

Lynch Mining Pty Ltd

EL7/2005 “Luina”

Annual Report for the period 30 June 2010 to 30
June 2011.

By: Bright Phase Pty Ltd

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Attachments:

- Bright Phase Pty Ltd Scoping Study + Appendix
- Draft Metallurgical testwork report
- Memo from Andrew Glastonbury to Bright Phase Pty Ltd
- Draft mining report from CSA Global

1.0 Introduction

This report details all development work undertaken on Exploration License 7/2005, "Luina" during the Annual Period 30th June 2010 to 30th June 2011.

Exploration License 7/2005, "Luina" covering an area of 18km² lies in Western Tasmania and is situated to the west of Waratah. Access is via the sealed Waratah-Savage River road and then via a series of old sealed and unsealed mine access and forestry roads/tracks.

The terrain is dominated by steep shrub and tree covered slopes, with a significant area of rehabilitated mine workings and former town site. Areas of former forestry clearing are also present. Both the Whyte River and Washington Creek transect the tenement areas.

Development work during the reporting period included:

- Consolidation and review of all available data on the historic Cleveland Mine Operations located at Burnie Research Labs, Burnie, Tasmania
- Beginning metallurgical testwork at Burnie Research Laboratory, using a composite sample that was taken from the 2008 testwork previously completed by Lynch Mining
- Completion of a desktop review of the potential to re-open the mine
- Completion of a preliminary review of mining operations and methods.
- Commencement of site-based water sampling and environmental monitoring
- Engagement of environmental consultants to begin the NOI, DPEMP and ML processes
- Continuing development work by Bright Phase Resources on drafting developmental programmes for geological reviews, development of 3-D geological models and financial modelling of potential mining operations at the site.
- A contract was negotiated for a 3rd party (Bright Phase Resources Ltd) to purchase this lease, subject to a number of conditions, from Lynch Mining. This contract is attached.

2.0 Tenure

Exploration license 7/2005 (Luina) covering an area of 18km² was granted to Lynch Mining Pty Ltd on 30th June 2005 for a period of 5 years. The lease is now granted on a year-by-year basis. Lynch Mining has negotiated a contract with Bright Phase Resources Ltd to sell ownership of this lease subject to a number of conditions being met.

3.0 Previous Exploration

The Cleveland deposit was discovered in 1898. Initial production of the surficial ore commenced in 1908, and ceased in 1914 after production of 295.5 tonnes of cassiterite. Tributing continued until 1917, during which time a further 48 tonnes were produced. Aberfoyle commenced mining in 1968, and mining was ceased in early 1986, primarily due to the collapse of the tin price.

At the cessation of the more recent mining activity, approximately 7 million tonnes of ore assaying 0.82% Sn and 0.35% Cu had been processed.

Various reports show that a significant resource remains in the deposit, with Measured and Indicated resources of 5.2 million tonnes @0.70Sn and 0.31%Cu and Inferred resources of 1.3 million tonnes @0.72% and 0.22% Cu at a 0.35% Sn cut-off. In addition, a large resource known as the Foley Zone located at the lower levels of the mine had been drilled and is reported to contain 3.8 million tonnes @0.28% WO₃ at a 0.2% cut-off. Data for these estimates has been sourced to archived reports currently stored at Burnie Labs and available online on the MRT website. The most important document for confirmation of this data is Aberfoyle's "close-out report".

4.0 Regional and Local Geology

The Cleveland ore bodies occur in a steeply dipping northeast trending succession of arenaceous, argillaceous, and chemical sediments and mafic volcanic rocks and ultramafic/mafic complexes. The sequence is unfossiliferous but has been correlated by rock type with the Cambrian Crimson Creek Formation of the Zeehan-Rosebery area. The Meredith Granite, a high level, late tectonic Late Devonian to Early Carboniferous granitic pluton believed to be genetically associated with the mineralisation, intrudes the sequence east and south of the mine. The Cleveland ore bodies are located in a dominantly fine grained sedimentary sequence that is thought to lie in an embayment on the margin of a basaltic eruptive centre.

The tin-copper ore bodies occur as a series of sub parallel, near vertical sulphide lenses within the Halls Formation. The mineralisation is composed largely of fine to medium grained quartz, tourmaline, fluorite, chlorite, and pyrrhotite plus chalcopyrite, cassiterite and stannites. Eleven lenses are recognised and are divided into two groups; the footwall and the hanging wall lodes separate by a micaceous greywacke unit and an overlying basic volcanic unit. The footwall lodes are composed of three relatively thick lenses (Henry, Lucks and Khaki) which have a strike length of less than 200 metres. The hanging-wall lodes (or Halls lenses) comprise 5 lenses and have a greater strike length and stratigraphic continuity than the footwall lodes, attaining a maximum thickness of about 30 metres and a maximum strike length of about 600 metres. The sulphide lenses are offset by a series of sub parallel, southeast dipping reverse faults.

5.0 Developmental Activities

Developmental activities during the period have principally involved development of a scoping study for undertaking tailings reprocessing (by Bright Phase Pty Ltd); beginning metallurgical testwork to match the testwork completed in 2008; completion of a desk-top study on re-opening the underground mine.

5.1 Scoping Study.

A scoping study that reviewed re-processing the tailings material was completed by Bright Phase Pty Ltd, a Brisbane based company that has signed a contract that will see them participate heavily in re-developing this project.

The essential summary points of the scoping study are:

- With reasonable recoveries, an economic operation can be established and successfully run
- "Mining" the tailings is a simple process

- There is a significant environmental problem at the site that will need to be addressed through tailings mining
- Plant used for the project need only be skid-mounted temporary plant.

A copy of the scoping study is attached to this report.

5.2 Metallurgical testwork

Metallurgical testwork has begun that is a repeat of the 2008 testwork completed at Burnie Research labs.

A draft testwork report is attached.

5.3 Desktop review of mine re-opening.

Mr. Andrew Glastonbury was tasked with completing a desktop review of re-opening the mine. Andrew completed this review and his memo is attached.

CSA Global were also commissioned to complete a desktop study of the re-opening of the mine, with a view on the mining operations and operating costs. This was completed, but no formal report was issued, just the draft with mark-ups as attached.

5.4 Environmental monitoring work begun

Environmental monitoring, in particular water monitoring in the Whyte River, has begun at the EL. In addition, Pitt and Sherry Consultants from Hobart have begun working on the NOI to be issued prior to the DPEMP and ML processes proceeding.

6.0 Expenditure

Expenditure over the license area totalled \$129,981 during the reporting period to 30th June 2011, and is broken down by expense in the table below:

Bright Phase Consultants	89,425.51
CSA Global	6,572.50
Glastonbury Mining	13,493.28
Environmental Consultants	8,232.40
Water monitoring onsite	1,735.00
Geological review	5000
Metallurgy	980
Tenement Fees	1189.02
Management and General	20.42
Burnie Laboratories	3,333.00
Total	129,981.13

7.0 Conclusions and Recommendations

EL 7/2005 covers the old Cleveland Sn/Cu mine area in western Tasmania, which has been subject to a long period of exploration and mining.

Recommendations for future activities:

- Re-drill the tailings and complete more detailed metallurgical testwork after handling the material properly prior to it reaching the laboratory
- Begin detailed feasibility study on tailings reprocessing
- Begin detailed scoping study on re-opening the mine
- Progress JORC compliance on the tailings and hard rock resources

Appendix 1 – Bright Phase Pty Ltd Scoping Study + Appendix



BRIGHT PHASE

Report No. B107/03

PRE-FEASIBILITY STUDY

**Cleveland Tin-Copper-Silver
Tailings Retreatment**

5th September 2010

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Executive Summary

Bright Phase Pty Ltd (BPPL) have signed a Letter of Intent with Lynch Mining (Lynch) on 18th August 2010 to enter into an Unincorporated Joint Venture and Option to Purchase (the Agreement) over their Cleveland property whereby BPPL agreed to undertake a feasibility study on the re-treatment of the Cleveland tin/copper tailings. The Cleveland tin-copper tailings deposit is located at Luina, close to the Tasmanian town of Waratah and some 50 km southwest of the coastal city of Burnie. The original Cleveland tin mine opened near Waratah in 1908 and operated until 1917. Aberfoyle later reopened Cleveland in 1968, continuing to work the deposit until 1986 and processed about 7-million tonnes of ore assaying 0.82% Sn and 0.35% Cu. The mill tin recovery was only 59% for the life of the mine and copper recovery was about 60%. The mine sold some 35,500 t of tin metal and 15,000 t of copper in concentrates assaying 55-64% Sn and 20-24% Cu with 3-6% Sn. Hard rock mining operations ceased in 1986 due to the collapse of tin prices and the depletion of economical mine reserves. There has been no hard rock mining activity since that time.

The current resource¹ comprises about 4.4-million tonnes of mill tailings grading ~0.32% Sn and 0.15% Cu, stored in two tailings dams. Some 3.2-4.0 million tonnes of Heavy Media Float (grading 0.25% Sn and 0.08% Cu) were produced at Cleveland, but only an estimated 750,000 t remain, as most were sold off for use as road base. In late 1986 Aberfoyle Ltd. the mine owner used the Cleveland mill for testing a new processing route for its Hellyer base-metal ore and processed some 300,000 t, of which about 80,000 t were placed on top of Tailings Dam 2 and 220,000 t were discharged into the Cleveland mine through a rise drilled from near the Cleveland mill. These grades are comparable with grades of existing plants mining primary underground tin ores, however, tailings deposits such as at Cleveland do not require the expense of exploration, crushing and primary milling, presenting a clear upside.

Cleveland tailings, like many base-metal sulphide abandoned mine sites, contain large deposits of reactive waste rocks and tailings that have high concentrations of trace metals. For Cleveland tailings impact on the Whyte River these values were measured² in 2000 as sulfate 680 mg/L, metals 17 mg/L and pH 5.3, significantly exceeding the ANZECC (2000)³ recommended trigger values for several metal pollutants in aquatic ecosystems. Downstream impact from mass loading of these reactive waste rocks and tailings during major flood events is a serious threat to the receiving environment for decades². A Cleveland tailings retreatment project will potentially have a major positive impact on the long-term environmental pollution aspects from these historic mine workings by removing acidic sulphide and metal contaminants at source and this is a key incentive for the proposed project to proceed.

A comprehensive review¹ was carried out in 2007 to investigate the feasibility of retreating Cleveland tailings at current tin prices. The study defined and examined the process flowsheet based on testwork data and estimated the capital and operating cost of retreating the tailings as well as cashflow and revenue scenarios. This report showed that the Cleveland tin and copper tailings can be economically recovered into a high-grade concentrate and copper matte by using a combination of dry mining, gravity separation, tin flotation and fuming. In this way about 60-65% of the tin can be recovered into a concentrate that will assay about 70% Sn and contain less than 2% Fe. At base case 2007 metal prices



(USD\$11,400/t Sn; \$4,000/t Cu; \$0.31/g Ag) all capitals would be paid back in the first year of operation. Current prices are now almost double these values and the outlook for tin and copper prices remains buoyant. Cashflow analysis showed that this project is more sensitive to tin price and head grade rather than recovery.

Testwork has been continuing since 2008 under the management of respected independent tin metallurgist, Nick Moony of ESKER Milling and Processing Pty Ltd, confirming various aspects of the review and enabling flowsheet improvements to be recommended⁴. Additionally, BPPL has recently carried out a QEMSCAN quantitative mineralogy study on a representative composite sample of the Cleveland tailings⁵. The key outcome of this program was to define the deportment and mineral associations of tin and copper, along with the root cause for poor recovery by conventional means. It was concluded that economic recovery of value metals should be achievable from Cleveland tailings if additional liberation can be achieved and solutions for fine particle flotation or gravity separation can be implemented.

Significant advances in minerals processing and beneficiation technologies and equipment has occurred in the twenty-five years since closure of the Aberfoyle mill and forty years since it began mining. Separation and recovery of fine mineral particle by magnetic, flotation and gravity separation techniques has improved dramatically – including the high-intensity magnetic separator (WHIMS) (Downer EDI), high-intensity magnetic separator (HIMS) (Eriez) and SLon high-intensity magnetic separation (Outotec). The advent and commercial implementation of Imhoflot G-Cell pneumatic flotation (Maelgwyn Mineral Services) has proven success in flotation of fines, and the Jameson cell (Xstrata Technology) and the Microcell column have had some success in this area. Advances in reagents such as selective tin collectors and depressants, along with upfront removal of iron-bearing minerals and the use of fines flotation devices will be brought to bear on metals recovery from Cleveland tailings.

While FM1 spirals, Holman shaking tables are well proven gravity concentrators in tin processing, the J1300 Kelsey Jig (Downer EDI Mineral Technologies) has since 1990 become industry standard, and UF Falcon Centrifugal Concentrators and C2000 Falcon Continuous Concentrator (Sepro Systems) are good for fine particle recovery. The Ausmelt top submerged lance (TSL) smelting technology (Outotec) is also applicable to tin fuming in a single-stage operation, unlike conventional two-stage fuming operations.

The Cleveland tailings re-treatment project is potentially a technically feasible project using modern physical separation techniques and able to generate significant positive cash flow. Chemical leaching will not be used and the retreatment process will potentially have a major positive impact on the long-term environmental pollution aspects from these historic mine workings. A definitive engineering and feasibility study along with operation of a simplified demonstration pilot plant will refine and optimise the process and costs and establish the parameters for successful tin and copper concentrates production. The project has a high IRR (~100%) and will generate sales of around \$57-million per year with an operating cost estimated at \$17-million per year. The project therefore has an attractive return. The net present value (NPV) of the base case project is calculated as A\$74million yielding a payback period of 0.7 years. The capital costs are estimated to be a total of A\$29.1-million for the 1 M tpa processing facility, with operating costs of A\$17.35 per tonne tailings processed (A\$ 2,984/t tin concentrate produced).



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- Schedule
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1. INTRODUCTION

Bright Phase Pty Ltd (BPPL) have signed a Letter of Intent with Lynch Mining (Lynch) on 18th August 2010 to enter into an Unincorporated Joint Venture and Option to Purchase (the Agreement) over their Cleveland property whereby BPPL agreed to undertake a feasibility study on the re-treatment of the Cleveland tin/copper tailings.

Bright Phase has agreed, subject to final due diligence, to repay Lynch Mining monies spent to date of \$270,000 and to assume all Lynch Mining's obligations with respect to the tenements and to meet the cost of all ongoing test work, as well as all engineering, metallurgical and environmental studies necessary to undertake a definitive Feasibility Study and earn a 50% interest in the Cleveland Tailings Project. Bright Phase can elect to purchase the property outright at any time in the first six months after contract signing by payment of an additional AUD\$2-million (the Purchase Price); Bright Phase Pty Ltd is to have first right over the hard rock resources if they do not exercise this option.

Following the completion of the Feasibility Study with a positive result, Lynch Mining will decide whether or not to continue with a 50% co-contributing position or to take a 15% free-carried equity interest in the project. Bright Phase would then raise whatever capital is required to build and commission the tailings treatment plant and any working capital requirements. Lynch Mining is to have a seat on the Project's Steering Committee, thereby retaining the knowledge developed thus far. The additional knowledge of expert consultants who have had input into the project thus far is expected to continue.

Bright Phase will manage the project from contract signing through feasibility to completion of the operation. As part of the feasibility process, and to potentially start generating cashflow, it is intended to build and operate a simplified demonstration-scale pilot processing plant, nominally operating at ~10 t/h (one-tenth of full-scale throughput).

1.1. Location and Access

Exploration Licence 7/2005, "Luina" covering an area of 18 km², lies in western Tasmania and is situated to the west of Waratah and some 50 km southwest of the coastal city of Burnie. The terrain is dominated by steep shrub and tree-covered slopes, with a significant area of rehabilitated mine workings and former town site. Areas of former forestry clearing are also present. The Whyte River transects the tenement area. Access is via the sealed Waratah-Savage River road and then via a series of old sealed and unsealed mine access and forestry roads and tracks for the 15 km drive to Waratah, with some 60 km on the Murchison Highway to Burnie and a further 50 km on the Bass Highway to Devonport, the closest port city. The site is therefore readily accessible.

Figure 1 reveals the location of the Cleveland Tailings Project on a map of Tasmania showing major towns and mines. The location of Cleveland tailings and Lynch Mining hard rock tenements is depicted in

Figure 2, while Figure 3 shows a satellite view of the Cleveland tailings and surrounds.



— Tasmania — Major Mining and Mineral Processing Operations and Proposed Projects

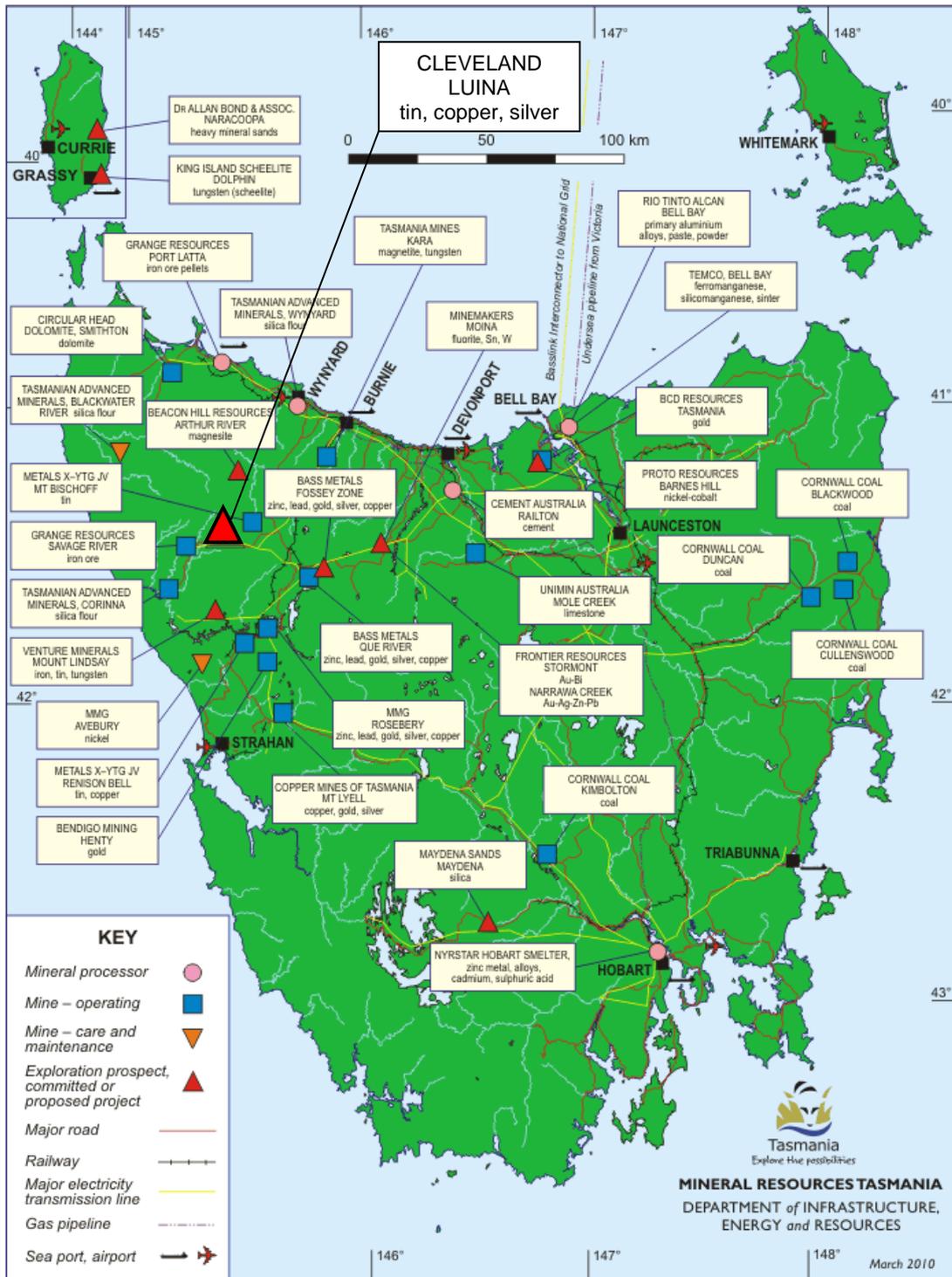


Figure 1 – Map of Tasmania showing major towns and mines and location of the Cleveland Tailings Project (adapted from www.mrt.tas.gov.au)

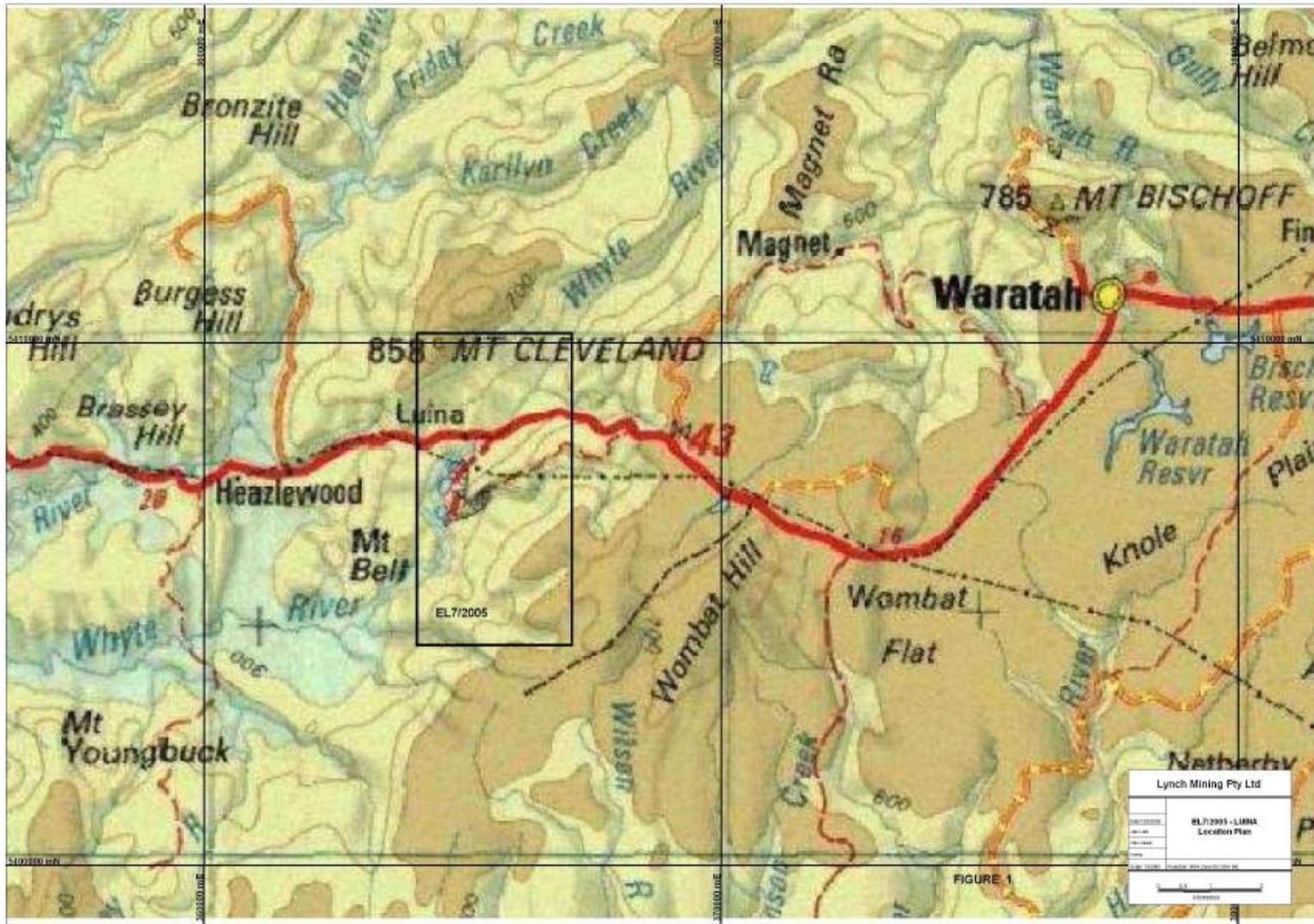


Figure 2 – Location of Cleveland tailings and Lynch Mining hard rock tenements

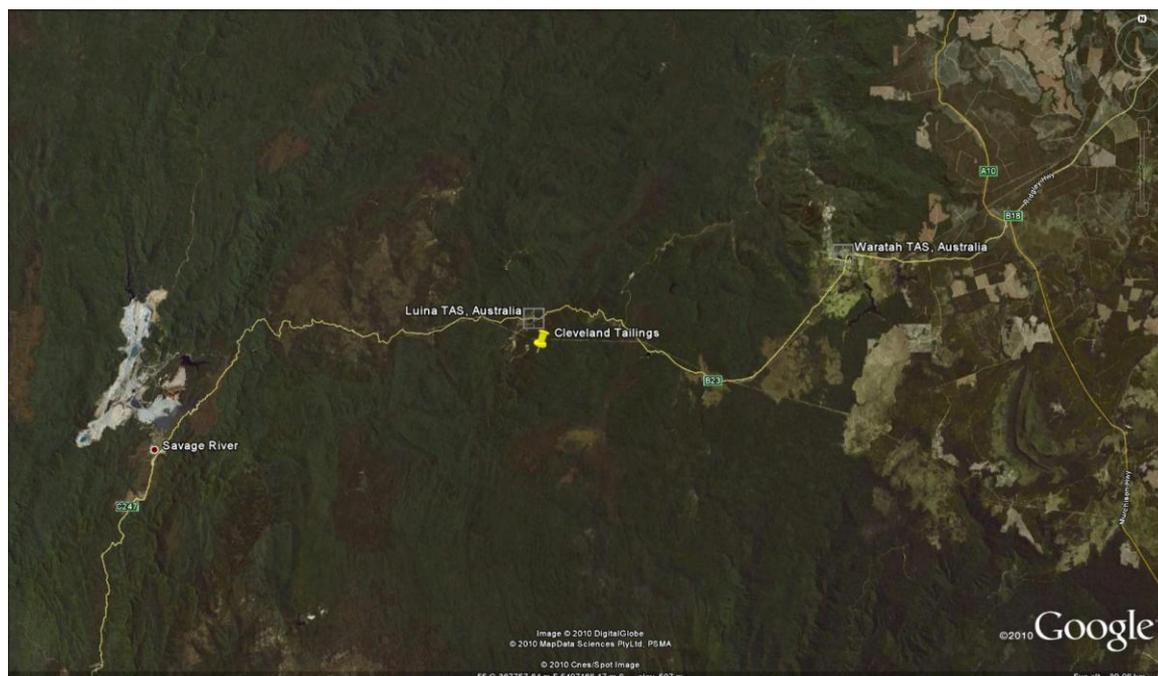


Figure 3 – Satellite view showing location of Cleveland tailings

1.2. Historical Background

The Cleveland deposit was discovered in 1898. Initial production of the surficial ore commenced in 1908, and ceased in 1914 after production of 295.5 tonnes of cassiterite. Tributing continued until 1917, during which time a further 48 tonnes were produced. Aberfoyle commenced mining in 1968, and mining was ceased in early 1986, primarily due to the collapse of the tin price. About 7-million tonnes of ore assaying ~0.82% Sn and ~0.35% Cu was processed. The mill tin recovery was only 59% for the life of the mine and copper recovery was about 60%. The mine sold some 35,500 t of tin metal and 15,000 t of copper in concentrates assaying 55-64% Sn and 20-24% Cu with 3-6% Sn. There has been no hard rock mining activity since that time. Exploration Licence 7/2005 (Luina) covering an area of 18 km² was granted to Lynch Mining Pty Ltd on 30th June 2005 for a period of 5 years, extended for a further year to expire on 1st July 2011.

1.3. Regional and Local Geology

The Cleveland ore bodies occur in a steeply dipping northeast trending succession of arenaceous, argillaceous, and chemical sediments and mafic volcanic rocks and ultramafic/mafic complexes. The sequence is unfossiliferous but has been correlated by rock type with the Cambrian Crimson Creek Formation of the Zeehan-Rosebery area. The Meredith Granite, a high level, late tectonic Late Devonian to Early Carboniferous granitic pluton believed to be genetically associated with the mineralisation, intrudes the sequence east and south of the mine. The Cleveland ore bodies are located in a dominantly fine-grained sedimentary sequence that is thought to lie in an embayment on the margin of a basaltic eruptive centre. The tin-copper ore bodies occur as a series of sub



parallel, near vertical sulphide lenses within the Halls Formation. The mineralisation is composed largely of fine to medium grained quartz, tourmaline, fluorite, chlorite, and pyrrhotite plus chalcopyrite, cassiterite and stannite. Eleven lenses are recognised and are divided into two groups; the footwall and the hanging-wall lodes separated by a micaceous greywacke unit and an overlying basic volcanic unit. The footwall lodes are composed of three relatively thick lenses (Henrys, Lucks and Khaki) which have a strike length of less than 200 metres. The hanging-wall lodes (or Halls lenses) comprise five lenses and have a greater strike length and stratigraphic continuity than the footwall lodes, attaining a maximum thickness of about 30 metres and a maximum strike length of about 600 metres. The sulphide lenses are offset by a series of sub parallel, southeast dipping reverse faults.

Various reports⁴ show that a significant resource remains in the deposit, with Measured and Indicated resources of 5.2-million tonnes @ 0.70% Sn and 0.31% Cu and Inferred resources of 1.3-million tonnes @ 0.72% Sn and 0.22% Cu at a 0.35% Sn cut-off. In addition, a large resource known as the Foley Zone located at the lower levels of the mine had been drilled and is reported to contain 3.8 million tonnes @ 0.28% WO₃ at a 0.2% cut-off. Data for these estimates has not yet been either sourced or verified.

1.4. Tailings

Aberfoyle processed about 7 million tonnes of ore assaying ~0.82% Sn and ~0.35% Cu. The mill tin recovery was only 59% for the life of the mine and copper recovery was about 60%. The mine sold some 35,500 t of tin metal and 15,000 t of copper in concentrates assaying 55-64% Sn and 20-24% Cu with 3-6% Sn.

The current resource comprises about 4.4 million tonnes of mill tailings grading ~0.32% Sn and ~0.16% Cu, stored in two tailings dams. Some 3.2-4.0 million tonnes of Heavy Media Float (grading 0.25% Sn and 0.08% Cu) were produced at Cleveland, but only an estimated 750,000 t remain, as most were sold off for use as road base. In late 1986 Aberfoyle Ltd. the mine owner used the Cleveland mill for testing a new processing route for its Hellyer base-metal ore and processed some 300,000 t, of which about 80,000 t of non-value waste material were placed on top of Tailings Dam 2 and 220,000 t were discharged into the Cleveland mine through a rise drilled from near the Cleveland mill. The overall tin grade (0.32%) of the Cleveland tailings resource is comparable with grades of existing plants mining primary underground tin ores, however, tailings deposits such as at Cleveland do not require the expense of exploration, crushing and primary milling, presenting a clear upside.

The dams comprise significant amounts of slimes that particularly in Tailings Dam 1 (completed in 1971) predated the introduction of flotation at Cleveland. The beach and wall areas are highly oxidized and are cemented in large agglomerates⁶.

A satellite view of Cleveland tailings and old workings and Luina abandoned townsite is shown in Figure 4, with a detailed view of the tailings deposit shown in Figure 5.

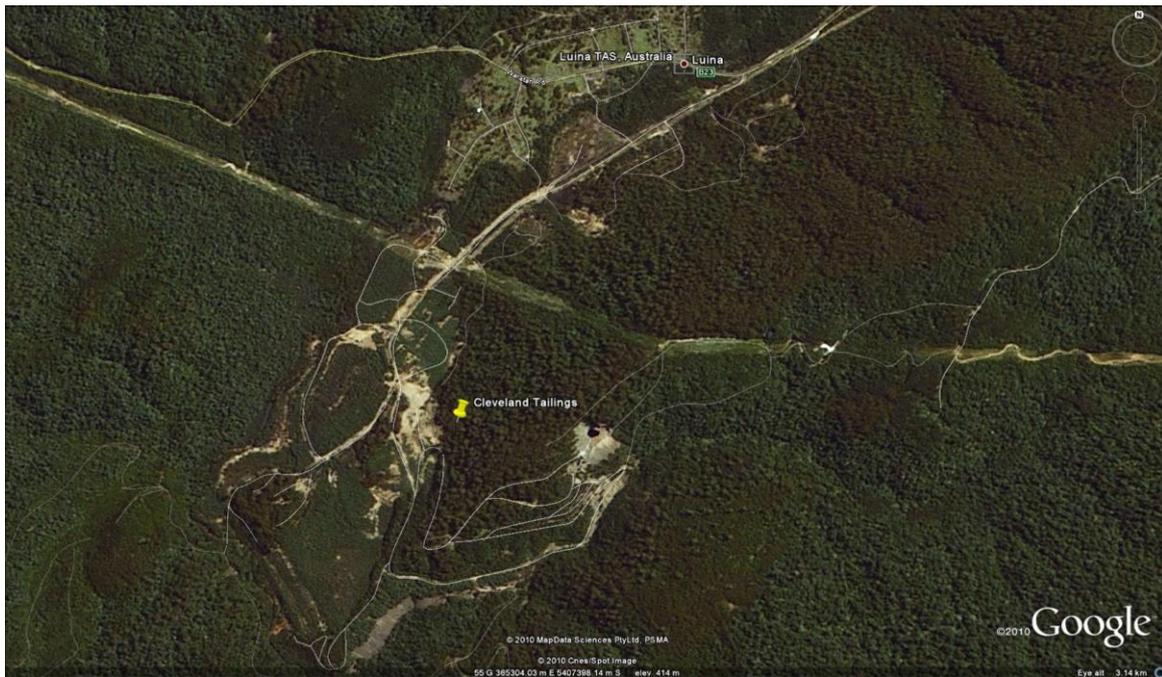


Figure 4 – Satellite view of Cleveland tailings and old workings and Luina abandoned townsite

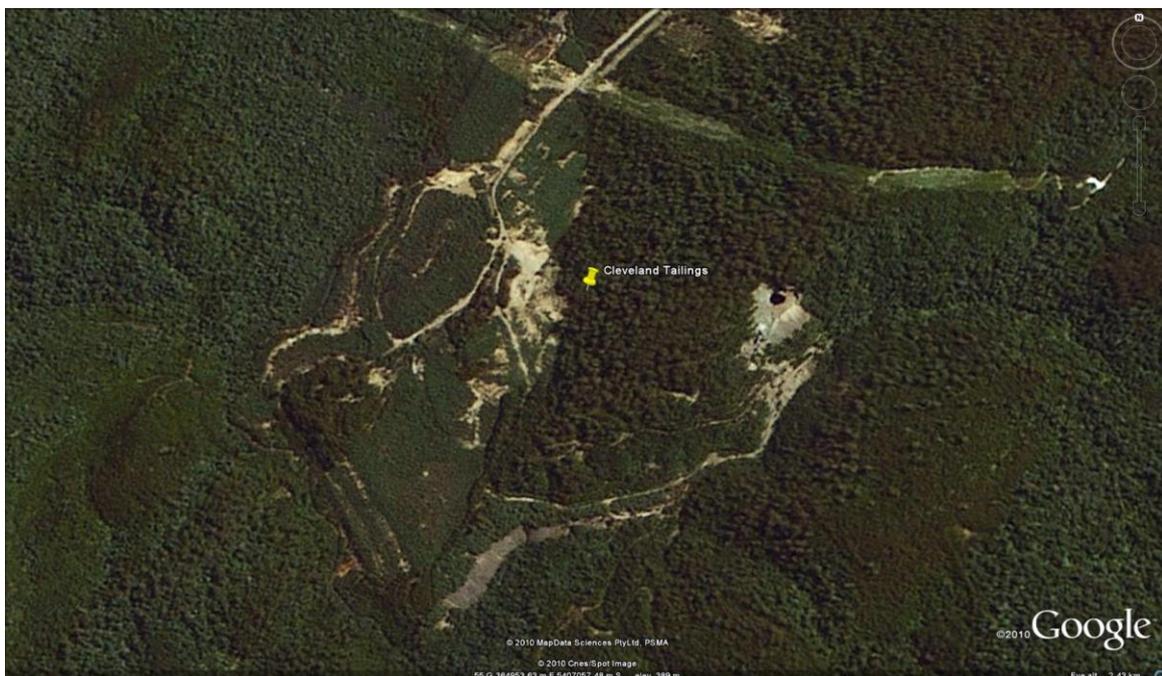


Figure 5 – Detailed satellite view of Cleveland tailings TD2

Schematic representations of TD1 and TD2 dam profiles are depicted in Figure 7. The heterogeneous composition of the deposit is immediately obvious from the indicated variation in sulfur grade and particle size. In particular, significant slimes are present in TD1, whereas TD2 suffers the complication of the Hellyer tailings overdeposit (containing 35% S), as well as the lower-grade HMS Float product abutment.

The Whyte River flows relatively close to the western base of both the Northern TD1 and Southern TD2 tailings dams, before flowing south and joining the Pieman River near Corinna, meeting the Great Australian Bight at Misery Island.

Figure 6 – Cleveland tailings dam profiles, showing variation zones in sulfur grade and particle size



1.5. Environmental Incentive – Acid Drainage

Cleveland tailings, like many base-metal sulphide abandoned mine sites, contain large deposits of reactive waste rocks and tailings that have high concentrations of trace metals. For Cleveland tailings impact on the Whyte River these values were measured² in 2000 as sulfate 680 mg/L, metals 17 mg/L and pH 5.3, significantly exceeding the ANZECC (2000)³ recommended trigger values for several metal pollutants (Cd, Cu, Mn, Pb, Zn) in aquatic ecosystems, as shown in Table 1. The trigger values are given by ANZECC in terms of Level of Protection (LoP), depicting expected per cent of species protected on exposure. Actual recommended values are site specific and determined after assessment by the relevant authorities. This holds both at the recommended 95 percentile protection limit (slight to medium impact) and the 80 percentile protection limit (high impact).

Downstream impact from mass loading of these reactive waste rocks and tailings during major flood events is a serious threat to the receiving environment for decades². A Cleveland tailings retreatment project will potentially have a major positive impact on the long-term environmental pollution aspects from these historic mine workings by removing acidic sulphide and metal contaminants at source and this is a key incentive for the proposed project to proceed.

Table 1 – Whyte River Assays at Cleveland Run-off and Recommended Trigger Values³

SO4	As	Cd	Cu	Fe	Mn	Pb	Zn	TDM	Unit
680	0.008	0.125	0.31	2.75	13.3	0.29	0.054	16.837	mg/L
680,000	8	125	310	2,750	13,300	290	54	16,837	ug/L
									LoP
-	1	0.06	1	-	1200	1	2.4		99%
-	24	0.2	1.4	-	1900	3.4	8		95%
-	94	0.4	1.8	-	2500	5.6	15		90%
-	360	0.8	2.5	-	3600	9.4	31		80%

Exceeded trigger values in red type; TDM – Total Dissolved Metals; LOP – Level of protection (% of species); assay data² from 10/03/2000 Whyte River @ Cleveland mine MD 364710 5407570 SRO



2. CLEVELAND TAILINGS RETREATMENT PROJECT

2.1. Concentrator Records and Previous Work

2.1.1. Concentrator Records

The mine treated about 7.0-million tonnes of ore between 1968 and 1986, the average feed grade was reportedly¹ 0.65-0.95% Sn and 0.25-0.45% Cu. Recovery ranged from 50% to 73% and averaged about 59% for the life of mine. About 36,500 tonnes of tin metal and 15,000 of copper metal were sold. This means that some 19,500 tonnes of tin have been deposited as tailings, but some 6,000 tonnes of tin has been lost in sold-off HMS Float. The production metal balance calculation is shown in Table 2, from which the Cleveland combined tailings resource estimate has been extracted into Table 3. Validation estimation using the reported average feed grades, total tonnes processed and metal sold returned an estimated combined tailings grade of 0.38% Sn and 0.18% Cu, comparable with the grades reported in Table 3.

Table 2 – Cleveland Mill Records Metal Balance

Product	Tonnes	Sn%	Tonnes		Dist'n		
			Sn	Cu%	Sn%	Cu%	
Copper Conc.	65,000	4.5	2,925	23.01	14,957	5.1	63.2
Tin Conc.	63,000	55	34,650	0.01	6	60.1	0.0
Sold HMS Float	2,450,000	0.25	6,125	0.08	1,960	10.6	8.3
HMS Float on site	750,000	0.25	1,875	0.08	600	3.3	2.5
No. 1 Tailings Dam	950,000	0.36	3,420	0.19	1,805	5.9	7.6
No. 2 Tailings Dam	2,700,000	0.32	8,640	0.16	4,320	15.0	18.3
Mined Ore	6,978,000	0.83	57,635	0.34	23,648	100	100

Table 3 – Cleveland Combined Tailings Resource Estimate

Product	Tonnes	Sn%	Tonnes		Dist'n		
			Sn	Cu%	Sn%	Cu%	
HMS Float on site	750,000	0.25	1,875	0.08	600	13.5	8.9
No. 1 Tailings Dam	950,000	0.36	3,420	0.19	1,805	24.5	26.8
No. 2 Tailings Dam	2,700,000	0.32	8,640	0.16	4,320	62.0	64.2
Combined Tailings	4,400,000	0.32	13,935	0.15	6,725	100	100



2.2. Metallurgical Testwork and Sampling

2.2.1. Sampling

A detailed sampling programme of the current tailings dams was carried out by Aberfoyle⁶ in 1981. While it was possible to sample walls, beach and interior of the completed and dry Tailings Dam No. 1 (TD1), the interior of the wet and active Tailings Dam No. 2 (TD2) was not possible. Therefore The TD2 samples would be expected to be biased towards more oxidized and agglomerated (and hence, less metallurgically responsive) samples. Table surveys were carried out on the samples; however, this data is not yet available for study.

A resource estimate was carried out as at 31st May 1983. TD1 was completed in 1971, while TD 2 was active. The combined tailings deposit was at that point estimated as 3.45-million tonnes containing 11,250 tonnes of tin. This yields an estimated grade of 0.33% Sn, consistent with current estimates.

Lynch Mining carried out aircore drilling in June 2008⁷ and collected 1 m samples from 32 completed holes (see Appendix 3 for map and logs for 11 holes transecting TD1 and 21 holes transecting TD2). Composited samples from each of the 32 completed holes were metallurgically evaluated at the Burnie Research Laboratory and these assays are included in Appendix. The individual samples for each metre of drilling were not assayed due to the sticky nature of the material and consequent fluidization during sampling. These samples have been stored off for potential feasibility study variability work, subject to scrutiny of potential cross-mixing.

Historical data which has been used for underground resource estimation will be sourced if possible and recompiled to a modern dataset for further evaluation. In addition, archived mineralogical and metallurgical testwork reports shall be sourced.

2.2.2. Metallurgical Testwork

A series of testwork programs have been completed by various parties over the last 25 years on development of a viable process for retreatment of the Cleveland Tailings. An extensive programme of pilot and flotation testwork was presented⁶ by Aberfoyle Metallurgy Services (AMS) in 1984, the findings of which have since been further developed in a period beginning 2006 by Lynch Mining. To support the more recent programme, Bright Phase performed a ‘sighter’ QEMSCAN mineralogical evaluation of fundamental processing characteristics in the Cleveland Tailings in November 2009.

Aberfoyle Metallurgy Service (AMS) – Flotation Testwork and Piloting

Significant metallurgical testwork on re-treatment of tailings was undertaken by Aberfoyle in the 1979-1984 period; however, this data has not yet been located for review, and this would be part of definitive feasibility activities.

The programme undertaken by AMS and presented⁶ in 1984 was focused on the evaluation of conventional flotation options and examination of a process for pre-



concentration with matte fuming. The results of the AMS study indicated that recovery of tin by conventional gravity and flotation was between 33% and 45% to yield a concentrate grade of 25-35% Sn. An option for pre-concentration of tin with subsequent matte fuming was shown to return better recoveries to a Sn concentrate grading 60-65% Sn. Recoveries by this process showed significant differences between TD1 (48-53% Sn recovery) and TD2 (64-69% Sn recovery). It was noted that slimes were a significant issue in TD1 along with more complex flotation chemistry associated with the presence of oxidized siderite and Fe sulfide minerals.

Lynch Mining – Magnetic, Gravity and Flotation Testwork

In early 2008 the Burnie Laboratory, Burnie, Tasmania (an Ammtec subsidiary) was commissioned to carry out a laboratory investigation on the retreatment of Cleveland Mine tin tailings (0.35-0.4% Sn) by way of enhanced gravity (Falcon centrifugal concentrator) pre-treatment followed by flotation processes to achieve tin concentrate grades of greater than 15% Sn. The primary focus of the investigation was to evaluate the effectiveness of the Falcon Concentrator Ultrafine series (FCUF) in simultaneously upgrading the tin head grade and in rejecting waste material such as gangue minerals and slimes. The latter have been generally known to be unfriendly to flotation processes, especially to cassiterite tin flotation. This option was investigated following the success of a similar process at the Renison Rentails project on material that appeared analogous to the Cleveland tailings.

The outcome of this programme was that the direct superpanning results (indicating gravity-recoverable tin) on a grind, deslime, sulfide flotation, gravity flowsheet were good and 33% of the tin was recovered at concentrate grades of 40% Sn. The FCUF did not perform for Cleveland tailings as it had for Rentails – without sacrificing recovery, the upgrading of head grade remained mediocre. Operating changes to a higher feed slurry density (10-15% solids) or to a higher energy spin (300-420 G) did not result in lowering tin losses to reject tails. Furthermore, flotation results of the concentrate fraction from FCUF had not improved, indicating that FCUF had not fundamentally removed from tailings materials problematic to flotation. The adverse impact to flotation resulting from the by-products of oxidised iron (Fe)-bearing sulphides and carbonates in tailings had been previously highlighted in the 1984 AMS report⁶.

It is also indicated that, because of the *in situ* oxidation by-products, Cleveland tailings are metallurgically more complex than Renison tailings for flotation on the samples treated – including sulfide flotation. All the flotation results reiterated the findings by AMS – in that prerequisites to successful cassiterite flotation appear to be: vigorous desliming (and washing of soluble metal ions by extension); sulfide removal (flotation complemented by low intensity magnetic separation); and effective rejection of oxidized sulfide and sideritic carbonates (by high magnetic intensity separation). It was recommended that the next stage of testwork on Cleveland tailings retreatment was to go back to the basics in removing and/or suppressing these flotation-interfering components and that may have to also include attrition-cleaning (of mineral surfaces) in the feed preparation stage and the use of an effective dispersant during cassiterite flotation.



The most recent testwork program by Lynch Mining was focused on removal of deleterious sideritic carbonates and Fe oxides by Low Intensity Magnetic Separation (LIMS) and high intensity magnetic separation (WHIMS), prior to flotation of sulfide minerals and cassiterite. The magnetic separation step may be carried out in the laboratory using rapid Dry High Intensity Magnetic Separation (DHIMS). In the Testwork in November 2009 on a grind, deslime, sulfide flotation, tin flotation flowsheet recovered 78% of the tin from beneficiated feed to a low-grade (2.3% Sn) concentrate. Testwork in February 2010 on a grind, deslime, rapid DHIMS, iron oxide prefloat, sulfide flotation, tin flotation flowsheet recovered ~60% of the tin from beneficiated feed to a rougher concentrate suitable for cleaning and blending with the gravity tin concentrate.

Bright Phase – QEMSCAN Evaluation of Cleveland Tailings

Examination of Cleveland Tailings (CTD 201-220) sourced from the Cleveland project, Tasmania, Australia (EL7/2005) was completed using QEMSCAN by Bright Phase Pty Ltd. The primary focus of this programme was to define the deportment of tin and copper, along with the root cause for poor recovery by conventional flotation and gravity separation.

The key outcomes of this study included:

- All tin (Sn) occurred in Cassiterite (SnO₂), which is known to be a floatable mineral.
- All copper (Cu) occurred with floatable copper iron sulfide minerals.
- Average grain size of Sn minerals was 5-20 µm, which is below practical recovery size for conventional flotation or gravity processes.
- Liberation of Sn and Cu minerals was poor in coarse particles (see Figure 7). However, coarse particles consisted of agglomerated fine particles and should be easily broken down, promoting liberation.
- Major contaminant minerals were pyrite and pyrrhotite, which must be depressed in flotation processes.

Fine grain size and poor liberation caused by agglomeration of value minerals, adequately described why conventional gravity and flotation methods have been unsuccessful in obtaining saleable tin and copper concentrate.

To overcome these difficulties a number of solutions may be examined, which must address the following key issues:

- Liberation of copper and tin minerals must be promoted by deagglomeration of coarse particles.
- Any solution must have the capability to recover fine particles, typically in the range of 5-20 µm.

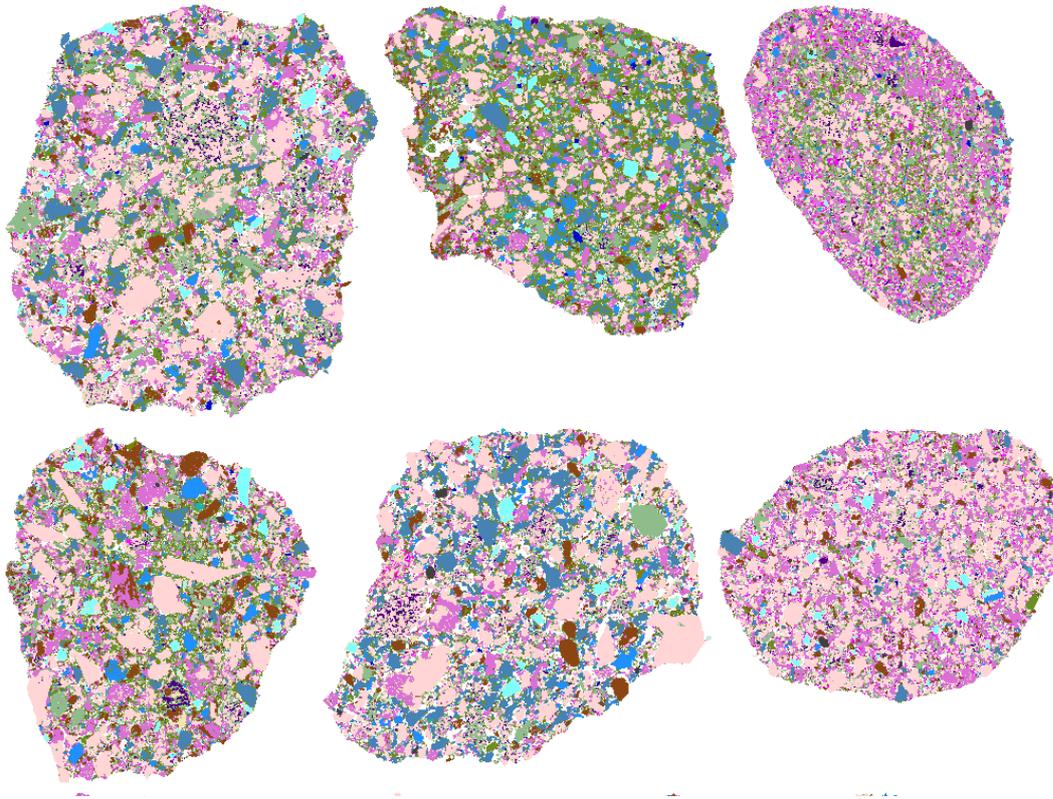


Figure 7 – Particle map from QEMSCAN PMA analysis of +106 µm fraction of Cleveland tailings composite (resolution @ 2 µm)

2.3. Conventional and Hydraulic Mining

Several other tailings retreatment operations, particularly in Africa, have implemented hydraulic mining to reduce mining costs. This option, amongst other variants, warrants consideration.

2.3.1. Plant and New Tailings Dam Location

The presence of the old mill floor, some pads, culverts and bitumen around the old plant area and Luina township may influence decisions on placement of new plant and infrastructure. Decisions on plant and new tailings dam location shall depend on the outcome of hydrological and other assessments conducted as part of the definitive feasibility study. To this end, expert consultants shall be engaged during definitive feasibility study to develop a hydrogeological model for the Cleveland Tailings Project to be utilized for mine, process and tailings dam design. The potential need to remove and stockpile the high-sulfide low grade Hellyer tailings would also be taken into account.

After mining and processing of the tailings, the original tailings repository can be revegetated and returned to its original native forest state. Re-treatment of the tailings can essentially be viewed as a clean-up operation by proxy, with the additional benefit of helping alleviate future acid drainage issues.



2.3.2. Mining

A common method of tailings reclamation is hydraulic mining using high pressure monitors to pulp the tailings, which, after screening, can be pumped to the plant. Conventional mining is currently favoured for the Cleveland tailings retreatment over hydraulic mining. The latter option would likely increase the existing acid drainage issue, making hydraulic mining unattractive from a water containment point of view as well as increase the risk of environmental impact during flood events. Moreover, the relatively dilute and variable pulp density in the pumped slurry stream would demand a feed thickener, adding significantly to capital costs.

Both dams are very heterogeneous and will require mining by a “strip” method, where the slimes, fines and sands will be blended at the mill subject to more a refined ore typing protocol. It is expected to use a loader, excavator, dozer and 3 35 JCB type trucks to transport the tailings to the mill feed mixing pad, where the different zones will be blended and fed into a trommel. A float crushing circuit will use the loader and trucks.

2.3.3. Plant Pre-Screening

Given the close location of the proposed plant to the resource no provision has been made for slurrification and pre-screening at the dump face. Given the growth of foliage on the dump, close attention to vegetation and scrap removal from the dump should be made prior to mining.

2.4. Metallurgical Process Description

A techno-economically successful operation would require the application of modern micro-characterization, ore typing and mineral processing techniques as well as access to fast turnaround of specialist assays and metallurgical testing.

A range of particle liberation, separation and concentration techniques shall be required, incorporating a blend of established wisdom and modern commercially-proven innovations. Separate front-end processing is required for Tailings and for HMS Float material, the latter requiring hammer milling. The base case process flowsheet recovers coarse cassiterite using spirals, fine cassiterite with centrifugal concentrators and fine froth flotation. Copper is recovered from a sulfide-rich concentrate by fine froth flotation. Figure 8 depicts the layout of such a circuit on the old plant floor.

For the purpose of this study the simplified process flowsheet broadly outlined above and described by Moony¹ in 2007 (see Figure 9) has been allowed for to determine the potential feasibility of the Cleveland tailings retreatment project on a preliminary basis (updated for 2010 financials). A simplified flowsheet mass balance based on this revised flowsheet and taking the BPPL QEMSCAN data into account has been developed as part of the current study and is included in Appendix 2.

The testwork and development work carried out in the interim by Lynch, Esker and Bright Phase has identified several processing aspects likely to impact positively on



recoveries from selected streams (the revised flowsheet by Moony to 100 tph and with additional Falcon centrifuges is included in Appendix 2). Assessment of these options is best done, after appropriate laboratory-scale testing, on a demonstration pilot plant at reasonable scale and located onsite, given known scale-up constraints in some of the equipment.

- Magnetic separators for the possible removal of deleterious magnetics fractions (eg, WHIMS, LIMS, SLan).
- Centrifugal gravity concentrators to enhance recovery of fine mineral particles (eg, Kelsey jig, UF and C Falcons).
- Pneumatic flotation cells to enhance recovery of fine mineral particles (eg, MMS Imhoflot G-Cells).
- Ultrafine grinding mill to enhance liberation of value minerals from flotation feed streams (eg, Isamill, Deswik, Metso Vertimill, Metprotech, MMS Ro-Star).
- Single-stage fumer to produce high-grade tin concentrates from low-grade concentrates and mags rejects, thereby increasing overall recovery. This option is still under review (eg, Ausmelt).



2.4.1. Milling and Attritioning

Blended tailings is fed at a nominal 70-84 tph into a feed hopper and across a vibratory grizzly screen to remove trash material such as residual vegetation and dumped waste materials, which report to a trash dump. The screen underflow is washed with water and fed into a trommel, which is a screened cylinder used to separate materials by size. Oversize reject at +2 mm is produced from this beneficiation. Some disaggregation of cemented particles will also occur via the scrubbing action within the trommel. Discharge is fed into a ball mill with Derrick screen oversize recycle. HM Float material is fed at a nominal 10-16 tph into a bin feeding a hammer mill screening at 12 mm to produce a product for feeding into a ball mill (separate or blended with the tailings mill feed).

Agitated high-energy attritioners have not been allowed for but may assist with cement disaggregation and mineral surface cleaning. Ultrafine grinding has not been allowed for on circuit streams but may be beneficial for float recoveries.

2.4.2. Gravity Separation

Tin processing relies on the high specific gravity (SG of cassiterite (6.9 g/cm³) compared with the lighter gangue minerals (quartz, micas, chlorite ~2.6 g/cm³) and the iron-containing minerals (pyrite, pyrrhotite, hematite ~5 g/cm³). Hydrocyclones are used to deslime and produce a bulk heavies stream containing the coarse tin particles. Bulk gravity separation via FM-1 spirals, which have been successfully used many times for cassiterite recovery, is used to further separate coarse heavy mineral particles from fine gangue and clays, producing a coarse tin concentrate. These may be used in conjunction with Kelsey jigs and centrifugal concentrators such as the UF and C Falcons, to recover fine cassiterite and sulfides (containing much of the copper minerals).

2.4.3. Flotation

Cassiterite flotation has been deployed at many commercial installations over the last fifty years. Addition of collector, depressant and frother reagents is made to an aerated pulp in a suitably designed cell to ensure adherence of value mineral particles to bubbles for recovery as a froth overflow, whilst allowing the gangue minerals not to adhere to bubbles and to separate via gravity. A combination of roughers, cleaners and scavenging cells is typically used for the separation.

Pneumatic flotation devices with external aerators decrease bubble size and turbulence, allowing fines to be more readily floated without disengaging in the cell itself, with gangue particles separating via a combination of gravity and centrifugal forces. Improvements have been made in reagent development, with several options available. This technology will be potentially useful for this project, as the cassiterite is fine-grained.

Recent testwork on Cleveland tailings suggests that a combination of magnetic separators may be useful in the upfront removal of minerals that are deleterious to flotation. Allowance is not made for magnetic separation in the flowsheet. Sufficient improvements



in recoveries made via particle liberation and fine particle centrifugal concentration may also simplify all flotation circuits.

2.4.4. Tin Fuming

Tin fuming at 1250 °C is commonly used to upgrade tin concentrates. No allowance has been made in the current circuit for tin fuming; however it may be a useful economic decision

Current testwork results and simplified mass balance (see Appendix 2) suggest that a 35% Sn concentrate blend can be produced from Cleveland tailings. Single-stage fuming may be considered to produce high-grade tin concentrates (~65% Sn) from low-grade concentrates and mags rejects, thereby increasing overall recovery. The Ausmelt technology is an example that has been successfully commercialized. The potential use of tin fuming in this project will be considered in the definitive feasibility study.

2.4.5. Tailings

Tailings would be pumped to a new facility. If the remnant stopes within the old Cleveland mine are separately determined to be uneconomic (beyond the scope of this study), then mine backfill disposal of tailings would be an attractive option economically and environmentally. Alternatively, high-sulfide tailings only could be disposed in known spent stopes, leaving the higher-grade stopes available for possible future mining.

2.4.6. Infrastructure

The tailings are situated near an old mine and consequently the area is well accessed by infrastructure. The tailings area only a few kilometres from the nearest small town and reasonably sized city. Roads are present at the site but will require refurbishing. Power runs very near the minesite and access is not considered to present a problem.

The process plant will need a reliable supply of water throughout the year. The Whyte River would appear to be able to supply an adequate amount of water.

Only relatively small consumption of chemicals such as flotation reagents and grinding media are required and are easily supplied, given that several other operating mines and processing plants exist in the general area.

2.4.7. Relevant Case Studies

It is useful to compare the proposed process flowsheet with other tin operations and projects in Australia. The Rentails and Mt Lindsey projects are relevant in terms of tin grades, shown in context in Table 4. By way of comparison, Cleveland tailings project comprises 4,400 kt @ 0.32%, with 13,935 t contained tin metal, placing it at eighth position in tin value alone but at a higher position in terms of equivalent contained tin metal if by-product copper and silver are included.



Table 4 – Top 10 Largest JORC Compliant Tin Resources in Australia

Ranking	Project	Company	Resource Category	Tonnes or bcm	Tin Grade	Contained Tin Metal
1st	Renison Bell	Metals X Ltd	Measured	645kt	1.87%	12,090t
			Indicated	1,401kt	2.01%	28,200t
			Inferred	2,351kt	2.04%	48,000t
			Total	4,397kt	2.01%	88,290t
2 nd	Rentails	Metals X Ltd	Measured	18,176kt	0.42%	76,600t
			Indicated	-	-	-
			Inferred	-	-	-
			Total	18,176kt	0.42%	76,600t
3rd	Mount Lindsay	Venture Minerals	Measured	-	-	-
			Indicated	-	-	-
			Inferred	23,000kt	0.2%	49,000t
			Total	23,000kt	0.2%	49,000t
4 th	Ardlethan	Marlborough Resources NL	Measured	2,290kbcm	2.95kg/bcm	6,750t
			Indicated	2,132kbcm	2.66kg/bcm	5,670t
			Inferred	8,729kbcm	1.26kg/bcm	11,000t
			Total	13,151kbcm	1.78kg/bcm	23,420t
5 th	Mount Bischoff	Metals X Ltd	Measured	3kt	0.82%	20t
			Indicated	1,405kt	1.10%	15,500t
			Inferred	1,071kt	0.70%	7,730t
			Total	2,479kt	0.94%	23,250t
6 th	Doradilla	YTC Resources	Measured	-	-	-
			Indicated	-	-	-
			Inferred	7,810kt	0.28%	22,300t
			Total	7,810kt	0.28%	22,300t
7 th	Gillian	Consolidated Tin Mines Ltd	Measured	-	-	-
			Indicated	2,300kt	0.84%	19,320t
			Inferred	-	-	-
			Total	2,300kt	0.84%	19,320t
8 th	Scotia	Van Dieman Mines PLC	Measured	5,320kbcm	1.02kg/bcm	5,440t
			Indicated	-	-	-
			Inferred	21,350kbcm	0.27kg/bcm	5,850t
			Total	26,670kbcm	0.42kg/bcm	11,290t
9 th	Collingwood	Metals X Ltd	Measured	39kt	1.26%	490t
			Indicated	269kt	1.34%	3,620t
			Inferred	346kt	1.06%	3,680t
			Total	654kt	1.19%	7,790t
10 th	Baal Gammon	North Queensland Metals Ltd	Measured	-	-	-
			Indicated	3,561kt	0.21%	7,610t
			Inferred	-	-	-
			Total	3,561kt	0.21%	7,610t

Source: Venture Minerals website

Metals X – Rentails Project, Tasmania

The Rentails project is aimed at the re-processing and recovery of tin from an estimated 18.2 million tonnes of tailings at an average grade of 0.42% tin and 0.20% copper (containing over 76,000 tonnes of tin and over 36,000 tonnes of copper metal) that remain at the site from the historic processing of tin ores from the Renison Bell mine.

The project has been through a conceptual pre-feasibility study in 2005, which identified that a combination of sulfide flotation and tin flotation separation techniques could produce a low grade concentrate which would be fused to produce a saleable tin product.

Metals X subsequently completed a Feasibility Study for the Rentails project, which showed that the average production costs after copper credits was US\$9,500 assuming a copper price of US\$5,000/t at a USD exchange rate of \$0.80. The capital costs under this



development philosophy were estimated at A\$194 million \pm 15%. Alternative project concepts and development methods are under consideration. The process route uses proven technology and has developed a robust process for the recovery of both tin and copper. In addition the project allows for the treatment of other tin sulfide (stannite) ore bodies within the region, which are not currently viable under conventional tin processing routes requiring tin fuming. The study outcomes demonstrate that although the project is marginal at current levels, it remains exponentially exposed to upside from Australian dollar tin and copper prices. A significant consideration is that the timing of the study and its capital and engineering construction estimates were completed before the global financial crisis.

The Rentails Project concept and flowsheet comprises the reclaiming or mining of tailings from the dams at a rate of 2 Mt per annum to produce approximately 5,300 t of tin and 2,000 t of copper contained in concentrates per annum. The tailings ore is finely ground to increase the liberation of the tin bearing minerals before removing the gangue sulfides by flotation. The sulfide flotation tails containing the fine tin is then processed through classification, gravity recovery and tin flotation circuits to produce a 10% tin concentrate that will be smelted to produce a tin fume product assaying in excess of 68% tin. A by-product from the fuming process will be a saleable copper matte assaying 70% copper.

Venture Minerals – Mt Lindsay Project, Tasmania

The Mt Lindsay Project is located 15 km northwest and along strike of the Renison Bell Tin deposit and immediately south of the Savage River magnetite mine. Venture has already defined a substantial resource base in tin, tungsten and iron and is currently completing a 10,000 m diamond core drill program aimed at significantly increasing that resource base.

The Company has adopted an exploration programme through 2010 which currently involves five drill rigs testing high priority targets at the Mt Lindsay Project. Following the completion of the current exploration phase the Company anticipates outlining a development schedule for the project in the second half of 2010.

2.5. Tailings Retreatment Project Economics

The economic viability of the tailings retreatment operation depends on the capital outlay required to build the plant and associated infrastructure and the operating cost. Economic evaluation data is presented in Appendix 4.

2.5.1. Capital Cost

The capital costs are estimated to be a total of A\$29.1-million for the 1 M tpa processing facility (assuming a USD/AUD exchange rate of 0.85). This cost includes a contingency of 30%. Capital equipment estimates are based on the feasibility review¹ performed for Lynch Mining by Esker Pty Ltd in 2007 and updated by Nick Moony of Esker on



10/08/2008 to a 1 M tpa processing facility. Consumer Price Index (CPI) adjustments have been made. Additional items such as new TSF, engineering procurement and construction management (EPCM), civils and owner's costs including permitting and environmental impact assessment (EIA) have been included based on a 2008 engineering scoping study carried out by BPPL on a 1 Mtpa tailings re-treatment plant.

The front-end comminution circuit for processing the HMS Float material is excluded from the capital requirements and will be funded from operating revenues after Year 1.

The cost of freight for all capital equipment is excluded as it has been integrated into the installed capital costs of the equipment.

2.5.2. Operating Cost

The overall operating cost was estimated at A\$17.35 per tonne tailings processed, or expressed in terms of end-product, A\$ 2,984/t tin concentrate produced. Additional revenue for the copper concentrate and silver credits would also accrue.

Operating cost estimates are based on those on the feasibility review¹ performed for Lynch Mining by Esker Pty Ltd in 2007, adjusted for CPI. The estimates are drawn from tin processing industry experience Australia-wide.

2.5.3. Royalties and taxes

All calculations are on a pre-tax basis. The proposed federal Mining Resources Rent Tax (MRRT) does not apply to the metals recovered in the Cleveland tailings re-treatment project. Royalties under Tasmania's Mineral Resources Development Act 1995 (as at 1 July 2010) is payable at a rate of 1.6% of net sales plus profit (maximum of 5% of net sales). The project team is investigating the removal of the state royalty as a state government assistance for BPPL helping clean up the long-term acid drainage environmental problem.

2.5.4. Discussion

The net present value (NPV) of the base case project is calculated as A\$74-million, yielding a payback period of 0.7 years. Plant equipment at the end of the 5-year project would be available for use in a potential underground mine at Cleveland or another tailings retreatment project. Tin constitutes ~78% of revenue, with by-product Cu (~13%) and silver (~9%).

The analysis also highlights the possibility that the demonstration pilot plant capital could be paid for from the revenues it generates, potentially making this important component of the feasibility assessment process close to self-funding.



Sensitivity analysis across the key components was carried out and the results are summarized in Table 5, showing that the project viability is relatively insensitive to capital and operating cost changes, somewhat sensitive to variations in tin recovery and grade, and most sensitive to changes to the commodity price.

Table 5 – Key Sensitivities (Pre-Tax) for the Project

		NPV (\$M)
<i>Base Case</i>	<i>Change</i>	74
Commodity price change	+10%	88
	-10%	60
Operating Costs	+10%	70
	-10%	77
Capex	+10%	70
	-10%	76
Tin Recovery	+10%	83
	-10%	64
Tin Head Grade	+10%	81
	-10%	67

- **Commodity Price.** The tin and copper metal prices have recovered (see Figure 10) and look set to remain stable for some time, to feed the supply of tinsplate and copper for building in growing economies such as India and China. However, for the purpose of this pre-feasibility study, direct sale of low-grade concentrate to smelters has been assumed. Sale price of concentrate is impacted primarily by impurity type and level, as summarized in Appendix 4. Increasing commodity prices could be obtained by way of one or a combination of the following:
 - Upgrading the tin concentrate (to ~65%) by fuming, either onsite or at a toll-treatment facility (e.g., Ausmelt), thereby also removing impurities.
 - Circuit modifications to enhance concentrate cleaning and recleaning efficiencies (the project is less sensitive to capital variations).
- **Tin Recovery.** The deposit is quite heterogeneous, with zones and layers of slimes, fines and other waste materials present, so circuit variability will be a challenge requiring ore typing and good blending planning. Increased tin recoveries could result from the following:
 - Application of fine particle gravity recovery technologies (e.g., Falcon).
 - Application of fine particle flotation recovery technologies (e.g., G-Cells).
 - Circuit modifications to enhance scavenging of value minerals.
 - Optimization of grinding conditions to increase particle liberation.
 - Introduction of ultrafine grinding of selected stream(s).

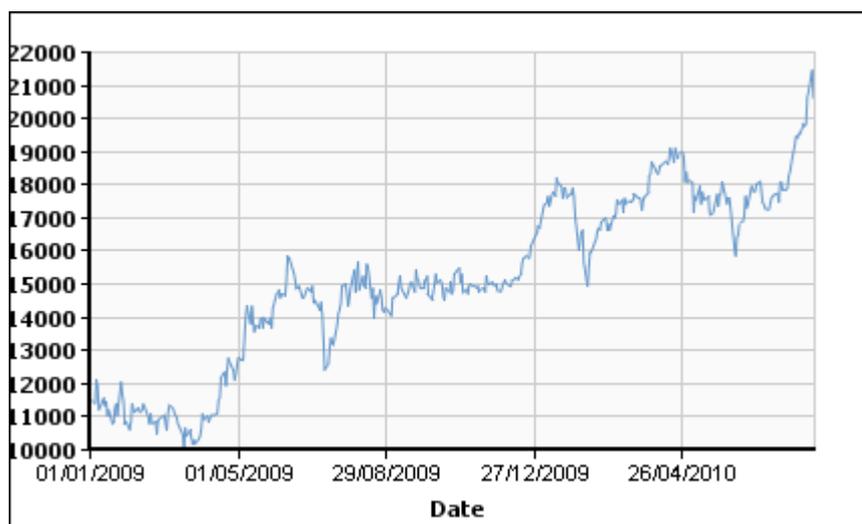


Figure 10 – Tin prices since 1/01/2009

Therefore, additional capital investments that achieve the above objectives would likely be warranted, either upfront or as modifications during the course of the operation. The incentive for onsite demonstration plant operation as part of the feasibility process should serve to enhance these benefits.

It is possible that some fabrication and installation costs could be substantially reduced by sourcing second-hand equipment from neighbouring countries where a number of operations have wound up recently.

The project has a high IRR and will generate sales of around \$57-million per year with an operating cost estimated at \$17-million per year. The project therefore has an attractive return and project economics can be further increased by optimizing the plant to increase concentrate grades and metal value recoveries.

3. MINERAL EXPLORATION AND HARD ROCK RESOURCES

Regional and local geological setting has been discussed in Section 1.3. While not part of the scope of this pre-feasibility study, worthy of mention is the potential for future hard rock mining, possibly using the full-scale process plant envisaged for the tailings retreatment project.

3.1. Hard Rock Mining Potential Project

The Cleveland mine has had a long history, and operations were ceased in 1989 primarily due to collapse in the tin price rendering the mine uneconomical. As shown in Fig. Figure 10, the tin price has now recovered, along with the copper price, and there is further upside. Various reports⁴ show that a significant resource remains in the deposit, with



Measured and Indicated resources of 5.2 million tonnes @ 0.70% Sn and 0.31% Cu and Inferred resources of 1.3-million tonnes @ 0.72% Sn and 0.22% Cu at a 0.35% Sn cut-off. In addition, a large resource known as the Foley Zone located at the lower levels of the mine had been drilled and is reported to contain 3.8 million tonnes @ 0.28% WO₃ at a 0.2% cut-off. Data for these estimates has not yet been either sourced or verified.

4. CONCLUSIONS

Several key aspects of the Cleveland tailings re-treatment project have been identified as drivers towards the next stage of development:

- The project has a high IRR and will generate sales of around \$57-million per year with an operating cost estimated at \$17-million per year. The project therefore has an attractive return.
- The net present value (NPV) of the base case project is calculated as A\$74-million, yielding a payback period of 1.8 years.
- Project economics can be further increased by optimizing the plant or post-processing to increase concentrate grades and metal value recoveries.
- The Cleveland tailings as a tin-copper resource represent a significant value proposition for company-making development by BPPL, with the potential to generate revenue from sale of tin and copper concentrates.
- The incentive for environmental remediation by removal of significant acid-drainage generating material is an important driver; however, this shall not be to the detriment of the business plan.
- Micro-characterization techniques applied by BPPL have assisted in identifying the causes of value lockup in the tailings, and have suggested innovative but commercially-proven aspects to the process design.
- Liberation of fine mineral particles and separation by magnetic, gravity and flotation using modern centrifugal and pneumatic equipment in a tested flowsheet configuration is considered key to operational success.
- Operation of a onsite demonstration pilot plant at sufficient scale, duration and flexibility is warranted as an important part of the feasibility process.

5. PROPOSED DEVELOPMENT PLAN AND SCHEDULE

It is proposed that three key aspects of the development programme shall run concurrently over a period of one year from decision to proceed and mobilization capital injection:

1. Demonstration Pilot Plant.
2. Definitive Feasibility Study.
3. Permitting.



5.1. Demonstration Pilot Plant

The proposed flowsheet options for efficient recovery of value from Cleveland tailings include innovative solutions that would benefit significantly from evaluation and demonstration at pilot scale prior to commissioning of a full-scale operation. To achieve this BPPL will develop a demonstration pilot plant for key unit operations in the proposed process flowsheet, to be operated on-site. This will provide scale-up data to assist with feasibility analysis and process design with the added benefit that cashflow from the concentrate produced through this operation could be generated. This would assist in funding the feasibility study and permitting process.

The demonstration pilot plant would focus on basic evaluation of rougher gravity concentration for cassiterite, followed by pneumatic flotation for recovery of fine-grained cassiterite. Initially the demonstration pilot plant would not include magnetic separation although vendor equipment could be integrated to assess options once additional laboratory testwork had been completed.

The initial targets for the demonstration pilot plant would be:

1. Operate at 1/10th process scale with nominal throughput of 10 tph.
2. Target recovery of 65% of tin from tailings to a medium grade (35% Sn) concentrate. No effort would be made in the demonstration pilot plant to recover copper or silver, which will become penalty elements in the concentrate, although potential exists to carry out a short copper and sulfide flotation trial as part of the feasibility assessment process.
3. Operate over an evaluation period of one year. In this time a comprehensive program of sampling, assaying and data collection would be undertaken to provide inputs into the detailed process design and scale-up completed in the feasibility study.
4. Tailings from the demonstration pilot plant would be retained on existing tailings dams to maximize future economic benefit and to minimize environmental impacts.
5. An option to continue production using the demonstration pilot plant for an additional year during commissioning of the full-scale plant would be retained to generate additional cashflow.
6. Key pilot-scale equipment (grinding, gravity and support) would be depreciated through the life of the demonstration pilot plant to ensure availability as modular units for future projects.
7. Generate sufficient cashflow to provide substantial offset payments on demonstration pilot plant costs and overheads.

It is anticipated that the proposed demonstration-scale pilot facility will provide BPPL with an opportunity to generate cashflow early in the development process. Further benefits will be gained through generation of large pilot-scale production information on several flowsheet options from an on-site setting that will greatly assist in reducing risk in detailed process design. Importantly, testing of fine particle recovery equipment can in several cases only be achieved using plant equipment of this size. The general



requirements of working at the site and producing a saleable product will also minimize problems associated with commissioning of the full-scale operation.

5.2. Definitive Feasibility Study

Concurrent with operation of the demonstration pilot plant, a suitably qualified engineering company as well as other expert mining, process, logistical and environmental consultants shall be engaged to assist with development of a definitive feasibility study to be completed in the same one-year timeframe. Where possible, consultants with prior input and experience in considering the Cleveland tailings retreatment or historical operations shall be utilized as appropriate. Archived historical and more recent production, mineralogical and metallurgical reports shall be sourced and compiled for input into a physical and virtual data room for the duration of the study.

5.3. Permitting

To help ensure smooth progress in the other activities and to avoid delays in site works for an eventual full-scale plant, permitting activities shall be initiated once the go-ahead decision has been made and start-up mobilization funding has been allocated. A logistical consultant with knowledge of local area, regional issues and a relationship with state authorities shall be sourced and mobilized to commence detailed permitting and logistics activities. These shall include provision of services (power, water, access roads) for both demonstration pilot-scale and full-scale operations and assistance with sourcing and delivery of plant equipment and auxiliaries to site for the demonstration facilities.

In conjunction, a suitable environmental consultant group with knowledge of local ecological, water and environmental issues and a relationship with state authorities shall be sourced and mobilized to commence immediate preliminary environmental support for the piloting activities, as well as generation of an environmental impact assessment (EIA) for the full-scale operation, and in negotiation of the relevant permits.



REFERENCES

¹ Moony, N. (2007). *Review of Cleveland Tin Tailings Deposit in Tasmania for the Lynch Group*, ESKER Milling and Processing Pty Ltd, 3 October 2007.

² Gurung, S. (2001). *Tasmanian Acid Drainage Reconnaissance. 1. Acid drainage from abandoned mines in Tasmania*, Tasmanian Geological Survey Record 2001/05, Mineral Resources Tasmania, Rosny Park.

³ ANZECC (2000). *An Introduction to the Australian and New Zealand Guidelines for Fresh and Marine Water Quality*, Australian and New Zealand Environment and Conservation Council and Agriculture and Resource Management Council of Australia and New Zealand, October 2000.

⁴ Brewer, A.M. EL7/2005 “LUINA” Annual Report for the Period 30th June 2008 to 30th June 2009, Lynch Mining Pty Ltd, May 2009.

⁵ Goodall, W. Mineralogical Evaluation of Cleveland Tailings by QEMSCAN, *Bright Phase Report No. B107/1*, November 2009.

⁶ Stribley, D.J., Tapp, G.C. and Meik, S.S. (1984). Cleveland Tailings Retreatment Pilot Plant and Laboratory Flotation Results, *Aberfoyle Metallurgy Report No. AMS84/8*, 1 March 1984.

⁷ Brewer, A.M. EL7/2005 “LUINA” Annual Report for the Period 30th June 2007 to 30th June 2008 Lynch Mining Pty Ltd, May 2008.



Appendices

Appendix 1 – Resource

- Location, Profile and Plan
- Cleveland Historical Production Record
- Resource Estimation
- Environmental Acid Drainage Data

Appendix 2 – Process Plant

- Process Flow Diagram
- Conceptual Plant Layout
- Simplified Mass Balance

Appendix 3 – Metallurgical Testwork and Sampling

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- Previous Testwork List
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Appendix 4 – Financial Evaluation

- Financial Model
- Capital Cost Estimates
- Smelter Charges
- Sensitivity Analysis

Appendix 5 – Development Plan and Schedule

- Development Funding Proposal
- Schedule
- Budget



Appendix 1 – Resource

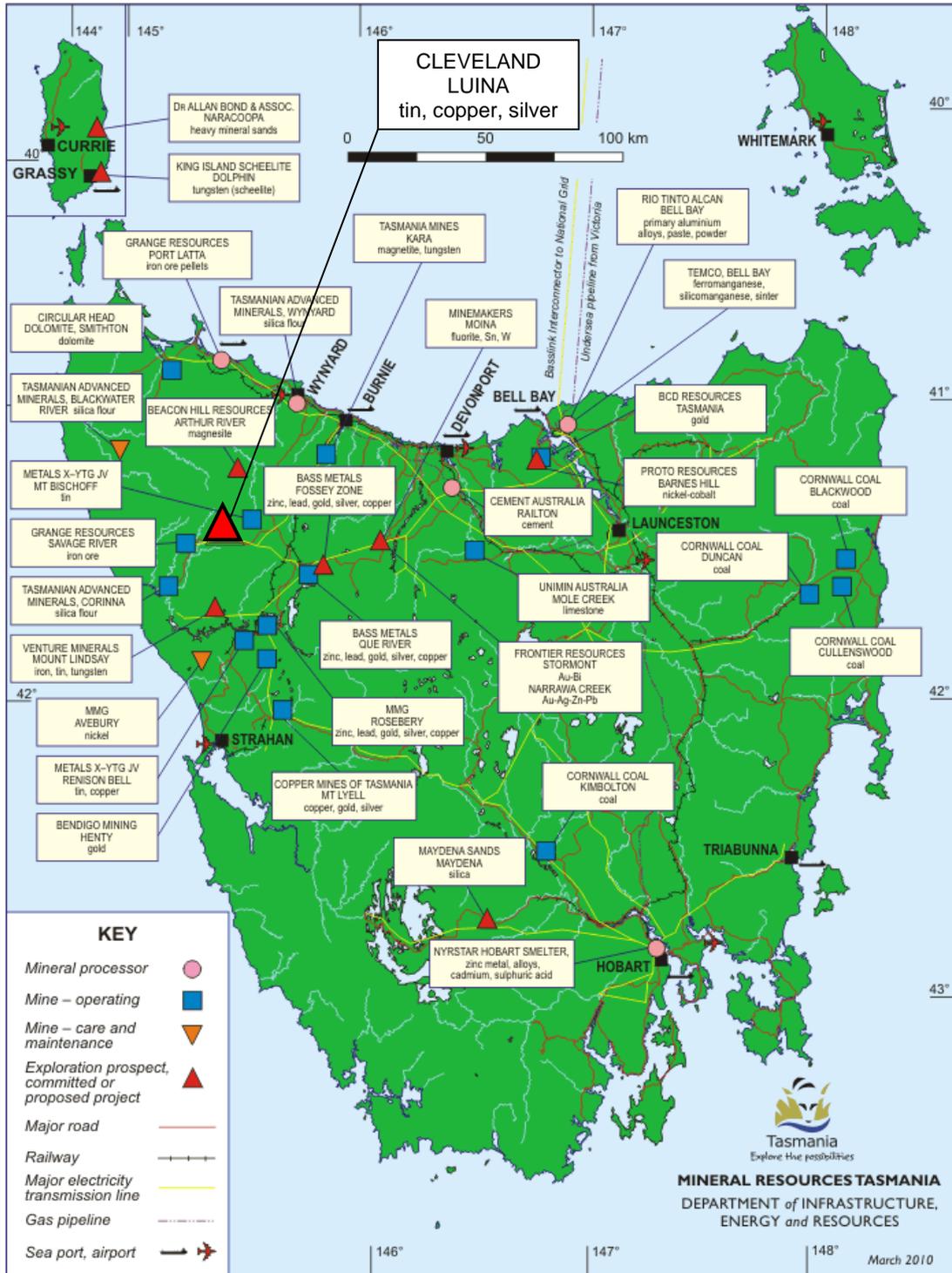
Location, Profile and Plan
Cleveland Historical Production Record
Resource Estimation
Environmental Acid Drainage Data



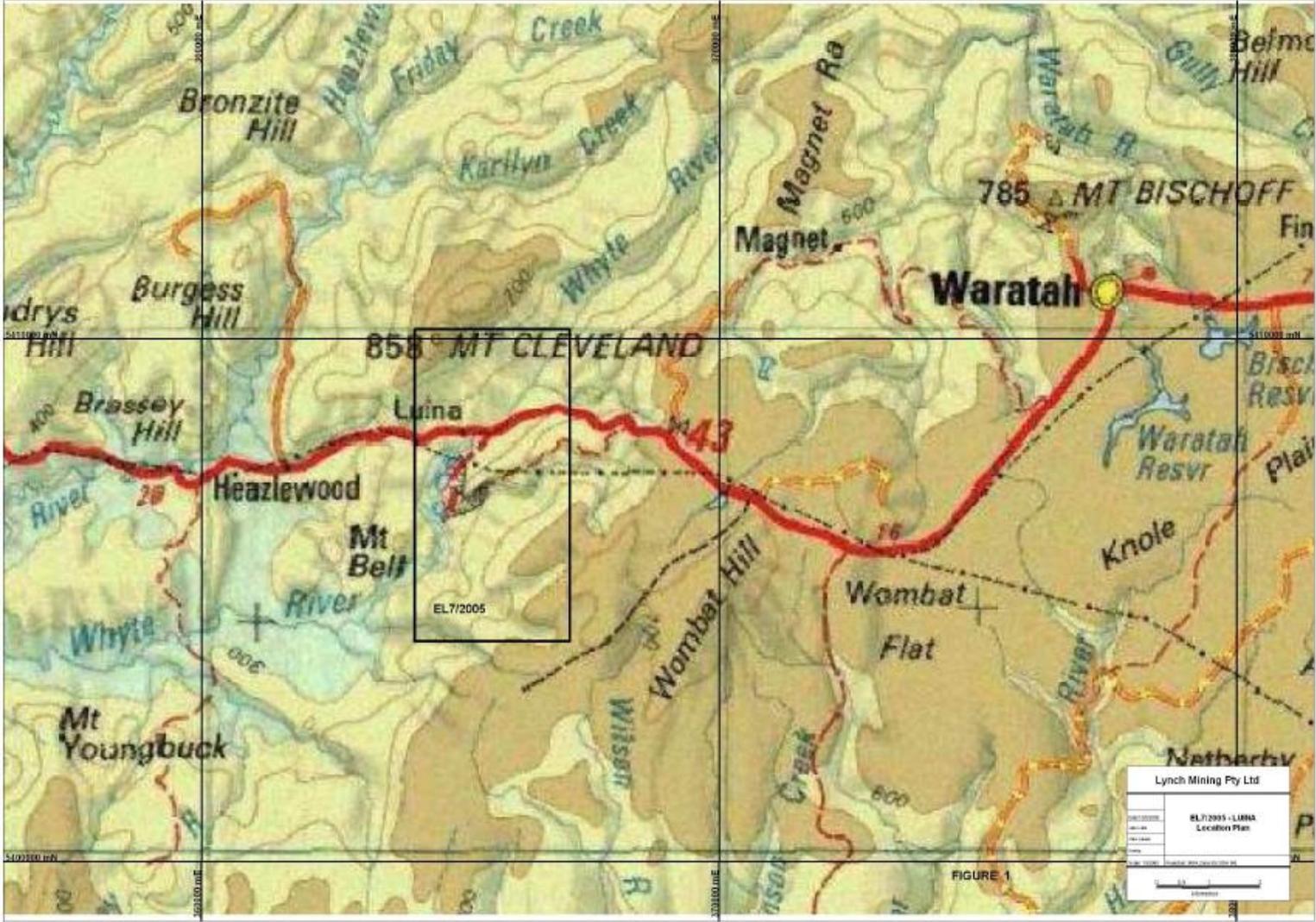
Location, Profile and Plan



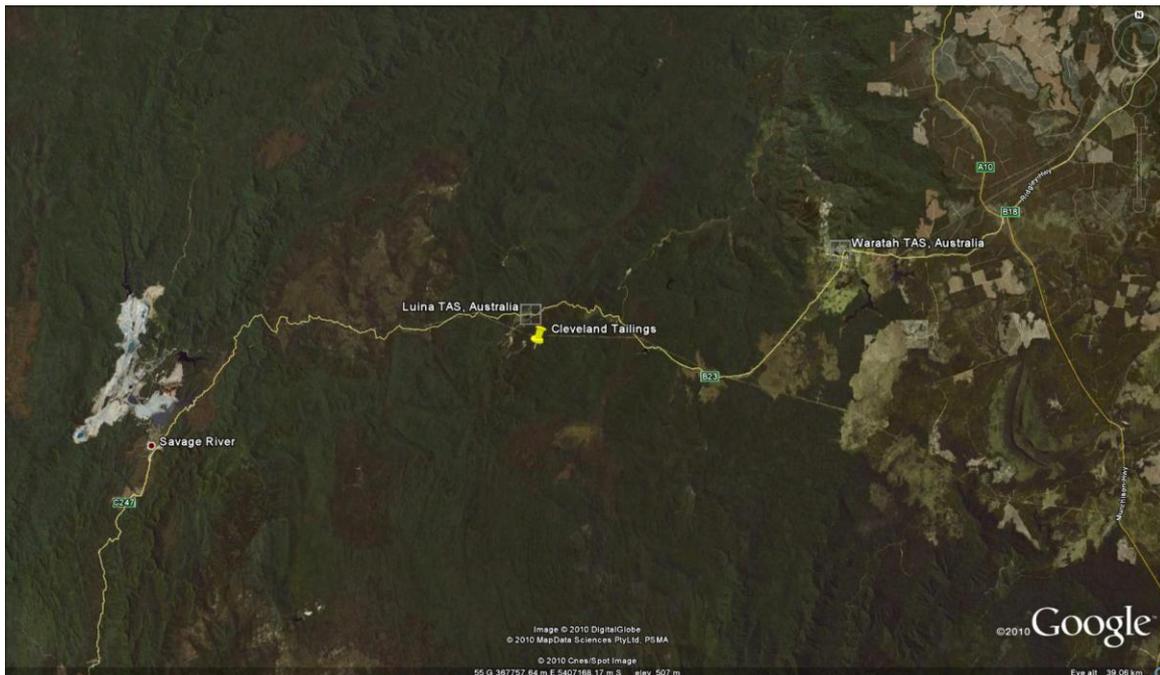
— Tasmania — Major Mining and Mineral Processing Operations and Proposed Projects



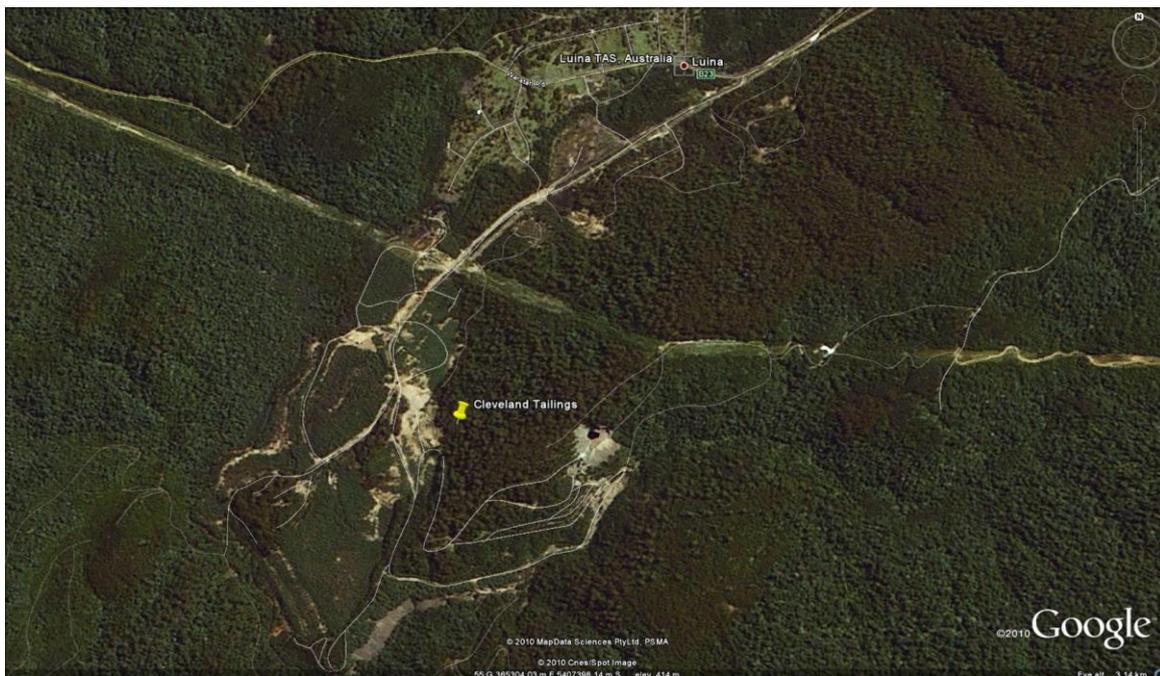
Map of Tasmania showing major towns and mines and location of the Cleveland Tailings Project (adapted from www.mrt.tas.gov.au)



Location of Cleveland tailings and Lynch Mining hard rock tenements



Satellite view showing location of Cleveland tailings



Satellite view of Cleveland tailings and old workings and Luina abandoned townsite

Detailed satellite view of Cleveland tailings TD2

Cleveland tailings dam profiles



Cleveland Historical Production Record



Cleveland Mill Records Metal Balance

Product	Tonnes		Tonnes		Dist'n	Dist'n	
	Tonnes	Sn%	Sn	Cu%	Cu	Sn%	Cu%
Copper Conc.	65,000	4.5	2,925	23.01	14,957	5.1	63.2
Tin Conc.	63,000	55	34,650	0.01	6	60.1	0.0
Sold HMS Float	2,450,000	0.25	6,125	0.08	1,960	10.6	8.3
HMS Float on site	750,000	0.25	1,875	0.08	600	3.3	2.5
No. 1 Tailings Dam	950,000	0.36	3,420	0.19	1,805	5.9	7.6
No. 2 Tailings Dam	2,700,000	0.32	8,640	0.16	4,320	15.0	18.3
Mined Ore	6,978,000	0.83	57,635	0.34	23,648	100	100



Resource Estimation



Cleveland Combined Tailings Resource Estimate

Product	Tonnes	Sn%	Tonnes		Dist'n		
			Sn	Cu%	Cu	Sn%	Cu%
HMS Float on site	750,000	0.25	1,875	0.08	600	13.5	8.9
No. 1 Tailings Dam	950,000	0.36	3,420	0.19	1,805	24.5	26.8
No. 2 Tailings Dam	2,700,000	0.32	8,640	0.16	4,320	62.0	64.2
Combined Tailings	4,400,000	0.32	13,935	0.15	6,725	100	100

Resource Grade Validation from Production Recovery Data

The mine treated about 7.0-million tonnes of ore between 1968 and 1986, the average feed grade was reportedly **Error! Bookmark not defined.** 0.65-0.95% Sn and 0.25-0.45% Cu. Recovery ranged from 50% to 73% and averaged about 59% for the life of mine. About 36,500 tonnes of tin metal and 15,000 of copper metal were sold. This means that some 19,500 tonnes of tin have been deposited as tailings, but some 6,000 tonnes of tin has been lost in sold-off HMS Float. The production metal balance calculation given earlier in this section was used to provide a Cleveland combined tailings resource estimate summarized in the table above. Validation estimation using the reported average feed grades, total tonnes processed and metal sold returned an estimated combined tailings grade of 0.38% Sn and 0.18% Cu, comparable with the grades reported above.

Resource Grade Validation from Previous Resource Estimate

A resource estimate was carried out as at 31st May 1983. TD1 was completed in 1971, while TD 2 was active. The combined tailings deposit was at that point estimated as 3.45-million tonnes containing 11,250 tonnes of tin. This yields an estimated grade of 0.33% Sn, consistent with current estimates.

Resource Estimation from Annualized Production Data

Aberfoyle production data is contained in archives held by a previous owner. Tailings resource estimation based on annualized metals production, feed tonnes and grade data has therefore been deferred until the definitive feasibility study stage.

Resource Estimation from Drilling

Drilling in 2008 yielded 1 m samples across single transects of TD1 and TD2 that are still available. Sub-samples were blended to produce 32 composite samples, the assay results of which are included in Appendix 3. The sticky nature of the material resulted in possible cross-contamination in both airhole and auger drilling as well as substantial water dilution so they were not assayed, apart from compositing. An assessment of the



likelihood of obtaining meaningful data from the available 1 m samples or more extensive drilling has therefore been deferred until the definitive feasibility study stage.

Environmental Acid Drainage Data



Whyte River Assays at Cleveland Run-off and Recommended Trigger Values

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SO4	As	Cd	Cu	Fe	Mn	Pb	Zn	TDM	Unit
680	0.008	0.125	0.31	2.75	13.3	0.29	0.054	16.837	mg/L
680,000	8	125	310	2,750	13,300	290	54	16,837	ug/L
									LoP
-	1	0.06	1	-	1200	1	2.4		99%
-	24	0.2	1.4	-	1900	3.4	8		95%
-	94	0.4	1.8	-	2500	5.6	15		90%
-	360	0.8	2.5	-	3600	9.4	31		80%

Exceeded trigger values in red type; TDM – Total Dissolved Metals; LOP – Level of protection (% of species); assay data **Error! Bookmark not defined.** from 10/03/2000 Whyte River @ Cleveland mine MD 364710 5407570 SRO



Appendix 2 – Process Plant

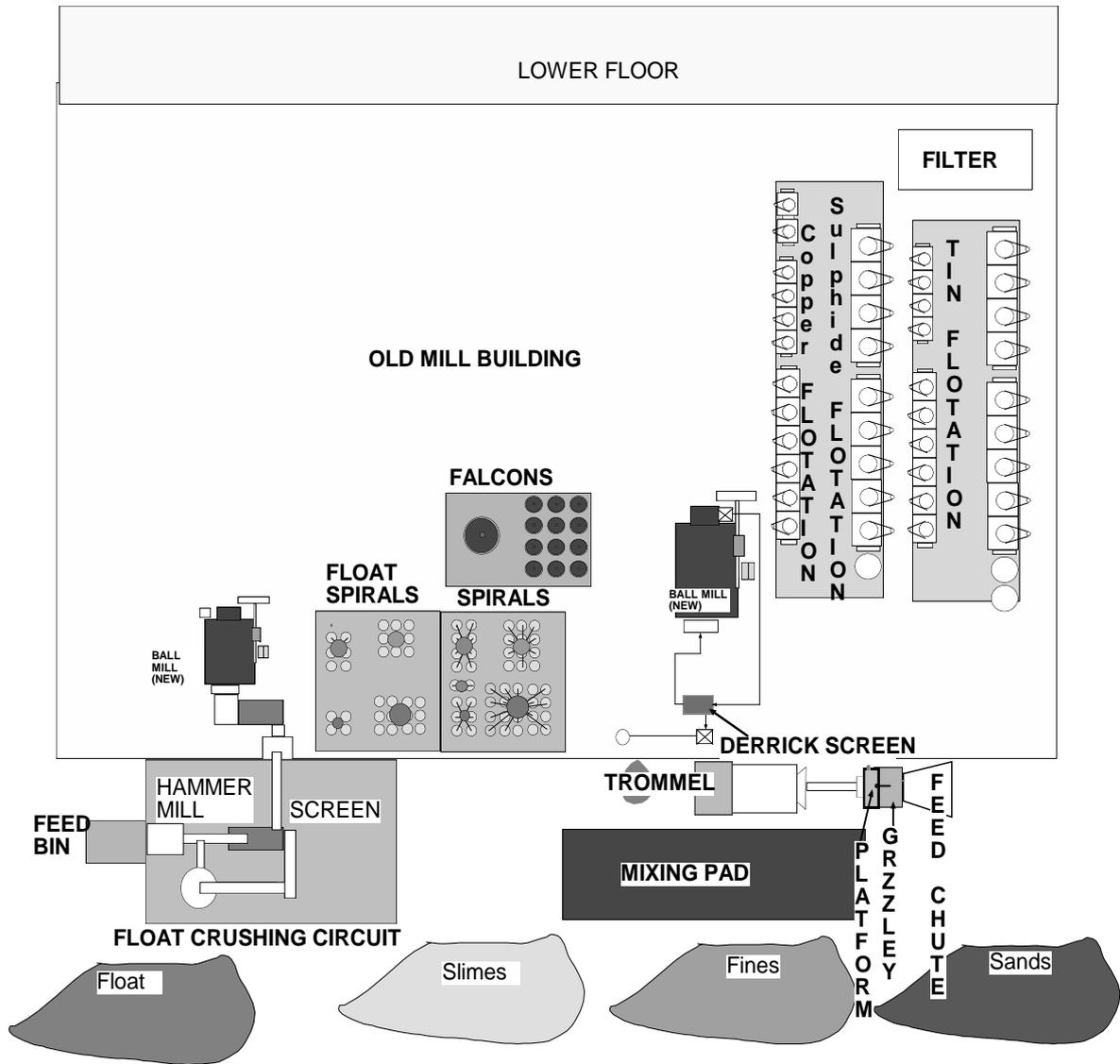
Process Flow Diagram
Conceptual Plant Layout
Simplified Mass Balance



Process Flow Diagram



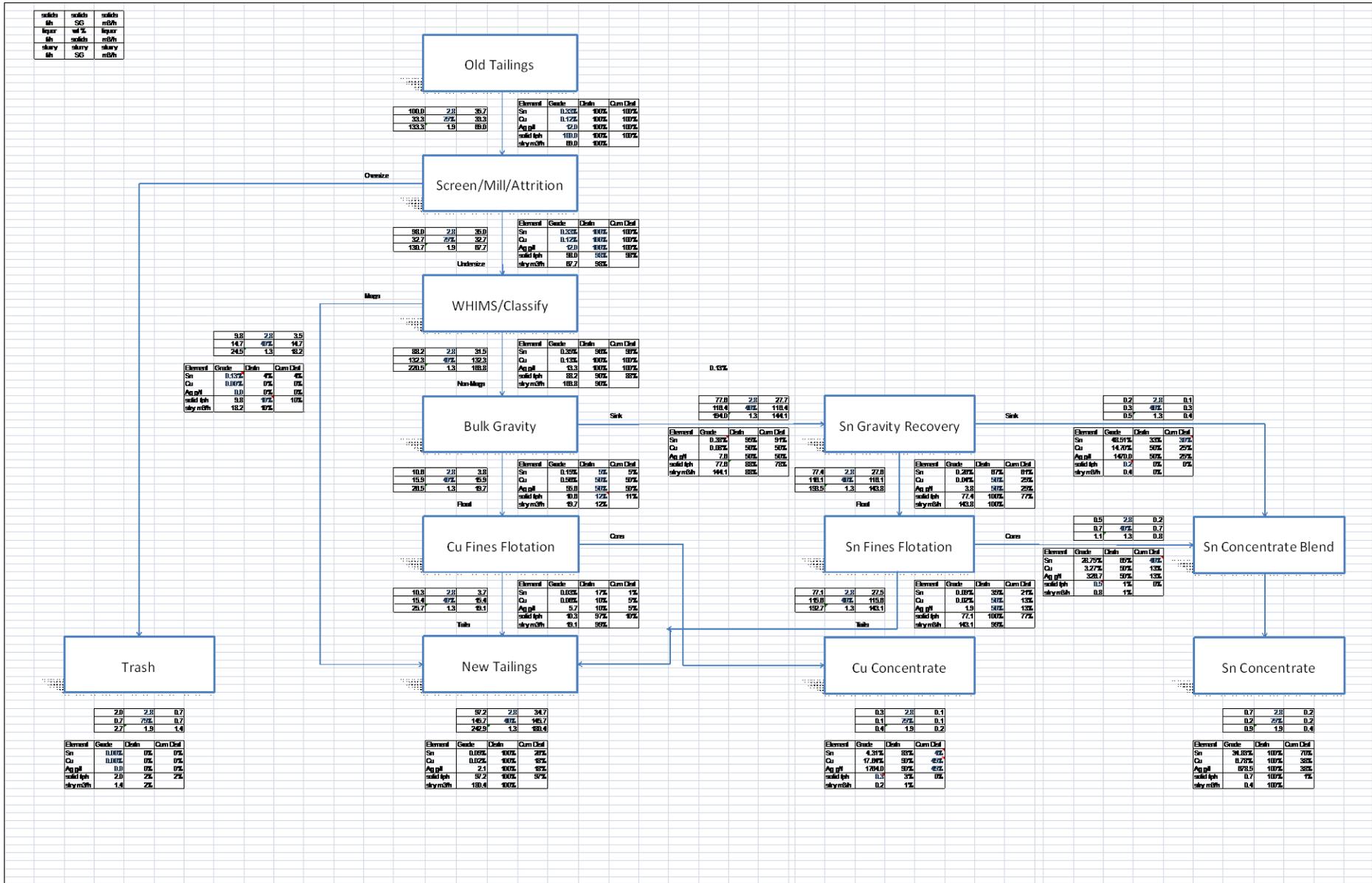
Conceptual Plant Layout



Indicative Cleveland tailings retreatment processing plant layout



Simplified Mass Balance



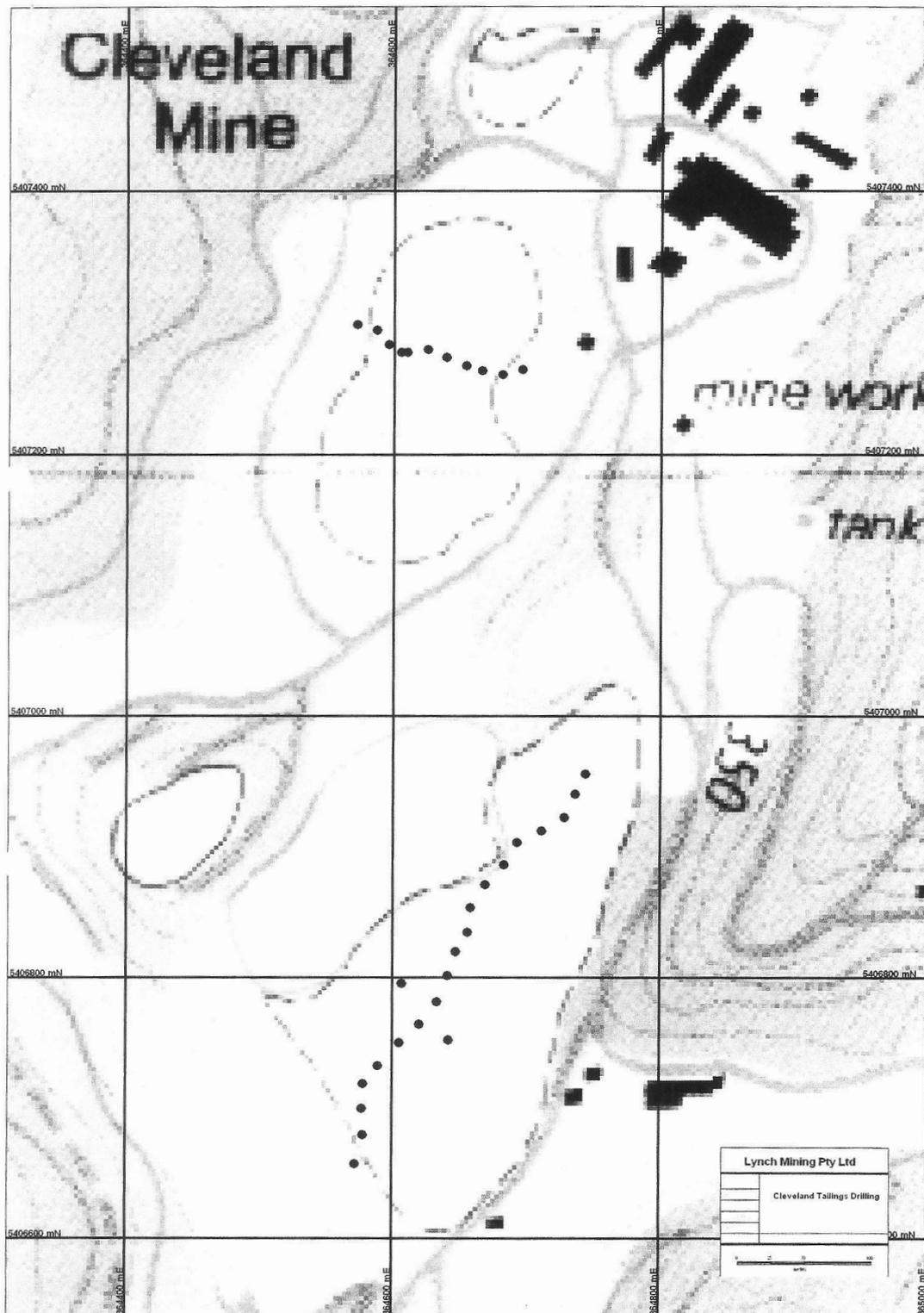


Appendix 3 – Metallurgical Testwork and Sampling

Sampling Map, Logs and Assays
Previous Testwork List
BPPL Testwork Summary



Sampling Map, Logs and Assays



Sampling transects taken in 2008



CLEVELAND DRILL COLLARS

Hole ID	Easting	Northing	Eoh Depth
CTD101	364572	5407299	19.5
CTD102	364587	5407295	19.0
CTD103	364596	5407284	19.0
CTD104	364610	5407278	14.0
CTD105	364625	5407280	13.0
CTD106	364639	5407274	12.0
CTD107	364654	5407268	10.0
CTD108	364666	5407264	6.5
CTD109	364681	5407261	5.0
CTD110	364696	5407265	4.0
CTD111	364605	5407278	16.1
CTD201	364572	5406658	35.0
CTD202	364578	5406680	35.0
CTD203	364577	5406700	31.5
CTD204	364578	5406719	33.0
CTD205	364589	5406733	33.0
CTD206	364605	5406751	30.0
CTD207	364620	5406765	26.0
CTD208	364607	5406796	29.0
CTD209	364642	5406753	28.0
CTD210	364633	5406782	26.0
CTD211	364641	5406802	25.0
CTD212	364647	5406820	25.0
CTD213	364656	5406835	24.0
CTD214	364658	5406854	17.0
CTD215	364669	5406872	14.0
CTD216	364683	5406887	10.0
CTD217	364693	5406904	9.0
CTD218	364711	5406913	11.0
CTD219	364728	5406923	13.0
CTD220	364736	5406941	9.0
CTD221	364744	5406956	10.0



Cleveland Tailings Composites Assay Data

Sample Description	Number	Pulv	Sn	As	Fe	MgO	Ca	Mn	Al	Cu	SiO2	S
			Fusion	%								
		y/n	XRF	CS2000								
CTD 101 1-2	383001	y	0.40	0.02	16.3	0.73	1.96	0.23	2.89	0.09	51.2	7.20
101 2-2	383002	y	0.32	0.04	19.1	1.54	1.81	0.24	2.83	0.16	44.4	8.92
102 1-2	383003	y	0.33	0.04	19.8	1.19	1.87	0.25	2.89	0.14	44.2	9.50
102 2-2	383004	y	0.30	0.01	16.4	0.82	2.03	0.24	2.96	0.07	50.6	7.17
103 1-2	383005	y	0.26	0.02	16.0	1.07	1.74	0.21	2.95	0.08	50.5	7.36
103 2-2	383006	y	0.29	0.03	16.2	2.44	1.83	0.24	3.38	0.12	45.3	6.29
104 1-2	383007	y	0.34	<0.01	15.3	1.56	2.15	0.25	2.96	0.18	37.7	7.08
104 2-2	383008	y	0.41	0.01	17.4	1.23	2.35	0.30	3.45	0.12	44.4	6.69
105 1-2	383009	y	0.39	0.04	16.7	1.64	2.33	0.32	3.63	0.14	46.1	5.81
105 2-2	383010	y	0.43	0.03	16.1	1.62	2.38	0.33	3.87	0.12	45.1	4.80
106 1-2	383011	y	0.41	<0.01	15.5	1.52	2.99	0.36	4.06	0.11	45.7	3.86
106 2-2	383012	y	0.46	0.01	15.3	1.53	2.51	0.37	4.22	0.11	46.3	3.36
107 1-2	383013	y	0.30	<0.01	12.4	4.33	2.73	0.29	4.15	0.08	46.3	1.89
107 2-2	383014	y	0.44	<0.01	14.2	2.23	2.50	0.37	4.47	0.11	46.6	2.33
108 1-1	383015	y	0.42	<0.01	13.5	2.29	3.01	0.37	4.39	0.10	46.6	1.70
109 1-1	383016	y	0.42	<0.01	14.4	1.19	3.15	0.40	4.36	0.11	48.1	1.85
110 1-1	383017	y	0.56	0.01	11.6	1.77	2.50	0.28	4.42	0.10	54.0	2.39
111 1-2	383018	y	0.43	0.01	19.0	1.28	2.46	0.29	3.10	0.26	42.0	7.80
111 2-2	383019	y	0.40	0.02	19.7	0.90	2.17	0.28	3.06	0.11	43.6	9.05
CTD 201 1-3	383020	y	0.28	<0.01	16.9	0.77	2.67	0.29	3.08	0.06	49.0	5.62
201 2-3	383021	y	0.31	<0.01	18.7	0.88	2.24	0.30	2.69	0.10	46.5	8.53
201 3-3	383022	y	0.30	0.02	19.9	0.79	2.51	0.30	2.89	0.10	43.6	8.62
202 1-4	383023	y	0.32	<0.01	18.9	0.78	2.60	0.31	3.01	0.08	45.7	7.65
202 2-4	383024	y	0.30	<0.01	18.9	0.71	2.96	0.30	2.81	0.09	43.4	7.81
202 3-4	383025	y	0.28	0.01	22.0	0.69	2.23	0.29	2.54	0.12	41.3	11.6
202 4-4	383026	y	0.34	0.03	18.9	0.91	3.00	0.36	3.56	0.12	43.2	6.67
203 1-2	383027	y	0.31	<0.01	18.5	0.78	2.73	0.33	3.08	0.09	44.3	7.46



Cleveland Tailings Composites Assay Data

Sample Description	Number	Pulv	Sn	As	Fe	MgO	Ca	Mn	Al	Cu	SiO ₂	S
			Fusion	%								
		y/n	XRF	CS2000								
203 2-2	383028	y	0.29	<0.01	21.5	0.69	2.39	0.31	2.64	0.14	41.9	10.6
204 1-4	383029	y	0.28	<0.01	18.1	0.84	2.75	0.33	3.24	0.08	47.0	6.73
204 2-4	383030	y	0.29	<0.01	21.0	0.70	3.00	0.32	2.81	0.14	40.9	9.79
204 3-4	383031	y	0.34	0.01	18.4	0.81	3.01	0.35	3.18	0.11	44.5	7.07
204 4-4	383032	y	0.35	<0.01	16.8	0.94	3.09	0.39	3.66	0.11	46.1	4.61
205 1-4	383033	y	0.28	<0.01	18.6	0.84	2.88	0.34	3.22	0.08	45.7	6.99
205 2-4	383034	y	0.33	<0.01	19.1	0.81	3.39	0.38	3.19	0.16	410.5	7.64
205 3-4	383035	y	0.35	0.01	17.5	0.92	2.98	0.38	3.51	0.12	46.6	5.37
205 4-4	383036	y	0.36	<0.01	16.9	0.96	3.26	0.40	3.68	0.09	46.2	4.30
206 1-4	383037	y	0.30	<0.01	18.8	0.82	2.69	0.34	3.17	0.10	45.8	7.38
206 2-4	383038	y	0.29	<0.01	16.6	0.82	3.61	0.39	3.38	0.07	46.0	4.67
206 3-4	383039	y	0.38	<0.01	16.3	0.91	3.52	0.40	3.71	0.11	47.5	4.25
206 4-4	383040	y	0.37	<0.01	16.5	0.98	3.35	0.40	3.83	0.09	46.3	3.84
207 1-3	383041	y	0.30	<0.01	18.4	0.82	2.88	0.35	3.22	0.09	46.4	6.77
207 2-3	383042	y	0.30	<0.01	15.5	0.86	3.74	0.37	3.63	0.08	48.4	3.90
207 3-3	383043	y	0.37	<0.01	16.8	0.91	3.61	0.41	3.69	0.10	45.7	4.41
208 1-4	383044	y	0.32	<0.01	17.1	0.98	2.73	0.36	3.59	0.09	47.7	5.16
208 2-4	383045	y	0.30	<0.01	16.5	0.87	3.75	0.38	3.52	0.08	44.9	4.38
208 3-4	383046	y										4.79
208 4-4	383047	y	0.39	0.02	16.1	1.04	3.54	0.43	4.06	0.13	46.7	3.09
209 1-4	383048	y	0.31	0.02	19.3	0.81	2.6	0.35	3.06	0.08	45.8	7.66
209 2-4	383049	y	0.27	0.01	17.4	0.87	3.67	0.40	3.55	0.07	45.2	4.92
209 3-4	383050	y	0.38	<0.01	15.2	0.97	3.94	0.38	4.09	0.11	46.8	2.99
209 4-4	383051	y	0.43	0.01	15.7	1.02	3.71	0.45	4.03	0.12	46.9	2.88
CTD 210 1-3	383052	y	0.27	<0.01	18.4	0.90	2.96	0.35	3.36	0.08	45.8	6.62
210 2-3	383053	y	0.32	<0.01	16.6	0.87	3.83	0.39	3.57	0.09	46.0	4.47
210 3-3	383054	y	0.40	0.00	16.2	0.99	3.85	0.43	3.82	0.11	46.6	3.62



Cleveland Tailings Composites Assay Data

Sample Description	Number	Pulv	Sn Fusion	As Fusion	Fe Fusion	MgO Fusion	Ca Fusion	Mn Fusion	Al Fusion	Cu Fusion	SiO2 %	S %
		y/n	XRF	XRF	XRF	XRF	XRF	XRF	XRF	XRF	XRF	CS2000
211 1-3	383055	y	0.29	<0.01	18.0	0.92	2.95	0.37	3.50	0.08	47.0	5.65
211 2-3	383056	y	0.32	<0.01	16.8	0.84	3.89	0.42	3.54	0.08	44.7	4.28
211 3-3	383057	y	0.13	<0.01	15.2	0.99	4.13	0.44	4.00	0.11	45.9	2.58
212 1-3	383058	y	0.28	0.00	16.9	0.98	2.97	0.36	3.60	0.09	47.3	5.06
212 2-3	383059	y	0.31	<0.01	17.7	0.85	3.81	0.44	3.45	0.10	43.0	5.04
212 3-3	383060	y	0.45	0.00	14.9	1.05	4.05	0.47	4.33	0.12	44.9	1.85
213 1-3	383061	y	0.31	0.01	17.3	0.99	3.02	0.38	3.63	0.09	46.3	5.00
213 2-3	383062	y	0.30	<0.01	18.2	0.87	3.86	0.46	3.54	0.11	42.0	5.41
213 3-3	383063	y	0.46	<0.01	14.9	10.60	4.09	0.46	4.29	0.14	44.8	1.76
214 1-2	383064	y	0.25	<0.01	16.8	0.95	3.41	0.36	3.77	0.10	46.6	4.82
214 2-2	383065	y	0.34	0.01	18.5	0.85	3.86	0.49	3.47	0.10	41.2	5.46
215 1-2	383066	y	0.30	<0.01	16.7	1.01	2.96	0.36	3.91	0.12	47.5	4.39
215 2-2	383067	y	0.26	0.00	17.9	0.82	3.68	0.49	3.30	0.11	40.1	5.38
216 1-1	383068	y	0.28	<0.01	16.5	1.06	3.06	0.37	3.99	0.14	46.9	4.38
217 1-1	383069	y	0.27	<0.01	17.1	1.00	2.91	0.34	3.75	0.11	47.2	4.90
218 1-1	383070	y	0.30	<0.01	16.6	1.00	3.06	0.36	3.84	0.13	46.8	4.23
219 1-2	383071	y	0.27	<0.01	18.1	0.89	2.78	0.31	3.36	0.11	45.2	7.14
219 2-2	383072	y	0.27	<0.01	16.7	1.09	3.30	0.44	4.23	0.14	46.1	3.39
220 1-1	383073	y	0.29	0.00	18.2	0.92	2.78	0.33	3.51	0.16	45.3	6.95
221 1-1	383074	y	0.36	0.00	19.4	0.87	2.37	0.32	3.28	0.20	44.0	8.68



Previous Testwork List



Previous Testwork List

Date	Company	Testwork	Samples	Targets
1961	Aberfoyle	Cassiterite Flotation	ROM tailings	1. Implemented cassiterite flotation in operating flowsheet
1978-79	Aberfoyle	Sn Fumer	Unknown	1. Fuming technically feasible.
1984	Aberfoyle	Pilot plant and laboratory flotation	TD1 – Beach, Wall and interior TD2 – Beach and Wall	1. Conventional flowsheet – gravity, grinding, classification, sulphide/cassiterite flotation and magnetic separation 2. Preconcentrate + Matte fuming
1991	Lachlan Resources	Kelsey Jig amenability	Unrepresentative (0.55% Sn)	1. Amenability to gravity concentration
2008	Lynch Mining	UF Falcon – Gravity + Sulphide flotation + superpanning	Composite samples from drilling program	1. Amenability to recovery of fine grained cassiterite by UF Falcon
2008	Lynch Mining	- Regrind (P80 38um) - LIMS, WHIMS magnetic separation; - Falcon split coarse and fine separation - Cyclone deslime	Composite samples from drilling program	1. Removal of Fe bearing oxide minerals detrimental to flotation. 2. Evaluation of alternative flotation reagents.
2009	Bright Phase	QEMSCAN mineralogy	Composite samples from drilling program	3. Department and grain size of Sn and Cu bearing minerals



BPPL Testwork Summary

Mineralogical Evaluation of Cleveland Tailings by QEMSCAN

Date: January 2010

Prepared by: Dr Will Goodall

Sample: Head Sample CTD 201-221

Summary of Findings

Examination of Cleveland Tailings (CTD 201-220) sourced from the Cleveland project, Tasmania, Australia (EL7/2005) was completed using QEMSCAN by Bright Phase Pty Ltd. The primary focus of this program was to define the deportment of Sn and Cu, along with the root cause for poor recovery by conventional flotation and gravity separation.

The key outcomes of this study included:

- All tin (Sn) occurred in Cassiterite (SnO₂), which is known to be a floatable mineral.
- All copper (Cu) occurred with floatable copper iron sulphide minerals.
- Average grain size of Sn minerals was 5-20 µm, which is below practical recovery size for conventional flotation or gravity processes.
- Liberation of Sn and Cu minerals was poor in coarse particles. However, coarse particles consisted of agglomerated fine particles and should be easily broken down, promoting liberation.
- Major contaminant minerals were pyrite and pyrrhotite, which must be depressed in flotation processes.

Fine grain size and poor liberation caused by agglomeration of value minerals, adequately described why conventional gravity and flotation methods have been unsuccessful in obtaining saleable tin and copper concentrate.

To overcome these difficulties a number of solutions may be examined, which must address the following key issues:

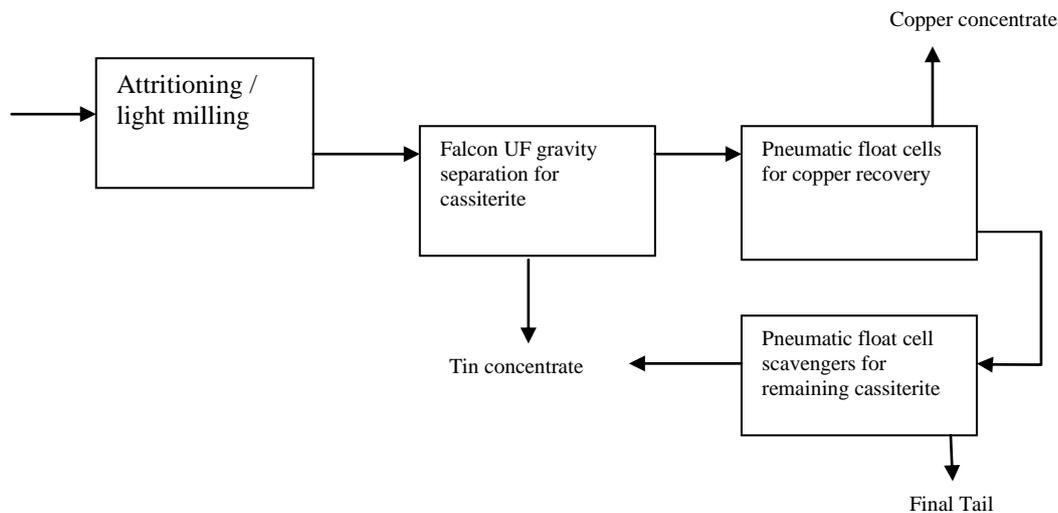
- Liberation of copper and tin minerals must be promoted by deagglomeration of coarse particles.
- Any solution must have the capability to recover fine particles, typically in the range of 5 – 20 µm.



Bright Phase Pty Ltd proposes that the key processing criteria for the Cleveland tailings may be addressed through use of the following technologies:

- Attritioning of agglomerated particles through use of high shear devices or light milling.
- Ultra-fine gravity separation devices, such as the new Falcon UF series centrifugal concentrators.
- Fine particle flotation systems, such as pneumatic float cells.

A potential block diagram flowsheet that may be suggested from the findings of this study may be seen below:



Economic recovery of Sn and Cu should be achievable from Cleveland Tailings if additional liberation can be achieved and solutions for fine particle flotation or gravity separation can be implemented

Next Steps:

1. Undertake amenability testing of Cleveland tailings with vendors of pneumatic float cells and ultra-fine gravity recovery solutions.
 - a. BPPL has contacts with a number of vendors in this area and will be able to facilitate amenability testing in a fast and efficient manner.
 - b. Consideration of the degree of deagglomeration required and amenable methods should be considered at this stage.
2. With positive results proceed to pilot scale testing of amenable flowsheet.
 - a. Pilot scale plants are available for most technologies and may be set up directly on-site.
3. Using amenability and pilot scale testing results undertake engineering design and construction of process plant.



Appendix 4 – Financial Evaluation

Financial Model
Capital Cost Estimates
Smelter Charges
Sensitivity Analysis



Financial Model

Financial Summary: Cleveland Tailings Retreatment - Including Tin, Copper and Silver recovery - **DRAFT**

Financials	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	
Production	Demo Pilot Plant	Demo Pilot Plant						
Production (t)	18,000	18,000	400,000	1,000,000	1,000,000	1,000,000	1,000,000	
Tin Grade	0.32%	0.32%	0.32%	0.32%	0.32%	0.32%	0.32%	
Recovery (%)	64.6%	64.6%	64.6%	64.6%	64.6%	64.6%	64.6%	
Tin Produced (t)	37	37	814	2,035	2,035	2,035	2,035	
Copper Grade		0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	
Recovery %		0%	65.5%	65.5%	65.5%	65.5%	65.5%	
Copper produced		-	393	983	983	983	983	
Silver Grade		11.8	11.8	11.8	11.8	11.8	11.8	
Recovery %		0%	64.6%	64.6%	64.6%	64.6%	64.6%	
Silver produced (g)		-	3,049,120	7,622,800	7,622,800	7,622,800	7,622,800	
Concentrate	% Sn tonnes	35% 104.65	35% 104.65	35% 2,325.60	35% 5,814.00	35% 5,814.00	35% 5,814.00	35% 5,814.00
Currency	USD	USD (smelter)	AUD	Currency Conversion			0.85	
Tin Price (\$/t)	18675	18675	21,970.59					
Copper (\$/t)	6484.5	6484.5	7,629					
Silver (\$/oz)	16.542	16.542	19.46	Kg to ounces conversion			31.1	
Capital cost (Pilot Plant)	\$	587,500		NPV Discount Rate			0.15	
Capital cost (Processing)	\$	29,056,871						
Cashflows								
REVENUE								
Tin Revenue	804,743	804,743	17,883,180	44,707,950	44,707,950	44,707,950	44,707,950	
Copper revenue	-	-	2,998,128	7,495,319	7,495,319	7,495,319	7,495,319	
Silver Revenue	-	-	1,908,021	4,770,053	4,770,053	4,770,053	4,770,053	
TOTAL REVENUE	804,743	804,743	22,789,329	56,973,322	56,973,322	56,973,322	56,973,322	
COSTS								
Mining incl. rehab	55,800	55,800	1,240,000	3,100,000	3,100,000	3,100,000	3,100,000	
\$/t	3.1	3.1	3.1	3.1	3.1	3.1	3.1	
Processing (milling)	18,000	18,000	400,000	1,000,000	1,000,000	1,000,000	1,000,000	
\$/t	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Processing (gravity)	27,000	27,000	200,000	500,000	500,000	500,000	500,000	
\$/t	1.5	1.5	0.5	0.5	0.5	0.5	0.5	
Processing (Mag Sep)	-	-	200,000	500,000	500,000	500,000	500,000	
\$/t	-	-	0.50	0.50	0.50	0.50	0.50	
Processing (Flotation)	27,000	27,000	400,000	1,000,000	1,000,000	1,000,000	1,000,000	
\$/t	1.50	1.50	1.00	1.00	1.00	1.00	1.00	
Concentrate transport	14,400	14,400	320,000	800,000	800,000	800,000	800,000	
\$/t	0.80	0.80	0.80	0.80	0.80	0.80	0.80	
Smelter Charge	169,429	169,429	1,788,318	4,470,795	4,470,795	4,470,795	4,470,795	
\$/t	9.41	9.41	4.47	4.47	4.47	4.47	4.47	
Administration	44,640	44,640	992,000	2,480,000	2,480,000	2,480,000	2,480,000	
\$/t	2.48	2.48	2.48	2.48	2.48	2.48	2.48	
Labour	63,000	63,000	1,400,000	3,500,000	3,500,000	3,500,000	3,500,000	
\$/t	3.5	3.5	3.5	3.5	3.5	3.5	3.5	
TOTAL COSTS	419,269	419,269	6,940,318	17,350,795	17,350,795	17,350,795	17,350,795	
\$/t	23.29	23.29	17.35	17.35	17.35	17.35	17.35	
\$/t tin concentrate	4,006	4,006	2,984	2,984	2,984	2,984	2,984	
OPERATING MARGIN	385,475	385,475	15,849,011	39,622,527	39,622,527	39,622,527	39,622,527	
%	92%	92%	228%	228%	228%	228%	228%	
Capital Costs	587,500		\$ 29,056,871					
Net Cash Flow	-	587,500	385,475	14,528,436	1,320,575	39,622,527	39,622,527	39,622,527
\$/t Sn	-	587,500	385,475	396,646.18	1,622.41	19,471.49	19,471.49	19,471.49
NPV	73,901,197				NPV (Pilot)	39,169		
IRR	123%				IRR (Pilot)	20%		
Payback	0.7 years				PayBack (Pilot)	1.52 years		

Recovery:	Sn, Cu and Ag recovery defined from feasibility review performed for Lynch Mining by Esker Pty Ltd, 2007
Commodities	Commodity prices determined from LME, August 2010 with a 10% discount to allow for price fluctuations
Concentrate transport:	Defined from feasibility review performed for Lynch Mining by Esker Pty Ltd, 2007. Estimation of \$0.8 per tonne processed based on experience of competent Sn metallurgist
Smelter Charge	Determined based on impurity penalties provided by Sn smelter 17/08/2010 for offtake agreement
Administration	Includes corporate, camp, travel, vehicles, administration staff, concentrate marketing etc.
Capital cost	Based on equipment value (adjusted for CPI) defined in Esker feasibility review updated to 1 Mtpa tailings. EPCM, TSF and ancillary capital expenses defined from similar scale projects evaluated by BPPL
Demo Pilot Plant	To be operated as part of feasibility analysis to provide scale-up and site-specific processing information on bulk samples of Cleveland tailings

Sensitivity calculations

Operating Cost sensitivity	100%					
Capital cost sensitivity	100%					
Tin recovery sensitivity	100%					
Sn head grade	90%					
Commodity Price - Sn	100%	-	13,207,860	39,622,527	39,622,527	39,622,527
Commodity Price - Cu	100%					
Commodity Price - Ag	100%					



Capital Cost Estimates

Summary

Processing operation			
Summary	2008	2010	
Buildings	806,150	855,949	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Front End Preparation	1,172,988	1,245,448	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Gravity Plant costs	4,700,000	4,990,338	Spirals and centrifugal concentrator (Canada) - Esker 2007 feasibility review for Cleveland
Flotation Plant Costs	2,000,000	2,123,548	Flotation circuit sourced from China - Esker 2007 feasibility review for Cleveland
Concentrate handling equipment	650,000	690,153	Thickener and filter - Esker 2007 feasibility review for Cleveland
Electrical and Instrumentation	2,000,000	2,123,548	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Misc - including reagent storage, plant piping, civils, miscel	2,553,171	2,710,891	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Tailings Storage Facility	1,804,926	1,916,424	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Construction and Installation	3,108,330	3,300,344	Calculated from BPPL scoping study for 5Mt tailings retreatment project
Freight	-	-	Equipment cost estimates based on installed capital
EPCM	2,255,468	2,394,797	Ratio 12%
Owner's Costs	1,650,000	1,751,927	Corporate overheads and project staff
Subtotal	21,051,033	22,351,439	
Contingency	30%	6,315,310	6,705,432
Total	27,366,343	29,056,871	

Demonstration Pilot Plant			
Item	Capital		
Pebble Mill	\$ 50,000		Estimated
Trommel	\$ 10,000		Estimated
Gravity Concentrator (centrifugal)	\$ 50,000		Estimated
Pilot flotation cells (Cu pre-float)	\$ 30,000		Estimated
Pneumatic flotation pilot cells	Lease		Assumed
Pumps + pipework	\$ 20,000		Estimated
Equipment transport	\$ 20,000		Estimated
Sub-Total	\$ 180,000		Estimated
Office/Admin Building	\$ 115,000		Estimated
Laboratory + equipment	\$ 172,500		Estimated
Reagent storage	\$ 40,000		Estimated
Vehicles (2 x utes)	\$ 60,000		Estimated
Earthworks	\$ 20,000		Estimated
Sub-Total	\$ 407,500		
Capital requirement	\$ 587,500		



Smelter Charges

Smelter Charges and Concentrate

Concentrate

<i>Pilot Plant</i>					
Sn	Fe	Cu	Ag	S	Si
35.0	20.0	5.0		10.0	20
Charge \$ 850.00 Per tonne concentrate					
<i>Process</i>					
Sn	Fe	Cu	Ag	S	Si
35.0	20.0	0.5		10.0	20
Charge \$ 175.00 Per tonne concentrate					

Source: Libra Minerals, 17/08/2010

Smelter Charge	USD	
S	<0.5%	0 Per tonne ore
	0.5% to 0.99%	2 Per tonne ore
	1%-1.49%	4 Per tonne ore
	1.5%-1.99%	6 Per tonne ore
	2%-2.49%	8 Per tonne ore
	>2.5%	10 Per tonne ore
As	1%	80 Per tonne ore
Pb,Zn,Ag,Ni,Co	1%	80 Per tonne ore
Bi & Cu	1%	150 Per tonne ore
Sb	1%	240
Fe, Mn & WO3	<1%	Free
	>1%	10% of gross Sn assay
F and Cl	1%	60 Per tonne ore



Sensitivity Analysis

Sensitivity Analysis Summary: Cleveland Tailings Retreatment - Including Tin, Copper and Silver recovery -
DRAFT

Key Sensitivities (pre-tax)				
Base Case: Tin recoveries: 64.6%; Copper recoveries: 65.5%; Silver recoveries: 64.6%; Capex:\$25M; Start-up: tbd; mine life 5 years; USD/AUD rate: 0.85				
		IRR (%)	NPV (\$M)	Payback (yrs)
<i>Base Case</i>	<i>Change</i>	<i>102</i>	<i>54</i>	<i>1.6</i>
Commodity price change	+10%	113	68	1.5
	-10%	90	40	1.9
Operating Costs	+10%	98	50	1.7
	-10%	105	59	1.6
Capex	+10%	93	49	1.8
	-10%	112	60	1.5
Tin Recovery	+10%	111	65	1.5
	-10%	92	43	1.8
Tin Head Grade	+10%	111	65	1.5
	-10%	92	43	1.8



Appendix 5 – Development Plan and Schedule

Development Funding Proposal
Schedule
Budget



Development Funding Proposal



Development Funding

Bright Phase is an environmentally focused tailings retreatment company seeking to generate value from waste generated in historic mining operations. Through the use of innovative characterization techniques Bright Phase seeks to define focused processing options for complex tailings material. Targeted processing options will be developed to maximize value recovery from historic tailings material and minimize legacy environmental impacts of mining operations.

Mission Statement

To create value for our shareholders through the successful application of new technologies and sustainable mining practices to mine tailings, wastes and slags.

Cleveland Project, Tasmania

The Cleveland tailings project is located approximately 150 km SSW of Burnie in Tasmania, situated between the Savage River township and the historic and abandoned township of Waratah and Mt. Bischoff.

The tailings were generated by Aberfoyle Ltd during their underground mining of the Cleveland deposit. Mining and minerals processing was ceased in 1986.

The Cleveland tailings and remaining hard rock resource is owned by Lynch Mining (led by Brisbane-based mining identity John Lynch).

Project Summary:

Tonnage: 4,400,000 tonnes

Grade: ~ 0.32 Sn, 0.15% Cu and 12 g/t Ag).

In addition to the tailings, there is a hardrock resource of 5 Mt of Sn and Cu followed by another 5 Mt of low-grade Mo and W.

Results from initial characterization: Prior to BPPL getting involved, Lynch Mining had completed gravity testwork on the tailings that failed to generate the recoveries expected. BPPL reached an agreement in principle and proceeded with completing a QEMSCAN analysis of a composite tailings sample. The result of this study was the identification of Sn and Cu sitting in agglomerates of liberated, floatable minerals at small grain size (5-20 microns). This data has assisted BPPL to design a conceptual flowsheet to address the process requirements, resulting in the recommendation of attritioning with subsequent fine value minerals recovery by flotation and gravity techniques.



Ownership/Legal status: BPPL has negotiated development rights for the Cleveland tailings material with the option to extend ownership to hard rock opportunities on the Cleveland leases.

Why Cleveland?: BPPL view Cleveland as the “company making” project. The tonnage is of economic size and the mineral grades are very good. The global tin market is in slow recovery and is expected to improve over the next 24 months. The global copper market is certainly in recovery, as copper prices are ascending towards their prices pre-GFC. There is little variability across the tailings that will impede a standard flowsheet development for this material. The outcome from the Cleveland operation will be to make two (Sn and Cu) saleable concentrates for sale into the global markets. BPPL’s focus with the Cleveland project will be to:

- Make a profit.
- Understand the technicalities of processing tin when present as agglomerated fine particles. This approach may have application elsewhere in Tasmania and around the world.
- Understand the effectiveness of modular plant design and operation including design for a future grinding circuit if BPPL also pursue processing of the remaining hard rock material.
- Understand the operating processes required for a “remote” site.
- Benefit the local environment by cleaning up an environmental problem (acid mine drainage). The tailings dam is located around a delicate rainforest environment.
- Generate long term contacts with the global Cu and Sn smelting companies.

Local Community: Cleveland is in a relatively isolated area of north western Tasmania. The project personnel would be sourced from either Burnie or Rosebery. Any mining or industrial project developments are reviewed favourably by the Tasmanian Government.

Pre-Feasibility study analysis:

A pre-feasibility study has been completed to evaluate the viability of alternative processing options for the Cleveland prospect

Next steps: The next steps in developing the Cleveland tailings project are:

- Visit the site to carry out more complete Due Diligence, understand site requirements and logistical and environmental issues.
- Complete detailed metallurgical and if indicated, mineralogical QEMSCAN/MLA characterization on additional samples.
- Develop a preliminary flowsheet and a feasibility study (accuracy to be determined).

Demonstrate viability of the retreatment process flowsheet via economic operation of a simplified onsite processing facility at 1/10 scale, with provision to systematically test a range of recommended process flowsheet variations and innovative commercially proven process technologies, whilst producing revenue.



Schedule



Schedule

The focus of development and operations for BPPL will be on the Cleveland Sn-Cu-Ag project to ensure timely generation of cash-flow.

Task	Aug-10	Sep-10	Oct-10	Nov-10	Dec-10	Jan-11	Feb-11	Mar-11	Apr-11	May-11	Jun-11	Jul-11	Aug-11	Sep-11
Funding for Cleveland development	█													
Site visit for location and environmental evaluation		█												
Secure equipment for Cleveland demonstration plant		█												
Planning for drilling of Cleveland tailings dams		█												
Commissioning of Cleveland demonstration plant			█	█	█	█	█	█	█	█	█	█	█	█
Cleveland Demonstration plant evaluation period					█	█	█	█	█	█	█	█	█	█
Drilling program for Cleveland tailings dams 1 and 2			█	█	█	█	█	█	█	█	█	█	█	█
Variability analysis of Cleveland tailings dams 1 and 2						█	█							
Preliminary process design - pre-feasibility	█													
Feasibility study - Detailed process flowsheet design					█	█	█	█	█	█	█	█	█	█
Feasibility study - Environmental and mining permitting			█	█	█	█	█	█	█	█	█	█	█	█
Feasibility study - Detailed economic evaluation						█	█	█	█	█	█	█	█	█
Tender and assign environmental consultants for EIS		█	█	█										
Environmental Impact Study			█	█	█	█	█	█	█	█	█	█	█	█
Secure EPCM manager			█	█	█									



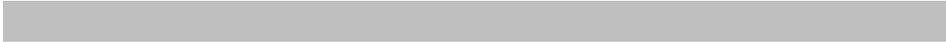
Budget



Budget

BPPL is seeking funding to progress development of the Cleveland Sn-Cu-Ag project in Tasmania and to provide the platform for future opportunity development. The following budget has been developed based on an 18 month development programme for the initial BFS phase.

Cleveland		
Project management	Managing Director	\$270,000
Technical	Technical Director	\$270,000
	Snr Technical Consultant	\$270,000
	Directors fees	\$60,000
	Legal Fees	\$85,000
	Finance and admin support	\$50,000
	Office lease and IT set-up	
Capital Raising costs		\$50,000
Travel		\$30,000
		\$1,135,000
Variability Analysis	Drilling	\$15,000
	Assay	\$10,000
<i>sub-Total</i>		\$25,000
Metallurgical Testwork		\$100,000
Demonstration Pilot Plant	Capital	\$560,000
	Operating (6 months)	\$580,000
<i>sub-Total</i>		\$1,240,000
Feasibility Study		\$500,000
Environmental Impact Study		\$250,000
Cleveland Development		\$3,150,000
Payment for work completed to-date		270,000
Total Cost		\$3,420,000



Appendix 2 - Draft Metallurgical testwork report

Cleveland Tailings Sn-Cu Project

Project #: B107

Overview of sighter metallurgical testwork program for Sn and Cu recovery from Cleveland Tailings composite

Aims:

The aim of this program was to undertake initial 'sighter' analysis testwork, using an existing composite sample of Cleveland tailings material, for validation of the BPRL flowsheet approach. Key process parameters for the proposed Cleveland Tailings flowsheet were examined to determine amenability and provide the basis for more detailed metallurgical testwork once a fresh tailings composite sample was generated.

Sample:

The sample used for this program was a composite generated from a drilling program undertaken by Lynch mining on the Cleveland tailings dams in 2008.

Summary	Sn (t) %	Sn (s) %	Sn (cas) %	Cu %	P80 micron	Total S %	Sulphide %	Sulphate %
Cleveland tail Head	0.28	0.03	0.25	0.11	150	5.32	3.15	2.17
Dist		9.4%	90.6%				59.2%	40.8%

The assayed Sn grade was 0.28% Sn, of which 9.4% was soluble Sn, attributed to Stannite.

A significant proportion of the sulphur present was as sulphate (41% w/w), suggesting that significant oxidation of the sulphide minerals had occurred. The sample had been stored in bags outside since 2008 and it was unclear if oxidation had occurred during this time or in-situ in the tailings dumps. A fresh sample from a new drilling program will be used for subsequent material evaluation.

Program:

The program of testwork was developed as a 'first pass' investigation of key components anticipated to be in the process flowsheet. The components were defined based on knowledge from the historic Cleveland flowsheet, other Sn-Cu mixed flotation/gravity operations and discussions with John Glen of ALS Ammtec Burnie Research Laboratories.

The key components for investigation were:

1. Material Preparation
 - a. Comminution
 - i. Attrition vs. Stirred mill size reduction for Cleveland Tailings
2. Sulphide mineral removal
 - a. Magnetic separation
 - b. Copper and pyrite sequential sulphide flotation
3. Detrimental element removal
 - a. Acid conditioning of tailings material
4. Tin recovery
 - a. Cassiterite flotation
 - b. Gravity concentration of screen oversize material.



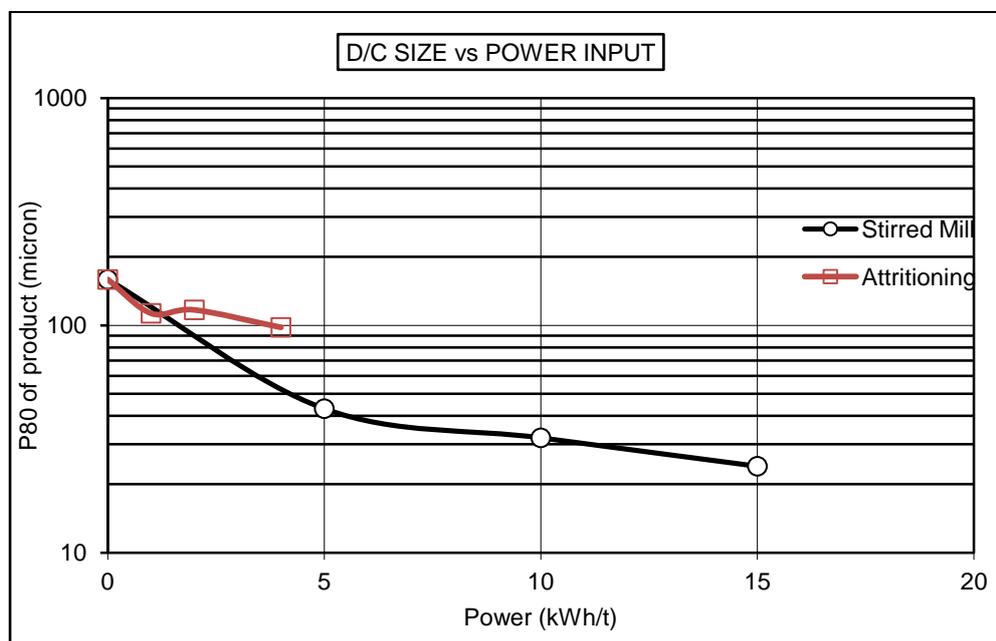
Results and Discussion:

Comminution

The nature of Cleveland tailings material meant that significant size reduction was not required. This had been previously defined by QEMSCAN analysis of the composite material utilised in this program (Goodall 2009). Comminution was therefore targeted at removal of oxidised and precipitation layers that were expected to have formed while material was in the tailings dumps. It was also visually noted that significant aggregation of fine particles had occurred and action would be required to break these.

To achieve the comminution aims and avoid over-grinding of the material two separate methods were utilised. Attritioning (stirred mill with no media) was trialled to specifically target removal of oxidised and precipitate layers, while stirred milling at higher power was used to increase breakage of aggregates and promote additional liberation.

The impact of varying power input on the product P80 can be seen in figure **.



- Attritioning reduced P80 to minimum of 98 micron when 4 kWh/t used.
- Stirred milling had a much more pronounced effect, taking the PSD down to 43 micron at 5 kWh/t and then to a minimum of 23 micron at 15 kWh/t

These results suggested that attritioning achieved breakdown of a proportion of the aggregate particles that had formed in the tailings dumps. It should be noted that the target grind for the Aberfoyle Cleveland Mill was to a P80 of 68 μm . This suggested that attritioning had not achieved completed breakdown of aggregates and consequently could not be expected to have had a significant effect on removal of oxidized layers.

In contrast, stirred milling resulted in significant particle size reduction. This inferred that a mechanism between that of attritioning and actual stirred grinding may be required for effective aggregate breakdown and surface cleaning.

Next Steps

1. Evaluation of Sn and sulphide mineral liberation by QEMSCAN.
2. Evaluation of other breakage mechanisms for efficient aggregate breakdown without additional particle size reduction.
3. Effect that increased liberation has on recovery when flotation conditions have been



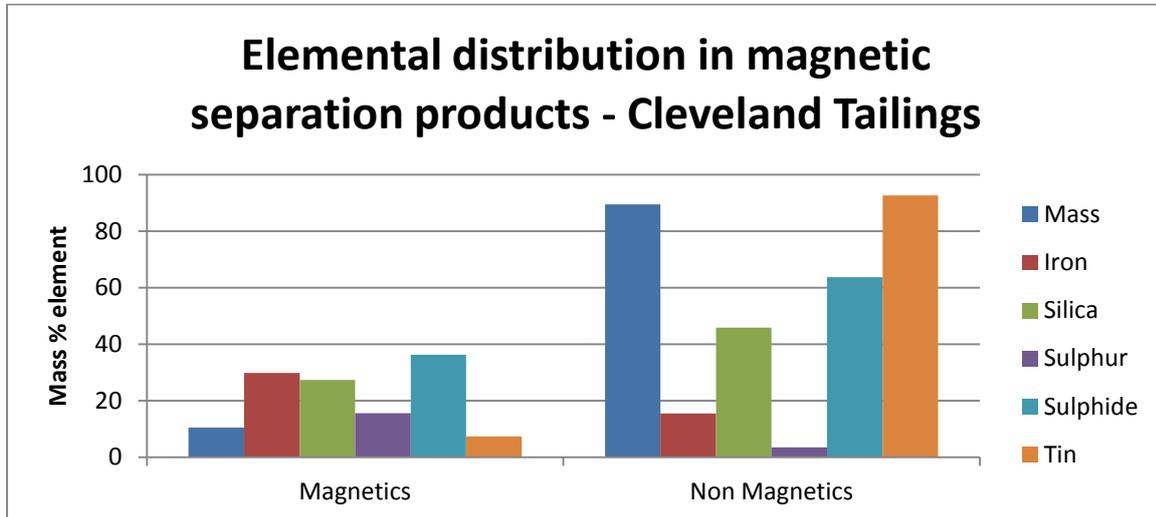
optimised.

4. Use of a regrind on the -53 micron screen fraction should be investigated

Flowsheet stage 1 - sulphide removal

Magnetic separation

Magnetic separation was included as a possibility for removing magnetic Fe oxide minerals and more specifically magnetic pyrrhotite.



- Feed for magnetic separation was from the attritioning tests at power input of 2 kWh/t. This resulted in a feed P80 of 119 micron. This feed was coarser than would be targeted in the process.
- Magnetic fraction contained 10.5% of mass, 7.5% of Sn and 34% of S.
- Certainly worthwhile looking at further optimization. An examination of the effect of further size reduction on the results is warranted. It is anticipated that further size reduction would reduce the total Sn loss at this stage.

The key finding from the magnetic separation testwork was that a significant proportion of the pyrrhotite was magnetic and amenable to separation in this manner with further optimization.

Next Steps

1. Analysis of magnetic and non-magnetic products by QEMSCAN to define cause of losses.
2. From QEMSCAN interpretation define program of optimization if theoretical Sn losses can be minimized.

Sulphide Flotation

Cu sulphide flotation

Initial tests without optimisation recovered 30% Cu with a loss of 3.2% of Sn. The interesting point from these initial tests was that when stirred milling was used rather than attritioning (117 micron > 75 micron), the Cu recovery increased to 34% but the Sn loss stayed the same. Mass pull of 3-4%. This potentially suggests a number of things that should be further investigated:

- Cu sulphide minerals require further liberation
- Cassiterite is independent of the Cu sulphide minerals.
- Stannite is well liberated already and was recovered at both grind sizes.



Optimisation of the flotation (increasing Cu collector, 9810, to 45 gpt from 21 gpt) saw an increase in Cu recovery to ~50%. Sn losses were also increased, up to ~9% of total Sn.

- 40-45% of the S was also recovered into the Cu concentrate, which had a mass pull of 10%.
- Analysis of soluble Sn in Cu and Py concentrates showed 40% recovery into Cu concentrate and 21% recovery into Py concentrate.
- This suggested that recovery of Stannite (CuSnS) into sulphide concentrates is roughly analogous to the recovery of cassiterite into Sn float concentrate.
- It is understood that a credit should be available for Sn in Cu concentrate but that Sn lost to Py concentrate is gone. It is therefore desirable to maximise the proportion of stannite reporting to the Cu concentrate.
- This will need some additional optimisation at a later date.

Next Steps

1. Evaluation of Cu rougher tail by QEMSCAN to determine deportment of Cu sulphide losses.
2. Evaluation of Cu concentrate by QEMSCAN and soluble Sn assay to determine deportment of Sn losses.
3. Re-do flotation testwork based on QEMSCAN interpretation to maximize Cu recovery to rougher concentrate and minimize cassiterite Sn losses.
4. Optimisation of Cu concentrate grade. Initial target of >10% Cu.

Sulphide flotation

Sulphide flotation was undertaken subsequent to Cu recovery for removal of Fe sulphide minerals from the system. Fe sulphide minerals, pyrite and pyrrhotite, can have a detrimental effect on Sn recovery in cassiterite flotation using SPA collector and their removal is an important step in any Sn recovery flowsheet.

Fe Sulphide	Mass % (Total)	Deportment
Pyrrhotite	7.5%	68%
Pyrite	3.5%	32%

The core requirement of this stage was to minimise Sn losses. Consequently, sulphide removal may not be optimised. A general target for sulphur recovery should be in the order of 80-90% with a Sn loss of <10% Sn. This should be taken over both the Cu and sulphide flotation stages.

- In the sulphide flotation the Cu recovery was still 15-20% and S recovery only 25-30%.
- Mass pull of 7-8% for a Sn loss of 6% total Sn.
- 21% of the stannite reported to the Py conc, which attributed only 2% of the total Sn lost as soluble Sn at this stage.

The recovery of Cu and S was improved by reducing the grind from 119 micron to 75 micron (3% improvement) but Sn loss and Fe recovery were not affected.

- This may suggest that Py/Po were well liberated but that the Cu sulphides were not.

Decreasing grind size had an effect on Sulphide (S₂-) grade in concentrate but not on total S grade.

- This may suggest that a greater proportion of the sulphide was available for concentration after cleaning of the surfaces by more intense grinding.
- Increasing PAX and Cu₂SO₄ only served to bring the mass pull down and had little effect on grade or recovery of S



The real problem seemed to be with kinetics. The flotation was very slow and after 22 minutes none of the test conditions had achieved a plateau. This is probably due to the presence of oxidised species.

- To increase the kinetics we may need to clean the surfaces even harder. This may require a different attritioning or cleaning process at the front end.

Overall there are a few key things that we need from the sulphide flotation stage of the process and we have not managed to achieve any of them with great success. they include:

- Recovery of 85-90% of S into a high grade S concentrate.
- At the moment we are well under this and can expect a negative flow on effect into the cassiterite flotation
- Total Sn loss across Cu and Py concentrates of <10%
 - Average total Sn loss of 15-16% is too high at present
- Cu recovery >60%
 - Cu recovery at 51-52% into a 1% Cu concentrate is far too low. This will reduce further when we clean the concentrate so we need to understand why we are missing Cu, its deportment and how we can fix it

The reason for all of this seems to be the slow kinetics of flotation. I need to do some background into speeding up the kinetics but it will come down to liberation. Poor liberation in this case will be caused by both insufficient grinding and by surface oxidation of the sulphide minerals in the tailings dam.

Next Steps

- Investigation of sulphide mineral liberation in flotation products by QEMSCAN
- Evaluation of effect on flotation rate that other cleaning methods have.

Conditioning

The concept of conditioning was to remove soluble Fe, Al and Ca from the material before the cassiterite float. The tests completed were designed simply to look at the acid consumption to reach various pH targets and the effect of going to those target levels on dissolution of detrimental metals. The target pH was set at below the optimum for cassiterite flotation (pH 4) to help ensure that anything that would dissolve at pH 4 was gone.

What we found was that with a target of pH 3 there was not much acid consumption and little dissolution of metals. At pH 2 the metal dissolution was ok and the acid consumption still acceptable. Once a target of pH 1 was taken the acid consumption went through the roof.

- Based on this alone a target of pH 2 was selected for the sighter tests

Next Steps

- Detailed balance of Fe and Ca distribution in these tests.
- Evaluation of residence time effects.

Cassiterite flotation

Sulphide scavenger float

Cassiterite flotation requires a very clean feed and to have as much sulphide as possible removed. However, this can't all be done in the initial sulphide float as the tin losses would be too high. Consequently it is necessary to use a sulphide scavenger immediately prior to the cassiterite float.



In the current test work the S scavenger only recovered 3-4% of the total S into a low (7-10%) S grade concentrate.

- Suggested that the conditions used were not very well optimised.

We will need to look at the mineral department in these concentrates and see what we are getting and what we are missing.

In effect we need to recover 85-90% of the remaining S in this stage.

- In this work we only got 25-30% of S in the deslimed, -53 micron fraction
- There is clearly a lot of scope here to improve.

We did only lose 2% of the Sn in the float feed though so that was a good outcome.

It is probable that the reason for poor S recovery at this stage was the same as for the first stage of Sulphide flotation.

- This indicates that the conditioning had not been as effective at cleaning the sulphide surfaces of oxidised layers as had been anticipated.
- Will require a combined investigation of conditioning and flotation if we decide to use it.

Cassiterite Float

A number of different conditions for cassiterite flotation using the industry standard collector, SPA, were investigated. These were then complemented with an additional test examining the effect of a new collector made available by Tall Bennet. The cassiterite flotation tests were targeted at Sn recovery and no specific energy was put into creating a saleable grade concentrate. For this reason the results can only be used as an indication of what might happen and not a definitive estimation of what Sn recovery we could expect. Sn cleaner stages were used, however, they effectively only included additional flotation time and as a rule did not show good results. The total recovery should therefore be considered based on the Rougher concentrate, which is the sum of the Cleaner concentrates and tails.

The feed material for each cassiterite float was sourced as the Rougher tail from a Cu + Py float test. This was necessary to remove the negative influence of sulphide minerals on Sn flotation.

The sulphide float conditions were maintained constant for all the Sn floats to help with comparability. A number of different preparation routines and reagent conditions were used, including:

- Acid conditioning (pH 2, no addition), deslime, Sn Ro + Cl
- Acid conditioning (pH 2, no addition), Screen (53 micron), deslime, Sn Ro + Cl
- Acid conditioning (pH 2, SSF addition), Screen (53 micron), deslime, Sn Ro + Cl
- No conditioning, Screen (53 micron), deslime, Sn Ro + Cl
- No conditioning, Screen (53 micron), Deslime, Sn Ro + Cl (new collector)

The recovery of Sn to Ro concentrate for the unscreened sample was 67% into a concentrate of only 1.16% Sn. This is obviously much lower than required but it forms the basis for further upgrading

- It should be considered for this test that screening was not performed in this test. Consequently cassiterite in the +53 micron fractions (~38% of total Sn) would not normally be in the flotation feed, having been screened off prior. That said, the total recovery of Sn (assuming 100% recovery in gravity, which would not occur) was analogous to the other tests where screening was used.



- This suggested that flotation recovery of the Sn in the +53 um fractions was very good.

Introduction of screening while keeping everything else consistent led to a combined recovery between the flotation Ro concentrate and Sn contained in the +53 micron fraction of 68%.

- This suggested that all of the Sn contained in the +53 micron fractions was being recovered in the float.
- The proportion of Sn reporting to the Sn Cleaner concentrate was very low, only 27% of recovered Sn, suggesting that cleaning wasn't doing much to upgrade.
- For this sample less SPA was added than the other tests. This suggested that the reagent consumption for SPA was still high.
- Conditioning hadn't done what it was supposed to by removing the SPA scavenging elements, such as Fe, Ca and Al.

Adding SSF to the conditioner resulted in a small increase in recovery and grade, plus a greater proportion of Sn in the CI concentrate. However, there was also an increase in SPA dosing, which is more likely to have had an effect.

- Can't draw any conclusions from this.

Removing the conditioning stage gave the best results for the SPA tests. Clearly this showed that the conditioning wasn't doing what it was destined for.

- Need to go back and look at the conditioning results in more detail.

The best results were achieved using the new Carboxylic acid collector being developed by Tall Bennett. Gave comparable recovery to the best SPA float but with about double the grade of Sn.

- Was performed in conditions optimised for SPA so better results should be expected.
- There was a smaller proportion of Sn reporting to the final Sn CI concentrate but this could be attributed to the lack of optimisation of conditions for this collector.

Next Steps:

- In progress



Appendix 3 - Memo from Andrew Glastonbury to Bright Phase Pty Ltd

MEMORANDUM

TO: **Chris Dunks**
COPY: **Will Goodall, Mike Adams**
FROM: **Andrew Glastonbury**
SUBJECT: **Cleveland Underground**
FILENO:
DATE: **8th March 2011**

P.O. Box 8383
Orange East NSW 2800
Telephone: (02) 6365 1392
Mobile : 0417 653 135
Email: andrewg@gmcmining.com.au

This memo is a summary of the Site Inspection trip to Tasmania where I accompanied Will Goodall in Burnie on Tues & Wed 1st & 2nd March & to site at Luina early on Thurs 3rd March. This is a summary of the key findings and actions required to better understand the issues associated with the underground potential associated with the Cleveland Tin project.

Archives for Cleveland Tin were inspected at the AMMTEC Metallurgical Laboratories in Burnie (soon to be rebadged as ALS Labs). Records consisted of plans (contained in 2 vertiplan files in the cleaner's room) and 17 archive boxes contained in a separate room along with archives from various other Tasmanian mines.

Larger Vertiplan File contained: in excess of 1,000 plans incl geological surface exploration plans from the 1970's and 80's, topographical maps, Geophysical IP / Magnetics plans, soil geochem plans, Stream sediment plans, mine section plans and face grade plans,

Smaller Vertiplan File contained: a smaller no of plans with sections of the mine from the 1970's through to 1981. These mine sections included various lenses & levels throughout the mine. On top of the smaller vertiplan file were some additional plans of long sections of the mine from 1980.

Archive Boxes: The 17 archive boxes contained the following information.

Box 100: Survey Note books, **Box 101:** Survey Note books / Rise pickups, **Box 102:** Survey Note books
Box 103 Drill logs C1 – C550, **Box 104** Drill logs C551 – C1125, **Box 105** Drill logs C1126 – C1500,
Box 106 Drill logs C1501 – C1800, **Box 107** Drill logs C1801 – C2050, **Box 108** Face Sampling Levels 1 – 7,
Box 109 Face Sampling Levels 8-13, **Box 110** Face Sampling Levels 14-21, **Box 111** Face Sampling Levels 22-24
(1982), **Box 112** Geology Files / Aerial Photos / Exploration Mtg Minutes / Exploration prospects,
Box 113 Geology Files / Mineralogical & Petrology Reports / Geochemistry reports / Exploration reports 1970's &
1980's / Foley resource report* / logging reports / Yr end report to Dept of Mines Tasmania 1972*, **(*copied)**
Box 114 Drill hole test survey reports Cominco 1973 / Mineralogical & Petrology Reports / Washington Hay mine
reports / Exploration reports 1970's, **Box 115** Geology Files, Soil & surface sampling, exploration proposals /
Petrology & exploration Reports late 1970's & early 1980's, **Box 116** EL 1/63 reports, early 1970's early 1980's /
statutory exploration reports, **Box 117** Drill hole survey data,

- Contact details for Dick Lewis of LMRC Lewis Mineral Resource Consulting have been provided to Will as someone suitably qualified & experienced to further discuss the requirements to develop a JORC compliant resource for both the Tailings & underground resources, I have worked with Dick's resource data on a number of other projects and believe he would be of great assistance to BPPL in this area.
- Spoke with Adrian Brewer of Brewer Geological Services 08 8389 3505, Mob 0427 569 274, email ambrewer@ozemail.com.au Adrian is based in the Adelaide Hills and assisted with compilation of information for Lynch Mining Annual report, he provided ore resource figures based upon historical reports from Aberfoyle.
- His comments were that he felt these u'g resources were based at depth and included the Foley resource.
- Adrian referred to information more recent than that found at the AMMTEC labs in Burnie and in fact has gone through these archived records some time ago. He referred to more recent archived records held by

Price Waterhouse Coopers in Melbourne or Perth as they were the administrators for Western Metals. Adrian believes there were 10 or more achieved boxes complete with information from the early 1980's, he went through some of this information back in May 2008.

- He will forward me an email from PWC with relevant contact details and archive details so as you can further follow up (Done). He believes the contact was a Michael Fung in Melb on 03 8603 1147 and the particular box we may be after with resource information was box No 1514 from archive account number 60608. He also had not seen any specific mine closure reports, technical mining reports or life of mine plans or schedules which I have been chasing.

Basic Geology / Stratigraphy (as described in 1972 yr end report to Tasmanian Mines Dept for ML 27M/71) in the immediate vicinity of the Cleveland Mine is as follows:

1. A basic volcanic unit (uppermost)
2. A lode bearing unit (Halls Formation)
3. A mica sandstone unit (lowermost)

Ore Controls / Lenses (as described in 1972 yr end report to Tasmanian Mines Dept for ML 27M/71)

Halls Formation contains 2 mineralised sequences, a lower sequence separated from the upper by a distinctive mica sandstone unit, contains Henry's, Luck's & Khaki's lodes which is a group of 5 lenses, from the base the sequence is A,B,C, D & E lenses. In Henry's lode, ore pods may be up to 15m wide, 30m along strike and 200m down pitch.

To the south of the mine area, it is apparent that the ore lenses become progressively separated from one another as a result of thickening by sediments between the lenses. This has been mapped between A & B lenses on the 8 & 9 levels on the mine.

Ore controls at Cleveland are essentially 1) Stratigraphic: the orebodies lie within discrete, conformable, mineralised layers within a well defined shale & chert formation, The shape & extent of these layers probably depends upon the persistence of the facies. 2) Mineralogical: ore is restricted to areas within the mineralised layers by zoning.

Not mentioned in this above report but ascertained from other reading and discussions are the additional lenses of Battery & the untouched Foleys resource at depth. It is also noted that the Henry's lens was mined by open pit methods on the top of the hill and also at depth from the underground workings.

Historical Production & Ore Reserves: Report AR 1985_86 report of the Director of Mines, for year ending 30 June 1986 indicates that over the 18 years of it's life Cleveland mine produced 5.6 mill tonnes of ore yielding 24,000 t of tin & 9,800 t of copper.

Report No 87-2624 Aberfoyle Resources Ltd Geological Resource Assessment Cleveland Tin Mine as at End of Milling – Mining Operations 12th June, 1986 is a most valuable document in assisting understand the remaining resource at the Cleveland Tin Mine.

This report states a total geological in situ resource of $\geq 0.35\%$ SnT as at 12th June 1986 were 6.5 mill tonnes at 0.7% SnT, 0.29% Cu.

In terms of measured & indicated resources the report indicates 5.2 mill tonnes at 0.7% SnT & 0.31% Cu

In terms of Inferred resources the report indicates 1.3 mill tonnes at 0.72% SnT & 0.22% Cu

In terms of the Foley Zone W/SN/Mo $\geq 0.3\%$ W_o_3 there is 1.2 mill tonnes at 0.35% W_o_3 , 0.05%SnT, 0.02% MoS_2

It appears that tin grades tend to reduce with depth and comments tend to indicate that metallurgical recoveries also have reduced with reducing grade. It also appears that there are perhaps not many resources remaining in the upper levels, having been the focus of remnant mining over the final years of operation.

It is recommended that a suitably qualified resource geologist refer to this report when seeking to establish a JORC resource figure for Cleveland underground.

Underground mining method & development: This commentary has been developed through discussions with both Bill Lannen & Dick Livingstone, fortunately I have known both Bill & Dick for quite some time from other projects so they were happy to share their knowledge of the operation. Bill is a Mining Engineer who did his time underground at Cleveland and who rose through the ranks whilst there and now is an executive director of Mancala Mining (mining contractor & raise driller). Bill left site before it was closed, he recalled that there was a large rib pillar removed from the 14 / 15 levels which may cause some geotechnical & ground control issues upon re-entry into this area and with potential mining in the area.

As an aside he indicated Mancala would be most interested to assist BPPL with establishing the site and ultimately with any aspects of the underground or tailings treatment operation, as they have of the order of 100 employees in Tasmania and have various items of mobile plant & equipment. They are currently working for Bass Metals as the project managers through a mining alliance agreement at their Que River operation and at the Fossey underground project. My understanding of the Mining Alliance is that Mancala provide the required statutory personnel, mining equipment, some technical staff & operators and charge Bass a cost plus o'heads (9-10%) + margin (of the order of

5%) + a % of free cashflow from the operation. Mancala have also undertaken various work at have worked at Henty, Rosebery, Renison, Avebury & Mt Lyell.

Dick Livingstone was a production engineer at Cleveland and was there in 1985 & 1986 which included the period of mine closure. Dick has sent me 2 emails which I have forwarded to you all following this memo. The 1st email is titled Cleveland Tin and contains some photos of the Henry's stope taken from 5L and 4L and shows the stope (mining void) breaking through to the surface and the stope being bogged with one of the first underground remote loaders used in the country, this email also contains an aerial photo of the site taken in 1986 which may be of use as well as a technical paper on mining practices at Cleveland from 1977 and confirms the size of the decline below 7L as 5m x 5m, which should be suitable for use with some of today's conventional u'g trucks.

The 2nd email titled Cleveland contains a little bit more information on Cleveland – plan of Halls and Henry's areas near top of hill and a couple of pages reporting on the mine closure status at May 1986. It indicates that there should be a set of plans sent to the Mines Department.

Dick recalls that in 1985 & 86 the only mining was remnant mining of the levels above 7 level. His understanding was that in the 1970's mining was being undertaken down the lower levels of the mine and in the early 1980's the tin quotas restricted output as tin prices were low and so the decision was made to mine the upper levels of the mine where the tin grades were higher, mining costs were lower and where the mill recoveries were higher and it was this strategy that kept the mine going for as long as it did. In areas of the upper 9 levels the tin grades were as high as 0.9%. The last 12 mths of the mining operations saw as much ore as possible being removed from the upper 9 levels.

It was about the time of the early 1980's that the decision was made to stop dewatering the lower levels and to just maintain the water level at the 14 level. Dick recalls that it required pumping out around 1ML/day ie 11 – 12 l/sec to maintain the water at this level.

Henry's open pit has been originally mined as a small open pit and then as underground mining has proceeded the underground workings have broken through into the bottom of the open pit and have continued through to the 7 level. One of Dicks emails contains a photo showing the extent of the stope created from the 5L through to the surface. Dick believes that this stope has collapsed and that this hole is now full of surrounding waste material through from the surface to the 7 level. Despite this collapse Dick believes that it should still be possible to access the lower levels from the 7 level decline. The "glory hole" in these upper levels will be allowing all precipitation that falls in the immediate area direct access into the underground workings.

An unknown still remaining is where the estimated 220,000 tonnes of Hellever tailings has been sent in the mine. It is thought this was pumped into the 7 level but no details have been found yet just where, if this was pumped in the 7 level and into the decline then it should be possible for these to be bogged out and may provide some additional tonnage to be added to the tailings resource. Shorty Halfacre also understands that asbestos building materials from the decommissioning of the nearby houses & buildings have been stockpiled underground also somewhere on the 7 level. These materials would need to be removed and disposed of in the appropriate manner.

Ground Conditions: The ground conditions underground appear to have been quite good as many stopes were left open and unfilled and in a number of cases were of significant dimensions. The decline and underground development appears to have been point anchored rockbolted and that this was generally only required to secure flat dipping joints to prevent slabs falling out.

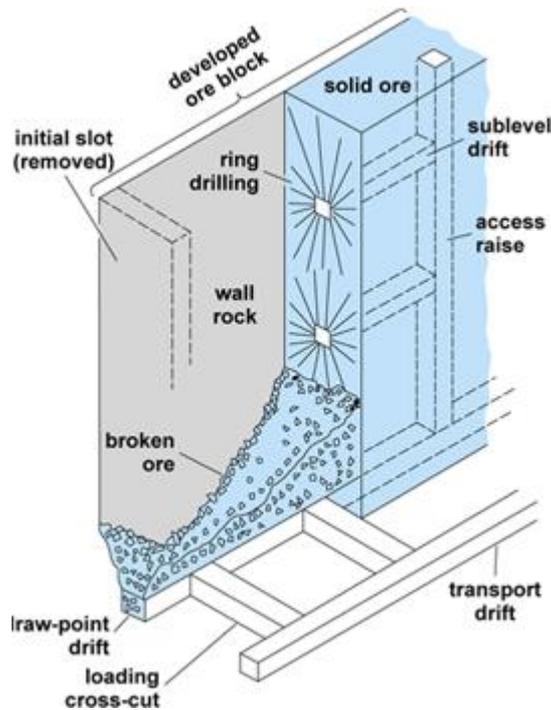
Electric power provided 415 V to the u'g site which was just for supplying the u'g pump stations and booster compressors. The mining fleet was pneumatic or diesel. Should we be recommencing u'g operations we would now require 1,000V power to be provided u'g as in most other u'g operations so as to power vent fans, compressors, pump stations, electric hydraulic jumbos (u'g drill rigs) and longhole drill rigs.

Underground water. Both Dick & Bill have differing recollections of the volumes. Bill felt that water volumes were of the order of 30 – 40 l/sec into the mine that were required to be removed whereas Dick recalled having to pump around 1 ML/day (11.6 litres/sec) to maintain the water at the 14 L (as in 1985 the mine had already been let flood to that level). Bills figure could be correct when mining at the deepest extent of the mine and possibly as water levels in the mine rose less water came into the mine.

Mining Method: Year end report for 1972 refers to airleg stoping of the Henry's North Lens mining ie narrow vs some section plans show areas of stopes with widths of up to 40m this will depend upon the various lenses and their dip as to the most suitable mining method given the local ground conditions and grades. My understanding is that the predominant mining method was a mechanised sublevel open stoping using uphole drilling (similar to the diagram attached below).

The paper presented to the AusIMM underground operators conference in Oct 1977 "Current Mining Practice at Cleveland Mine" of Abminco NL by HR Everett confirms:

- a mining method of mechanised open stoping and outlines the plant used.
- Production increased from 254,000 to 400,000 tpa in 1977 ave grades that year were 0.79% Sn & 0.3% Cu
- The waste rock of the hanging wall & footwall through which the decline is developed is competent.
- The ore is highly competent. Rarely is ground support required, and if required is in the form of expansion shell type bolts to support isolated sections of ground.
- The hangingwall & footwall of stopes are considered sufficiently competent to be self supporting, consequently no fill is used.
- The orebody above the 7L horizon is reached by adits into the hill on each level with these drives believed to be of smaller profile ie 4m x 4m or 4.5m x 4.5m. The portal at 7L (rl 343) is just above the valley floor and provides the only truck access.
- Levels below 343 rl are reached via decline of gradient 1:9, dimensions 5m x 5m, (5.5m wide on curves)
- A ventilation rise (upcast) & services rise (downcast) were extended with the decline in 20m lifts (providing a 2nd means of egress)
- The lowest level in 1977 was the 16 L (rl 175)



My understanding from discussions with Bill & Dick is that the sublevel spacing was around 20m and that the ground conditions were pretty good allowing wide stopes in areas where lenses were taken together, stope lengths of up to 50 -60 m was not unusual, although with today's standards and geotechnical understanding this may no longer be acceptable. The drill rings were usually charged and fired with ANFO indicating relatively dry conditions.

Stopes were usually left open, ie not backfilled, however stopes could be backfilled with waste and tailings as a means of providing support and to save on hauling all waste to the surface. It appears that most of the mined stopes remain open (ie have not been filled) although the effect of these stopes having been submerged under water remains unknown.

Mining from below the 26 level would result in ore haulage distances in excess of 4kms to the ROM pad. The haulage distance from the bottom of the mine ie 26 level to the ROM pad would be of the order of 387 vertical metres x 1:9 = 3,483 decline metres plus say 600m to ROM pad = 4.1 km haul using conventional u'g trucks which in this application would average around 80,000 tkm/mth ie 1,420 tkm/shift ie 346 tonnes over 4.1km, assuming an average load of 35 t this equates to 10 loads / shift. If targeted ore production was 450,000 tpa then this would require approx 2 trucks if the ore was sourced from the lower levels of the mine, additional trucks would be required to handle waste volumes.

Mining & haulage costs will increase with increasing depth of mining. Mine waste could be considered for use as backfill for some of the open stopes thereby removing the need to haul this waste to a surface waste dump.

Mine plans indicate:

- ✓ The main portal / mine office level and main haulage drive was the 7 L (rl 343)
- ✓ 1 – 6 levels appear to have independent adits accessing them and may not all be interconnected with decline or incline access, it may also be safe to assume that in terms of resources that these upper levels have all been mined out.
- ✓ 7L to 26L has decline access
- ✓ 23L (sea level)
- ✓ 26L (45m below Sea Level)
- ✓ It appears that the main ventilation raisebores went from the surface to the 17L

Surface Infrastructure: Much of the original surface infrastructure has been removed from site, items such as surface raisebore holes appear to have been secured and covered. Restoration of a suitable laydown area for storage of mobile plant & equipment & other mining support facilities, establishment of a fuel storage, distribution and bunding area, workshop, offices, muster area, ablutions and change room facilities including septic, etc are all required to be re-established onsite for the mining requirements.

Underground Infrastructure: It appears that much of the underground infrastructure has been removed from the mine and salvaged and what remains underground would now be of little use. It would be a requirement to re-establish the decline and level access from the surface portal and work from areas of safety and security into areas requiring rehabilitation in the form of scaling (removing loose material from the walls and roof) and rockbolting areas requiring support as well as re-establishing services into the lower areas of the mine such as ventilation, power, water, dewatering lines and compressed air lines. Raisebore holes should still be serviceable as should the 2nd means of egress and return ventilation rises that appear to have been developed as the mine deepened. Again perhaps these may need additional support upon inspection and would require new ladderways installed.

Mine closure was completed in 1986 and appears to have been done professionally and with planning. The numerous portals into 7L and above levels appear to have been all secured and barricaded off. Reference to the memo attached to email 2 from Dick Livingstone includes an internal memo addressing outstanding mine closure work.

Conclusions / Recommendations & Suggested steps forward :

The grades underground at Cleveland are on the lower end when compared to neighbouring tin mines at Renison & Mt Bischoff and show a tendency to decrease with depth along with metallurgical recoveries, however with the current strong commodity prices and a focus on mining & processing costs whilst maintaining grade and minimising dilution should mean that a profitable operation could be re-established at Cleveland.

- Prior to re-entry into the underground workings it is recommended that a suitably experienced team participate in a comprehensive risk assessment detailing the potential hazards and possible consequences and their likelihood thereby prioritising the potential issues and developing suitable ways to address those that present higher risks.
- Further geotechnical investigation and perhaps monitoring would be suggested to better understand the risk this presents to
 - access to the 7 level in light of the collapsed upper workings as a result of pillar removal
 - access from the 7 level to the workings below, particularly in light of the remnant mining that has taken place in the upper levels and the rib pillar removal on the 14/15 level.
 - the effect that the raised water level has had on stope stability and main access decline below the 7 level and on the previously installed ground support.
- Dewatering of the mine could take quite some time. At the relevant time and if required I could endeavour to estimate the volume of water that might be in the underground workings and estimate the likely time required to remove and maintain water levels so that work could continue from the lower levels of the mine.
- Perhaps the easiest method of commencing the mine dewatering process would be to lower a submersible borehole pump down one of the surface raisebore holes which I understand extend from 7L through to 17L.
- A significant issue to be considered and raised with the relevant authorities will be the quality of the underground water and where it can be released. If it is unsuitable for release into the adjacent Deep Ck (or Washington Ck) & Whyte Ck then it may need to be stored and treated onsite which would pose significant challenges.

- Perched water may also present a potential hazard.
- Consideration needs to be given to the close proximity in terms of horizontal and vertical clearance from the 7 level portal to the adjacent Deep Ck (or Washington Ck) immediately outside the portal area. Currently this could present a potential hazard of significant consequence should there be a major rainfall event or change in river course for any reason.
- Reestablishment of the ventilation will be an issue prior to any personnel re-entering. It is recommended that a safe work procedure be developed for this potentially high risk activity given there is no understanding of the air quality underground. It would be envisaged that after dewatering is well advanced that a team equipped with gas monitoring equipment and self contained breathing apparatus (similar to that used in mine rescue situations) would be used to monitor air quality and re-establish a ventilation network.
- Dick Livingstone has suitable experience in this area as he assists with coordination of mines rescue events as well as having a working knowledge of the Cleveland mine and may well be best placed to work on this aspect of the re-entry process if this is required. I would be pleased to assist with coordination / planning / budgeting of this process if required.
- There is also consideration of the cleanup of the estimated 220,000 tonnes of tailings that have been disposed underground, this may or may not be a significant issue depending upon where it has been placed.
- Whilst many of the items suggested or recommended above may not be required for a number of years it may be worthwhile if considering re establishing the underground operations to commence this in a slow and measured way in the short to medium term with a small crew of experienced underground operators commencing the dewatering and cleanup process with a defined checklist of priorities so as to gain a clearer understanding of the potential issues and plan for these as required, this could be done for an initial period of 3 to 6 mths and then reassessed.
- It is recommended that discussions be held with the power supplier so as to check the availability of power to the site for tailings retreatment and also for possible reestablishment of an underground mining & ore processing operation. The supply passing through the site to Savage River appears as though it would have sufficient capacity.
- The underground voids could present the opportunity to store retreated tailings thereby saving the additional costs of new tailings storage facilities on the surface as well as a suitable place to dispose of waste material generated from the decline and level development, thereby saving haulage to the surface waste dump.
- I would be happy to assist with the development of a more comprehensive checklist of items to be considered prior to re establishment of the underground operations so as this can start to be clarified, costed, planned, prioritised and scheduled for later implementation.
- It is recommended that a suitably qualified resource geologist refer to Report No 87-2624 Aberfoyle Resources Ltd Geological Resource Assessment Cleveland Tin Mine as at End of Milling – Mining Operations 12th June, 1986 when seeking to develop a JORC resource figure for Cleveland underground.

I believe that under current commodity conditions it is worth further consideration of re-establishing the underground operation and at first look into the project there do not appear to be any aspects or issues that could not realistically be addressed and cost effectively managed. Essentially there do not appear to be any issues that time, money and good planning could not address and overcome, however the volatility of tin prices needs to always be considered and managed where possible.

Fortunately the orebody appears to lend itself to a relatively cheap, efficient & effective mining method that has been proven successful over the past at Cleveland.

END OF MEMO

Appendix 4 - Draft mining report from CSA Global



DRAFT FOR CLIENT REVIEW AND DISCUSSION

31st March 2011

Our Ref: **BP.CT.01**

Dr Mike Adams
Process Director
Bright Phase Resources Ltd
PO Box 256
Holland Park
QLD 4121

Email: mikea@brightphase.com.au

Dear Mike

RE: Initial Review of the Cleveland Tin Project, Tasmania

Bright Phase Resources Ltd ("Bright Phase") has engaged CSA Global Pty Ltd ("CSA") to carry out consulting work in relation to Technical Assistance with the Cleveland Tin Project ("Cleveland Project"), Tasmania.

Background

Bright Phase is evaluating the possibility to of developing and operating the Cleveland Project in Tasmania. The project has three potential components:

1. Reclaim and treatment of the tailings
2. Re-opening and operation of the underground mine
3. Development of the underlying Tungsten Molybdenum deposit

CSA has been asked to provide technical assistance to Bright Phase in their review of this opportunity. This letter report documents the outcomes of the first part of mining and related aspects of the technical assistance, covering the first two elements of the scope of work:

1. Review the geological information and resource model to gain an understanding of the nature of the mineralisation and the host rocks.
2. Prepare a first pass estimate of mining capital and operating costs to populate a simple "what if" economic model

Assistance related to the main review of the geology and Resources will be covered under a separate Assignment Specification. Bright Phase will cover other aspects of the review through its own personnel or direct use of local and international expertise.

Bright Phase has provided CSA with a selected number of documents and records related to the resources and mining history at Cleveland. The main documents used as the basis for this review are listed below.

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Initial Review

In the time available, CSA has quickly reviewed the available documents at a level of detail required to identify the key issues and potential, either for risk or opportunity and in order to develop the input criteria for the “what if” model. No attempt is made here to compile a description of the deposits or the potential operation, as this information is available in the source documents. Such work would be part of the next phase of Scoping Study.

BBPL Power Point Presentation

1. At depth the tin lodes are thinning and the separation between the lodes is increasing.
2. The main mining targets at depth are the Foley zone (Tungsten Molybdenum) and the Henry's and Hall's Tin Zones

BPPL Information Memorandum

1. States the exploration potential of the tenements and the fact that modern exploration techniques have never been applied to the area
2. Refer to Section F of the IM for risk factors
3. First stage stated as tailings retreatment with the following parameters:
 - a. Resource 4.4Mt @ 0.32% Sn, 0.15% Cu, 12g/t Ag
 - b. Tin recovery 65% to a 60% concentrate
 - c. Parallel production of a ~22%Cu concentrate with ~5% contained Sn
 - d. Price assumptions: Sn \$20,750/t, Cu \$7205 /t Ag \$18/0z
 - e. Capex: \$22M to \$29M
4. Given CSA's experience with the Hellyer tails retreatment study in 2010 it is recommended that BPPL carefully review the practical mining recovery rates for the tailings.

BPPL Tailings Retreatment PFS

1. 1Mtpa throughput
2. Capex \$29.1M
3. Operating Cost \$17.35 per tonne treated = \$2984/tonne Sn Concentrate
4. Evaluations carried out pre-tax
5. Royalty 1.6% net sales plus profits to max 5% net sales. (BPPL in discussions to remove the profits component)
6. NPV (?) of 74M

1986 Resource Memo – Cassiterite Deposits

1. 1985 reconciliation from Resources to mill feed was +37% tonnes and -10% grade
2. Cut-off grade used for the resources was 0.35% Sn
3. Average width of the remaining reserves was 7m
4. SG = 3.05 t/m³
5. Refer to the attached spreadsheet for summary of the resources remaining in the Cassiterite deposits

1983 Resource Memo – Foley Deposit

1. Foley zone has been drilled from 1000mRL (Sea Level) down to 400mRL



2. At the top of the zone where it has been intersected by development it is relatively narrow (3m to 6m wide) with very irregular outlines (“feathered margins”)
3. At depth it widens to 30m but it still “locally very irregular in shape”
4. The geology, zoning and controls to mineralisation of the zone appear to be poorly understood
5. There were believed to be similarities with the Mt Pleasant Orebody in New Brunswick. Mt Pleasant at depth was a very regular “bee hive” orebody with increasing MOS2 at depth. The orebody was very amenable to large open stoping techniques (P.Davies personal experience)
6. The tungsten occurs mainly as Wolframite with only minor (2% of total WO3 as Scheelite)
7. The mineralized material did not show promising results from the early stage photometric sorting trials, due to the nature of liberation of the vein material
8. The Foley Zone is believed to post date the tin mineralisation at Cleveland
9. Ground conditions are referred to as excellent in the porphyry, but with some localized irregular zones of bleaching and clay alteration which are not related to geology or structure
10. Resources were calculated at a range of cut-off grades from 0.2% WO3
11. Refer to the attached spreadsheet for summary of the resources remaining in the Cassiterite deposits

Aberfoyle Concentrator Performance Report

1. There were major processing issues with both the Khaki Zone ores and the B-South Zone ores, which were never fully resolved (refer p17 of the report). The B-South ores were the most complex
2. Recovery in 1977 was only 49.7% of total Sn in the mill feed
3. This could represent an opportunity to BPPL in terms of the amount of Sn that can be recovered from the tailings
4. There was a change in reporting of the Sn grades and recoveries at the time when the Khaki Zone ores were giving problems. The change for reporting recovery against total tin (Snt) to against tin in Cassiterite only (Snc) gave a 5% to 8% increase in apparent recovery.

Glastonbury Visit Report

5. Remaining resources are:
 - a. Tin zones: 5.2Mt @ 0.7% Sn, 0.31% Cu, at a 0.35% Sn cut-off grade.
 - b. Foley Zone: 1.2Mt @ 0.35% WO3, 0.02% MOS2 – open at depth.
6. The tin grade reduces with depth in the identified deposits
7. Robbing of the crown and rib pillars between the 14 and 15 level late in the mine life (post 1980) may have created wide instability which could affect the access to the lower levels of the mine via the old decline.
8. The location of the 220kt of old Hellyer tailings which were dumped in the mine is unknown. These could cause problems/risk during mine re-opening
9. Similarly, asbestos building materials were dumped in the mine just before closure in locations unknown.
10. The decline size at 5m x 5m with widening on the curves to 5.5m would be suitable for modern truck haulage. The gradient at 1:9 is advantageous compared to current inclinations of 1:7 in terms of equipment wear, although the haul length will be 30% longer. It will also mean that there is a longer decline length to support during the rehabilitation of the underground mine
11. Ground conditions in the stopes and access development were good.
12. Water inflow rates are variously quoted as 1MI per day and 4MI per day. This needs to be resolved

Comment [RRE1]: In the identified deposits.

Comment [RRE2]: This is the key factor as it is more ground to re-support to modern day standards

Comment [RRE3]: Definitely a major issue.... Water management in terms of how to treat it to get emission levels to that required will be the key challenge.



13. The glory hole above the old stopes will act as a funnel for rainwater egress, adding to the dewatering load during mine re-opening and operations
14. The emission levels that must be met and the method and costs for water disposal will need careful consideration
15. Mine re-opening should be started as soon as possible in view of the unknowns stated above

Comment [RRE4]: Strongly agree

Development Ratios

The attached graphical representation of the annual development and ore production rates (prepared by mine staff late in the mine life) is a very useful guide to ongoing ratios that could be expected after re-opening of the underground mine. It is difficult to determine trends, because the variations were probably primarily driven by short term market conditions and cashflow, but the forecasts at the bottom of the chart are a good guide as follows:

Comment [RRE5]: Is it possible that we are mis-reading this table and the graph is cumulative..... and therefore the lateral waste development could be more like 2400m (which I think is still too high)

1. Mine production 450 ktpa
2. Decline development: 800m per year (equivalent to 80m per year deepening at 1:9. This seems high unless two declines were planned to access different areas. This will be checked with Bill Lannen next week)
3. Lateral development in ore: 1200m
4. Lateral development in waste: 2000m

Using current Australian development costs and allowing for raises and other miscellaneous development this would be equivalent to a cost of \$40/t. Adding \$25/t for stope break and \$10/t for haulage and mine services gives a total mining cost per tonne of \$75.

Outcomes from the What-if Modelling

[section to be written next week after a brainstorming session with the model]

General Observations

Based on the work that CSA has done to date, the following general observations are made regarding the possible re-opening of the Cleveland mine to exploit the remaining tin resources:

1. Historically the tin orebodies at Cleveland were amenable to the use of low cost large scale underground mining methods. The ore zones were regular and ground conditions were good.
2. The ore zones are believed to become lower grade and narrower at depth, but these disadvantages may be out-weighed by the greater efficiencies achievable with modern equipment and methodology
3. Geometry of the ore body is the major driver in the amount of lateral waste and ore development required. A better understanding of that geometry is required prior to understanding the viability of a future underground project. The absence of a three dimensional block model has made this difficult to assess in the time available for this preliminary review.
4. Metallurgical recoveries were historically relatively low (below 50% in some years). The advanced processing techniques that will be applied to the tails retreatment by BPPL are expected to have a corresponding positive impact on the hard rock tin recoveries.

Comment [RRE6]: additional

Comment [RRE7]: This is a key project opportunity that needs to be assessed and quantified... the gap or change in recovery is a key economic driver.



5. Suitably experience mining personnel, contractors and processing plant operating skills and are available in the local area.
6. Hydrogeology, water management and control of water emissions will be a critical aspect of the underground mine re-opening
7. Care needs to be taken when quoting tin grades and recoveries to distinguish between total tin and tin in Cassiterite.
8. With the project having three phases, further development of how the phases can interact to provide an improved project cash flow outcome would be advised.
9. Developing the project initially with a tail retreatment does reduce risk and capital requirements for a total project through funding of later stages from cash flow. There should be additional technical opportunities as well :
 - a. Future tails storage facilities (ie. underground tails storage, minimizing surface TSF's)
 - b. Optimum processing needs (ie. grinding required for hard UG ore can be installed early to fine grind tails to get higher recovery and offset cost of extra grinding)
 - c. Optimum infrastructure costs (ie. setting up footprints and power needs for a full project will be cheaper than doing upgrades)
10. Appropriate reconciliation factors should be applied to the resources used in the evaluation to reflect historical experience at Cleveland

Comment [RRE8]: Strongly agree especially the management of emissions.

Comment [RRE9]: additional

Comment [RRE10]: additional

Concluding Comments

We would like to thank you for giving us the opportunity to provide Technical Support for the evaluation and possible future development of the Cleveland Project. We would certainly welcome the opportunity to continue our involvement with the evaluation and development of the project.

Please contact Peter Davies at any time to discuss any issues related to the work.

Yours sincerely

Peter Davies
Director Mining and Projects

 Preliminary "What-if" Model Rev ____