	-0.0207	-2.0787	-0.7784	-1.9531	0.3415	-2.1550	20.9970
F	-0.0098	0.3569	0.0049	-0.4256	0.6526	1.4905	1.4679	0.3583	4.2445	3.9411	0.5127	-36.841

1, average of zone 4, average of zone 1, Table 2

2, average of zone 4, average of zone 2, Table 2

3, average of zone 4, average of zone 3, Table 2

4, molar gains and losses by the solution for the greisenized section; values are in moles/100 cm³ meters; intersections of zones used are: zone 1, 1.75 m; zone 2, 1.05 m; zone 3, 10.87 m; and zone 4, 21.49 m

5, replacement of quartzite: veined quartzite (9, Table 4), pure quartzite (8, Table 4)

6, replacement of veined quartzite (3, Table 4), pure quartzite (8, Table 4); veined quartzite is shown in Figure 4A

7, replacement of silty sediment pyroxene skarn (5, Table 4), silstone (2, Table 4); pyroxene skarn is shown in Figure 4B

8, replacement of marble; calc-silicate skarn (7, Table 4), average marble (21, Table 4); calc-silicate skarn is shown in Figure 4B

9, replacement of marble; wrigglyite skarn (20, Table 4), unreplaced average marble (21, Table 4)

10, replacement of marble; wrigglyite skarn (16, Table 4), unreplaced adjacent marble (15, Table 4); see figure 5C for relation

11, replacement of marble; calc-silicate skarn (18, Table 4), unreplaced adjacent marble (17, Table 4); see Figure 4E for relation

12, molar proportions of elements lost to and gained from solutions while traversing and reacting in section (SMD 12); values are in moles/100 cm³ m; thicknesses are: veined quartzite, 3.31 m; lower calc-silicate unit, 3.81 m; laminar skarn, 7.40 m; and upper calc-silicate unit, 0.97 m

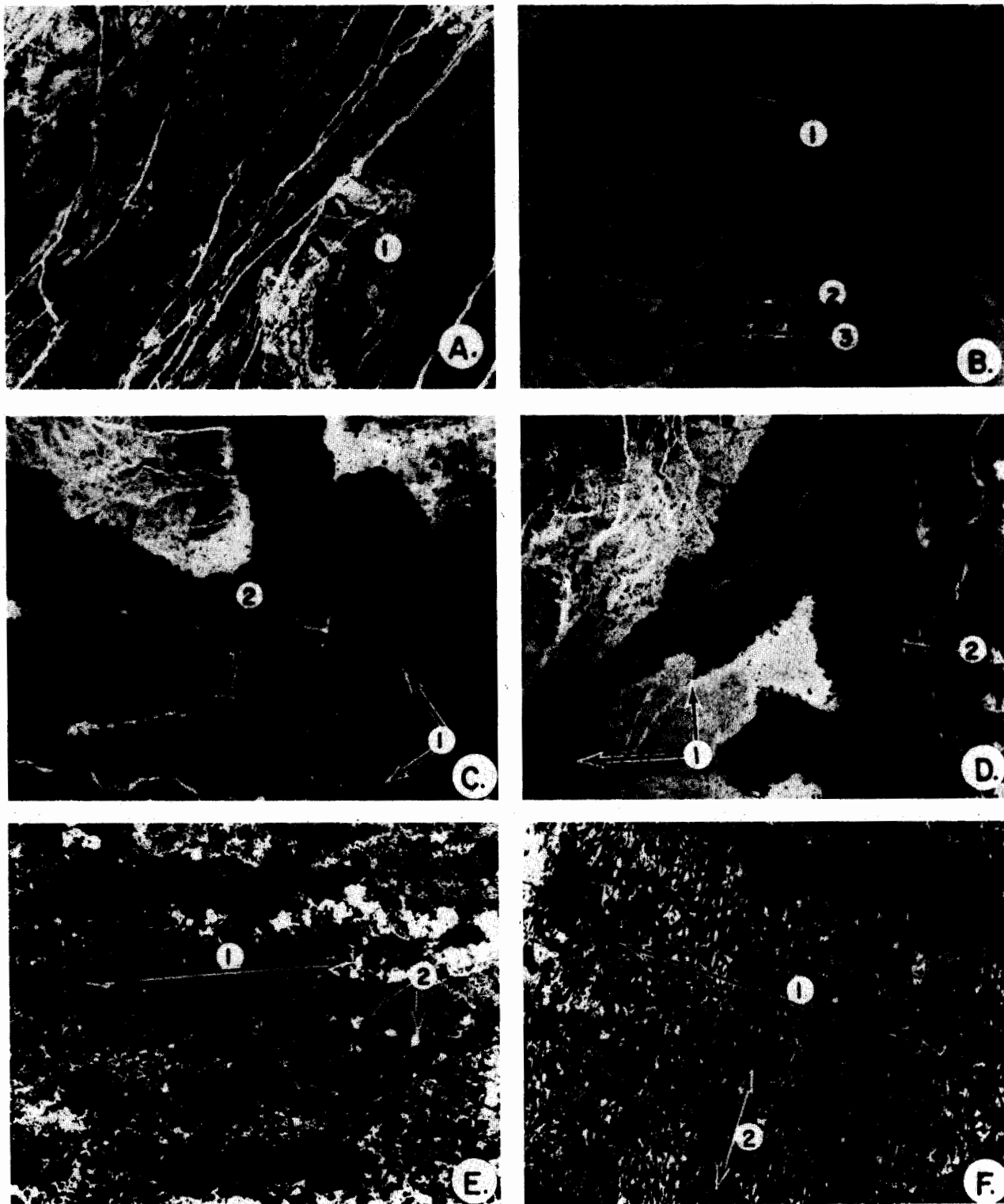
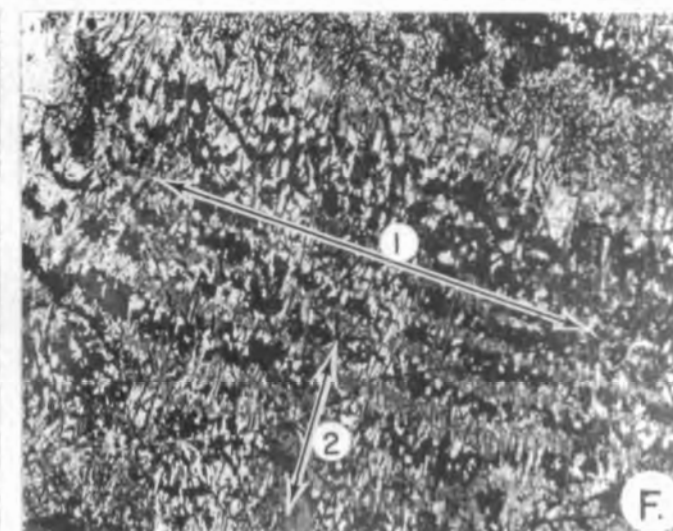
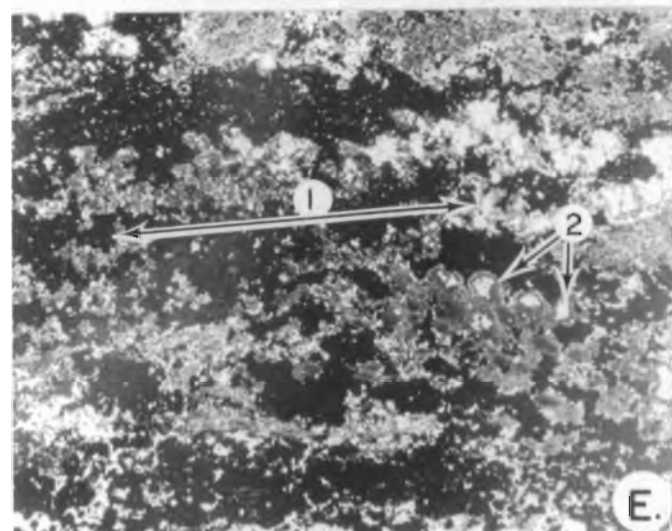
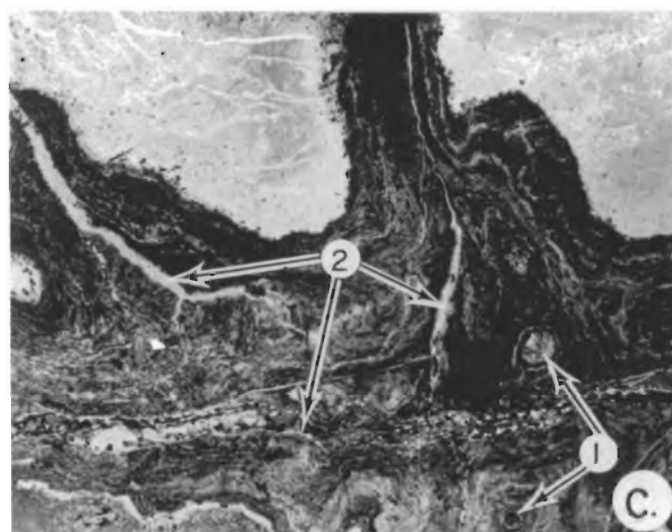
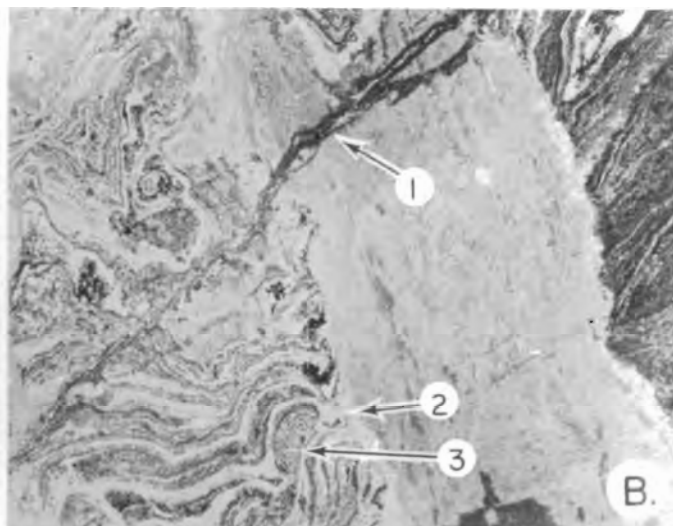
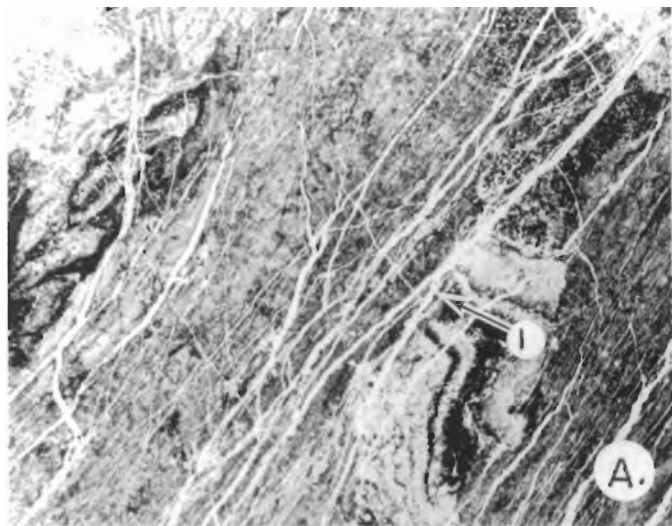


FIG. 5. Features of primary wrigglyite skarn and secondary sulfide replacement.

A. SMD 5, -56 m, shows the extremely fractured "sheeted" appearance of wrigglyite skarn and garnet skarn common near the skarn-quartzite contact. (1) shows a mineral-filled fracture. The field of view is 4 cm wide.



the quartz lodes?). This is consistent with the Bismuth Creek fault being active after as well as during wriggilite skarn deposition.

The Occurrence of Elements of Economic Importance in the Skarns

At Moína, Sn occurs in a number of ways, namely, in solid solution in garnet (to 0.70 wt %), in Sn sphene, and as cassiterite. AMDEL (1978) regarded the tin content partly as a solid solution in garnet (45%) and also as cassiterite of very fine grain size (55%). Green (1979) has shown that at Moína, cassiterite occurs as laminar inclusions of 1.5 to 5 μm long in garnet. These were not found by the present authors although cassiterite grains to 15 μm in size were observed in the magnetite-rich layers in the wriggilite skarn. No distinct Be mineral such as helvite was found and Be is presumed to occur in vesuvianite. Beus (1966, p. 144) found Be values in vesuvianite up to 9.20 weight percent and in garnet, 0.39. W is in scheelite in the skarn fabric and in wolframite plus scheelite in veins; no significant values of W in any of the other minerals was found. F is mainly in fluorite (>90%) with topaz + F-micas in veins; similarly, Bi is in bismuthinite, Mo in molybdenite, Cu in chalcopyrite, and Cd and In in sphalerite. Au may occur as a solid solution in bismuthinite, as the correlation data suggest, or simply as free gold.

Fluid Inclusions

Primary fluid inclusions in the skarn minerals were not found, reflecting their fine-grained nature, whereas measureable inclusions occur only sparingly in vein minerals. Fluid inclusions should represent those present during skarn genesis because the mineralogy of the veins is similar to that of the skarn; the intersecting nature of successive periods of vein deposition suggests cogenesis (Fig. 3). In quartz (SMD 12 at 155 m), liquid CO_2 is a common but not constant constituent (Fig. 8B, E, and F). The daughter products

are NaCl (D_1) and fluorite (D_2) as confirmed by SEM analyses, with a rare reddish opaque which is possibly hematite (Fig. 8D) and a number of minute grains of unidentified nonopaque daughter products. Homogenization temperatures were $331^\circ \pm 12^\circ\text{C}$ (average of 12; gas dissolved into the liquid phase in seven cases and liquid into gas in five). This variation, together with the fact that gas-rich and saline liquid-rich inclusions occur together (Fig. 8H), suggests the inclusions were trapped from a boiling solution. NaCl dissolved before the homogenization temperature was reached while CaF_2 did not, perhaps due to slow kinetics. The first melting of previously frozen inclusions gave temperatures of $-23.1^\circ \pm 0.4^\circ\text{C}$ (average of six) and the last melting of ice, $-15.3^\circ \pm 2.2^\circ\text{C}$ (average of six).

Fluid inclusions in fluorite in a fluorite-rich vein (SMD 16 at 43.9 m depth) contain no liquid CO_2 . Homogenization temperatures of the commonly five-phase inclusions are $482.8^\circ \pm 9.2^\circ\text{C}$ (average of eight; three homogenized into gas, five into liquid), with a first melting temperature of $-19.5^\circ \pm 3.0^\circ\text{C}$ (average of eight) and a last melting of $-7.0^\circ \pm 1.5^\circ\text{C}$ (average of eight). Fluid inclusions in topaz in a cassiterite-wolframite-bismuthinite quartz vein (stratigraphically below the skarn, reference points 1700E, 100S, Fig. 1A) contained no liquid CO_2 and generally two daughter products (Fig. 8G). Homogenization was at $416^\circ \pm 5.2^\circ\text{C}$ (average of 10; eight dissolved into liquid, two into gas), with a first melting temperature of $-15.0^\circ \pm 1.5^\circ\text{C}$ (average of seven) and a last melting of $-6.4^\circ \pm 0.8^\circ\text{C}$ (average of seven). The pressure correction for the homogenization temperatures is unknown but is probably small as the deposit was likely formed at near-surface conditions.

On the basis of the limited data available, the following conclusions are drawn: the homogenization of fluid inclusions constituents into both the liquid and the gas phase, with the large variation of ratios ob-

B. SMD 12, -101.28 m, showing the interface (2) between wriggilite and pyroxene skarn. (3) shows a concentrically zoned wriggilite skarn area. (1) shows a fluorite-rich fracture which obviously has altered pyroxene skarn to a greater extent than wriggilite skarn. Field of view is 2 cm wide.

C. SMD 12, -80 m; wriggilite skarn replacement of marble. The central fractures (2) have been filled in by a variety of vein materials. (1) refers to augen-type features near the intersection of major fractures. Field of view is 3 cm wide.

D. SMD 12, -80 m, wriggilite skarn replacement of marble showing the parallel relationship between the original fracture and the lamination of the skarn. (2) shows the interface between amphibole-sulfide replacement area and unreplaced wriggilite skarn. Field of view 3 cm wide.

E. SMD 16, -65 m, transmitted light photomicrograph of layered sphalerite base metal sulfide skarn after garnet or wriggilite skarn. The lamination is parallel to (1). (2) shows garnet partly altered to hematite quartz. Field of view is 1 cm wide.

F. SMD 9, -97.2 m, transmitted light photomicrograph of pyrrhotite replacement of primary laminar skarn. Relict wriggilite skarn lamination is parallel to (1) whereas a later superimposed fabric is shown by (2). Field of view is 1 cm wide.

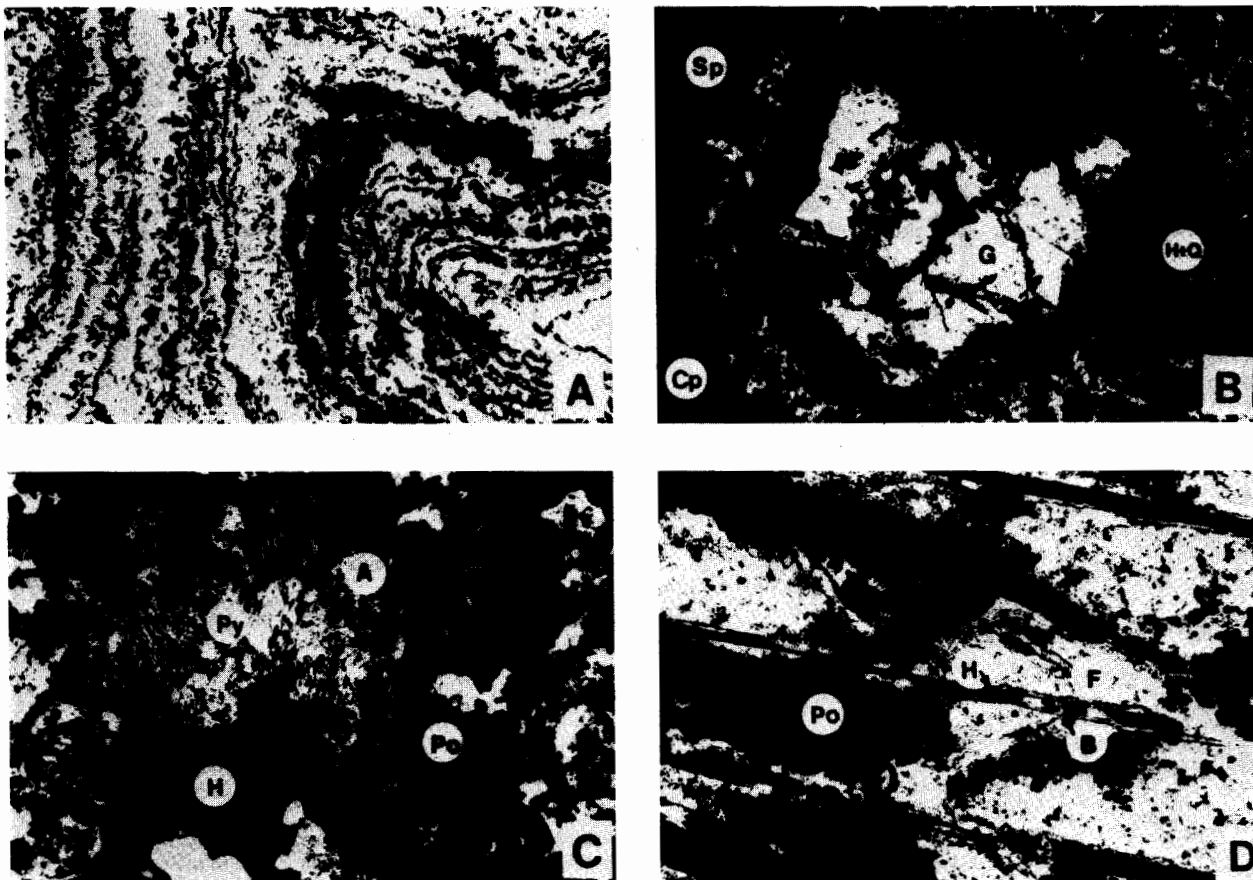


FIG. 6. Textures in primary wrigglyite skarn and replacement of this skarn.

A. Typical wrigglyite skarn showing that the dark (magnetite \pm cassiterite) lamellae are not continuous. The apparent foldlike form is due to irregularities in permeability (see text for explanation); SMD 5, -46.5 m. Field of view is 0.9 cm wide.

B. Replacement of garnet (G) by hematite + quartz (H + Q) with sphalerite (Sp) and chalcopyrite (Cp). Enlargement of replacement feature in Figure 5E; SMD 16, -65 m. Field of view is approximately 1 mm wide.

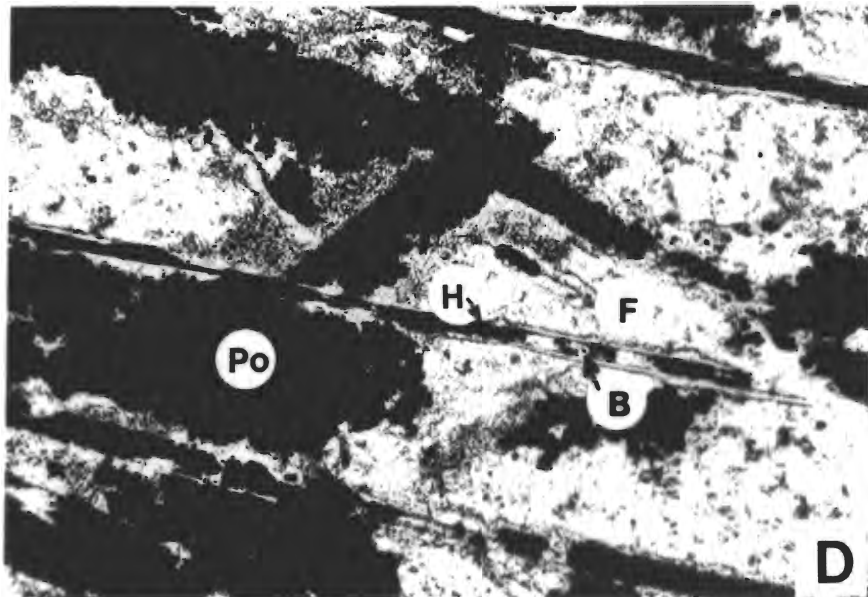
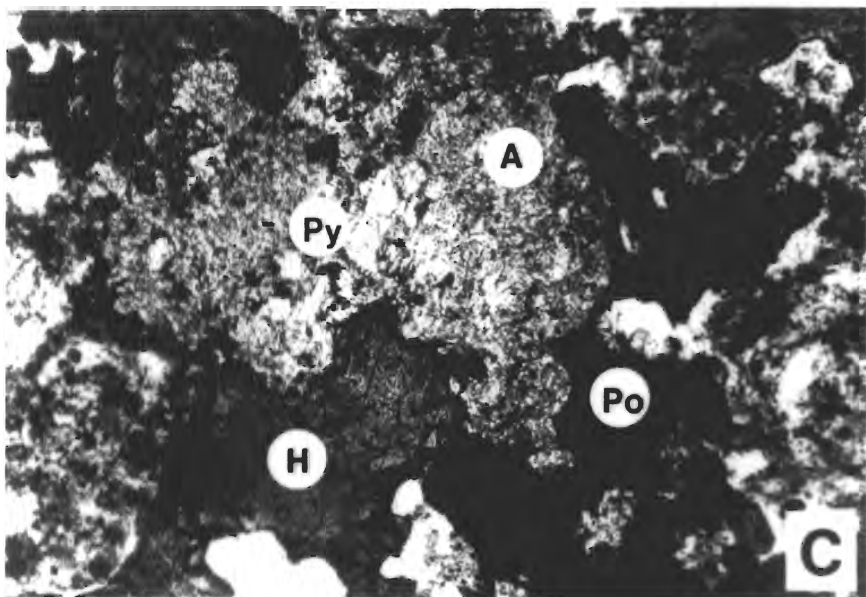
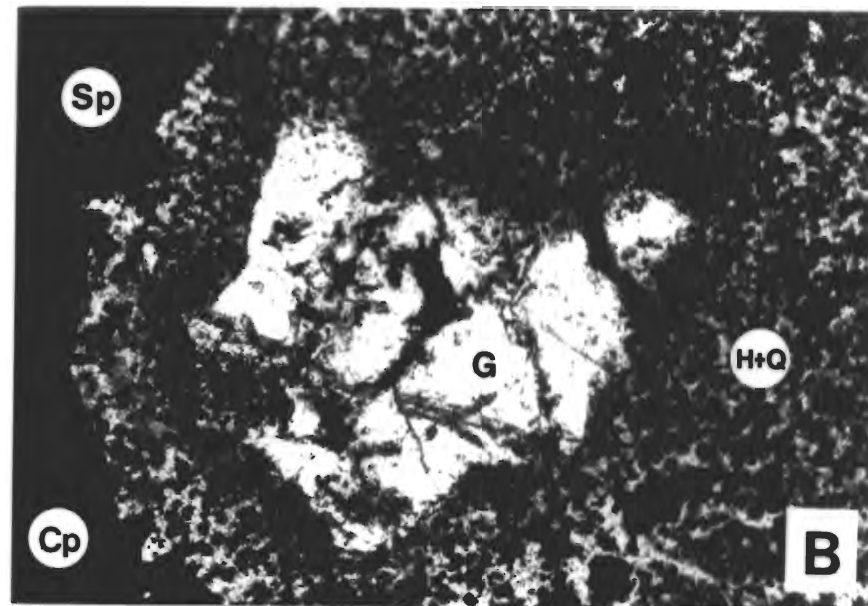
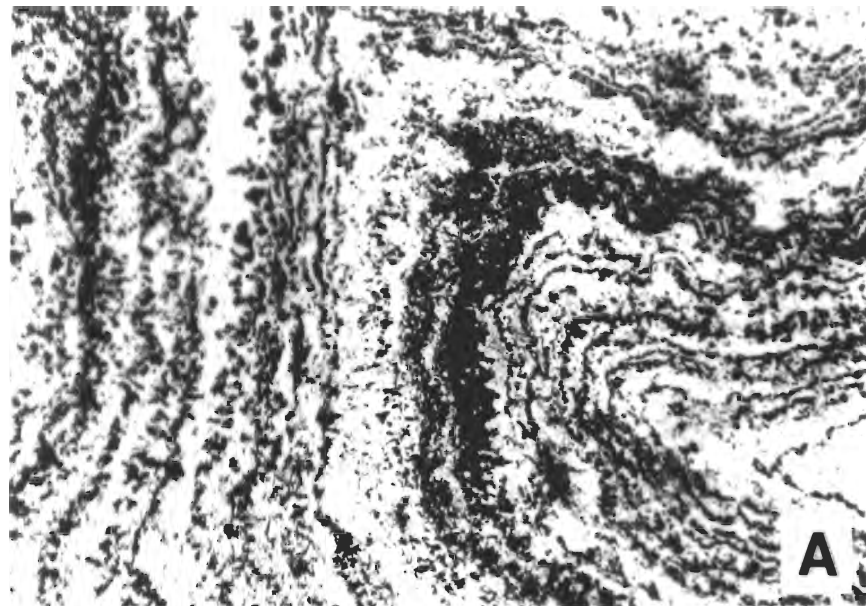
C. Replacement of pyroxene (Py) by amphibole (A) with associated hematite (H) and pyrrhotite (Po). Field of view is approximately 1.1 mm wide; SMD 16, -65 m. Some of the light areas are fluorite.

D. Replacement of wrigglyite skarn by pyrrhotite (Po) with fluorite (F) retained. Enlargement of a part of Figure 5F, SMD 9, -97.2 m. A later fabric consisting of hematite plates (H) with marginal F annite (B) has been superimposed upon the already replaced laminar skarn. This suggests hematite is a postsulfide replacement. Field of view is approximately 1 cm wide.

served, implies that boiling the hydrothermal solutions occurred at some time during genesis, and salinities were high (≈ 30 –40 equiv. wt % NaCl). The major constituents in the fluid phase are NaCl, (KCl?), H_2O , and CO_2 . Some CaF_2 occurs either as a daughter product or as accidental crystals. The first melting temperatures near $-20^\circ C$ confirm that NaCl (+KCl?) is the major salt present and that $CaCl_2$ is not present. Systems involving the latter salt show first melting temperatures near $-52^\circ C$ (the ternary eutectic in the system $CaCl_2$ -NaCl- H_2O).

Sulfur Isotopes

Table 8 shows sulfur isotope values from the Moina area done by Dr. Shen-Su Sun of the Division of Mineralogy, CSIRO. These values are relative to the Cañon Diablo standard and the analytical uncertainty is believed to be better than 0.2 per mil. As can be seen sulfides from wrigglyite, pyrrhotite-rich (amphibole), pyrite-rich (amphibole), and sphalerite-rich skarns have similar sulfur isotopic values (8.4 to 9.3‰) to those of pyrite in the greisenized granite



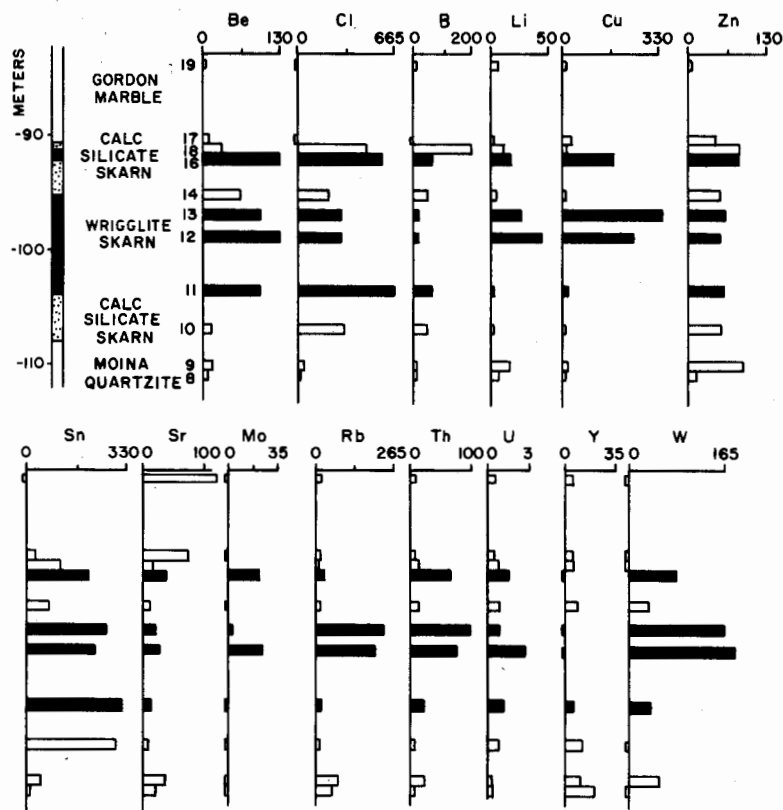


FIG. 7. Histograms of chemical variations through the quartzite-skarn-limestone section of SMD 12 (see Fig. 1A for location). The numbers 8 to 19 refer to the analyses in Table 4. The values are in moles per 100 cm³ of rock (to account for density differences). The dark bars to the right of the vertical line refer to wrigglyte skarn composition values. Values shown to the left of the vertical line indicate that the analysis yielded no values of the element.

(8.9‰). Pyrrhotite from SMD 14 (at 38 m depth below the collar) limestone and framboidal pyrite from the Moina sandstone have negative values (−4.2 and −6.9‰, respectively). Pyrite from metasomatized sandstone (SMD 18 at 102.5 m depth) and pyrite and pyrrhotite from the margin of the skarn (Moina) have intermediate values (≈4.5‰).

Sulfur isotope $\delta^{34}\text{S}$ values of hydrothermal fluid can be as much as 4 per mil larger than the $\delta^{34}\text{S}$ melt if the volume ratio fluid to melt is small (Ohmoto and Rye, 1979, p. 527). Average $\delta^{34}\text{S}$ values of silicic igneous rocks of as high as 10 ± 5 per mil have been suggested (Holser and Kaplan, 1966), and individual values as high as 30 per mil have been recorded (Shima et al., 1963). Presently accepted values, however, are approximately 0 ± 3 per mil (Ohmoto and Rye, 1979, p. 524).

The greisenized granite and skarn values (≈9‰) have a sulfur source which is quite constant but is unlikely to be purely magmatic in origin. Coexisting

pyrrhotite-pyrite pairs in SMD 9 (−102.50 m) and Moina 13 have similar values. On the basis of sulfur fractionation thermometers (Ohmoto and Rye, 1979, p. 518), this would suggest they crystallized at very high temperatures (>700°C), which is geologically unlikely, or they did not coexist stably (not formed at the same time). The latter explanation is most likely on textural evidence. Sulfur isotope values are known for the Renison Bell Sn replacement body (Patterson and Ohmoto, 1976). Here, $\delta^{34}\text{S}$ values of pyrrhotite in the ores are near 6.5 per mil and the primary (?) sulfide in the associated granite have $\delta^{34}\text{S}$ values near 3.5 per mil. Patterson and Ohmoto (1976) interpreted this to mean the sulfur has a magmatic source.

Discussion

Origin of wrigglyte skarn at Moina and elsewhere

Fluorite-rich wrigglyte skarns are known from many other localities including: Dagoon Mountains,

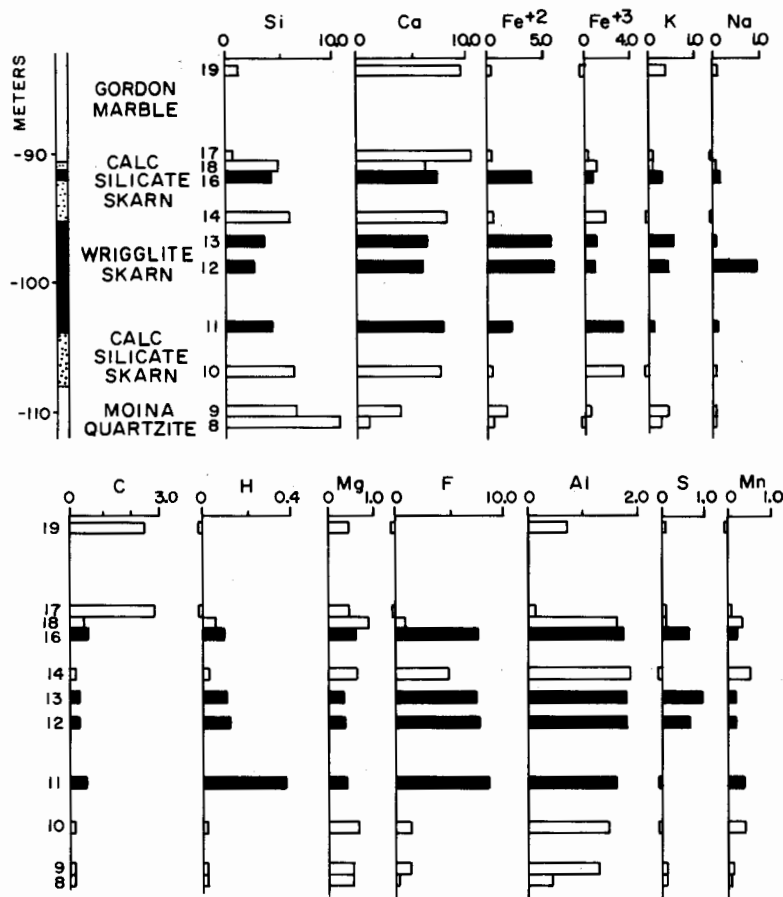


FIG. 7 (cont.).

Arizona, (Perry, 1964); Iron Mountain, New Mexico (Jahns, 1944b); Lost River, Alaska (Knopf, 1908; Sainsbury, 1964, 1969); south China (Meng, 1937; Hsieh, 1963; Beus, 1966); Chugako, Japan (Miyake, 1965); Kazakhstan, USSR (Beus, 1966; Zasedatelev, 1973; Ermilova and Senderova, 1959); Iten' Yrginsk, Chukotka, USSR (Shcherba, 1970; Getmanskaya, 1972); various other deposits in the Far Eastern Province, USSR (Govorov, 1958); Dal'negorsk, Primor'ye, USSR (Aleksandrov, 1975); Kristiana, Norway (Twelvetrees, 1913); Mt. Garnet, Queensland (Askins, 1976); and Mt. Bischoff, Tasmania (observed by one of us—P.W.A.). They are related to leucogranite plutons or dikes exhibiting greisenization and are usually found in Paleozoic limestone terrains (Govorov, 1968). There is invariably intense fracturing and faulting and, at least in some cases, boiling of the hydrothermal solutions can be demonstrated (evidence from the study of W. Brown at Mt. Garnet, pers. commun., 1979; at Moina and Mt. Bischoff, P. Collins, pers. commun., 1979). The primary skarn is enriched in

F, Fe, Sn, W, Be, Li, B, and Bi whereas secondary alteration commonly produces skarn rich in Zn, S, Pb, and Cu (e.g., at Mt. Garnet, Askins, 1976; Dal'negorsk, Aleksandrov, 1975; and at Moina). Anomalously high values of such elements as B, In, Cd, and Li may occur (Sainsbury, 1969).

Bulk chemical analyses of wrigglite skarn by Zasedatelev (1973), Miroshnchenko and Gulyayev (1978), Waite (1978); Sainsbury (1969), and the present authors indicate that when the fluorine content is less than about 9 weight percent the wrigglite structure does not form and granular skarn forms instead. In a cuspidine wrigglite skarn from Mt. Garnet (Askins, 1976), the lamination is indistinct and F \approx 9.0 weight percent.

Wrigglite skarn is fine grained, has a contorted structure, and is commonly associated with coarser grained, F-poor, granular skarn. Individual lamellae, which contain restricted mineral assemblages, pinch and swell and may contain crosscutting septalike veinlets of magnetite (at Moina) or tourmaline (at

TABLE 6. Correlation Coefficients of Elemental Abundances of Elements Relative to Others

	Pb	W	Bi	Sn	Mo	Cu	Zn	Be	F	Sc	Y	Cd	In	Ge	Au	Ag
Pb																
W	262															
Bi	263	344														
Sn	285	401	364													
Mo	229	313	305	321												
Cu	278	265	260	282	228											
Zn	290	324	324	347	284	287										
Be	284	359	358	382	316	281	346									
F	287	364	364	387	321	284	349	384								
Sc	56	130	132	145	109	57	109	50	148							
Y	56	125	127	140	104	57	105	145	143	144						
Cd	20	8	18	20	6	20	20	20	20	20	20					
In	41	25	33	40	16	42	42	42	41	42	42	18				
Ge	19	13	17	19	8	19	19	19	19	19	19	8	15			
Au	93	80	85	96	73	90	96	90	96	46	46	18	35	18		
Ag	25	20	24	25	23	17	25	19	24	0	0	0	0	0	20	

The upper right hand part of the table gives the correlation coefficients between elements and the lower left hand part gives the number of pairs used. When the concentration of an element was zero, the value was not used for correlation. The program used was written by Ms. Judy Hannah of the Department of Geology, University of California, Davis, California. The program fits the data to a linear function and calculates correlation coefficients.

TABLE 7. Correlation Coefficients of Elemental Abundance of Economically Important Elements in Weight Percent vs. Depth in the Skarn Profile

Also given are the range of values and the average value for each drill hole. All rock types that contain mineralization are included, of which wriggelite skarn predominates.

Traverse 1	W	Bi	Sn	Mo	Cu	Zn	Be	F
DDH 4 (32 analyses each)								
Cor. coef.	-0.214	-0.678	-0.755	-0.026	-0.249	-0.362	-0.650	-0.751
Range	70-2,220	0-780	30-2,100	0-150	5.350	18-1,240	70-700	0.63-12.89
Average	744	305	1,178	43	117	130	266	5.86
DDH 11 (42 analyses each)								
Cor. coef.	0.162	-0.678	-0.357	0.301	0.165	-0.027	-0.411	-0.535
Range	50-6,500	8-820	620-2,950	0-370	10-350	30-450	70-500	2.91-12.41
Average	1,040	402	1,490	50	120	116	252	7.98
DDH 6 (70 analyses each)								
Cor. coef.	-0.073	-0.266	-0.372	0.380	-0.234	0.133	-0.036	-0.587
Range	55-4,600	0-2,100	46-5,000	0-360	5-190	45-4,550	70-600	1.51-14.60
Average	879	430	1,302	62	45	216	2.44	7.74
DDH 15 (41 analyses each)								
Cor. coef.	0.171	-0.680	-0.672	0.355	ND	ND	-0.391	-0.699
Range	45-3,650	0-700	60-3,600	0-150	ND	ND	80-500	0.73-12.41
Average	1,092	403	1,321	43	ND	ND	256	7.88
DDH 7 (60 analyses each)								
Cor. coef.	-0.193	-0.305	-0.780	0.104	-0.232	-0.200	-0.264	-0.268
Range	95-3,400	0-2,050	180-2,750	0-290	2-440	55-9,800	80-300	1.31-14.07
Average	878	391	1,169	47	46	366	200	8.17
Traverse 2								
DDG ML-3 (38 analyses each)								
Cor. coef.	0.307	0.364	-0.481	-0.209	-0.078	0.000	-0.532	-0.432
Range	0-1,400	0-1,060	1-8,200	0-2,075	0-910	10-435	1-370	0-15.00
Average	317	259	2,223	245	64	147	180	6.69
DDH 16 (53 analyses each)								
Cor. coef.	-0.018	0.049	-0.059	0.052	ND	-0.096	0.292	0.120
Range	15-6,500	20-2,000	190-2,650	0-200	ND	55-154,000	50-5,000	0.34-12.05
Average	728	416	933	42	ND	11,852	160	5.14
DDH 13 (57 analyses each)								
Cor. coef.	0.213	-0.180	-0.361	0.434	-0.228	-0.082	0.312	0.244
Range	0-1,850	0-2,100	8-720	0-34	2-910	42-275,000	2-1,000	0.01-5.60
Average	69	266	299	9	151	24,182	81	0.60

The samples represent analyses of split core of 1 m length, in most cases. The positions of the drill holes are shown in Figure 1A. W, Bi, Sn, Mo, Cu, Zn, and Be are in terms of ppm while F is in terms of weight percent. Values of Pb, Sc, Y, Cd, In, Ge, Au, and Ag are not included because they are only minor constituents of the skarn. Their ranges in ppm are Pb (0-110), Sc (0-20), Y (0-70), Cd (0-1340), In (0-90), Ge (0-5), Au (0-4.5), and Ag (0-2).

Lost River). The following mineral assemblages, occurring in the alternating dark and light lamellae, respectively, were observed by the authors in skarn(s) from different areas: (a) magnetite \pm cassiterite: vesuvianite + fluorite (most common); (b) magnetite + cassiterite: adularia + vesuvianite + fluorite (rare, Moina); (c) magnetite \pm cassiterite: hedenbergite + fluorite \pm andradite (rare, Moina); (d) f annite + magnetite: fluorite (rare Mt. Garnet); (e) magnetite rimmed by gahnite: f cuspidine + vesuvianite + fluorite (rare, Mt. Garnet); (f) pyrrhotite + cassiterite

\pm stannite: F biotite + F-Na tourmaline + fluorite + sellaite (rare, Mt. Bischoff); and (g) chrysoberyl + F margarite + F-Na tourmaline: fluorite + F margarite (Lost River).

Other common minerals are: helvite, danalite, hematite, spinel, lithian mica, and Ca-rich plagioclase (Govorov, 1968; Shabynin, 1977). Wriggite-like skarn consisting of alternating magnetite and garnet has been recorded from the Kearney mine, Hanover, New Mexico (D. M. Burt, pers. comm., 1979).

Different authors believe that wriggite skarns rep-

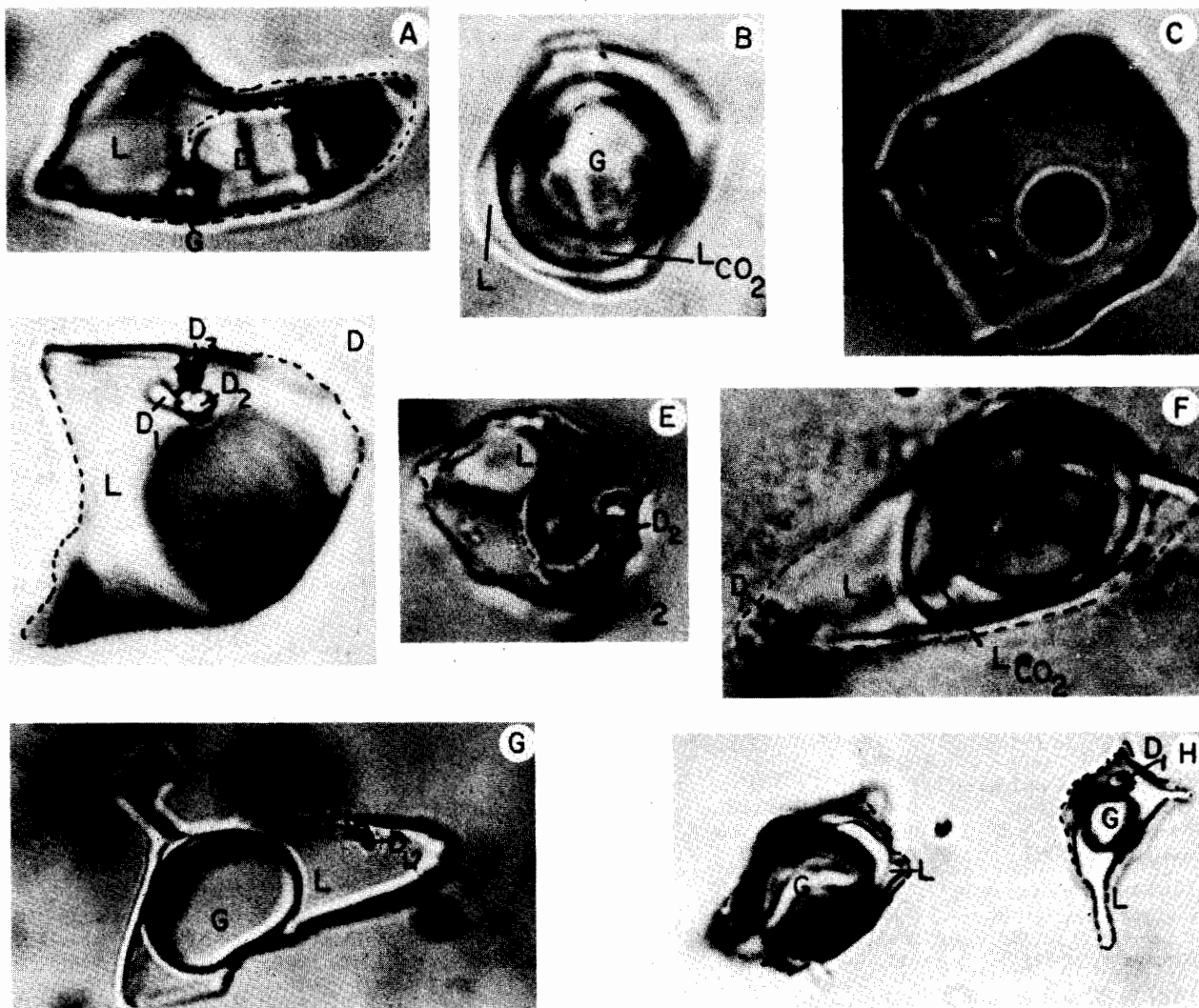


FIG. 8. Fluid inclusions in minerals associated with the wrigglyte skarn. A, B, C, D, E, F, and H are from a quartz (-cassiterite-fluorite) vein in SMD 12 at 155 m. Note the liquid CO_2 (L_{CO_2}) daughter products (D), gas (G), and liquid (L). D_1 = NaCl, D_2 = fluorite, and D_3 = probably hematite. The coexisting gas-rich and liquid-rich fluid inclusions (H) have been interpreted to indicate that boiling occurred during trapping. G shows a highly saline fluid inclusion in topaz (described in text). All the fluid inclusions are generally 15 to 30 microns wide.

resent: (1) bedding in an unusual sediment (Zasedatelev, 1973), (2) replacement of fine bedding and/or stromatolites (P. W. Stainton, pers. commun.), (3) colloform, solidified gel precipitates (as in Stevenson and Jeffery, 1964), (4) rhythmical deposition in cavities (Trustedt, 1907), and (5) deposition by replacement at diffusion fronts moving outward from fractures (Knopf, 1908; Jahns, 1944a; Eskola, 1951; Beus, 1966; Sainsbury, 1969; Georgievskaya, 1955; Shabynin, 1977).

In Zasedatelev's model, wrigglyte skarn is formed

as chemical sediment in small, strongly evaporated basins, the unusual components (F, Be, Sn, W, etc.) being attributed to thermal springs of volcanic association. The rhythmic layering is a possible seasonal effect and chemical deposition is a colloidal precipitate producing colloformlike textures. Diagenesis, regional metamorphism, and finally contact metamorphism by a granite caused recrystallization of the sediment to its present mineralogical composition. The evidence against such an origin is strong. The laminations consist of folded, commonly pipelike

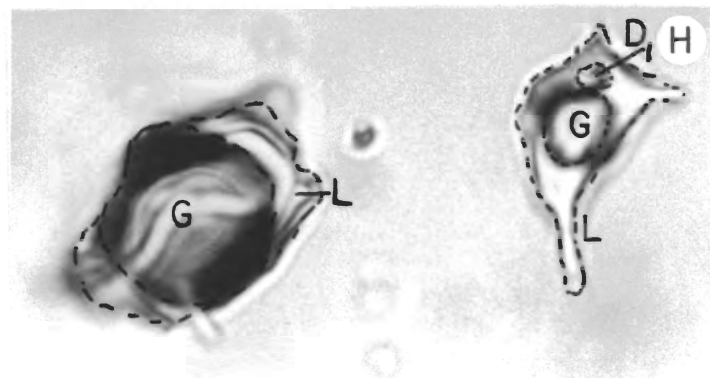
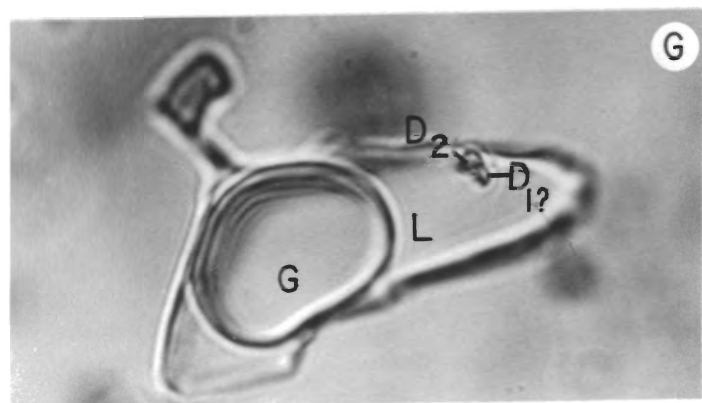
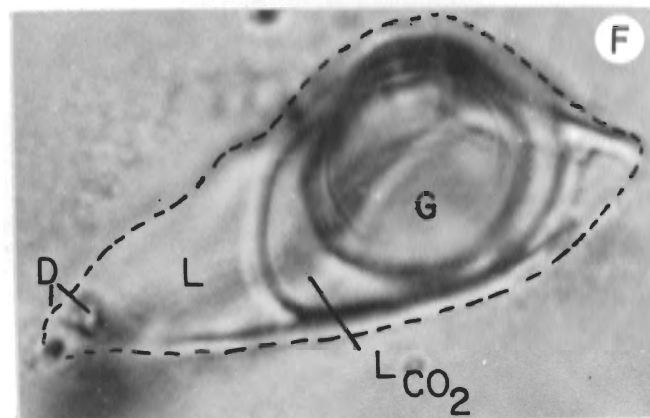
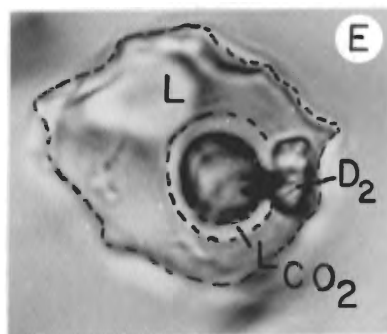
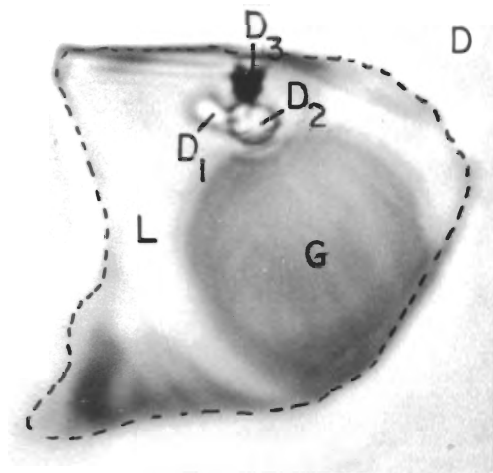
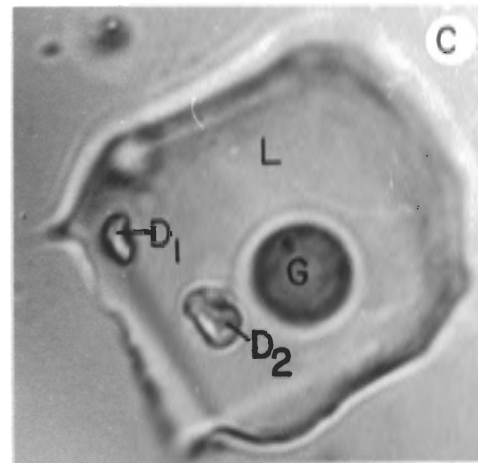
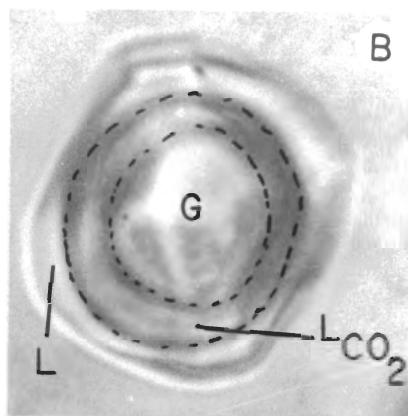
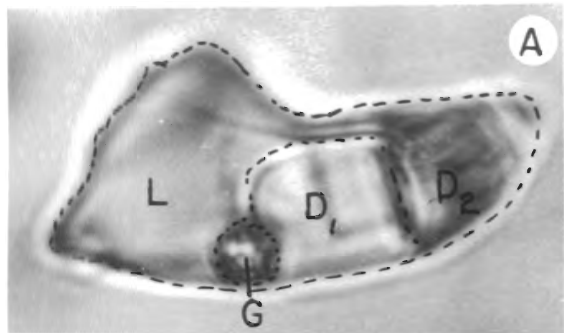


TABLE 8. Sulfur Isotope (δ^{34}) Values Relative to Meteoric Sulfur

The sample numbers refer to drill holes and depth shown in Figure 1A. The depths are in meters down from the collar of the drill hole. Numbers such as Pait, Moina 11, 12 refer to specific surface samples collected. The analyses were performed by Dr. Shen-Su-Sun, CSIRO Laboratories, North Ryde, N.S.W.

Drill hole and distance of sample down hole	Rock	Mineral	Value $\delta^{34}\text{S}$	Remarks
ML-1, 237 m	greisenized granite	pyrite	8.9	top of cupola is at 279 m
SMD 18, 102.50 m	metasomatized impure sandstone	pyrite	4.5	between wriggelite skarn and granite
Pait, Moina 11, 12	wriggelite	pyrite	8.8, 9.1	surface sample from near mill site, Shepherd and Murphy Mine
SMD 9, 102.50 m	pyrrhotite-rich skarn	pyrrhotite	8.6	pyrrhotite after magnetite wriggelite
		pyrite	8.4	
SMD 14, 38 m	fine-grained dark dolomitic interbed in limestone	pyrrhotite	-4.1	above wriggelite, unaltered or metamorphosed
SMD 13, 93.50 m	sphalerite-rich skarn	sphalerite	9.3	sulfide-replaced wriggelite?
		pyrite	8.4	on NE side of Bismuth fault
Surface sample	fine-grained dolomitic layer in limestone	pyrrhotite	4.4	1.5 km from skarn
		pyrite	4.6	
Pait, Moina 10	pyrite-rich skarn	pyrite	9.1	1.5 km from Moina skarn
Pait, Moina 7	framboidal pyrite, Moina sandstone	pyrite	-6.9	5 km from skarn

shapes which are not geometrically equivalent to folds in sedimentary bedding. The early stages of wriggelite skarn formation can clearly be demonstrated to be a replacement of preexisting limestone along fractures (Fig. 5C and D).

Superficially many wriggelite skarns appear to be a very selective specialized replacement of stromatolite layering (model 2). However, no stromatolites exist in the unreplaced carbonate host rocks known to the authors and in any case replacement demonstrably takes place outward from fractures, not specifically along a preexisting layering.

In model 3, limestone is progressively dissolved by acidic fluoride solutions and at the same time iron hydroxide and fluorite form as a gel precipitate, separating out by mutual attraction into separate layers as replacement proceeds. The gel crumples as replacement of the limestone proceeds irregularly, and the gel slowly ages to the present texture and mineralogy. Various similar processes have been proposed by Stevenson and Jeffery (1964), Boydell (1925) (quoted in Lovering, 1962), and Garrels and Dreyer (1952), for replacement of limestone by magnetite or sulfides. The model does explain the colloformlike texture of wriggelite skarn. However, Roedder (1968) demonstrated that colloformlike textures do not necessarily indicate that colloids have been present. Continuous layers in a gel when deformed would produce concentric folded shapes, not the reentrant angles commonly observed, and the volumetrically significant shrinkage cracks that occur in real gels are not

observed in the lamination. If a gel intermediate stage existed on a large scale, the rock would have little strength and would be structurally unstable.

The model proposing that wriggelite skarns are crustiform precipitates in cavities (model 4) was first favored by Trustedt (1907). Presumably acid fluids infiltrate through limestone channels and produce fissurelike and pipelike openings. In the forward zone of the advancing front of fluids, the fluid passing by a given point is neutral to alkaline, and fluorite and magnetite (etc.) begin to precipitate. Slight fluctuations in the pH of the fluid cause the rhythmic layering until the cavity is filled or nearly filled. Crustiform layers necessarily build up from the walls of a cavity and are parallel to the walls. However, at Lost River and Moina, delicate argillaceous beds are preserved in wriggelite skarn whose layering is continuous on either side of the bed (Fig. 4C; and Sainsbury, 1969) suggesting that cavities could not have been present.

In model 5, fluids infiltrate along areas of high permeability while the components in the fluid replace and diffuse into the marble due to activity (and concentration) gradients. Wriggelite skarn forms by replacement at diffusion fronts advancing from the highly permeable area until the effective limiting distance of diffusion from the fracture is reached or until the fluid infiltrating along the fracture is spent. There are irregularities in the advancing diffusion fronts caused by slightly differing rates of diffusion from one place to another resulting in local thickening

and thinning. This produces the irregular fold forms in the wrigglyte skarn. The last phase in the process is the filling of the fracture itself, to produce a coarser grained central vein of fluorite and other minerals, or further infiltration occurs producing alteration of wrigglyte skarn by fluid(s) of different composition.

The mechanism to explain the rhythmic nature of the skarn is probably that discovered by Leisegang (quoted in Knopf, 1908). If a drop of AgNO_3 is placed upon a gelatin plate impregnated with $\text{K}_2\text{Cr}_2\text{O}_7$, a series of concentric rings of $\text{Ag}_2\text{Cr}_2\text{O}_7$ forms. These become progressively more widely spaced with increasing distance from the center, although the widening of the rings outward does not take place if the concentration at the center is kept constant by a continuous influx of material (Watanabe, 1924b). Five theories to explain the phenomenon and about 600 papers have been produced, as summarized by Stern (1954). Stern concluded that Ostwald's 1897 explanation is the most satisfactory. He believed that dichromate ions diffuse inward as Ag ions move outward. The continued diffusion caused super saturation of $\text{Ag}_2\text{Cr}_2\text{O}_7$ and precipitation at a front normal to the diffusion direction. Just beyond the front there is a zone of low concentration of dichromate and here Ag diffuses outward through this zone until it reaches inward-diffusing dichromate and supersaturation occurs once more, forming a new ring. The process can take place in solids and even in water (e.g., Leisegang, 1931; Stern, 1954). Watanabe (1924a) conducted experiments on the diffusion of mixed zinc and iron sulfate solutions through a sodium sulfide-impregnated gel. He produced alternating ZnS and FeS rhythmic layers. One boundary condition is that if the solution contains much Zn^{+2} and less Fe^{+2} , no rhythmic layering occurs, presumably because FeS can only precipitate when Zn^{+2} is exhausted at the leading edge. This rhythmic layering was found to be metastable, with the system eventually reverting to a granular structure. This may also be true of wrigglyte skarns.

Beus (1966) and Eskola (1951) have attempted to explain the mechanism in exact terms. Beus believes that the leading front of the acidic solution is partly neutralized by reaction with carbonate and Fe oxide precipitates. The solution, now containing F-Al-Si-Be-Sn, etc., diffuses through this Fe oxide layer and a layer of fluorite plus other calc-silicate minerals precipitates. A new front of solution diffuses through this couplet, a new couplet forms, and so on. Beus's model is questionable because it does not explain the origin of wrigglyte skarn in which no Fe oxide phase is present, such as some that occur at Lost River, Alaska. It also offers no adequate explanation for the fact that wrigglyte skarn does not generally form in skarns poor in F, where granular skarn forms.

The mechanism proposed here is that extreme supersaturation of one component is necessary so that minerals containing that component (usually F) nucleate rapidly. Because of the extreme supersaturation of F in the carbonate environment, many small crystals of F minerals form from the first layer. When this supersaturated component is depleted in the solution, other elements diffuse or percolate through the first formed layer to form the next wrigglyte consisting of other minerals.

In the examples studied, when the thickness of successive couplets reaches a critical thickness of between 1 mm and 5 cm, fracturing and replacement of unreplaced marble occurs. At both Moina and at Mt. Garnet this critical thickness is greatest nearest the contact with the granite and least nearest the marble. Repeated periods of fracturing and vein filling explain the sequential nature of the vein contents shown in Figure 3. By sequential fracturing and replacement the entire marble unit is replaced (Fig. 4C) or if repeated fracturing does not occur, unreplaced marble is retained (Fig. 5C and D). The fracturing is interpreted to be in response to clogging of the system when the constituents of the solution can no longer diffuse across the distance to unreplaced marble and instead precipitate in the fracture system. The pressure needed to cause the fracturing may be caused by development of a CO_2 overpressure due to reactions occurring elsewhere in the skarn and, possibly, due to a tectonic component. This hypothesis best explains why fracturing is most intense within and peripheral to the skarn unit but not nearer the granite at Moina. The augen-type features containing granular skarn (Fig. 5C) are believed to represent the replacement of embayed marble where the conditions necessary for wrigglyte skarn growth were not met (slow, systematic loss of CO_2 ?).

The rhythmic nature of the lamination could be caused by a constant supply of new solution passing through the fracture system as described by Watanabe (1924a). Alternatively, it could be caused by periodic opening and closing of the system at the end of each period of couplet formation. Spent CO_2 -rich fluid would be flushed out of the system and wrigglyte skarn formation would proceed from new introduced fluid. The periodic loss of volatiles (CO_2) may be essential for wrigglyte skarn genesis because some skarns do contain much fluorine with Ca-Si silicates but do not have a wrigglyte structure. This needs more study.

A model for the genesis of the skarn, quartz veins, and greisen at Moina

The assertion that the skarn, Sn-quartz veins, and greisen are cogenetic is supported by their close spatial relations and unique chemistry. As is shown by

the relict granite fabric and relict mineralogy in the greisen, fine-grained granite crystallized first. Its mineralogy was K-feldspar, plagioclase, quartz, F annite, and fluorite. The anomalous mica composition suggests the granite crystallized from a high f_{F_2} melt having high Fe/Fe + Mg and/or f_{O_2} , and relatively high SiO_2 . The aqueous phase which evolved could not have been greatly out of equilibrium with the granite and some other process was needed to produce the greisenization.

When the pluton developed a solid granite carapace and dense surrounding hornfels, evolving aqueous solutions from lower levels were partly trapped. Rupturing of the carapace either by tectonic or fluid overpressure decreased P_{fluid} from values of $P_{lithologic}$ or more to values near $P_{hydrostatic}$. Boiling occurred producing an acidic, volatile-rich (HF, HCl, H_2S , etc.) vapor phase and a basic saline aqueous phase. As such, the system resembles a (small) porphyry copper system. The vapor phase caused greisenization of the granite and hornfels and facilitated the transport of much F and associated elements.

At Moina and many other wriggilite skarn deposits, massive granular skarns commonly occur between granite and wriggilite skarn (Govorov, 1958). The granular skarns consist mainly of garnet, vesuvianite, magnetite, pyroxene, and some fluorite. The garnet may be Sn rich (0.70 wt % at Moina to 0.3 at Iron Mountain, Jahns, 1944b) whereas vesuvianite commonly is Be bearing (to 1.30 wt % at Iron Mountain, Jahns, 1944b). Sn values in granular skarn are commonly high (e.g., in SMD 12, this study; at Iron Mountain, Jahns, 1944b), indicating that associated granular skarn is genetically related to the F-Sn-Be-bearing mineralization. However, this granular skarn has invariably been cut and partly altered by fluorine-adularia (-scheelite) veins (i.e., see Fig. 4B of this study for Moina; p. 80 of Sainsbury, 1969 for Lost River; p. 59 of Jahns, 1944b, for Iron Mountain). The veins are interpreted to have been the conduits for solutions which replaced marble to form wriggilite skarn higher in the sequence. The interpretation suggested here is that the granular skarn and hornfels were produced cogenetically with the granite and that the processes of greisenization, alteration of granular skarn, and formation of wriggilite skarn are associated with later boiling of the solutions.

The alteration of primary wriggilite and calc-silicate skarn by Zn-Cu-S-Cd-In-Au (group 2) precipitating solutions has been reported elsewhere (e.g., Aleksandrov, 1975) and occurs at Moina. At Moina, a zonal arrangement occurs on a regional scale with wolframite-cassiterite and accessory molybdenite nearest the Dolcoath granite; cassiterite with accessory wolframite and bismuthinite farther out; and Bi-

Au and Ag-Pb-Au(-Zn) mineralization at the outer edge (Jennings, 1965, p. 513). A similar zonation associated with Sn-F skarn has been reported by Miroshnechenko and Gulyayev (1978, p. 54). The replacement of group 1 elements by group 2 metals occur outward from the Bismuth Creek fault. The alteration is interpreted to have occurred during the time when the system telescoped inward near the end of the mineralizing episode, superimposing base metal mineralization on group 1 mineralization. That primary (group 1) mineralization occurred by way of multiple fractures and faults (see Fig. 1A, magnetics), best explains the relatively even concentration of Sn, F, and Be in different drill holes. These fractures were eventually sealed by the precipitation of Sn-W-quartz veins.

The local alteration of the skarn to a hematite-quartz assemblage may have postdated the sulfide replacement as suggested by textural evidence (Fig. 5F). Whether the crystallization of type 2 mica, found associated with hematite in the greisen, is associated with hematite crystallization in skarn is not known. Presumably these minerals could have been produced when cooler, mainly meteoric water circulated through the system near the end of the mineralizing episode. Laumontite, which occurs in fractures in the skarn, may also relate to this episode, or may have formed later. The unreplaced limestone and siltstone above and peripheral to the skarn are of low metamorphic grade because they contain epidote + tremolite and calcite + quartz assemblages (Fig. 4E). The skarns, however, represent higher grade metamorphism as is indicated by the assemblage wollastonite + garnet + pyroxene. This, with the unique composition of all the skarns, suggests the skarn forms a pocket of high-grade metamorphism superimposed on a lower grade environment. The hydrothermal solutions may thus be the main source of the heat for the high-grade metamorphism and the heat was not produced from outward-moving isograds in response to conductive heat gained by the rocks from the cooling pluton.

Relation to other Sn-F replacement deposits

The relation between Sn-bearing wriggilite skarn deposits and massive Sn sulfide replacement deposits such as the Renison-Bell (Patterson, et al., 1981), Mt. Bischoff (Orr, 1976; Groves et al., 1972), and Cleveland (Palmer, 1976) deposits of Tasmania is unclear. These examples are replacements of both dolomite and limestone, and the mineralogy includes cassiterite, fluorite, and pyrrhotite. In each case extensive faulting occurred. However, magnetite-bearing wriggilite skarns occur near granite contacts (<250 m) and the other sulfide Sn-F deposits occur farther away (>500 m). Parts of the Mt. Bischoff and part of the

Moina deposits (Fig. 5E) appear to be transitional in having the assemblage pyrrhotite + fluorite + F-biotite \pm cassiterite and also in being wriggilite.

Limited fluid inclusion data indicates that wriggilite skarns are at least partly produced from a boiling, high-temperature, saline solution whereas the non-wriggilite Sn replacement deposits are produced from nonboiling, lower temperature, and less saline solutions. The data of Collins (1979) suggest that for fluid inclusions studied from the Renison Bell and Cleveland deposits, homogenization temperatures range from 250° to 430°C, salinities are 10 to 15 weight percent (NaCl equivalent), and no evidence of boiling occurs. At Mt. Bischoff boiling appears to have occurred (Collins, 1979) with homogenization temperatures near 530° to 510°C and with fluid inclusions containing cubes of sylvite and halite (therefore, >26 equiv. wt % NaCl). Similar data has been produced in this study and in a study of the Mt. Garnet wriggilite skarns (study in progress by W. Brown and T. A. P. Kwak).

The relative importance of dolomites versus limestones as hosts is unclear because wriggilite skarn forms in dolomites (e.g., at Pitkaranta, USSR, Trustedt, 1907; at Chukotka, USSR, Getmanskaya, 1972; at Mt. Bischoff) as well as limestones.

Although more study is clearly needed, the above data can be used to imply that when suitable carbonate horizons exist near a "Sn"-leucogranite (<250 m) with the required plumbing system (i.e., faults), the result is the precipitation of a magnetite-cassiterite-fluorite wriggilite skarn from a boiling, high-temperature saline solution. However, if the carbonate horizon occurs at great distances from the pluton (>700 m) and the required permeability exists (such as the mineralized Bassett Federal fault at the Renison Bell deposit) and there is a source of sulfur, a massive cassiterite-pyrrhotite-fluorite-stannite replacement body forms from a lower temperature, nonboiling, more dilute solution. This has important economic consequences because only the latter type contains cassiterite in a form and quantity that is economically recoverable under present conditions.

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MOINA ORE AS A SOURCE FOR PRODUCING ACID GRADE FLUORSPAR**A METALLURGICAL APPRAISAL FOR JORC COMPLIANCE****1.0 Introduction and Purpose of Appraisal**

TNT Mines holds the title for the poly-metallic deposit located at Moina in northern Tasmania. Extensive exploration of the licence area during the 1970s and 1980s established the presence of substantial mineralisation containing fluorspar, magnetite, scheelite, tin, copper, zinc, bismuth, molybdenum, gold, and garnet. The mineralisation lies in a fine-grained and rhythmically fine layered skarn in which the valuable minerals are closely associated with each other and with gangue. The primary values are fluorspar, tungsten, tin, and bismuth and a sample of recent drill core submitted for detailed mineralogical analysis was assayed with the following head grade:

F	Sn	W	Bi	Zn	Mo	Fe
7.90%	0.16%	0.06%	0.05%	0.02%	0.005%	18.50%

In order to advance the development of the project, TNT Mines intend to complete sufficient further drilling to define the deposit as a JORC compliant resource. After determining the JORC compliant grade of the resource, the resource limits, and the cut-off grade, TNT Mines intends to select representative drill core for metallurgical feasibility testing. However the presence of fluorspar, as an industrial mineral, imposes the need for a judgement in respect of technical and economic factors likely to influence the prospect of economic beneficiation of the fluorspar in whole or in part as a marketable product.

The purpose of this brief appraisal is therefore to provide that assessment. As with all forward-looking judgements, it is important to clarify that the purpose of the feasibility testwork is to seek technical solutions to potential processing difficulties and compile

robust empirical evidence which justifies the metallurgical process design. The feasibility testwork data will also underpin the economics of processing the ore into a marketable product. Any form of forecast or judgement made in advance of that empirical evidence becoming available will be based on similar industry practice and experience rather than robust empirical data; thus this forecast will be entirely subject to the limitations of subjective specialist knowledge and cannot be construed as definitive. This appraisal will focus on the fluorspar component of the ore and will include the other valuable minerals only in so far as they impact on the economics of the fluorspar recovery.

2.0 Background

For clarity, the key background information relevant to this appraisal is listed in bullet format:

- By world standards, the Moina deposit is one of the largest fluorspar deposits.
- The minerals which comprise the fine-grained rhythmically contorted layers of mineralisation known locally as ‘wrigglite’ are not difficult minerals to separate when coarse in grain size; it is the unusually fine grain size and unusually intimate mineral associations in Moina ore which present the processing challenges.
- Preliminary conventional flotation testwork was undertaken in the 1970s which, after only nine tests, concluded with a 93.1% CaF_2 concentrate containing approximately 2.0% Fe_3O_4 , 2.5% SiO_2 , 1.0% CaCO_3 , and 0.4% WO_3 at a recovery of 65% CaF_2 . Although the grade did not reach acid grade specification, this was a commendable result after so few tests and demonstrates the order of magnitude of fluorspar recovery that one might expect when approaching acid grade specification.
- During the same period, research was carried out to search for potential roasting and pyrohydrolysis processing routes which would allow extraction of the fluorine without liberation of the fluorspar. Unfortunately both routes were found to be uneconomic.
- In February 2011 a detailed Qemscan® mineralogical study was completed which provided key data on all the mineral values and their liberation sizes.
- In addition the Qemscan® raw data was reprocessed to simulate the mineralogically-limiting grades and recoveries potentially achievable by gravity separation, flotation, and magnetic separation. This reprocessed data created the primary justification for forecasting the processing potential of Moina wrigglite ore.

3.0 Key Technical Data

A summary of the key currently known technical data is listed below:

- 3.1 Not all the fluorine (F) in wriggly ore originates from fluorite. Fluorite accounts for 92% of the total F and the remaining 8% is contained in vesuvianite, calc-silicates, biotite, commingtonite, chlorite, sericite, and diopside.
- 3.2 Calcite and calc-silicates comprise 17% of the ore, thus the combination of non-fluorite F and non-fluorite Ca adds to the complexity of assaying samples for fluorite (including drill core samples) and interferes with the interpretation of metallurgical testwork results.
- 3.3 The majority of acid grade fluorspar sold internationally meets a particle size specification with $K_{80} > 63 \mu\text{m}$. At this particle size the Moina mineralogically-limiting CaF_2 recovery is ~65% which equates to ~59% recovery of total F.
- 3.4 After grinding to $K_{80} = 45 \mu\text{m}$ the mineralogically-limiting CaF_2 recovery increases to ~70% and after additional grinding to $-20 \mu\text{m}$, the mineralogically-limiting CaF_2 recovery rises again to ~83%.
- 3.5 Using the raw Qemscan® data, a processing simulation sequence was completed as follows:
 - Primary grind $K_{80} = 75 \mu\text{m}$;
 - Magnetic separation to remove free magnetite;
 - Bulk sulphide flotation;
 - Bismuth separation;
 - Reverse flotation of easily floatable free gangue;
 - Bulk calcium mineral flotation;
 - Regrind calcium mineral concentrate to $K_{80} = 45 \mu\text{m}$;
 - Magnetic separation to remove newly-liberated magnetite;
 - Cleaning of bulk Ca mineral concentrate to remove newly-liberated gangue;
 - Gravity separation to remove scheelite and cassiterite concentrates;
 - Regrind fluorspar concentrate to $-20 \mu\text{m}$;
 - Fluorspar cleaner flotation to acid grade spec.
- 3.6 The resulting mineralogically-limiting CaF_2 recovery was 83% at $-20 \mu\text{m}$ which equates to 76% total F recovery.

- 3.7 Similarly, at a target grade of 60% WO_3 , the mineralogically-limiting scheelite recovery was 87% which equates to a WO_3 recovery of 83%.
- 3.8 Using the same flowsheet and a target grade of 70% Sn, the mineralogically-limiting cassiterite recovery was 45% which rose to 75% after fine regrinding to -20 μm . However, as only 48% of the Sn is contained in cassiterite (the balance is in garnet) the mineralogically-limiting recovery of Sn was 22% without regrinding, rising to 36% after regrinding to -20 μm .
- 3.9 Finally, since the bismuth concentrate will likely be leached in the subsequent up-grading process, the recovery will benefit from a low grade concentrate in the range 10% Bi to 20% Bi. At 10% Bi the mineralogically-limiting Bi recovery was 73%.

4.0 Potential Processing Route

Based on the above key technical data and standard fluorspar industry practice it is feasible that magnetite, bismuth, pyrite, scheelite, cassiterite, and fluorspar concentrates could be produced by conventional gravity, magnetic, and flotation separation circuits. The bismuth, scheelite and cassiterite would meet saleable concentrate specifications without further processing but the magnetite and fluorspar would require agglomeration treatment before meeting currently accepted saleable specs. For the magnetite, simple low-cost conventional briquetting would make the product saleable, but the fluorspar would require either agglomeration or high-pressure briquetting followed by controlled attrition neither of which have a track record in the fluorspar industry. There are no obvious technical reasons to suggest that either of these routes would not be feasible or economic, but, since no fluorspar industry experience currently exists in this area, further predictions cannot be made without testwork results.

Having established that there appear to be no currently known fatal flaws with the grades of the value streams, it can be assumed that all products are potentially marketable. Thus compliance with JORC requirements rests on the economics, or recovery, of the each of the valuable minerals, especially the fluorspar. From fluorspar industry experience, overall commercial plant recovery generally lies in the range 10% to 15% lower than the mineralogically-limiting recovery depending on the complexity of the plant and whether pre-concentration is applicable. Since processing of Moina ore will require unusually fine

regrinding, selectivity of fluorspar in the flotation cleaner circuit will be lower than industry norms. Furthermore, due to the recovery of the other value streams the process flowsheet will be fairly complex, therefore the predicted overall plant recovery will likely be at the low end of the recovery range. Thus a rough estimate of the likely CaF_2 recovery after fine regrinding to $-20\ \mu\text{m}$ would be $\sim 68\%$ which would equate to 62% recovery of total F. This estimate compares well with the empirical result achieved during the 1970s flotation testing.

5.0 Concluding Judgement and Forecast

The economic break-even for most fluorspar operations around the world varies within the range 60% to 75% overall plant CaF_2 recovery depending on whether the mine has underground or open pit operations, low or high grade mineralisation, coarse or fine grinding, developed or developing world labour costs, short- or long-haul inland freight costs, tailings dam or tailings re-treatment costs, etc, etc. The low head grade of Moina ore will raise the break-even recovery within the range but this will be off-set by the additional revenue streams from tungsten, tin, and bismuth, albeit at lower than industry norm recovery, as outlined in the above mineralogically-limiting simulation data.

In compensation for the above-average processing costs, the Moina project would benefit significantly from lower than industry norm costs in most other areas of operation. The large scale shallow horizons outcropping at surface and extending to $150\ \text{m}$ depth lend themselves to low-cost open pit mining. Abundant supplies of good quality soft water needed for fluorspar flotation are available on site, and low-cost hydro-electricity is generated near-by and the transmission lines pass adjacent to the site. Essential infrastructure already exists and inland transport to the deep water bulk port of Burnie is just $84\ \text{km}$ on good quality roads. Moina lies close to an environmentally sensitive area in Tasmania however the Moina site includes an area suited to tailings dam construction which could be readily environmentally restored at the time of mine closure. The sparse population in the area creates no re-settlement or displacement difficulties and the well-established mining history in Tasmania supports a pre-existing trained workforce together with specialised mining and maintenance services.

Consequently after taking into account the combination of the currently available data, worldwide fluorspar industry practice, and the locally available infrastructure and services, the processing component of this project will be technically and economically challenging but there is no currently available evidence to suggest that these technical and economic factors should prevent JORC compliance.

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15 May 2012

Appendix B - Plant Layout and Site Images

APPENDIX B

PLANT LAYOUTS AND SITE IMAGES

B1. Plant Site Plan

B2. Aerial Image of Moina Site

B3. Burnie Harbour

B4. Mine Site and surrounds

B5. Mine Access Roads and Powerlines

B6. General Moina Area Countryside

B7. Core Sampler and Sample extractions

B2. AERIAL IMAGE OF MOINA SITE



B3. BURNIE HARBOUR



Burnie Harbour. Large vessels can use the harbour to load/drop containers.



Woodchip Wharf (right of photo) and container storage area (left of photo).

B4. MINE SITE AND SURROUNDS



Back of the dam and the proposed mine site behind the ridge (to the right).



Mine site ridge.



Mine Site with Original Processing Plant Foundations.



Mine Site Regrowth.



Mine Site with Original Process Plant.



Original Mine Tunnel Access Sign.



Original Mine Tunnel.



Original Mine Site Trenching (to left of photo).

B5. MINE ACCESS ROADS AND POWERLINES



Mine Access Road (mine site to left of photo.)



Road to Burnie and site powerlines.



Road to Burnie and site powerlines #2.

B6. GENERAL MOINA AREA COUNTRYSIDE



General Landscape of Moina Area.

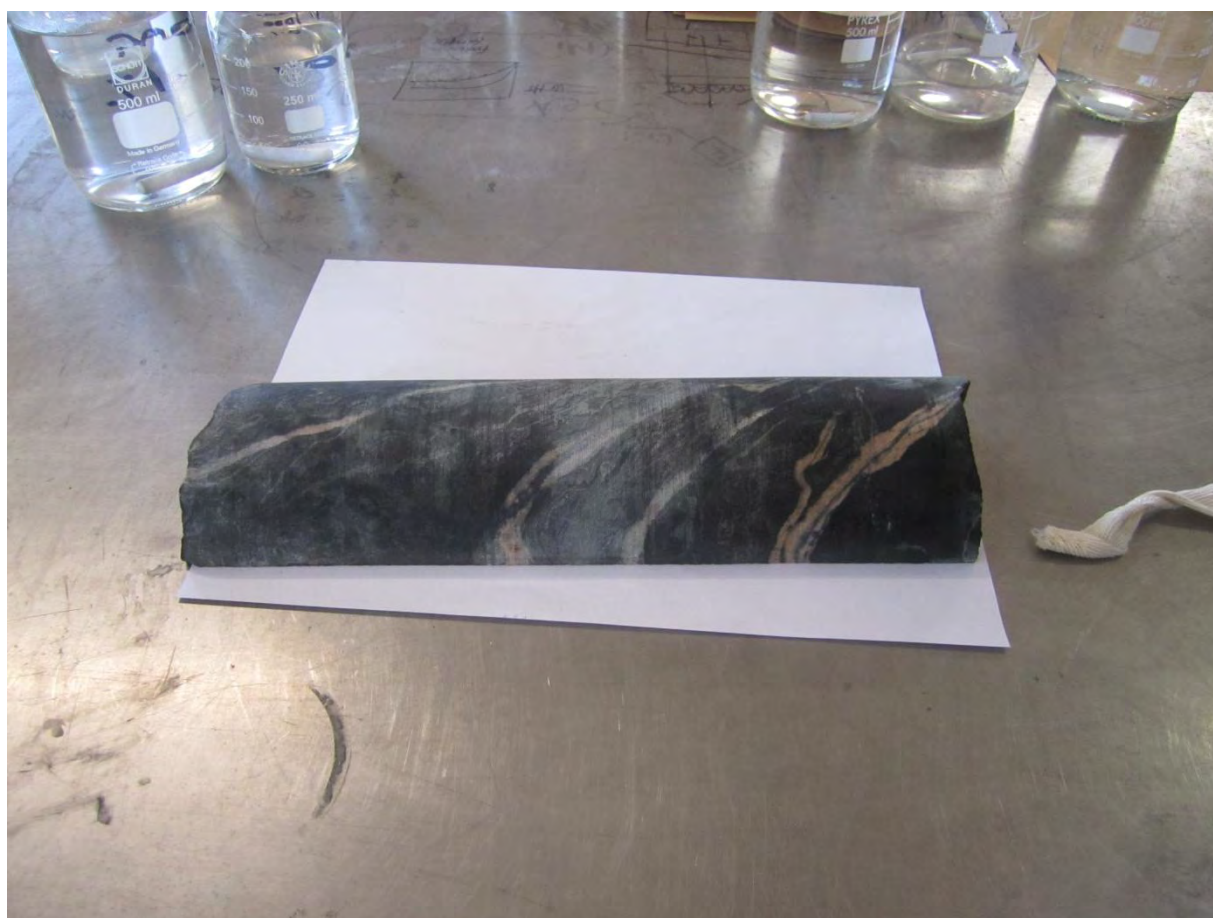
B7. CORE SAMPLER AND SAMPLES



Core Sampler.



Sample Extracted by Core Sampler.



Sample extracted by core sampler.



DETAIL B

JACOBS	CLIENT No.	REV.
	REQ. No.	REV.
	114-90-0000-A0001	P1

Appendix C - Electrical Supply and Costs

APPENDIX C

ELECTRICAL SUPPLY, LOGISTICS AND TRANSPORT COSTS

C1. De Bruyn's transport quote

C2. Aurora Tariffs

C3. Road Transport Conditions and Logistics

C4. Shipping cost quote – Toll ANL

C5. Shipping fee change by Toll

C6. Toll port costs

C7. Toll Shipping advice

Olsen, Kobie

From: John GM De Bruyn <jgmdebruyn@dbtrans.com.au>
Sent: Thursday, 14 June 2012 10:20 PM
To: Meija, George
Subject: RE: Information on Moina Mine

Hi George

Following on from our conversation earlier this week.

We have struggled with this and see the 20 micron as a issue, as you know the moisture content for shipping needs to be around 6%. We have some concerns as to being able to bulk ship this material and also when loaded into containers it will be extremely difficult if not impossible to come out of the container. We handle about 200,000 ton of magnetite and also load some into bulk bags. This material is around 40 microns and we have issues at times with moisture content. When loading into bulk bags they material is like big lumps and is extremely difficult to handle blocking the loading equipment etc. So it will be critical that the moisture content is around 6%. The magnetite we currently handle we load into bulk bags 1.2 ton per bag and 20 bags go into the container for a total of 24 ton. We stack the bags 2 high on pallets 10 pallets per container.

Budget prices are requested

Transport from the mine to Burnie \$15.00 per ton plus Fuel surcharge (currently 14.3%) and GST

Loading in to bulk bags \$500.00 per 20 bags at 1.2 ton each

Supply of bags \$20.00 each 20 per container

Supply of hardwood pallets \$25.00 each 10 per container

Delivery of container to wharf \$100.00 per container

I understand the above works out very expensive so there may be an option of loading the containers with bulk product in standard GP containers using a special plastic liner with ply board along the sides of the container to insure it does not luck into the grooves. This may require some experimenting but would certainly save some money. How they would tip on the receivers end and what happens to the liner would also need so experimenting.

Loading at the mine with mine's own loader utilising conveyor system provided by de Bruyn's Transport inclusive of liner and ply board and delivered direct to Burnie Wharf. \$1,000.00 per container plus Fuel surcharge (currently 14.3%) and GST.

Sorry about the delay hope this helps with your study.

Feel free to call me if your require anything further or if you wish the discuss the content of this email

Regards

John de Bruyn
General Manager



☎ 03 6430 2308 📠 03 6431 8184 📞 0419 371 210

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From: Meija, George [<mailto:George.Meija@jacobs.com>]
Sent: Thursday, 24 May 2012 10:41 AM
To: John GM De Bruyn
Subject: Information on Moina Mine

Hello John,

These are the quantities:

Tungsten:
Wet WO₃ Concentrate: 773 tpa

Fluorspar:
Wet CaF₂ Concentrate: 98,035 tpa

Magnetite:
Wet magnetite concentrate: 113,960 tpa

I need a cost per tonne transported from the mine to Burnie harbour. It will need to be in containers at harbour side. It will need to be contained for the journey as – 20 microns.

Regards

George

George Meija

Manager Studies
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Aurora Energy – Network Tariffs

Network Tariff Prices for Period 5 1 July 2011 – 30 June 2012

As Approved by the Regulator

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CONTACT

This document is the responsibility of the Commercial Management Group within the Network Division of Aurora Energy Pty Ltd (ABN 85 082 464 622). Please contact the indicated owner of the document with any queries or suggestions.

REVIEW DATE

This document is due for review not later than 31 May 2012

RESPONSIBILITIES

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IMPLEMENTATION

Commercial Management – Commercial Team

AUDIT

Periodic audits to establish conformance with this document will be conducted by Aurora's Commercial Management Group.

COMPLIANCE

All Group Managers

DOCUMENT MANAGEMENT

Commercial Management – Commercial Team

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AURORA ENERGY PTY LTD
Network Tariff Application Guide

Version History

Version No	Date of Revision	Authorised By	Details of Amendment
1.0	17 June 2011	Revenue Assurance Manager	Initial Version
2.0	17 June 2011	Regulatory Manager	Final Version
2.0	17 June 2011	Commercial Manager	Approved for publication

Amendments to each version of this document will be tracked through Aurora's document management system.

GST

All rates and prices in this document do not include GST.

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1. Distribution Tariffs

1.1. General Network – Residential

Aurora Code - N01 MSATS Code - AURESGEN	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
First 500kWh per Quarter	8.983 c/kWh
Next 1000kWh per Quarter	8.758 c/kWh
Remaining Consumption	8.758 c/kWh
TUoS Charge	
First 500kWh per Quarter	3.914 c/kWh
Next 1000kWh per Quarter	3.816 c/kWh
Remaining Consumption	3.816 c/kWh
Meter Charge	
Daily Charge	8.347 c/day
Total Charge (NUoS)	
Daily Charge	45.885 c/day
First 500kWh per Quarter	12.897 c/kWh
Next 1000kWh per Quarter	12.574 c/kWh
Remaining Consumption	12.574 c/kWh

1.2. General Network – Business

Aurora Code - NO2 MSATS Code - AUBLVGEN	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
First 500kWh per Quarter	8.983 c/kWh
Remaining Consumption	8.758 c/kWh
TUoS Charge	
First 500kWh per Quarter	3.914 c/kWh
Remaining Consumption	3.816 c/kWh
Meter Charge	
Daily Charge	12.042 c/day
Total Charge (NUoS)	
Daily Charge	49.580 c/day
First 500kWh per Quarter	12.897 c/kWh
Remaining Consumption	12.574 c/kWh

1.3. General Network – Business, Nursing Homes (Obsolete)

Aurora Code - NO2a MSATS Code - AUBLVNURSE	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
First 500kWh per Quarter	8.983 c/kWh
Remaining Consumption	4.052 c/kWh
TUoS Charge	
First 500kWh per Quarter	3.246 c/kWh
Remaining Consumption	2.410 c/kWh
Meter Charge	
Daily Charge	12.042 c/day
Total Charge (NUoS)	
Daily Charge	49.580 c/day
First 500kWh per Quarter	12.229 c/kWh
Remaining Consumption	6.462 c/kWh

1.4. General Network – Business, Curtilage (Obsolete)

Aurora Code - NO2b MSATS Code - AUBLVCURT	2011/12 Tariff
DUoS Charge	
Daily Charge	11.261 c/day
First 500kWh per Quarter	8.983 c/kWh
Remaining Consumption	8.758 c/kWh
TUoS Charge	
First 500kWh per Quarter	3.914 c/kWh
Remaining Consumption	3.816 c/kWh
Meter Charge	
Daily Charge	12.042 c/day
Total Charge (NUoS)	
Daily Charge	23.303 c/day
First 500kWh per Quarter	12.897 c/kWh
Remaining Consumption	12.574 c/kWh

1.5. Uncontrolled Energy

Aurora Code - N05 MSATS Code - AUHEATUNCO	2011/12 Tariff
DUoS Charge	
Daily Charge	3.754 c/day
All Energy	1.627 c/kWh
TUoS Charge	
All Energy	2.400 c/kWh
Meter Charge	
Daily Charge	8.285 c/day
Total Charge (NUoS)	
Daily Charge	12.039 c/day
All Energy	4.027 c/kWh

1.6. Controlled Energy

Aurora Code - N06 MSATS Code - AUHEATCONT	2011/12 Tariff
DUoS Charge	
Daily Charge	7.371 c/day
All Energy	0.428 c/kWh
TUoS Charge	
All Energy	0.701 c/kWh
Meter Charge	
Daily Charge	8.487 c/day
Total Charge (NUoS)	
Daily Charge	15.858 c/day
All Energy	1.129 c/kWh

1.7. Controlled Energy

Aurora Code - N06a MSATS Code - AUHEATCONN	2011/12 Tariff
DUoS Charge	
Daily Charge	7.371 c/day
All Energy	0.418 c/kWh
TUoS Charge	
All Energy	0.684 c/kWh
Meter Charge	
Daily Charge	8.487 c/day
Total Charge (NUoS)	
Daily Charge	15.858 c/day
All Energy	1.102 c/day

1.8. LV PAYG (Obsolete)

Aurora Code - N13 MSATS Code - AUPAYG	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
All Energy	5.034 c/kWh
TUoS Charge	
All Energy	2.438 c/kWh
Total Charge (NUoS)	
Daily Charge	37.538 c/day
All Energy	7.472 c/kWh

1.9. LV TOU – Residential

Aurora Code - N13r MSATS Code – AURESTOU	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
Peak Energy	8.720 c/kWh
Shoulder Energy	5.395 c/kWh
Off-Peak Energy	0.418 c/kWh
TUoS Charge	
Peak Energy	3.799 c/kWh
Shoulder Energy	2.541 c/kWh
Off-Peak Energy	0.684 c/kWh
Meter Charge	
Daily Charge	10.128 c/day
Total Charge (NUoS)	
Daily Charge	47.666 c/day
Peak Energy	12.519 c/kWh
Shoulder Energy	7.936 c/kWh
Off-Peak Energy	1.102 c/kWh

1.10. LV TOU – Business

Aurora Code – N13b/N13c MSATS Code – AUBUSTOU / AUBUSTOUB	2011/12 Tariff
DUoS Charge	
Daily Charge	37.538 c/day
Peak Energy	8.871 c/kWh
Shoulder Energy	5.568 c/kWh
Off-Peak Energy	0.418 c/kWh
TUoS Charge	
Peak Energy	3.865 c/kWh
Shoulder Energy	2.614 c/kWh
Off-Peak Energy	0.684 c/kWh
Meter Charge	
Daily Charge	15.261 c/day
Total Charge (NUoS)	
Daily Charge	52.799 c/day
Peak Energy	12.736 c/kWh
Shoulder Energy	8.182 c/kWh
Off-Peak Energy	1.102 c/kWh

1.11. LV kW Demand (Obsolete)

Aurora Code - N03 MSATS Code - AUBLVDMKW	2011/12 Tariff
DUoS Charge	
Daily Charge	205.175 c/day
All Energy Charge	1.982 c/kWh
First 250kW Demand	34.733 c/kW/day
Additional Demand	31.310 c/kW/day
TUoS Charge	
All Energy Charge	1.030 c/kWh
First 250kW Demand	28.183 c/kW/day
Additional Demand	25.399 c/kW/day
Meter Charge	
Daily Charge	15.261 c/day
Total Charge (NUoS)	
Daily Charge	220.436 c/day
All Energy Charge	3.012 c/kWh
First 250kW Demand	62.916 c/kW/day
Additional Demand	56.709 c/kW/day

1.12. LV kVA Demand

Aurora Code - N09 MSATS Code - AUBLVDMKVA	2011/12 Tariff
DUoS Charge	
Daily Charge	183.192 c/day
All Energy Charge	1.770 c/kWh
First 250kVA Demand	26.360 c/kVA/day
Additional Demand	23.762 c/kVA/day
TUoS Charge	
All Energy Charge	0.920 c/kWh
First 250kVA Demand	21.389 c/kVA/day
Additional Demand	19.276 c/kVA/day
Meter Charge	
Daily Charge	15.261 c/day
Total Charge (NUoS)	
Daily Charge	198.453 c/day
All Energy Charge	2.690 c/kWh
First 250kVA Demand	47.749 c/kVA/day
Additional Demand	43.038 c/kVA/day

1.13. HV kW Demand (Obsolete)

Aurora Code - N11 MSATS Code - AUBHVDMKW	2011/12 Tariff
DUoS Charge	
Daily Charge	140.548 c/day
All Energy Charge	0.123 c/kWh
Demand Charge	31.945 c/kW/day
TUoS Charge	
All Energy Charge	1.576 c/kWh
Monthly Demand Charge	2.822 c/kW/day
Meter Charge	
Daily Charge	65.865 c/day
Total Charge (NUoS)	
Daily Charge	206.413 c/day
All Energy Charge	1.699 c/kWh
Monthly Demand Charge	34.767 c/kW/day

1.14. HV kVA Demand (Obsolete)

Aurora Code - N10 MSATS Code - AUBHVDMKVA	2011/12 Tariff
DUoS Charge	
Daily Charge	125.489 c/day
All Energy Charge	0.110 c/kWh
Demand Charge	24.244 c/kVA/day
TUoS Charge	
All Energy Charge	1.407 c/kWh
Demand Charge	2.142 c/kVA/day
Meter Charge	
Daily Charge	65.865 c/day
Total Charge (NUoS)	
Daily Charge	191.354 c/day
All Energy Charge	1.517 c/kWh
Demand Charge	26.386 c/kVA/day

1.15. LV Day / Night (Irrigation) (Obsolete)

Aurora Code - N08 MSATS Code - AUIRRIG	2011/12 Tariff
DUoS Charge	
Daily Charge	180.386 c/day
Day Energy Charge	8.758 c/kWh
Night Energy Charge	0.418 c/kWh
TUoS Charge	
Day Energy Charge	4.383 c/kWh
Night Energy Charge	0.684 c/kWh
Meter Charge	
Daily Charge	14.915 c/day
Total Charge (NUoS)	
Daily Charge	195.301 c/day
Day Energy Charge	13.141 c/kWh
Night Energy Charge	1.102 c/kWh

1.16. LV Irrigation (TOU)

Aurora Code - N08a /N08b MSATS Code – AUIRRIGTOU / AUIRRITOU	2011/12 Tariff
DUoS Charge	
Daily Charge	163.046 c/day
Peak Energy	8.758 c/kWh
Shoulder Energy	5.418 c/kWh
Off-Peak Energy	0.408 c/kWh
TUoS Charge	
Peak Energy	4.383 c/kWh
Shoulder Energy	2.896 c/kWh
Off-Peak Energy	0.667 c/kWh
Meter Charge	
Daily Charge	15.261 c/day
Total Charge (NUoS)	
Daily Charge	178.307 c/day
Peak Energy	13.141 c/kWh
Shoulder Energy	8.314 c/kWh
Off-Peak Energy	1.075 c/kWh

1.17. Small LV Unmetered

Aurora Code - N07 MSATS Code - AUUMS	2011/12 Tariff
DUoS Charge	
Daily Charge	28.153 c/day
All Energy Charge	10.216 c/kWh
TUoS Charge	
All Energy Charge	4.821 c/kWh
Total Charge (NUoS)	
Daily Charge	28.153 c/day
All Energy Charge	15.037 c/kWh

1.18. Street Lighting

Aurora Code - N20 MSATS Code - AUUMSSL	2011/12 Tariff
DUoS Charge	
Demand Charge	0.087 c/lamp watt/day
TUoS Charge	
Demand Charge	0.036 c/lamp watt/day
Total Charge (NUoS)	
Demand Charge	0.123 c/lamp watt/day

Note: Does not include charge for light fitting.

1.19. Import Energy

Aurora Code - N21 MSATS Code - AU_IMPORT	2011/12 Tariff
DUoS Charge	
Energy charge	0.000 c/kWh
TUoS Charge	
Energy charge	0.000 c/kWh
Meter Charge (if applicable)	
Daily Charge	8.285 c/day
Total Charge	
Total Charge	8.285 c/day

2. Specified Demand Tariffs

2.1. HV kVA Specified Demand

Aurora Code - N10s MSATS Code - AUHVSPECDM	2011/12 Tariff
DUoS Charge	
Daily Charge	125.489 c/day
Peak Energy	0.099 c/kWh
Shoulder Energy	0.079 c/kWh
Off-Peak Energy	0.049 c/kWh
Specified Daily Demand Charge	18.183 c/kVA/day
Excess Demand Charge	181.830 c/kVA/day
TUoS Charge	
Peak Energy	1.267 c/kWh
Shoulder Energy	1.011 c/kWh
Off-Peak Energy	0.626 c/kWh
Specified Daily Demand Charge	1.607 c/kVA/day
Excess Demand Charge	16.070 c/kVA/day
Meter Charge	
Daily Charge	0.000 c/day
Total Charge (NUoS)	
Daily Charge	125.489 c/day
Peak Energy	1.366 c/kWh
Shoulder Energy	1.090 c/kWh
Off-Peak Energy	0.675 c/kWh
Specified Daily Demand Charge	19.790 c/kVA/day
Excess Demand Charge	197.900 c/kVA/day

2.2. HV kVA Specified Demand (>2.0MVA)

Aurora Code - N15	2011/12 Tariff
MSATS Code - AUCHVDM2	
DUoS	
Daily Charge	16.989 \$/day
Specified Demand Charge	9.074 c/kVA/day
Excess Demand Charge	45.370 c/kVA/day
Peak Energy Charge	1.428 c/kWh
Shoulder Energy	0.387 c/kWh
Off-Peak Energy	0.049 c/kWh
Connection	
Specified Demand Charge	0.329 c/kVA/day
Excess Demand Charge	1.645 c/kVA/day
Meter Charge (if applicable)	
Daily Charge	0.000 c/day
Transmission Charge	
Specified Demand Charge	As per nodal charge in table 25
Excess Demand Charge	5 x nodal charge

3. Standard Transmission Charges

Average nodal connection charges have been based on demand. Aurora's schedule of charges for 2011-12 is reproduced in the following table.

Hobart (Virtual) is based on the Chapel Street, Creek Road, Lindisfarne, North Hobart, Risdon and Rokeby connection sites.

Tamar (Virtual) is based on the Norwood, Trevallyn, Mowbray, George Town and Hadspen connection sites.

Transmission Charges 2011/12

Connection Site	TNI	Daily Charge (c/kVA) 11/12
Arthurs Lake	TAL2	33.790
Avoca	TAV2	31.802
Bridgewater	TBW2	23.731
Burnie	TBU3	21.584
Derby	TDE2	68.025
Derwent Bridge	TDB2	192.740
Devonport	TDP2	25.010
Electrona	TEL2	31.766
Emu Bay	TEB2	35.773
Kermandie	TKE2	49.093
Kingston	TKI2	21.475
Knights Rd	TKR2	29.554
Meadowbank	TMB2	25.510
New Norfolk	TNN2	27.048
Newton	TNT2	47.384
Palmerston	TPM3	28.579
Port Latta	TPL2	32.336
Queenstown	TQT2	39.191
Railton	TRA2	24.106
Rosebery	TRB2	25.405
Savage River	TSR2	27.670
Scottsdale	TSD2	58.588
Smithton	TST2	32.898
Sorell	TSO2	28.724
St Marys	TSM2	38.319
Triabunna	TTB2	53.326
Tungatinah	TTU2	102.563
Ulverstone	TUL2	23.941
Waddamana	TWA2	39.593
Wesley Vale	TWV2	18.338
Hobart Virtual	TVN1	23.061
Tamar Virtual	TVN2	20.470

Transport and Logistics

Road transport.

The condition of the road from Moina to Burnie, the port of destination for shipping, is good in that it is fully recently sealed, marked and well detailed. Therefore there should not be any difficulty to carry containers by trucks.

The full distance travelling by the Cradle Mountain Development Road (C132) and then by the Murchison Highway (A10), from Moina to Burnie is approximately 130 km.

Rate Submission



QUOTE NUMBER:RF0672

Company Name:Jacobs

Contact: George Meija

Address:

Telephone:07 30187261

FAX:

MOBILE:0407193503

Email Address:George.Meija@jacobs.com

Payment Terms:

Date Quoted: 28/5/12

COMMENTS:

DESCRIPTION	DIMENSIONS	FROM / TO	PRICE PER UNIT	TOTAL PRICE	Fuel	GST	TOTAL
20' GP	Mining Products	Burnie Melbourne	\$850.00	\$850.00	\$106.25	\$95.63	\$1,051.88
TOTAL:				\$850.00	\$106.25	\$95.63	\$1,051.88

Fuel Surcharge %: 12.50 JUN

This Quotation is subject to the following conditions:

1. The quotation number (above) being clearly shown on the Consignment Note
2. The item(s) must be Pre-Booked at the respective wharf. Melbourne (03) 9299 8445 or Burnie (03) 6441 0013
3. Where applicable - cargo must have appropriate lashing points (e.g. trailers)
4. The consignment being shipped within 60 days of the above date
5. The above described item(s) only and is not applicable to future consignments.
6. The above measurements being actually confirmed upon receipt of consignment by Toll ANL Bass Strait Shipping
7. Subject to our terms and conditions please refer to following link
<http://www.tollanl.com/downloads/TermsConditions.pdf>
8. Credit Card payments will have an additional fee of 3.2% for Amex/Diners & 1.4% for Visa/Mastercard
9. Customers with current trading accounts must pay within 30 days
10. All Cargo must be free of all dirt and plant material as per AQIS regulations. Cargo that does not meet these regulations will be impounded for Quarantine clean.
11. Fuel Surcharge is variable and adjusted in accordance with the monthly Toll ANL Shipping fuel levy advice
12. The Total Price is inclusive of GST (10%) and applicable monthly fuel levy surcharge
13. Cabotage rates supplied are subject to space availability on Cabotage service
14. Cabotage rates supplied are based on equipment availability, additional charges will apply if repositioning is required

For and on behalf of

TOLL ANL Bass Strait Shipping Pty Ltd

Name: ROD FRY
 Telephone: 03 92998417
 Fax: -
 Email: rod.fry@tollgroup.com

Signature.....Date.....

Customer acceptance of rates and conditions

Customer Name

Name:

Telephone:

Fax:

Email:

Signature.....Date.....



Toll ANL Bass Strait Shipping
120-178 Williamstown Road
Port Melbourne VIC 3207 Australia
PO Box 420
Port Melbourne VIC 3207 Australia

T +61 3 9299 8400
F +61 3 9299 8430
www.tollanl.com

Toll ANL Bass Strait Shipping Pty Ltd
ABN 74 138 616 645

Friday 11TH May 2012

Dear Customer,

RE: CHANGE IN FEES FOR TOLL ANL BASS STRAIT SHIPPING SERVICES

From 1 July 2012, Toll ANL Bass Strait Shipping will be impacted by a number of governmental decisions that will affect the net cost of providing logistics services to its customers.

Unfortunately, Toll ANL Bass Strait Shipping is unable to absorb these costs and will need to recover the following costs from its customers:

- Costs passed through to Toll ANL Bass Strait Shipping by the Port of Melbourne Corporation (**PoMC**) following the imposition of an annual Port License fee on PoMC by the Victorian Government from 1 July 2012. Final confirmation of port charges to be applied to Tasmanian freight is yet to be confirmed; and
- The net cost increase to Toll ANL Bass Strait Shipping of providing logistics services due to the Federal Government's Clean Energy Legislation from 1 July 2012.

At this stage, we are unable to confirm the full impact of any price increase due to the PoMC Port License fee but will keep you advised of developments as we further clarify our position.

We expect that the majority of increased carbon costs to our customers will be reflected in Toll's bunker fuel charge, due to the net benefit of fuel tax credits reducing by 6.2 cent per litre from 1 July 2012. The increase in the net cost of fuel to Toll ANL Bass Strait Shipping will be recovered on a monthly basis under the existing fuel surcharge mechanism.

We apologise for the impact that these Government imposed changes will have on your business but trust you appreciate the changes are outside of our control. We continue to implement changes to our business in an effort to reduce our carbon footprint.

Please contact your Account Manager if you would like to discuss any questions or concerns you may have.

We look forward to continuing our relationship with you and wish to reiterate our ongoing commitment to meeting your logistics needs.

Yours faithfully

Paul Giotas
National Sales & Marketing Manager

Toll ANL Bass Strait Shipping
120-178 Williamstown Road
Port Melbourne VIC 3207 AUSTRALIA

Rate Submission



QUOTE NUMBER:RF0672

Company Name:Jacobs

Contact: George Meija

Address:

Telephone:07 30187261

FAX:

MOBILE:0407193503

Email Address:George.Meija@jacobs.com

Payment Terms:

Date Quoted: 28/5/12

COMMENTS:

DESCRIPTION	DIMENSIONS	FROM / TO	PRICE PER UNIT	TOTAL PRICE	Fuel	GST	TOTAL
20' GP	Mining Products	Burnie Melbourne	\$850.00	\$850.00	\$106.25	\$95.63	\$1,051.88
TOTAL:				\$850.00	\$106.25	\$95.63	\$1,051.88

Fuel Surcharge %: 12.50 JUN

This Quotation is subject to the following conditions:

1. The quotation number (above) being clearly shown on the Consignment Note
2. The item(s) must be Pre-Booked at the respective wharf. Melbourne (03) 9299 8445 or Burnie (03) 6441 0013
3. Where applicable - cargo must have appropriate lashing points (e.g. trailers)
4. The consignment being shipped within 60 days of the above date
5. The above described item(s) only and is not applicable to future consignments.
6. The above measurements being actually confirmed upon receipt of consignment by Toll ANL Bass Strait Shipping
7. Subject to our terms and conditions please refer to following link
<http://www.tollanl.com/downloads/TermsConditions.pdf>
8. Credit Card payments will have an additional fee of 3.2% for Amex/Diners & 1.4% for Visa/Mastercard
9. Customers with current trading accounts must pay within 30 days
10. All Cargo must be free of all dirt and plant material as per AQIS regulations. Cargo that does not meet these regulations will be impounded for Quarantine clean.
11. Fuel Surcharge is variable and adjusted in accordance with the monthly Toll ANL Shipping fuel levy advice
12. The Total Price is inclusive of GST (10%) and applicable monthly fuel levy surcharge
13. Cabotage rates supplied are subject to space availability on Cabotage service
14. Cabotage rates supplied are based on equipment availability, additional charges will apply if repositioning is required

For and on behalf of

TOLL ANL Bass Strait Shipping Pty Ltd

Name: ROD FRY
 Telephone: 03 92998417
 Fax: -
 Email: rod.fry@tollgroup.com

Signature.....Date.....

Customer acceptance of rates and conditions

Customer Name

Name:

Telephone:

Fax:

Email:

Signature.....Date.....



Shipping Advice

Thankyou for choosing to ship with Toll ANL Bass Strait Shipping. Below listed are points that need to be adhered to when shipping with us:

- Lashing points must be fitted (Ship's Master will reject cargo that cannot be lashed to the deck securely), please refer to the lashing point information document
- All containers presented at the wharf must have a current compliance plate
- Any hazardous cargo must be declared (hazardous goods for must be faxed through to our office 24hrs prior to shipping – Melbourne 9299 8444, Burnie 6431 3420. Failure to comply may result in cancellation of booking)
- Cargo will only be accepted into the wharf on the day of sailing
- Cargo must be collected on the day of arrival
- Payment is required prior to shipping otherwise freight will be rejected
- Cut off times will be organised before the date of shipping and must be adhered to otherwise shipping cannot be guaranteed
- All cargo must be free of dirt and plant matter as per Tasmanian Quarantine Regulations
- Stowage position will be determined by Toll Shipping
- Failure of drivers/passengers to show when booked without cancellation will be charged full fee
- There may be a requirement to split B Double trailers for stowage
- All mobile freight presented at the wharf, must be in a condition by which it is safe to operate and must be able to get on and off the ship under its own power ie. Not requiring towing unless organised prior to the shipping date.

Please note; although every endeavour will be made to ensure that your freight will sail on the day it is booked, please understand that occasionally freight conditions and volumes can change drastically and may result in your freight being rolled to the next day's sailing.

Please refer to the Terms and Conditions for comprehensive detail.