

**Gladstone Tin Pty Ltd
Scotia Project, Northeast Tasmania
Report on Bulk Sample Mining and Processing
September-November 2009**

**Peter Campbell & Ken Morrison
30 January 2010**

Contents

	Page Number
Introduction	1
Sample Mining	1
Jig Plant Processing	4
Concentrate Dressing	6
Results and Conclusions	7
Appendix A Metallurgical & Analytical Results	
Figure 1	Bulk sample mining operation-photo
Figure 2	Flooded test Pit with stable water level-photo
Figure 3	Texture of gravel wash on stockpile-photo
Figure 4	Small grab sample jig plant-photo
Figure 5	Circular jig-photo
Figure 6	Concentrating table-photo

Introduction

In September-November 2009 a program of bulk sampling and gravity plant processing was conducted at the southern end of the Scotia pit. The aim of the program was to produce a representative sample of basal gravel wash from the deposit, and to process a sufficiently large proportion of the wash using scaled down versions of the equipment in the main jig plant and tin dressing shed proposed for the run of mine operation. Despite extensive drilling coverage and major overburden pre stripping having been conducted at the modern Scotia operation, this would be the first actual sample of potential ore material excavated from the pit and the first modern tin concentrate to be produced from the deposit.

Several variables critical to evaluating the viability of the operation should be wholly or partly explained by this program:

- The physical and mining characteristics of the basal wash, in particular the materials handling and plant processing character of wash in comparison to overburden sediments.
- The water content of the basal wash and the suitability of alternative mining methods and equipment for dealing with a wet unconsolidated ore body.
- The nature of the heavy mineral suite contained in the basal wash, with respect to the abundance of cassiterite and any other minerals of value or detriment to the intended concentrate, the sizing of the cassiterite sand and the suitability of conventional gravity separation plant to produce a viable tin concentrate.
- The tin grade in comparison to predictions for that part of the lead derived from the 2008 drilling.

This report describes the methods used and discusses the results achieved in the context of both the original aims and the limitations imposed by the small scale of the operation, the difficult ground conditions and the need to construct a temporary small scale plant from improvised components available on site.

Sample Mining

A 20 tonne CAT excavator and a 25 tonne CAT articulated dump truck (Figure 1) were dry hired for a 17 day period to excavate the bulk sample. Initially the existing pit access ramp in the southwest corner of the pit was extended on siltstone basement rocks until the gradient became too steep for the truck to operate. This position coincides with the basement dropping abruptly at the western margin of the incised gutter which contains the basal wash and was anticipated from the basement contouring derived from drill hole data. It was also the position where basal wash could be accessed with the minimum overburden stripping required. Overburden stripping continued from the southwest until production slowed due to the steepness of the haul road gradient. The operation logistics then switched to access from the northeast, benching down through overburden to develop a test pit in wash, roughly circular in plan, with a 20 metre diameter and centred at approximately 584,205E, 5,466,955N (GDA 94 datum). The centre of the test pit is approximately 30 metres southwest of the southernmost east-west fence of 2008 VDM drill holes and the extracted wash is from the western side of the deposit.



Figure 1 Bulk sample mining operation - stripping dry overburden

According to projections mapped from the drill hole data, the area excavated comprised approximately 8 metres thickness of overburden overlying 8 metres thickness of wash. Under the mining plan being followed it would have been straightforward to excavate down to basement if the ground water inflow was moderate and able to be controlled by the pumps set up in advance for that purpose. At the overburden-wash interface a persistent heavy inflow of ground water saturated and fluidised the wash horizon and the basal two metres of overlying overburden and although continuous pumping was able to stabilise the water level, it was not possible to suppress the water table and achieve any access to dry wash. This situation ensured that all the wash excavated was in a wet slurry form and that the excavator was working on the platform of water saturated overburden. A major consequence of the abundance of water in the pit was that sampling of the top 4 metres only of the wash layer was possible and therefore the relatively higher grade zone immediately overlying basement was not sampled. A much larger pit involving major earthworks and water management infrastructure would have been required to reach basement and that would have required a budget several times the cost of the actual program. Another important conclusion in response to observing the ground water behaviour in the test pit is that excavator and truck mining of the wash is clearly not viable. (Figure 2).



Figure 2 Test pit flooded to stable water level-2 metres above top of basal wash

Despite these difficulties an effective sample of wash was excavated and stockpiled on the floor of the main pit, ready for processing through the portable jig plant which was subsequently set up approximately 50 metres southwest of the test pit. It was determined that one filled excavator bucket contained approximately one tonne of loose wet wash and on that basis the stockpiled wash sample was estimated at 428 tonnes. The material consists of sub angular to rounded, friable, free running vein quartz pebbles and granitic quartz granules and sand with a moderately well sorted texture (Figure 3). The wash is relatively clay free in contrast to the overburden and the fact that the matrix to the pebbles is predominantly sand and granules, with only minor clay, is the reason that the wet wash behaves as an unconsolidated slurry and would be much easier to pump as a fluid slurry than would be expected for the overburden sediments.



Figure 3 Gravel wash on stockpile

The wash from the site sampled is dark brown in colour, due to an impregnation by humic material precipitated from anaerobic ground water, post deposition of the deep lead sediments. This colour is not a problem for the tin concentrate as it washes out easily during processing but variable amounts of authigenic pyrite are preserved in the humic material and the pyrite behaves as a heavy mineral during processing and can be a contaminant species to the concentrate. The extent of this problem is unclear because the concentration of pyrite varies substantially and the pyrite is metastable – ie it breaks down when dry and exposed to air.

Several pan samples were washed from the stockpile and they consistently showed an encouraging relatively clean heavy mineral concentrate dominated by reddish brown cassiterite with a uniform particle size range, visually estimated to be in the 100-400 micron range. Laboratory sieve sizing results from the eventual tabled concentrate are discussed later in this report.

Jig Plant Processing

A sub sample of approximately 100 tonnes of basal wash was processed through a portable jig plant, put together from components on site.

The plant used is a one tonne per hour grab sampling unit using a 300 mm wide x 600 mm long main jig process unit. In order to increase the capacity of the plant a second jig was added to the discharge of the trommel screen and the plant was choke fed so the screen overflow reported to the second jig (Figures 4 & 5). This had the effect of increasing the throughput and reducing the overall test time.



Figure 4 Grab sampling plant utilised for testing



Figure 5 Circular jig for capacity increase

In order to emulate the production process plant, the test unit jig tails were recovered at the drop point in bins and reprocessed at the end of the 100 tonne sample lot. This was done to offset the lack of residence time in the test plant and to accommodate any obvious losses to tails.

The jigs were set at very conservative stroke lengths and pulse speeds creating a low grade concentrate but minimising any further losses. This was able to be done as the concentrates were going into the tin shed for retreatment across a Wilfley shaking table. It must be noted that losses of cassiterite will have occurred due to the capacity limitations associated with the testing plant.

Concentrate Dressing

The jig concentrates were recovered to the tin shed and hand fed across the table in order to produce a concentrate from the jig concentrates and a tailings product (Figure 6). In order to create a finished result similar to a production plant, the tails component of the table feed was recovered in total and reprocessed across the table. The table concentrates were also reprocessed with the tails being sent back to the table for retreatment.

In a conventional run of mine setting, the tables would have a middlings recovery from the table and this would be kept in closed recirculating circuit with a tail being constantly removed from the table.



Figure 6 Jig concentrates going over the shaking table

It can be observed from the photograph (Figure 6) that the cassiterite (heaviest) is the lighter pink coloured product coming across the table at the top. There is an indistinct line separating the next heaviest material so a high grade cut is made requiring a retreatment of the residual material, because some cassiterite will be left in the lower grade material.

Because of the limited tin dressing equipment available at Gladstone for the removal of contaminant heavy minerals from the concentrate, the final tin concentrate stage was completed at the Burnie Research Laboratory (BRL), Burnie, Tasmania, where assaying and size distribution determination were also carried out. It was noted by BRL that a cleaner final product than was possible for them to produce would be expected if their feed came from a well set up production scale jig and spiral plant.

Appendix A contains the flow sheets and analytical results from the lab. Approximately 14 kg of 40% tin concentrate were processed at BRL and the dressing method involved removing pyrite by flotation, monazite by dry magnetics and silicate and oxide species by a combination of gravity and screening. A final product of approximately 67% Sn was determined by BRL to be the “natural tin grade” (Appendix A) and this would be expected to increase to a concentrate grade >68% if the heavy silicate fraction, especially the fine -212 micron zircon fraction, was removed in a production scale circuit by a combination of spirals and buddle. Simple technology sizing produced some of the main grade gains in the metallurgical test work and if the mill concentrate was more uniformly sized before dressing, than was possible with the small scale test equipment available in the pit, then a much higher recovery of the >68% product would have resulted. The distribution table in Appendix A shows that the coarser cassiterite grades 70% Sn after floatation and magnetics but the -212 micron concentrate is contaminated by uniformly fine zircon, most of which could be removed by well run spirals, prior to final dressing, at the production scale.

Results and Conclusions

Several critical aspects of both the nature of the tin mineralisation and the potential for alternative mining methods to be viable at Scotia were demonstrated by this bulk sampling program and for that reason alone the program is considered successful and cost effective. The main conclusions are summarised as follows:

- The tin bearing basal wash is unconsolidated fine gravel with relatively low clay content. It appears ideal for pumping as wet slurry and for processing through a conventional alluvial process plant utilising trommel screens, vibrating screens with jigs and spirals as major plant. The inclusion of other heavy minerals will require a tin shed circuit for dressing jig plant concentrates.
- The basal wash at Scotia is water saturated and the heavy rate of ground water recharge indicates that mining of the wash by excavator and truck is not physically viable. Independent of cost considerations, the wash appears suitable for sluicing to a gravel pump, dredging or drag line mining.

- The particle size distribution of the final tin concentrate, determined by dry sieving at the Burnie Research Laboratory, Burnie, Tasmania, shows that only 0.7% of the recovered product is <75 microns and 8.4% is >300 microns. On a cumulative frequency curve 100% of the final concentrate product is within the 50-400 micron size range.
- Metallurgical test work indicates that a final sales product grading approximately 68% tin is achievable by dressing a jig/spiral concentrate with a combination of; tabling, screening, floatation and dry magnetics. The relatively high pyrite and monazite content requires floatation and magnetics to be added to the normal gravity/screening circuit.
- The grade distribution of tin concentrate produced equates to a wash grade of approximately 5 kg or recovered 68% Sn concentrate per 50 BCM. Given that only upper half of the basal wash layer could be sampled, it is reasonable to consider this as a conservative estimate of the actual in-ground grade. The sample was taken at the south-western end of the block covered by the 2008 drilling program, which demonstrated a range in grades from 119-3829 gm 70% Sn concentrate per BCM over the entire thickness of the basal wash layer and consistently the best grade is at the very base of wash layer. This high grade portion of the wash was not reached during the test program, as discussed above, therefore no meaningful estimate of the real grade in that part of the deposit is possible.

APPENDIX A
Metallurgical & Analytical Results

Burnie RESEARCH LABORATORY
GRAVITY SEPARATIONS

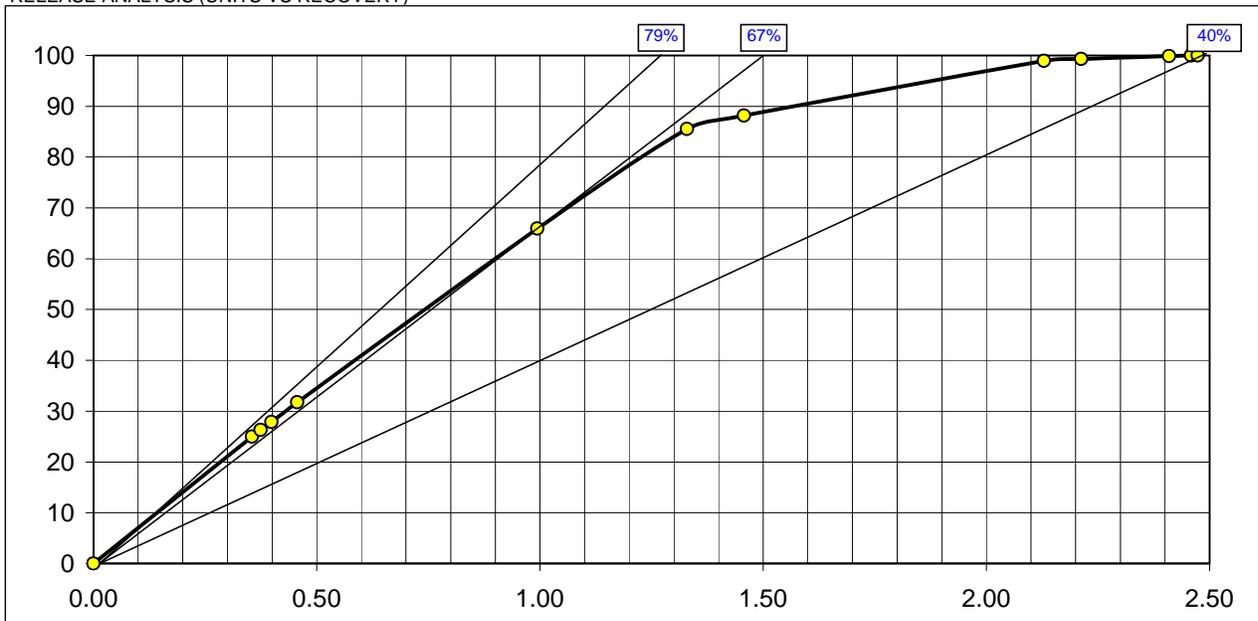
SAMPLE	tin concentrate
METHOD	stage separations to upgrade
NOTES	products arranged in order of tin grade

PROJECT	T0541
TEST NO	1
DATE	1/01/2010
TECHNICAL	ID

PRODUCT			TIN		SiO2		P2O5		S	
PRODUCT	WT (gm)	WT (%)	Sn(%)	DIST (%)	SiO2 (%)	DIST (%)	P2O5 (%)	DIST (%)	S (%)	DIST (%)
NM +212 CONC	2073.9	14.37	70.31	25.0	0.21	0.5	0.02	0.2	0.15	0.2
NM +300 CONC	110.1	0.76	69.01	1.3	0.61	0.1	0.03	0.0	0.37	0.0
NM +300 TAILS	140.0	0.97	66.42	1.6	1.12	0.2	0.03	0.0	0.57	0.1
NM +212 TAILS	340.6	2.36	66.07	3.9	0.64	0.3	0.04	0.1	0.52	0.1
NM-212 CONC	3136.1	21.74	63.54	34.2	1.23	4.7	0.53	6.5	0.90	1.9
NM-212 TAILS	1953.8	13.54	58.54	19.6	0.84	2.0	0.42	3.2	2.18	2.8
PNM +425	745.0	5.16	20.56	2.6	10.47	9.6	0.38	1.1	24.12	11.8
PNM GRAV TAIL	3920.0	27.17	16.06	10.8	16.04	77.3	2.46	37.8	19.54	50.4
PGC MAGS 2	486.5	3.37	4.77	0.4	1.59	1.0	20.38	38.8	0.36	0.1
PNM SULPHIDE CONC	1148.0	7.96	2.79	0.5	2.65	3.7	1.08	4.9	42.51	32.1
PRIMARY MAGS	286.0	1.98	2.20	0.1	1.31	0.5	3.03	3.4	2.25	0.4
PGC MAGS 1	87.6	0.61	0.54	0.0	1.02	0.1	11.98	4.1	0.75	0.0
CALC	14427.6	100.00	40.43	100.0	5.64	100.0	1.77	100.0	10.53	100.0

Wt (%)	WT (%)	Sn (%)	CUM Sn (%)	DIST (%)	CUM DIST	UNITS	CUM UNIT
NM +212 CONC	14.37	70.31	70.31	25.0	25.0	0.36	0.4
NM +300 CONC	0.76	69.01	70.24	1.3	26.3	0.02	0.4
NM +300 TAILS	0.97	66.42	70.01	1.6	27.9	0.02	0.4
NM +212 TAILS	2.36	66.07	69.51	3.9	31.8	0.06	0.5
NM-212 CONC	21.74	63.54	66.28	34.2	65.9	0.54	1.0
NM-212 TAILS	13.54	58.54	64.33	19.6	85.5	0.33	1.3
PNM +425	5.16	20.56	60.49	2.6	88.1	0.13	1.5
PNM GRAV TAIL	27.17	16.06	46.47	10.8	98.9	0.67	2.1
PGC MAGS 2	3.37	4.77	44.90	0.4	99.3	0.08	2.2
PNM SULPHIDE CONC	7.96	2.79	41.46	0.5	99.9	0.20	2.4
PRIMARY MAGS	1.98	2.20	40.68	0.1	100.0	0.05	2.5
PGC MAGS 1	0.61	0.54	40.43	0.0	100.0	0.02	2.5
	100.0	40.43		100.0			

RELEASE ANALYSIS (UNITS VS RECOVERY)

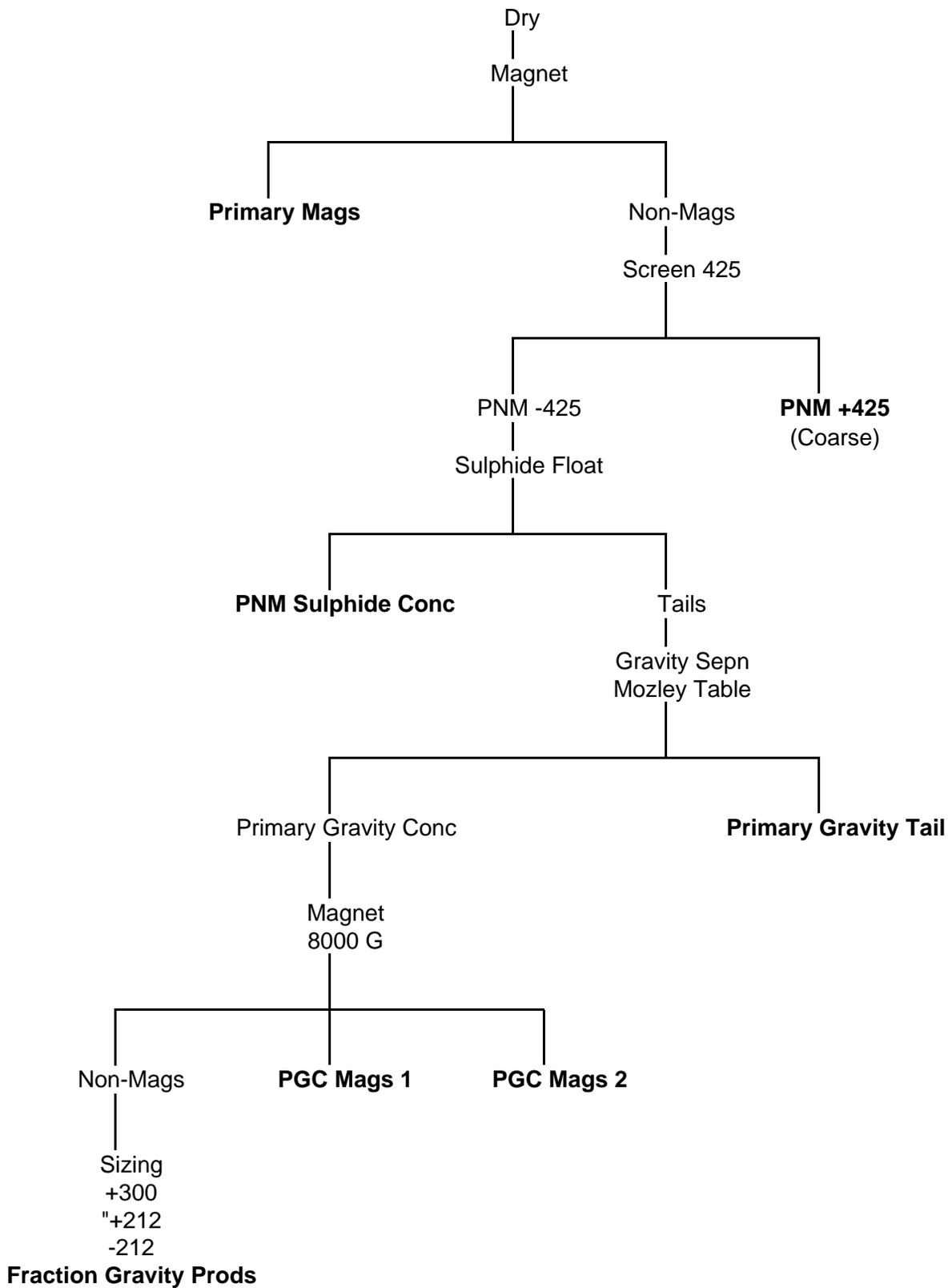


Burnie RESEARCH LABORATORY
TESTWORK TASK SHEET

GladstoneTin - Testwork

26/01/2010

T0541



GladstoneTin - Flowsheet

7/01/2010

T0541

