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WHEAL LUTWYCHE PTY. LTD.

EXPLORATION LICENCE 59/83

ANNUAL REPORT FOR THE YEAR ENDING 5th JUNE 1985.

AUTHORS: H. J. STACPOOLE
T. G. SUMMONS

DATE: 3rd MAY, 1985.

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SUMMARY

Investigations in EL 59/83 have resulted in the definition of a significant resource of fine tailings (slimes) at Aberfoyle, distributed over five dams, and totalling 198 000 tonnes (dry) grading 0.39% Sn, and 0.35% Wo_3. A less significant resource of coarse tailings amounts to 570 000 tonnes at a grade of 0.135% (Sn + Wo_3).

Cash flow studies of the c oarse tailings suggest that economic viability is not likely at present, principally due to the low head grade.

Research work has shown the fine tailings to be amenable to treatment to produce tin and tungsten concentrates with a 40% overall recovery.

Cash flow analysis of retreating these fine tailings indicates most encouraging rates of return, ranging from 16.3% to 36.0%. The most important factor controlling the construction of a fine tailings retreatment plant appears to be the granting of a tin production quota.

Reappraisal of the in situ (unmined) tin-tungsten mineralization in the Aberfoyle veinsystem suggests a potentially significant resource totalling approximately 14 000 000 tonnes of ore of imprecise grade, but which seems likely to be close to 0.20% CM (Sn + Wo_3).

1. INTRODUCTION

Exploration in Exploration Licence (EL) 59/83 during the twelve months ended 5. 6. 85 has included a range of activities both underground and on surface at the Aberfoyle tin - tungsten (Sn - W) deposit.

A considerable portion of the work to date has consisted of mineral processing research in to the tailings at Aberfoyle, with subsequent feasibility studies of these reserves. However, a surface core drilling program is currently in progress, as is the underground mapping of the upper levels of the old Aberfoyle Mine.

These latter two programs are designed to provide further data on the Aberfoyle vein system (unmined portion), and ultimately to permit a comprehensive assessment of its open pit potential.

2. ORE RESERVES

2. 1. FINE TAILINGS ("SLIME DAMS").

The fine tailings are situated in five dams to the north of the Aberfoyle main shaft (Figure 1); a total of thirty auger holes were drilled in dams 1, 2, 3, 4 and the "current" dam. These drill holes were intended as "in fill" holes between the earlier holes put down by Aberfoyle Ltd.

Each of the holes was assayed for Sn and W, following which the total samples from a given dam were bulked and reassayed for Sn and W.

The grade assigned to each dam was calculated by combining the earlier Aberfoyle Ltd. data with the present data (bulked assays).

The indicated ore reserve of these fine tailings is 198 000 tonnes (dry) grading 0.39% Sn, and 0.35% WO_3.

Details are shown in the table:

TABLE 1.

FINE TAILINGS RESERVES

Dam Number	Tonnes (dry)	Grade	
		Sn%	WO_3%
1	50 000	0.40	0.41
2	23 500	0.39	0.42
3	12 000	0.46	0.26
4	22 500	0.56	0.34
Current	90 000	0.33	0.40
TOTAL	198 000	0.39	0.35

2.2. COARSE TAILINGS

Previous work by Wheal Lutwyche showed the coarse (jig) tailings dump at Aberfoyle to contain 570 000 tonnes (dry) grading 0.135% CM (Sn + WO_3).

This reserve is also considered to have an indicated ore reserve status.

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006B

FIGURE. 1.

FINE TAILINGS DAMS
ROSSARDEN
EL 59/83

Sn	WO$_3$	
0.92	0.13	← HEAD GRADE (TOTAL)
0.96	0.20	← −20μ FRACTION.

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2.3. IN SITU (UNMINED) RESERVES

Previous work by Wheal Lutwyche on the unmined part of the Aberfoyle vein swarm (Summons 1983 a, b) has shown the dependence of ore grade on quartz grade and quartz volume.

Reappraisal of the parameters used earlier (Summons 1983 a) has changed the previous reserve estimate; over a 1000m strike length, including the 60m wide Aberfoyle vein swarm, down to a depth of 120m (4 level), and using the other values quoted by Summons (1983a), the inferred ore is estimated at 13.9×10^6 tonnes. This figure allows for the previous production of 0.25×10^6 tonnes above 4 level.

The production records for the Aberfoyle Mine indicate the grade of the "major veins" in the upper 4 levels was between 5% and 7% CM (i.e. quartz grades).

The present investigations are directed mainly at the "minor veins" (i.e. those veins $\leqslant$ 0.30m, and not previously mined), and especially at assessing the quartz volumes represented by these veins.

In summary, the inferred ore of 13.9×10^6 tonnes has an as yet undetermined grade, but which is regarded as lying between 0.10% and 0.35% CM (i.e. QG = 5% to 7%, QV = 2% to 5%, Sn/WO_3 = 10:1).

Detailed evaluation of the quartz volume (QV) is currently in progress via underground geological mapping, review of old drill logs etc., and surface core drilling.

3. MINERAL PROCESSING

3.1. FINE TAILINGS

3.1.1. Methods Used

A 1.7 tonne bulk sample was taken from the No.2 dam and despatched to the Department of Mines Launceston Laboratory for detailed testing.

The sample was screened in to +894µ, -894+315µ, -315+108µ, and -108µ size fractions.

The +894µ fraction was then rejected due to its very low Sn and W values, while the other fractions were concentrated using a range of equipment including screens, cyclones, flotation cells, both dry and wet magnetic separators, tables, vanners and spirals.

3.1.2. Results

3.1.2.1. -894+315µ Fraction

In this fraction, concentrates grading 63.4%Sn (1.1% overall), and 65.4% WO_3 (0.45% overall) were obtained. Overall values in the middlings and tailings products were respectively:

Sn 5.6%, 2.0%
WO_3 1.3%, 0.5%.

3.1.2.2. -315 +108µ Fraction

Concentrates in this fraction graded 67.6% Sn (1.91% overall) and 72.2% WO_3 (1.96% overall). However, overall values in the middlings, tailings and sulphide products were as follows:

Sn 3.3%, 5.1%, 8.3%
WO_3 3.7%, 1.2%, 1.0%.

3.1.2.3. -108µ Fraction

Concentrates in this fraction assayed 65.7% Sn (36.8% overall) and 65.1%WO_3 (35.0% overall). However, overall values in the middlings, tailings and sulphide products were as follows:

Sn 31.%, 21.4%, 7.7%
WO_3 11.8%, 36.6%, 3.2%.

Full details of this test work are contained in Appendix 1.

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3.1.3. SUMMARY

3.1.3.1. Concentrates

Total recovered Sn amounts to 39.8% of the total, this concentrate having a calculated value of 64.3% Sn; total recovered W amounts to 37.4% of the total, and having a calculated value of 67.4% WO_3.

3.1.3.2. Middlings

Approximately 12.0% of the Sn and 16.9% of the WO_3 occurs in the middlings products.

3.1.3.3. Tailings

Approximately 31.2% of the Sn, and 39.3% of the WO_3 was lost in the tailings reject, (including the +894μ oversize material).

3.1.3.4. Sulphides

Approximately 16.0% of the Sn, and 4.2% of the WO_3 is contained in/associated with the sulphide portion of the slimes; accordingly, sulphide retreatment may be necessary in any future plant, pending evaluation of the proportion of oxide to sulphide Sn.

3.1.4. COMMENTS

3.1.4.1. Grinding of the middlings, with an assumed 50% recovery would add a further 6% Sn and 8.5% WO_3 to the present overall recoveries.

3.1.4.2. The high Sn values recorded in the sulphides of the -315 +108μ fraction implies that grinding of the sulphides to -108 would improve the Sn recovery by approx. 4% overall. However, this figure may be as low as 1% if stannous sulphides are as high as 6%.

The similar Sn content (7.7%) in the -108μ fraction is notable.

3.1.4.3. Further Sn (4.7% of total Sn) may be recoverable from the +315μ size, and may boost overall recovery by 2%.

3.1.4.4. Thus, indicated recovery of Sn in the concentrates of 39.8% may be improved by 6 to 12% (to 46% and 52% overall) by further research in to the middlings, sulphides etc.

3.1.4.5. The low sulphur content of the +894μ fraction (reject) would render it suitable as a construction material.

3.1.4.6. Magnetic cassiterite is suspected in some of the dry magnetically separated gravity concentrates; however, its presence is apparently not significant as for example, in the -894 +315μ fraction, it only represents 0.02% of total Sn.

3.1.4.7. The sulphide concentrate in the -108μ fraction assayed 27.7%S, 4.4% Cu, 12.5% Zn and 580g/T of Ag; the marketability of such a concentrate has not been investigated in detail.

3.1.4.8. An independent review of the Department of Mines work by Bollen Metallurgical Services (Appendix 2) concluded that the aims of the test work had been achieved; specific observations included:

(i) The cycloning procedures on the -108μ fraction may not be as efficient in an operating plant, and that more than 21.4% of overall Sn may be lost in tailings.

(ii) The absence of analyses for penalty metals (As, Cu, Fe).

(iii) The possible unsuitability of the laboratory procedures for a commercial plant.

3.1.4.9. These observations have been acted upon, with H.J.Stacpoole taking the spiral tailings from the -108μ fraction to Cornwall (UK) for beneficiation on a Bartles Crossbelt Concentrator (Appendix 3). In addition, further work by Bollen Metallurgical Services has resulted in the stream lining of the original flow sheet devised by the Department of Mines; full details are given in Appendix 4.

3.2. COARSE TAILINGS

3.2.1. Method Used

The Department of Mines (Rhodes 1983) treated the material from two auger holes in to the coarse tailings dump by screening, rejection of a relatively barren oversize fraction (+4.75mm), and grinding of the undersize followed by concentration in jigs and spirals.

Although 11.8% of total Sn, and 5.8% of total WO_3 occurs in the -5.6mm +4.75mm fraction, due to a shortage of screen clothes, the work was based on the -4.75 mm fraction.

3.2.2. Results

The average distribution of Sn in the -4.75mm fraction was 73.2%, and for WO_3 it was 75.9%.

Overall recoveries of 51.0% for the Sn and 53.8% for the WO_3 were shown to be achievable at saleable grades, (representing 37.3% of total Sn, 40.8% of WO_3). The +4.75 mm fraction was regarded as suitable aggregate for concrete manufacture.

3.2.3. Comments

An independent review of this work by Bollen Metallurgical Services (Appendix 5), considered the overall recoveries to be 32.1% of total Sn, and 34.4% of total WO3, and that the test work was inadequate with regard to the calculated and assayed head grades.

4. GEOLOGICAL MAPPING

As indicated previously, underground geological mapping of the top four levels in the Aberfoyle Mine has been undertaken, and is still in progress.

Although the data has not been fully compiled, preliminary quartz vein widths and frequency suggest quartz volumes in the range 3% to 5%.

This range of values is considered encouraging, especially in combination with quartz grades of 5 to 7%.

5. SURVEYING

In preparation for the surface diamond drilling program (currently in progress), a detailed topographic survey was conducted over the Aberfoyle deposit by Peacock, Darcey and Anderson.

The data was presented in contoured form, and also relates the old Aberfoyle mine grid to the State (AMG) grid.

6. DRILLING

Preparatory to commencing the surface core drilling program, a review of previous cored drill holes (put in by Aberfoyle Ltd.) was made.

The down hole surveys indicated that the wireline drill holes flattened in the top 200m (downhole), the average rate of flattening being 2°/10m. ?

The azimuth deviation in the same holes was found to be 3°/20m. ?

The current drilling program has been designed to allow for these ground conditions, and thus allow the drilling to be as cost effective as possible.

7. FEASIBILITY STUDIES

7.1. FINE TAILINGS

The definition of indicated ore reserves for the fine tailings dams, in conjunction with detailed mineral processing research and flow sheet design, has led to preliminary studies of the economic viability of the retreatment project.

Financial calculations have been based on 15 tonnes/hour, single shift basis operation (28 000T/yr), with 35% to 40% recovery (of Sn and WO_3 combined), 90% realization after smelting charges, and 46% tax rate.

Details of the operating costs ($420 000/yr) are as follows:

Labour	- 7 men @ $40 000	...	$280 000
Plant operation	- $5/tonne	...	$140 000
	TOTAL		$ 420 000

The work force envisgaed would consist of the following personnel:

1 working supervisor
1 loader and carter
1 on feed preparation
1 on the table (Wilfley) and flotation unit
1 on the wet concentrator
1 on filtration/drying/drymagnetic separation
1 on maintenance (electrician).

Total operating costs are estimated to be $15/tonne.

Cash flow studies, based on tin price of $145/unit, tungsten price of $114/unit operating costs of $420,000/yr, and capitalization of $$1x10^6$ to $$1.5x10^6$, result in DCFROR values ranging from 16.3% to 36. %; full details are included in Appendix 6.

As indicated previously (Section 3), an overall recovery of 40% has been shown to be feasible, and may approach 50% pending certain factors outlined previously.

Accordingly, the project is viewed as most promising, and further detailed studies of reserves, processing etc. now await the granting of a tin production quota to this company.

7.2. COARSE TAILINGS

The coarse tailings retreatment project is based on an indicated ore reserve of 570 000 tonnes of material grading 0.135%CM, from which a recovery of 40% is achieved.

Likely capital and operating costs of mining, field preparation, HM cyclone plant (or grinding mill with jigs and spirals), and waste disposal are considered to render the project marginal at present.

However, the economics of the project would be profitable in the event of a large operation starting up to treat ore from the proposed open pit over the Aberfoyle vein system.

References

RHODES L.J., 1983: Tin and Tungsten distribution in the coarse tailings from the Aberfoyle Mine; Report R 829. Unpub. Dept. Mines Tas.

SUMMONS T.G., 1983(a) Proposed Aberfoyle Open Pit; Unpub. Rep. Wheal Lutwyche P/L.

" " 1983(b) Aberfoyle Deposit Quartz Grades; Unpub. Rep. Wheal Lutwyche P/L.

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14th November 1984

R833

WHEAL LUTWYCHE PTY LTD

Rossarden : No. 2 Tailings Dam Investigation

Introduction

This investigation is an extension of project R832 in which some preliminary metallurgical testing was carried out on Aberfoyle Rossarden tailings dam's samples submitted by Mr. H. Stacpoole. Investigation R832 indicated that Dam 2 appeared to have better economic prospects for retreatment than Dam 1.

The work undertaken in this investigation is thus limited to Dam 2 only, and was designed to extend and reinforce the previous possible recovery values obtained for both cassiterite and wolframite.

This report describes the treatment methods used and the results obtained in concentrating the Sn and WO_3 in the tailing without any further grinding.

Object

1. To recover 2 to 3 kg of market grade concentrates.
2. To compare the actual recovery of R833 with the predicted recovery of R832.

Sample

The sample, designated registered No. 830929 was stated to have been taken from around the edges of the Rossarden No. 2 Tailings Dam by means of a backhoe.

The tailings were received in six separate 44 gallon drums, were wet, and ranged from fine slime and sands to fairly coarse gravelly material.

The total dry weight of received tailings was found to be 1715 kg.

Chemical analysis of the tailings gave the following results:

1. 0.49% Sn
2. 0.42% WO_3
3. 3.63% S

A calculated sizing analysis, percent Sn and percent WO_3 distribution was found to be as follows:-

W

Table 1 Calculated Sizing Analysis Sn & WO_3 Distribution of Head.

Particle Size µm	Percent Mass		Percent Sn	Percent WO_3	Percent Distribution			
	Overall	Cum Overall			Sn		WO_3	
					Overall	Cum Overall	Overall	Cum Overall
+9530	0.02	0.02	0.09	0.02	0.10	0.10	0.03	0.03
+4760	0.61	0.63						
+4000	0.09	0.72	0.03	0.03	0.03	0.13	0.04	0.07
+3350	0.45	1.17						
+2800	0.48	1.65	0.17	0.07	0.15	0.28	0.08	0.15
+2360	0.48	2.13	0.04	0.03	0.04	0.32	0.03	0.18
+2000	0.79	2.92	0.08	0.04	0.11	0.43	0.07	0.25
+1700	1.03	3.95	0.09	0.04	0.16	0.59	0.09	0.34
+1400	1.86	5.81	0.11	0.02	0.37	0.96	0.08	0.42
+1180	2.02	7.83	0.10	0.02	0.36	1.32	0.09	0.51
+ 850	4.54	12.37	(0.10)	(0.02)	0.85	2.17	0.26	0.77
+ 600	4.80	17.17	(0.10)	(0.02)	1.00	3.17	0.25	1.02
+ 300	13.02	30.19	(0.19)	(0.04)	5.25	8.42	1.20	2.22
+ 150	13.49	43.68	(0.42)	(0.15)	11.98	20.40	4.39	6.61
+ 75	15.65	59.33	(0.34)	(0.14)	11.29	31.69	4.75	11.36
+ 38	11.77	71.10	(0.67)	0.55	16.74	48.43	14.03	25.39
C/S 1*	0.37	71.47	19.9	22.6	15.57	64.00	18.14	43.53
C/S 2	0.42	71.89	7.70	8.21	6.91	70.91	7.48	51.01
C/S 3	4.21	76.10	1.25	1.53	11.19	82.10	13.97	64.98
C/S 4	5.49	81.59	0.62	0.80	7.20	89.30	9.53	74.51
C/S 5	3.22	84.81	0.49	0.67	3.31	92.61	4.68	79.19
O/F	15.19	100	(0.23)	(0.63)	7.39	100	20.81	100
H	100		(0.47)	(0.46)	100		100	

*C/S Temperature 7.5°C

Method

1. A Sweco vibrating screen was used to screen the tailing into the following size fractions.

(a) + 894 µm
(b) - 894 µm + 315 µm
(c) - 315 µm + 108 µm
(d) - 108 µm

During each screening operation small amounts of undersize material were automatically sampled for examination and design of further treatment methods. Appendix I is a flowsheet of the distribution from screening.

2. The + 894 µm Fraction

The + 894 µm fraction was found to contain very low Sn and WO_3 values and thus received no further treatment being set aside as reject.

3. The - 894 µm + 315 µm Fraction.

The + 315 µm material was fed to a Vickers 5 turn series C C spiral at a rate of 0.7 tonnes per hour at 35 percent solids weight for weight. The resulting spiral concentrate was subjected to concentration by Deister table flotation. The reagents used for table flotation were,

(a) Sulphuric acid 1.5 kg/tonne
(b) Potassium ethyl xanthate 2.5 kg/tonne
(c) Kerosene 0.5 kg/tonne

The resulting table concentrate was dry magnetically separated with both the non magnetic product and weakly magnetic product being separately retabled.

The spiral middling was separately subjected to identical concentration and separation treatments as indicated for the spiral concentrate.

Appendix II is a flowsheet detailing the concentration of the - 894 µm + 315 µm fraction.

4. The -315 µm + 108 µm Fraction.

The + 108 µm material was fed to a Vickers 5 turn series C C spiral at a rate of 1.38 tonnes per hour at 35 percent solids weight for weight. The spiral concentrate product was subjected to batch froth flotation to facilitate sulphide removal. The flotation reagents used were as follows:

(a) Sulphuric acid 0.2 kg/tonne
(b) Copper sulphate 0.4 kg/tonne
(c) Sodium ethyl xanthate 0.15 kg/tonne
(d) Sodium amyl xanthate 0.15 kg/tonne
(e) Methylisobutyl carbinol (MIBC) 0.02 kg/tonne.

The primary flotation concentrate was subjected to cleaner flotation and then set aside. The flotation tailings from both flotation stages were bulked, table concentrated and the table concentrate dry magnetically separated.

The spiral middling product was separately subjected to identical concentration and separation processes as detailed for the spiral concentrate with the addition of a further table concentration operation for the final non magnetic product and weakly magnetic product.

Appendix III is a flowsheet detailing the concentration of the -315 μm + 108 μm fraction.

5. The - 108 μm Fraction.

(a) The -108 μm product was fed to a Warman 3" Series R cyclone fitted with a 23mm diameter vortex finder and a hydraulic spigot assembly of initial spigot diameter 14mm. The hydraulic spigot was set at fully closed. The -108 μm material was fed at 35 percent solids weight for weight, The cyclone conditions used for the bulk of material were chosen only after five small scale tests were performed. Details of the conditions used in the small scale tests are described in the text of the results obtained section.

The cyclone was fed by means of a 5.5 horsepower Warman 1½" B-M pump which delivered the cyclone underflow C (U/F) as a rope discharge. The resulting cyclone overflow (C O/F) was set aside as reject.

(b) The resulting cyclone underflow was subjected to batch sulphide flotation in a 10kg Agitair flotation cell. The following reagents were used.

(1)	Sulphuric acid	0.2 kg/tonne
(2)	Copper sulphate	0.4 kg/tonne
(3)	Sodium ethyl xanthate	0.15 kg/tonne
(4)	Sodium amyl xanthate	0.15 kg/tonne
(5)	Methylisobutylcarbinol (MIBC)	0.02 kg/tonne.

The resulting sulphide concentrate was subjected to a further cleaner flotation stage. The cleaner sulphide concentrate was set aside as a finished product, while the cleaner tailing was bulked with the primary flotation tailing ready for further treatment.

(c) The flotation tailing was fed to a Crockett wet magnetic separator to facilitate the removal of any strongly ferromagnetic material. The magnetic product was set aside as a finished product while the non magnetics were further treated.

(d) The non magnetic product was fed to a Vickers 5 turn series C C spiral at 1.43 tonnes per hour at 40 percent solids,

(e) The spiral concentrate was fed to a Deister table for gravity concentration. The table concentrate product was hand screened at 106 µm to remove any tramp material present and then dry magnetically separated. The non magnetic product was retabled to produce a tin concentrate while the weakly magnetic product was retabled and subjected to two further separate magnetic separations to produce a final first grade wolfram concentrate and several second grade wolfram concentrates.

(f) The spiral tailings S/10T were riffled into the following parts.

(1) ½ - for vanner concentration

(2) ¼ - for table concentration

(3) 3/16 - for Jones wet magnetic separation.

(4) 1/16 - for test work by the investigation originator.

Spiral tailings were separately concentrated on both a Deister table and a Fieldhouse vanner to allow a comparison of unit concentration performance. The vanner was fed at a rate of 0.36 tonnes per hour at 21 percent solids. The primary vanner concentrate (V1C) was returned to the vanner for further concentration after the vanner slope was increased. Vanner products were then set aside for assay as the vanner concentrate (V2C) was expected on further treatment, to react similarly to the table concentrate.

The table concentrate was separated on a dry magnetic separator with the resulting non magnetic product being further tabled to produce an Sn concentrate. The weakly magnetic product was also tabled and the resultant gravity concentrate dry magnetically separated to produce a WO_3 concentrate.

Three sixteenths of the spiral tailing product was riffled out and fed to a Jones wet magnetic separator with the electromagnet current set at 10 amps. The feed rate was set at 44 kg per hour at 20 percent solids. Wash water was maintained at 1.5 litres per minute and a magnetic scour water pressure of 140 kPa was used. The resulting non magnetic product was again returned to the separator with the electromagnetic current being increased to 18 amps. The feed rate was 32 kg per hour at approximately 20 percent solids. The resulting non magnetic product was again returned to the magnetic separator with the electromagnetic current being again increased, this time to 30 amps. The feed rate on this occasion was 31 kg per hour at 20 percent solids. The various Jones magnetic separator products were then set aside for assay.

Appendix IV and V are flowsheets of the concentration of the -108 µm material.

(g) Equipment used in this investigation was as follows:-

1. Sweco screen 450mm diameter.
2. Warman 3" cyclone.
3. Agitair 10 kg flotation cell.
4. Crockett (permanent magnet) wet magnetic separator.
5. Vickers 5 turn series C C spiral.
6. Deister laboratory table.
7. Rapid high intensity dry magnetic separator.
8. Jones wet magnetic separator.

Results Obtained.

Sweco Screening and Overall Mass Balance.

Table 2

	Percent				Percent Distribution		
Product	Mass	Sn	WO_3	S	Sn	WO_3	S
-108 µm	44.80	(0.76)	(0.84)	(3.14)	69.9	88.8	38.8
+108 µm	21.07	0.43	0.16	6.80	18.6	7.9	39.5
-315 µm	65.87	(0.66)	(0.63)	(4.31)	88.5	96.7	78.3
+315 µm	19.62	0.22	0.05	3.50	8.8	2.3	18.9
-894 µm	85.49	(0.56)	(0.50)	(4.12)	97.3	99.0	97.2
+894 µm	14.51	0.09	0.03	0.71	2.7	1.0	2.8
Head	100	(0.49)	(0.42)	(3.63)	100	100	100

Particle size, and Sn and WO_3 distributions of the Sweco screening products were as follows:

Table 3	+894 µm
Table 4	-894 µm + 315 µm
Table 5	-315 µm + 108 µm

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Table 3 +894 µm Sweco O/S

Particle Size µm	Percent Mass		Percent Sn	Percent WO_3	Percent Distribution			
					Sn		WO_3	
	Overall	Cum Overall			Overall	Cum Overall	Overall	Cum Overall
+9530	0.02	0.02	0.09	0.02	0.10	0.10	0.03	0.03
+4760	0.61	0.63						
+4000	0.09	0.72	0.03	0.03	0.03	0.13	0.04	0.07
+3350	0.45	1.17						
+2800	0.48	1.65	0.17	0.07	0.15	0.28	0.08	0.15
+2360	0.48	2.13	0.04	0.03	0.04	0.32	0.03	0.18
+2000	0.79	2.92	0.08	0.04	0.11	0.43	0.07	0.25
+1700	1.03	3.95	0.09	0.04	0.16	0.59	0.09	0.34
+1400	1.86	5.81	0.11	0.02	0.37	0.96	0.08	0.42
+1180	2.02	7.83	0.10	0.02	0.36	1.32	0.09	0.51
+1000	2.21	10.04	0.11	0.03	0.43	1.75	0.15	0.66
+ 850	2.24	12.28	0.10	0.02	0.40	2.15	0.10	0.76
+ 710	1.31	13.59	0.10	0.03	0.23	2.38	0.09	0.85
+ 600	0.53	14.12	0.11	0.03	0.11	2.49	0.04	0.89
+ 300	0.35	14.47	0.30	0.12	0.21	2.70	0.11	1.00
+ 150	0.02	14.49						
- 150	0.02	14.51						
H	14.51	—	(0.10)	(0.03)	2.7	—	1.00	—

Table 4 -894 µm + 315 µm Sweco O/S

Particle Size µm	Percent Mass		Percent Sn	Percent WO_3	Percent Distribution			
	Overall	Cum Overall			Sn		WO_3	
					Overall	Cum Overall	Overall	Cum Overall
+710	1.68	1.68	0.12	0.04	0.42	0.42	0.13	0.13
+600	2.81	4.49	0.12	0.03	0.72	1.14	0.17	0.30
+425	9.07	13.56	0.16	0.04	3.09	4.23	0.73	1.03
+300	5.53	19.09	0.32	0.08	3.76	7.99	0.88	1.91
+150	0.51	19.60	0.71	0.36	0.81	8.80	0.39	2.3
+ 75	0.01	19.61						
+ 38	0.01	19.62						
- 38	Trace	19.62						
H	19.62	—	(0.21)	(0.06)	8.8	—	2.3	—

Table 5 -315 µm + 108 µm Sweco O/S

Particle Size µm	Percent Mass		Percent Sn	Percent WO_3	Percent Distribution			
	Overall	Cum Overall			Sn Overall	Sn Cum Overall	WO_3 Overall	WO_3 Cum Overall
+300	0.16	0.16	0.34	0.09	1.38	1.38	0.40	0.40
+250	1.87	2.03						
+212	2.80	4.83	0.40	0.13	2.25	3.63	0.80	1.20
+180	2.70	7.53	0.47	0.18	2.55	6.18	1.07	2.27
+150	4.43	11.96	0.45	0.17	3.98	10.16	1.66	3.93
+106	6.27	18.23	0.43	0.15	5.39	15.55	2.07	6.00
+ 90	1.14	19.37	0.50	0.21	1.15	16.70	0.53	6.53
+ 75	0.89	20.26	0.40	0.17	0.71	17.41	0.34	6.87
+ 38	0.64	20.90	0.66	0.42	0.84	18.25	0.59	7.46
- 38	0.17	21.07	1.03	1.16	0.35	18.60	0.44	7.90
H	21.07		(0.44)	(0.17)	18.6	—	7.90	—

Concentration of the -894μm + 315μm Product

(a) Spiral Concentration

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
S/L 1 C	0.47	3.3	1.87	2.9	1.1
S/L 1 M	4.14	0.55	0.14	4.2	0.8
S/L 1 T	15.01	0.06	0.02	1.7	0.4
H +315 μm	19.62	(0.24)	(0.09)	8.8	2.3

(b) The spiral concentrate (S/L1C) was further concentrated on a Deister table and gave the following results after table flotation and gravity concentration.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
T 1 C	0.03	24.9	14.1	1.36	0.64
T 1 M	0.04	1.75	0.81	0.11	0.04
T 1 T	0.40	2.2	0.77	1.43	0.42
H S/L1C	0.47	3.8	1.72	2.90	1.10

The high T1T assay value of 2.2 percent Sn is due to the table flotation operation and the close association between the Sn and the sulphide minerals.

(c) The table middling (T1M) and tailing (T1T) products above were set aside as middling products requiring further grinding. The table concentrate (T1C) was dry magnetically separated as follows.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
M/S 1 N	0.02	39.5	0.43	1.27	0.01
M/S 1 M/A 2	0.01	(2.8)	(40.1)	0.05	0.55
M/S 1 M/A 1	Trace	6.6	14.0	0.04	0.08
H T1C	0.03	(24.5)	(14.0)	1.36	0.64

(d) The non magnetic product M/S 1 N was concentrated on a Deister table and gave the following result.

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
T 2 C	0.01	63.4	0.62	1.13	0.01
T 2 T	0.01	9.8	0.20	0.14	Trace
M/S 1 N	0.02	(39.5)	(0.43)	1.27	0.01

(e) The magnetic product M/S 1 M/A 2 was concentrated on a Deister table and gave the following result.

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
T 3 C	Trace	2.7	65.4	0.02	0.45
T 3 T	0.01	2.9	15.1	0.03	0.10
M/S 1 M/A 2	0.01	(2.8)	(40.1)	0.05	0.55

Note: The product M/S1 M/A1 was not further treated because of the small mass available.

(f) The spiral middling S/L1M was further concentrated on a Deister table and gave the following results after table flotation and gravity concentration.

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
T 4 C	0.52	0.77	0.37	0.76	0.35
T 4 M	1.04	0.09	0.04	0.18	0.08
T 4 T	2.58	0.67	0.08	3.26	0.37
S/L 1 M	4.14	(0.54)	(0.11)	4.20	0.80

The high Sn value of 0.67 in the tailing T4T is again due to table flotation and the close association of Sn and sulphine minerals. T4T has therefore been considered as a middling product. ? Stannite

(g) The table concentrate T4C from the spiral middling was subjected to magnetic separation and gave the following result.

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
M/S 2 N	0.42	(0.73)	(0.05)	0.60	0.04
M/S 2 M/A 2	0.04	(0.92)	(0.29)	0.07	0.02
M/S 2 M/A	0.06	0.78	2.9	0.09	0.29
H T 4 C	0.52	(0.75)	(0.37)	0.76	0.35

(h) The non magnetic product M/S 2 N was subjected to Deister table concentration and gave the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 5 C	0.04	6.4	0.23	0.50	0.02
T 5 T	0.38	0.13	0.03	0.10	0.02
H M/S 2 N	0.42	(0.73)	(0.05)	0.60	0.04

(i) The magnetic product M/S 2 M/A2 was subjected to Deister table concentration and gave the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 6 C	Trace	5.4	1.10	0.04	0.01
T 6 T	0.04	0.42	0.20	0.03	0.01
H M/S 2 M/A 2	0.04	(0.92)	(0.29)	0.07	0.02

The small quantity of mass available coupled with the relatively coarse particle size made this separation very difficult. For this reason, the similar magnetic product M/S2 M/A1 was not further gravity concentrated.

(j) A summary of the concentration of the +315µm product material can be deduced from separately combining the appropriate middling and tailing products previously listed and is as follows.

		Assay %		% Distribution		% Recovery Individual Fraction	
Product	% Mass	Sn	WO_3	Sn	WO_3	Sn	WO_3
Total Sn C	0.01	63.4	0.62	1.13	0.01	14.0	0.4
Total WO_3 C	Trace	2.7	65.4	0.02	0.45	0.3	21.3
Total Mid.	3.14	(0.99)	(0.27)	5.64	1.33	64.0	55.0
Total Tail.	16.47	(0.06)	(0.02)	2.01	0.51	21.7	23.3
H +315µm	19.62	(0.25)	(0.08)	8.80	2.30	100	100

From the above, it can be seen that 14 (overall 1.13) percent of the Sn, and 21.3 (overall 0.45) percent of the WO_3 have been separately recovered from the +315µm product material at grades of 63.4% Sn and 65.4% WO_3 respectively. The WO_3 concentrate was found to contain 2.7 percent Sn most of which is probably present as magnetic cassiterite.

The total +315µm middling product requiring regrinding, amounted to 3.1 percent of the original mass, and was found to contain

5.6 percent of the overall Sn and 1.3 percent of the overall WO_3 at grades of 0.99 percent Sn and 0.27 percent WO_3 respectively.

The total +315µm product material that could be rejected as tailings amounted to 16.5 percent of the original mass and contained 2.0 percent of the overall Sn and 0.5 percent of the overall WO_3 at grades of 0.06 and 0.02 percent respectively.

Concentration of the -315µm + 108µm Product

(a) Spiral concentration of the -315µm + 108µm product material gave the following results.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
S/L 2 C	0.19	7.3	7.6	3.1	3.0
S/L 2 M	5.50	0.88	0.34	10.7	3.9
S/L 2 T	15.38	0.14	0.03	4.8	1.0
H -315µm +108µm	21.07	(0.40)	(0.18)	18.6	7.9

(b) The spiral concentrate S/L 2 C was subjected to batch sulphide flotation results of which are as follows.

		Assay %			% Distribution	
Product	% Mass	Sn	WO_3	S	Sn	WO_3
F (1+2) T	0.04	(25.6)	(27.5)	-	2.13	2.33
F 2 C	0.15	2.8	1.9	31.5	0.97	0.67
H-S/L 2 C	0.19	(7.2)	(6.8)	-	3.10	3.00

(c) The flotation tailing F (1+2) T was concentrated on a Deister table giving the following results.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 7 C	0.03	(32.5)	(34.5)	2.07	2.24
T 7 T	0.01	2.8	4.4	0.06	0.09
H-F (1+2) T	0.04	(25.6)	(27.5)	2.13	2.33

(d) The table concentrate T7C was dry magnetically separated to give the following results.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
M/S 3 M/A 1	Trace	1.6	21.9	Trace	0.08
M/S 3 M/A 2	Trace	27.9	30.9	0.15	0.17
M/S 3 M/A 3	0.01	0.46	72.2	0.01	1.96
M/S 3 N	0.02	67.6	1.1	1.91	0.03
H- T7C	0.03	(32.5)	34.5	2.07	2.24

(e) The spiral middling was subjected to batch sulphide flotation, results of which are as follows:-

		Assay %			% Distribution	
Product	% Mass	Sn	WO_3	S	Sn	WO_3
F(3+4) T	3.47	0.46	0.51	-	3.38	3.61
F 4 C	2.03	1.70	0.07	38.2	7.32	0.29
S/L 2 M	5.50	(0.92)	(0.35)	-	10.70	3.90

(f) The flotation tailing F (3+4) T was concentrated on a Deister table, results of which are as follows.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 8 C	0.24	5.8	3.4	2.95	1.67
T 8 M	1.04	0.07	0.83	0.15	1.76
T 8 T	2.19	0.06	0.04	0.28	0.18
H-F(3+4) T	3.47	(0.46)	(0.51)	3.38	3.61

(g) Magnetic separation of the table concentrate T 8 C gave the following results:-

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
M/S 4 M/A 1	0.06	0.63	7.9	0.08	1.01
M/S 4 M/A 2	0.06	(2.2)	(4.6)	0.30	0.59
M/S 4 N	0.12	(10.5)	(0.28)	2.57	0.07
H-T 8 C	0.24	(5.8)	(3.4)	2.95)	1.67

(h) Deister table concentration of the non magnetic product M/S 4 N gave the following results.

Product	% Mass	Assay %		% Distribution	
		Sn	WO$_3$	Sn	WO$_3$
T 9 C	0.03	43.2	0.71	2.53	0.04
T 9 T	0.09	0.19	0.15	0.04	0.03
H M/S 4 N	0.12	(10.5)	(0.28)	2.57	0.07

(i) Deister table concentration of the magnetic product M/S 4 M/A 2 gave the following result.

Product	% Mass	Assay %		% Distribution	
		Sn	WO$_3$	Sn	WO$_3$
T 10 C	Trace	16.2	47.3	0.17	0.48
T 10 T	0.06	1.05	0.97	0.13	0.11
H-M/S 4 M/A 2	0.06	(2.2)	(4.6)	0.30	0.59

The magnetic product M/S 4 M/A 1 was not further concentrated due to its relatively small mass and the consequent difficulty that would occur in any treatment process. Product T10C likewise, was not subjected to the further magnetic separation that it required.

(j) A summary of the concentration of the -315µm + 108µm product material can be deducted by separately combining the appropriate middling and tailing products previously listed and is as follows:-

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3	% Recovery Individual Fraction Sn	% Recovery Individual Fraction WO_3
Total Sn Conc.	0.02	67.6	1.1	1.91	0.03	9.8	0.4
Total WO_3 "	0.01	0.46	72.2	0.01	1.96	Trace	23.1
Total Middlings	1.20	(1.28)	(1.51)	3.27	3.74	18.0	49.4
Total Tailings	17.66	(0.13)	(0.03)	5.12	1.21	26.9	15.3
Total Sulphides	2.18	(1.8)	(0.20)	8.29	0.96	45.3	11.8
H -315µm + 108µm	21.07	(0.41)	(0.17)	18.6	7.9	100	100

From these results it can be seen that 9.8 (overall 1.91) percent of the Sn and 23.1 (overall 1.96) percent of the WO_3 have been separately recovered from the -315µm + 108µm material at grades of 67.6 percent Sn and 72.2 percent WO_3 respectively. The WO_3 concentrate was found to contain 0.46 percent Sn while the Sn concentrate was found to contain 1.1 percent WO_3.

The total -315µm + 108µm middling product requiring regrinding, amounted to 1.20 percent of the original mass and was found to contain 3.27 percent of the overall Sn and 3.74 percent of the overall WO_3 at grades of 1.28 percent Sn and 1.51 percent WO_3 respectively.

17.66 percent of the original mass was rejected as tailings and found to contain 5.12 percent of the overall Sn and 1.21 percent of the overall WO_3 at values of 0.13 and 0.03 Sn and WO_3 respectively.

The sulphides, which amounted to 2.18 percent of the original mass, were found to contain 8.29 percent of the overall Sn and 0.96 percent of the overall WO_3 at values of 1.8 and 0.20 percent Sn and WO_3 respectively. -315µm + 108µm sulphide retreatment is seen as an essential operation.

Concentration of the -108μm Product

(a) Hydraulic cycloning of the -108μm product material was seen to be an essential pre-concentration stage. In order to achieve the maximum recovery of Sn and WO_3 in the cyclone underflow (C U/F) coupled with the maximum mass rejection in the cyclone overflow (C O/F), it was necessary to run a series of small scale tests to determine the required cyclone conditions.

The following cyclone conditions were kept constant for each test.

(a) 23mm diameter vortex finder.
(b) 5.5 Hp Warman 1½ B-M pump
(c) 35 percent solids.

Results of the small scale tests were as follows:-

Test No.	Fraction	Percent			Individual Percent Distribution		Cyclone Conditions		
		Mass Indiv.	Sn	WO_3	Sn	WO_3	Spigot Dia. mm	Hydraulic Valve Post.	Type of C U/F Dis
N 1	C U/F	80.0	0.87	1.00	91.8	87.3	18	Half open	Spray
	C O/F	20.0	0.31	0.58	8.2	12.7			
	H	100	(0.76)	(0.91)	100	100			
N 2	C U/F	66.8	1.02	1.20	87.6	82.3	18	Closed	Rope
	C O/F	33.2	0.29	0.52	12.4	17.7			
	H	100	(0.78)	(0.97)	100	100			
N 3	C U/F	64.5	1.01	1.20	85.9	80.1	14	Open	Rope
	C O/F	35.5	0.30	0.54	14.1	19.9			
	H	100	(0.76)	(0.97)	100	100			
N 4	C U/F	38.9	1.35	1.61	71.7	66.0	14	Closed	Rope
	C O/F	61.1	0.34	0.53	28.3	34.0			
	H	100	(0.73)	(0.95)	100	100			
N 5	C U/F	58.2	1.05	1.26	82.5	76.1	11	Open	Rope
	C O/F	41.8	0.31	0.55	17.5	23.9			
	H	100	(0.74)	(0.96)	100	100			

The cyclone conditions from test N 4 (namely, a 14mm spigot with the hydraulic valve closed) were chosen for the bulk pre concentration test as they gave the following results.

(a) 61.1 percent individual fraction mass rejection or 27.4 percent overall mass as cyclone overflow.

(b) 71.7 percent individual Sn recovery in the cyclone underflow or 49.7 percent overall.

(c) 66.0 percent individual WO_3 recovery in the cyclone underflow or 58.6 percent overall.

Any Sn or WO_3 passing into the C O/F was considered to be finer than cyclosizer fraction C/S 3 and thus be unrecoverable by conventional gravity concentration methods. A rope discharge from the cyclone was chosen such as to affect the maximum mass rejection as cyclone overflow material.

Treatment of the bulk of the -108µm product gave the following results.

Product	Percent Mass			Assay Percent		Percent Distribution					
	Individual Fraction		Overall	Sn	WO_3	Sn			WO_3		
	Sought	Actual				Individual Fraction		Overall	Indiv. Fraction		
						Sought	Actual		Sought	Actual	O'al
C U/F	38.9	42.4	18.98	(1.33)	(1.29)	71.7	74.2	51.9	66.0	64.6	57.4
C O/F	61.1	57.6	25.82	0.34	0.52	28.3	25.8	18.0	34.0	35.4	31.4
H -108µm	100	100	44.80	(0.76)	(0.84)	100	100	69.9	100	100	88.8

A particle size analysis, and a Sn, and a WO_3 distribution of the cyclone overflow product was found to be as follows:-

Table 7 Sweco - 108µm   C O/F

Particle	Percent Mass		Percent	Percent	Percent Distribution			
Size µm	Overall	Cum Overall	Sn	WO_3	Sn		WO_3	
					Overall	Cum Overall	Overall	Cum Overall
+106	0.01	0.01	0.20	0.01	0.29	0.29	0.02	0.02
+ 90	0.02	0.03						
+ 75	0.13	0.16						
+ 53	0.59	0.75						
+ 38	1.78	2.53	0.06	0.02	0.21	0.50	0.08	0.10
C/S1*	0.07	2.60	2.70	3.20	0.36	0.86	0.51	0.61
C/S2	0.34	2.94	0.51	0.61	0.33	1.19	0.47	1.08
C/S3	3.64	6.58	0.43	0.51	3.02	4.21	4.23	5.31
C/S4	4.45	11.03	0.47	0.65	4.03	8.24	6.59	11.90
C/S5	2.53	13.56	0.43	0.64	2.10	10.34	3.69	15.59
O/F	12.26	25.82	(0.32)	(0.57)	7.66	18.0	15.81	31.40
H	25.82	-	(0.36)	(0.53)	18.0	-	31.4	-

* C/S Temperature 20°C.

(b) Sulphide flotation of the -108µm C U/F gave the following overall results from one stage of sulphide concentrate cleaning.

Product	Percent				Percent Distribution		
	Mass	Sn	WO_3	S	Sn	WO_3	S
F(5+6) T	15.32	(1.41)	(1.51)	0.17	44.2	54.2	0.72
F 6 C	3.66	1.02	0.38	27.7	7.7	3.2	27.94
H-108µm C U/F	18.98	(1.33)	(1.29)	(5.48)	51.9	57.4	28.66

The sulphide concentrate contained 7.7 percent of the overall Sn and 3.2 percent of the overall WO_3 in a mass of 3.66 percent. Little work was done to establish optimum flotation conditions but the result obtained indicates, when compared with other Rossarden Aberfoyle sulphide samples, that the Sn and WO_3 losses were perhaps at a minimum. ? Hale.

Sample	Assay % Sn	Assay % WO_3
R833 F 6 C	1.02	0.38
792085	1.6	1.3
792086	1.3	0.89

The flotation concentrate was found to contain:-

(a) 27.7 percent sulphur
(b) 4.4 percent copper
(c) 12.5 percent zinc
(d) 580 gms/tonne silver.

It is anticipated that a market could be found for such a sulphide concentrate. The sulphur recovery from the -108µm C U/F was 97.5 percent.

(c) Wet magnetic separation of the flotation tailing F (5+6) T was performed to remove the ferrosilicon which was seen to be present so that it would not be present during later gravity concentration operations.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
M/S 6 M/A	0.09	1.17	1.85	0.2	0.4
M/S(5+6) N	15.23	1.41	1.51	44.0	53.8
F(5+6) T	15.32	(1.41)	(1.51)	44.2	54.2

The mass of magnetics removed was found to be 0.1 percent overall, and contain 0.2 percent of the overall Sn and 0.4 percent of the overall WO_3.

Some concentration of WO_3 did occur in the magnetics which assayed 1.85 percent. It is felt that a market could be found for a ferrosilicon by product.

(d) A summary of the results of the preliminary -108µm treatment is as follows:-

		Assay %			Percent Distribution		
Product	% Mass	Sn	WO_3	S	Sn	WO_3	S
M/S 6 M/A	0.09	1.17	1.85	0.23	0.2	0.4	Trace
M/S(5+6) N	15.23	1.41	1.51	0.17	44.0	53.8	0.7
F(5+6) T	15.32	(1.41)	(1.51)	(0.17)	44.2	54.2	0.7
F 6 C	3.66	1.02	0.38	27.7	7.7	3.2	28.0
C U/F	18.98	(1.33)	(1.29)	(5.48)	51.9	57.4	28.7
C O/F	25.82	0.34	0.52	1.42	18.0	31.4	10.1
H -108µm	44.80	(0.76)	(0.84)	(3.14)	69.9	88.8	38.8

A sizing analysis, and Sn and WO_3 distribution of the product M/S (5+6) N was found to be as follows:-

Table 8 -108µm M/S (5+6) N (or Spiral Feed)

Particle Size µm	Percent Mass		Percent Sn	Percent WO_3	Percent Distribution			
					Sn		WO_3	
	Overall	Cum Overall			Overall	Cum Overall	Overall	Cum Overall
+106	0.42	0.42	0.18	0.11	0.16	0.16	0.11	0.11
+ 90	0.84	1.26	0.12	0.06	0.21	0.37	0.11	0.22
+ 75	4.08	5.34	0.18	0.09	1.53	1.90	0.84	1.06
+ 63	1.15	6.49	0.35	0.19	0.84	2.74	0.50	1.56
+ 53	2.33	8.82	0.57	0.34	2.77	5.51	1.81	3.37
+ 45	2.11	10.93	1.37	0.95	6.04	11.55	4.57	7.94
+ 38	0.97	11.90	2.29	2.18	4.64	16.19	4.83	12.77
C/S1*	0.41	12.31	21.8	25.6	17.29	33.48	23.95	36.72
C/S2	0.46	12.77	4.50	5.2	4.00	37.48	5.46	42.18
C/S3	1.42	14.19	1.76	2.3	4.83	42.31	7.45	49.63
C/S4	0.48	14.67	1.18	1.57	1.10	43.41	1.72	51.35
C/S5	0.14	14.81	0.91	0.92	0.25	43.66	0.29	51.64
O/F	0.42	15.23	0.42	(2.3)	0.34	44.0	2.16	53.80
H	15.23	-	(1.38)	(1.55)	44.0	-	53.8	-

*C/S Temperature 20°C.

(e) Spiral Concentration.

The -108µm product M/S (5+6) N was fed to a spiral with the following result.

		Assay %		% Distribution	
Product	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
S/L 3 C	4.42	(2.44)	(2.51)	21.8	24.8
S/L 3 T	10.81	1.02	1.20	22.2	29.0
M/S (5+6) N	15.23	(1.43)	(1.58)	44.0	53.8
Assay		1.41	1.51		

The result obtained indicated that the -108µm material was unsuitable for spiral concentration.

(f) The spiral concentrate S/L 3C was tabled on the Deister table to give the following result.

		Assay %		% Distribution	
Product	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
T 11 C	0.57	18.0	18.3	20.7	23.3
T 11 M	0.31	0.36	0.42	0.2	0.3
T 11 T	3.54	0.12	0.15	0.9	1.2
S/L 3 C	4.42	(2.44)	(2.51)	21.8	24.8

(g) The table concentrate T 11 C was hand screened on a 106µm screen and the tramp oversize was set aside. The -106µm fraction was magnetically separated to give the following result:-

		Assay %		% Distribution	
Product	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
Tramp+106µm	0.04	1.55	0.86	0.1	0.1
M/S 7 M/A 1	0.09	0.34	5.9	0.1	1.2
M/S 7 M/A 2	0.21	(1.23)	(42.2)	0.5	21.0
M/S 7 N	0.23	(44.7)	(1.75	20.0	1.0
T 11 C	0.57	(18.9)	(17.5)	20.7	23.3
Assay		18.0	18.3		

(1) The non-magnetic product from the above magnetic separation M/S 7 was concentrated on the Deister table to give the following result:-

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 12 C	0.15	69.5	2.6	19.8	0.9
T 12 M	0.06	1.50	0.18	0.2	Trace
T 12 T	0.02	1.38	0.53	Trace	0.1
M/S 7 N	0.23	(44.7)	(1.75)	20.0	1.0

(i) The magnetic product M/S 7 M/A 2 was tabled on the Deister table to give the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 13 C	0.13	(1.82)	(70.8)	0.4	20.8
T 13 M	0.06	0.44	1.20	0.1	0.2
T 13 T	0.02	0.27	1.50	Trace	Trace
M/S 7 M/A 2	0.21	(1.23)	(42.2)	0.5	21.0

(j) This table concentrate T 13C was again magnetically separated with the following result:-

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
M/S 8 M/A 1	Trace	1.10	35.3	Trace	0.1
M/S 8 M/A 2	0.01	22.3	38.2	0.3	0.6
M/S 8 M/A 3	0.12	0.61	73.0	0.1	20.1
T 13 C	0.13	(1.82)	(70.8)	0.4	20.8

(k) The magnetic product M/S 8 M/A 3 was again magnetically separately with the following result

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
M/S9(M/A2+N)	0.01	(1.96)	(65.8)	Trace	1.8
M/S 9 M/A	0.11	0.46	73.8	0.1	18.3
H - M/S 8 M/A 3	0.12	0.61	73.0	0.1	20.1

(1) The tailing from the spiral concentration test contained more than half of the tin and WO_3 that was present in the feed. Washing some of the tailing with a vanning plaque showed that recoverable tin was still present.

Accordingly the spiral tailing S/L 3 T was dried and riffled to provide a half sample a quarter sample, a three sixteenth sample and a one sixteenth sample.

A one quarter sample was concentrated on the Deister table to give the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 14 C	0.83	(11.3)	(10.1)	18.2	22.5
T 14 M	3.49	0.22	0.27	1.5	2.5
T 14 T	6.49	0.20	0.23	2.5	4.0
S/L 3 T	10.81	(1.06)	(1.00)	22.2	29.0

(m) The table concentrate (T 14 C) was dried and magnetically separated to give the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
M/S 10 M/A1	0.24	0.25	8.9	0.1	5.7
M/S 10 M/A2	0.30	(1.12)	(19.8)	0.7	15.9
M/S 10 N	0.29	(30.6)	(1.18)	17.4	0.9

(n) The non magnetic product M/S 10 N from this separation was tabled on a Deister table to give the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 15 C	0.14	61.4	2.3	17.0	0.9
T 15 T	0.15	1.33	0.11	0.4	Trace
H-M/S 10 N	0.29	(30.6)	(1.18)	17.4	0.9

(o) The magnetic fraction M/S 10 M/A 2 was tabled on the Deister table and gave the following result.

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
T 16 C	0.10	(2.7)	(55.6)	0.6	14.9
T 16 T	0.20	0.28	1.9	0.1	1.0
H-M/S 10M/A2	0.30	(1.12)	(19.8)	0.7	15.9

(p) This table concentrate T 16 C was magnetically separated as follows:-

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
M/S 11 M/A1	0.01	0.51	43.6	Trace	0.7
M/S 11 M/A2	0.04	4.7	58.3	0.4	6.2
M/S 11 M/A3	0.05	1.46	59.7	0.2	8.0
H-T 16 C	0.10	(2.73)	(55.6)	0.6	14.9

(q) The half sample of the spiral tailing was treated on the Fieldhouse vanner. The feed rate to the vanner was 0.36 t/h at 21% solids. On the first pass over the vanner, too much concentrate was produced. The slope of the vanner deck was increased and the vanner concentrate was fed over the vanner a second time with the following result:-

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
V 2 C	0.53	15.9	17.3	17.6	23.7
V 2 T	7.31	0.22	0.19	3.4	3.6
V 1 C	7.84	(1.28)	(1.35)	21.0	27.3
V 1 T	2.97	0.20	0.22	1.2	1.7
S/L 3 T	10.81	(0.98)	(1.04)	22.2	29.0
Assay		1.02	1.20		

(r) 30kg of the -108µm spiral tailing product were riffled out and fed to a Jones wet magnetic separator. The electromagnet current was set at 10 amps. The resulting non-magnetic product was returned to the separator and the electromagnet current was increased to 18 amps. The non magnetics were again returned to the separator with the electromagnet current being finally set at 30 amps. The following results were obtained.

Product	% Mass	Assay %		Electro Magnet Current Amps	% Distribution	
		Sn	WO_3		Sn	WO_3
M/S 14 N	8.79	1.07	0.05	30	20.7	1.1
M/S 14 M/A	0.71	0.35	0.84	30	0.6	1.5
M/S 13 N	9.50	(1.02)	(0.11)	18	21.3	2.6
M/S 13 M/A	0.34	0.42	5.0	18	0.3	4.1
M/S 12 N	9.84	(1.00)	(0.28)	10	21.6	6.7
M/S 12 M/A	0.97	0.29	9.4	10	0.6	22.3
H -S/L 3 T	10.81	(0.93)	(1.09)		22.2	29.0

The results obtained are inconclusive and no definite comment on the test can be made.

(s) A summary of the concentration of the -108µm product material can be deduced from separately combining the appropriate concentrates, middlings and tailings previously listed. The vanner concentration test results and the Jones wet magnetic separation results are not included in the summary which is as follows:-

Product	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
T 12 C	0.15	69.5	2.6	19.8	0.9
T 15 C	0.14	61.4	2.3	17.0	0.9
Final Sn Concentrate	0.29	(65.7)	(2.5)	36.8	1.8
M/S 9 M/A WO_3 Concentrate (Firsts)	0.11	0.46	73.8	0.1	18.3

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
M/S 9 (M/A2 + N)	0.01	(1.96)	(65.8)	Trace	1.8
M/S 11 M/A 2	0.04	4.7	58.3	0.4	6.2
M/S 11 M/A 1	0.01	0.51	43.6	Trace	0.7
M/S 11 M/A 3	0.05 7	1.46	59.7	0.2	8.0
WO_3 Conc. (Seconds)	0.11	(2.8)	(57.8)	0.6	16.7
WO_3 " (Firsts)	0.11	0.46	73.8	0.1	18.3
Total WO_3 Conc.	0.22	(1.60)	(65.1)	0.7	35.0

Compiling middling and tailing figures in the same manner as concentrate figures, the final summary becomes.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
Total Sn Conc.	0.29	(65.7)	(2.5)	36.8	1.8
Total WO_3 "	0.22	(1.60)	(65.1)	0.7	35.0
Total Middlings	4.69	(0.34)	(0.97)	3.1	11.8
Total Tailings	10.03	(0.17)	(0.20)	3.4	5.2
C O/F	25.82	0.34	0.52	18.0	31.4
Total Flotation Conc	3.66	1.02	0.38	7.7	3.2
"Ferro Silicon	0.09	1.17	1.85	0.2	0.4
H - 108µm	44.80	(0.79)	(0.82)	69.9	88.8

These figures indicate that 36.8 percent of the overall Sn (or 53.9 percent of the Sn in the -108µm product) has been recovered at a grade of 65.7 percent Sn. Similarly 35.0 percent of the overall WO_3 (or 39.1 percent of the WO_3 in the -108µm product) has been recovered at a grade of 65.1 percent WO_3. The total middling was found to contain 3.1 percent of the overall Sn and 11.8 percent of the overall WO_3 at grade of 0.34 percent Sn and 0.97 percent WO_3 in 4.69 percent of the original mass.

The total amount of material that could be rejected as tailings (gravity tailings plus cyclone overflow) was found to be 35.85 percent of the overall mass, contain 21.4 percent of the overall Sn, 36.6 percent of the overall WO_3 at a grade of 0.29 percent Sn and 0.43 percent WO_3.

(t) A comparison between Fieldhouse Vanner and Deister table gravity concentration of the spiral tailing (S/L 3 T) was as follows:-

		Assay %		% Distribution		Remarks
Product	% Mass	Sn	WO_3	Sn	WO_3	
V 2 C	0.53	15.9	17.3	17.6	23.7	Concentrate
T 14 C	0.83	(11.3)	(10.1)	18.2	22.5	Concentrate
V 2 T	7.31	0.22	0.19	3.4	3.6	Middling/Tail.
T 14 M	3.49	0.22	0.27	1.5	2.5	Middling/Tail.
V 1 T	2.97	0.20	0.22	1.2	1.7	Tailing
T 14 T	6.49	0.20	0.23	2.5	4.0	Tailing
or combining middlings with tailings.						
V 1 T+V2T	10.28	(0.21)	(0.20)	4.6	5.3	Tailing
T14T +T14M	9.98	(0.21)	(0.24)	4.0	6.5	Tailing

(1) Sn Recovery

The vanner and table gravity concentrating units have given very similar concentration performances for the spiral tailing, with overall recoveries of 17.6 and 18.2 percent respectively at grades of 15.9 and 11.3 percent Sn respectively.

(2) WO_3 Recovery

Separate Vanner and table concentration of the spiral tailing gave overall recoveries of 23.7 and 22.5 percent respectively at grades of 17.3 and 10.1 percent WO_3 respectively.

Overall Concentration Summary

(a) Concentrates

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
+315µm Sn Conc.	0.01	63.4	0.62	1.13	0.01
+108µm Sn Conc.	0.02	67.6	1.1	1.91	0.03
-108µm Sn Conc.	0.29	(65.7)	(2.5)	36.80	1.80
Total Sn Conc.	0.32	(64.3)	(2.3)	39.84	1.84
+315µm WO_3 Conc.	Trace	2.7	65.4	0.02	0.45
+108µm WO_3 Conc.	0.01	0.46	72.2	0.01	1.96
-108µm WO_3 Conc.	0.22	(1.60)	(65.1)	0.70	35.00
Total WO_3 Conc.	0.23	(1.6)	(67.4)	0.73	37.41

(b) Middlings

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
+315µm Middlings	3.14	(0.99)	(0.27)	5.64	1.33
+108µm Middlings	1.20	(1.28)	(1.51)	3.27	3.74
-108µm Middlings	4.69	(0.34)	(0.97)	3.10	11.80
Total Middlings	9.03	(0.69)	(0.80)	12.01	16.87

(c) Tailings

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
+315µm Tailings	16.47	0.06	0.02	2.01	0.51
+108µm Tailings	17.66	0.13	0.03	5.12	1.21
-108µm Tailings	10.03	(0.17)	(0.20)	3.40	5.20
Total Tailings	44.16	(0.12)	(0.07)	10.53	6.92
-108µm C O/F	25.82	0.34	0.52	18.00	31.40
+ 895µm O/S Reject	14.51	(0.09)	0.03	2.70	1.00
Total Tailing Reject	84.49	(0.18)	(0.20)	31.23	39.32

(d) Sulphides

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
-895+315µm	Nil	(Table Flotation)			
-315+108µm	2.18	(1.8)	(0.20)	8.29	0.96
-108 µm	3.66	1.02	0.38	7.70	3.20
Total Sulphides	5.84	(1.30)	(0.31)	15.99	4.16

(e) Overall Balance

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3	
Total Sn Conc.	0.32	(64.3)	(2.3)	39.84	1.84	
Total WO_3 Conc.	0.23	(1.6)	(67.4)	0.73	37.41	1.06
Total Middlings	9.03	(0.69)	(0.80)	12.01	16.87	0.71
Total Tailings	84.49	(0.18)	(0.20)	31.23	39.32	0.79
Total Sulphides	5.84	(1.30)	(0.31)	15.99	4.16	
-108µm Ferrosilicon	0.09	1.17	1.85	0.20	0.40	
Head	100	(0.50)	(0.42)	100	100	0.88 ave

/
3.5% Sn

The concentration techniques adopted have enabled

(1) The overall recovery of 39.8%of the Sn present in the tailing at a marketable concentrate grade of 64.3 percent Sn in 0.32 percent of the original mass.

(2) The overall recovery of 37.4 percent of the WO_3 in the tailing at a marketable concentrate grade of 67.4 percent WO_3 in 0.23 percent of the original mass.

(3) Production of middling material amounting to 9.0 percent of the original mass, containing 12.0 percent of the overall Sn and 16.9 percent of the overall WO_3 at values of 0.69 percent Sn and 0.80 percent WO_3 respectively.

(4) The rejection of 84.5 percent of the original mass as tailings containing 31.2 percent of the Sn and 39.3 percent of the WO_3 at assay values of 0.18 percent and 0.20 percent Sn and WO_3 respectively.

(5) The flotation as sulphides of 5.8 percent of the original mass containing 16.0 percent of the overall Sn and 4.2 percent of the overall WO_3 at assay values of 1.3 percent Sn and 0.31 percent WO_3 respectively.

(6) The production of a ferrosilicon concentrate amounting to approximately 0.1 percent of the original mass.

Conclusion

1. Recovery of Sn & WO_3

An overall recovery of 39.8 percent of the Sn at a grade of 64.3 percent Sn was obtained from the tailing. A similar recovery of 37.4 percent was achieved for the WO_3 at a grade of 67.4 percent WO_3.

Both of these recovery figures from the retreatment of the tailing sample, do not include any Sn or WO_3 values derived from grinding of middling products as this was not performed. Middling products however, amounted to 9 percent of the original mass, and contained, 12.0 percent of the overall Sn, and 16.9 percent of the overall WO_3 at values of 0.69 percent Sn and 0.80 WO_3 respectively. Not all the Sn and WO_3 in the middling products could be expected to be recovered after grinding. If a conservative middling metal value recovery of 50 percent is assumed, then the overall recovery could be enhanced by 6 and 8 percent Sn and WO_3 respectively. That is, a final overall recovery value of 45 percent for both Sn and WO_3 is quite feasible and realistic if grinding is included in the retreatment process. Investigation report R832. estimated a recovery value of approximately 50 percent for both Sn and WO_3 on sample 830791, also from Dam 2.

The close association between the cassiterite and the sulphide minerals present in the tailing has decreased the Sn recovery obtained. Any retreatment process should include the grinding of sulphides to -108µm. The +108µm cleaner sulphide concentrate F 2 C was found to contain 1 percent of the overall Sn (0.67 percent WO_3) in 0.15 percent of the original mass at an assay value of 2.8 percent Sn. Similarly the +108µm cleaner sulphide concentrate F 4 C was found to contain 7.3 percent of the overall Sn (0.29 percent WO_3) in 2.0 percent of the original mass at an assay value of 1.7 percent Sn. Grinding of +108µm sulphides to -108µm and retreatment could therefore enhance the overall Sn recovery by approximately half the available 8.3 percent. Grinding of -315µm + 108µm sulphides would thus bring the overall Sn recovery value to approximately 49 percent. 4.7 percent of the overall Sn was also found in the table flotation tailings (T 1 T and T 4 T) of the +315µm spiral concentrate and spiral middling products. The products T 1 T & T 4 T were however compiled as middlings in this report and thus do not report as sulphides.

8.3%

2. Rejection of +895µm Material

The +895µm product material contained very low metal values namely 0.09 percent Sn and 0.03 percent WO_3. Grinding of this material would not be an economic proposition and it should therefore be excluded from further consideration. In rejecting the +894µm material, 2.7 percent of the overall Sn and 1.0 percent of the WO_3 are lost in 14.5 percent of the original mass. The low sulphur content of the +894µm product should however make it suitable for use as a construction gravel.

3. Spiral Operations

(a) The Vickers spiral proved to be an ideal pre-concentration unit for both the -894µm + 315µm fraction and the -315µm + 108µm fraction with tailing rejection being as follows.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
-894µm+315µm S/L 1 T	15.01	0.06	0.02	1.7	0.4
-315µm+108µm S/L 2 T	15.38	0.14	0.03	4.8	1.0
Total +108µm S/L T	30.39	(0.10)	(0.03)	6.5	1.4

That is, by using the spiral a bulk spiral tailing amounting to 30.4 percent of the original mass could be rejected, as relatively barren material containing only 6.5 percent of the overall Sn and 1.4 percent of the overall WO_3 at assays values of 0.10 percent Sn and 0.03 percent WO_3 respectively.

(b) The Vickers spiral was unsuitable for concentration of the -108µm M/S (5+6) N product material and probably acted more as a classifier or sizing device than a concentrator.

4. -108µm Hydraulic Cyclone Operation.

The hydraulic cycloning of the -108µm material produced a reject cyclone overflow of 25.8 percent of the original mass containing 18.0 percent of the overall Sn and 31.4 percent of the overall WO_3 at grades of 0.34 Sn and 0.52 percent WO_3 respectively. Only 1.2 percent of the overall 18.0 percent Sn and 1.1 percent of the overall 31.4 percent WO_3 in the cyclone overflow was found to be coarser than cyclosizer fraction C/S 3 and thus recoverable by conventional gravity concentration methods. Thus, the cyclone operation was found to be an ideal method of pre-concentrating the -108µm material, increasing the head values of 0.76 percent Sn and 0.84 percent WO_3 to 1.33 percent Sn and 1.29 percent WO_3 as well as rejecting 25.8 percent of the original mass as slime material.

5. The Presence of Magnetic Cassiterite

Some of the table concentrates subjected to dry magnetic separation gave weakly magnetic products containing significant quantities of Sn. Whilst it may be possible to dismiss some of the Sn in the magnetics as entrainment, or magnetic composites it is reasonable to assume that a proportion of the Sn is present as magnetic cassiterite. The very nature of the tabular monoclinic wolframite compared to the pyramidal tetragonal cassiterite means that in general, the cassiterite particle will usually be closer to the magnet of a belt magnetic separator during separation.

Jones wet magnetic separation was employed on a 30kg sample of -108µm spiral tailing S/L3T to examine the presence of magnetic cassiterite.

The result however was inconclusive.

Summary

1. The following Sn Concentrates were produced.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
-894µm+315µm T 2 C	0.01	63.4	0.62	1.13	0.01
-315µm+108µm M/S 3 N	0.02	67.6	1.1	1.91	0.03
-108µm T 12 C	0.15	69.5	2.6	19.80	0.90
-108µm T 15 C	0.14	61.4	2.3	17.00	0.90
Total Sn Conc.	0.32	(64.3)	(2.3)	39.84	1.84

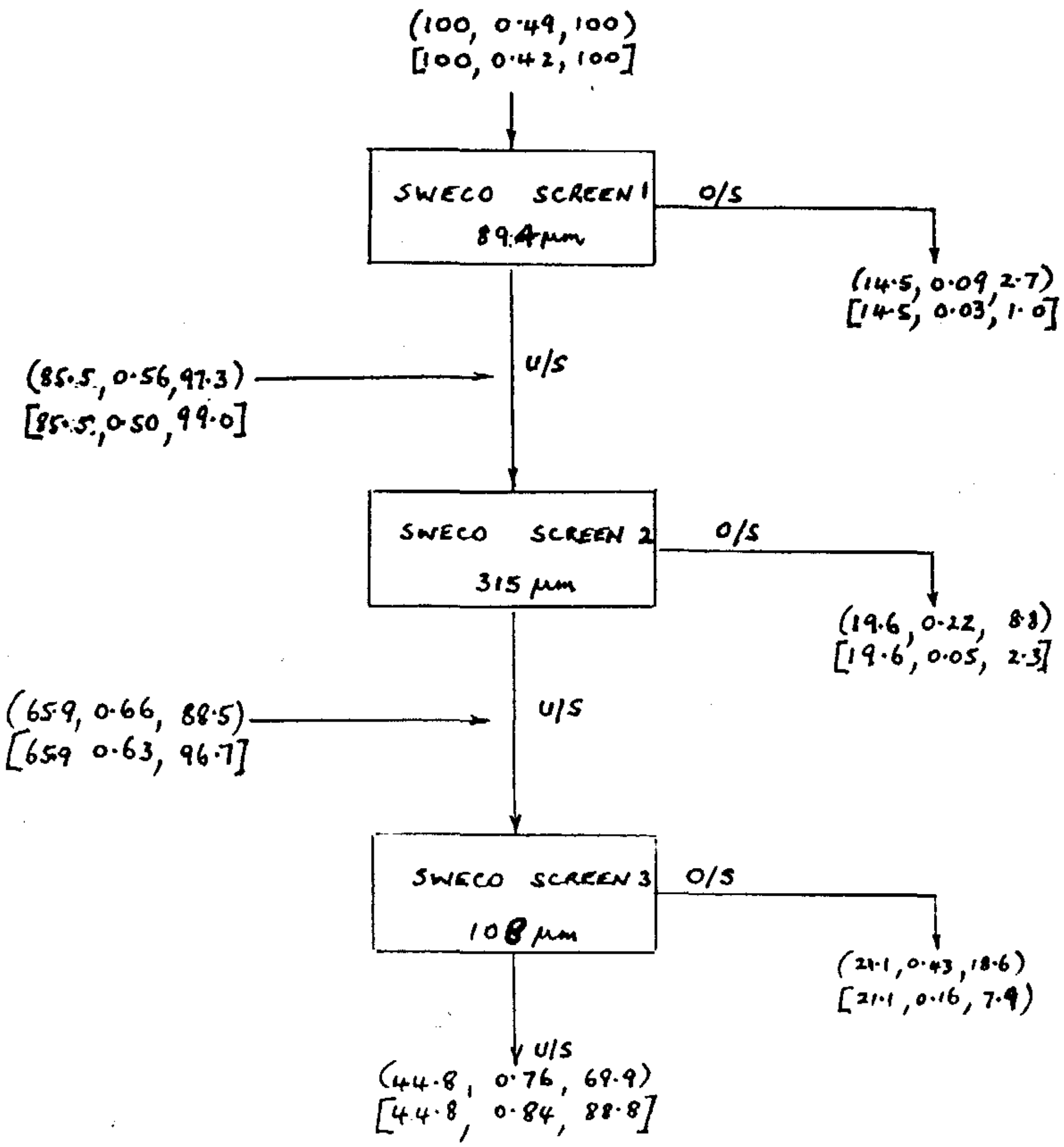
2. The following WO_3 Concentrates were produced.

Product	% Mass	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
-894µm+315µm T 3 C	Trace	2.7	65.4	0.02	0.45
=315µm+108µm M/S3 M/A3	0.01	0.46	72.2	0.01	1.96
-108µm M/S9 M/A1	0.11	0.46	73.8	0.10	18.30
-108µm M/S9 (M/A2+N)	0.01	1.96	65.8	Trace	1.80
-108µm M/S11 M/A2	0.04	4.7	58.3	0.40	6.20
-108µm M/S11 M/A1	0.01	0.51	43.6	Trace	0.7
-108µm M/S11 M/A3	0.05	1.46	59.7	0.20	8.00
Total WO_3 Conc.	0.23	(1.6)	(67.4)	0.73	37.41

Metallurgist..K.Austin...

(H. K. Wellington)
Chief Chemist & Metallurgist

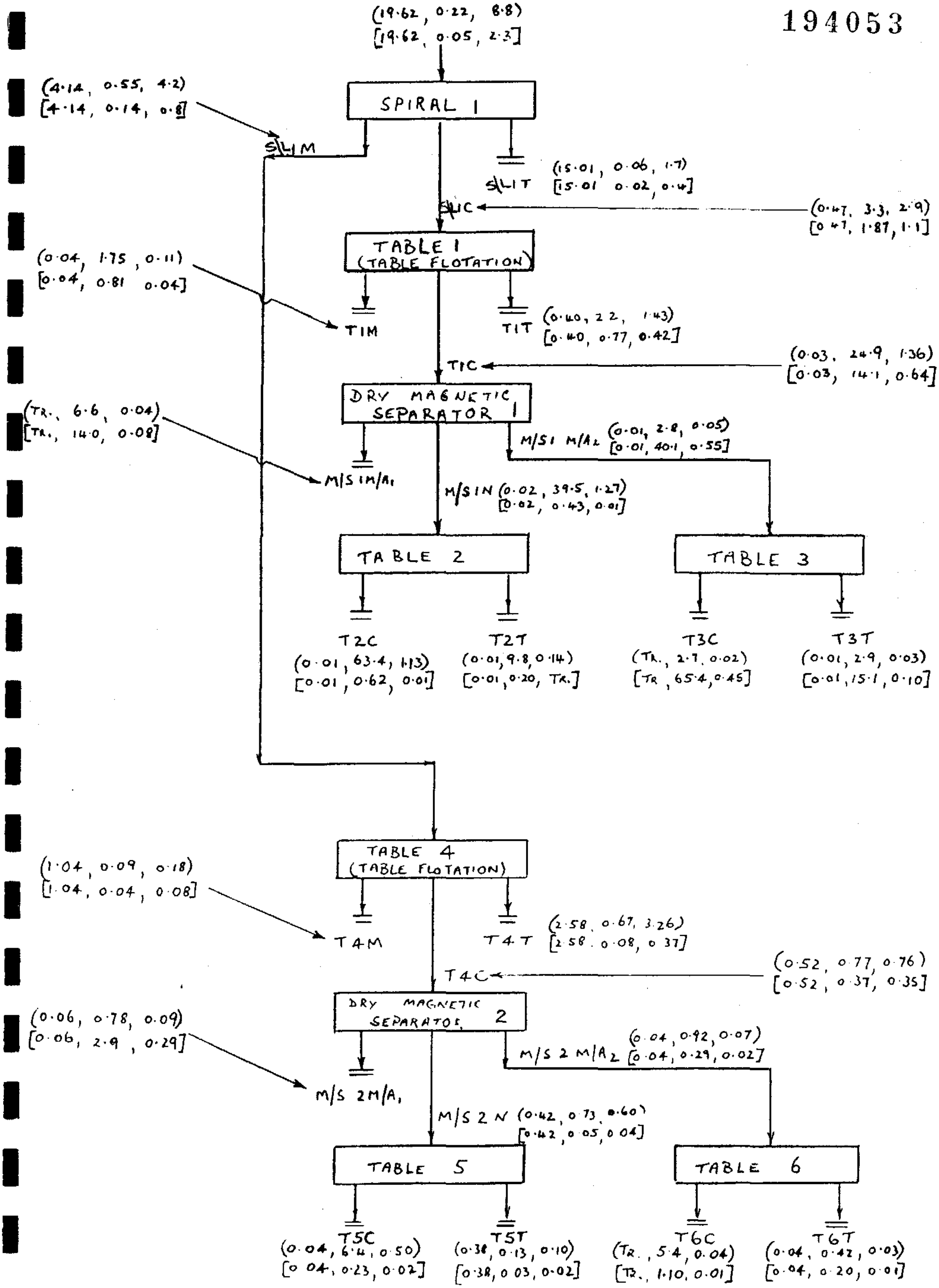
APPENDIX I FLOW SHEET - SWECO SCREENING



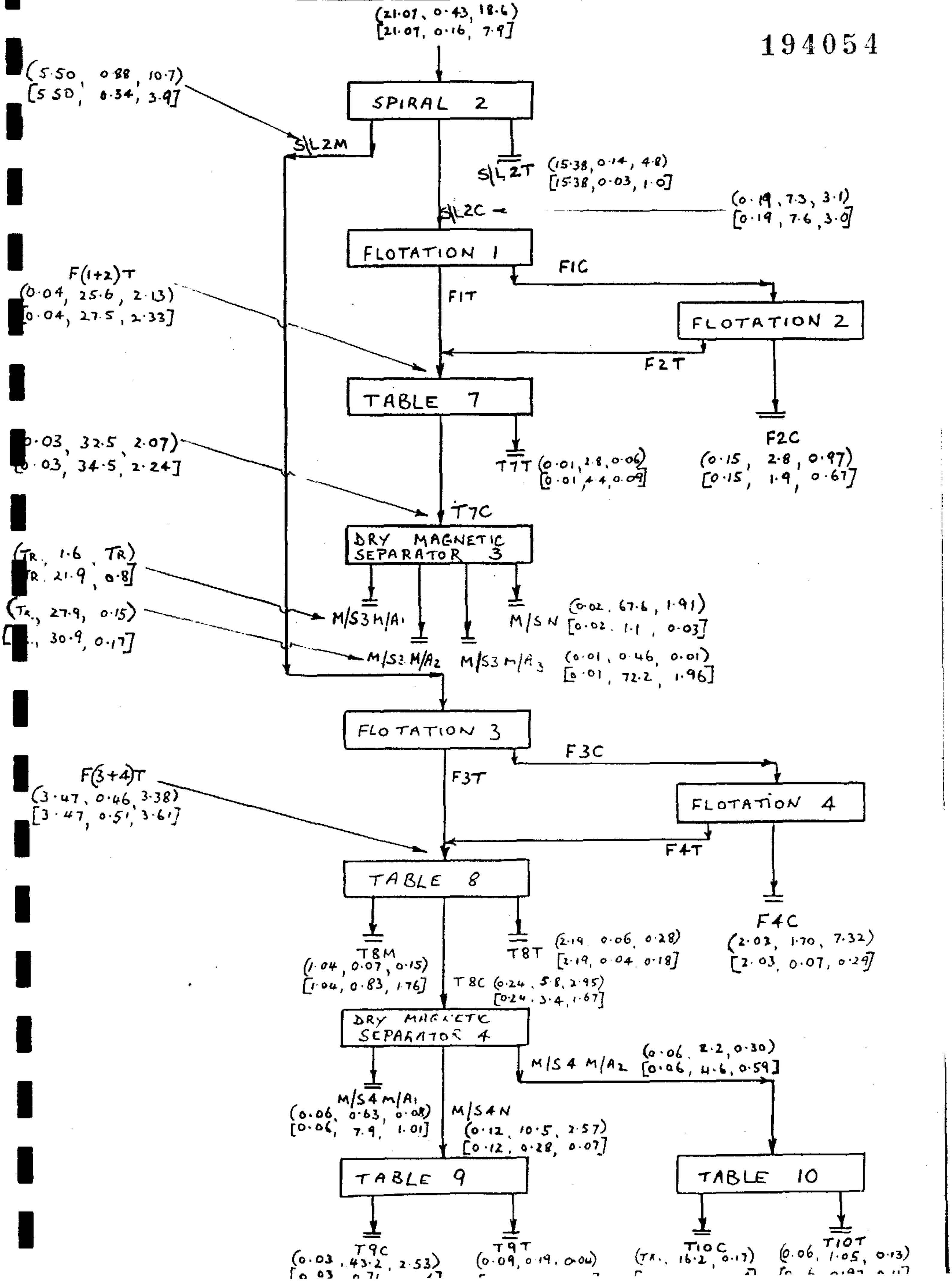
(a, b, c) TR = TRACE
[a, d, e]
a = PERCENT MASS
b = PERCENT Sn
c = PERCENT Sn DISTRIBUTION
d = PERCENT WO_3
e = PERCENT WO_3 DISTRIBUTION

APPENDIX II

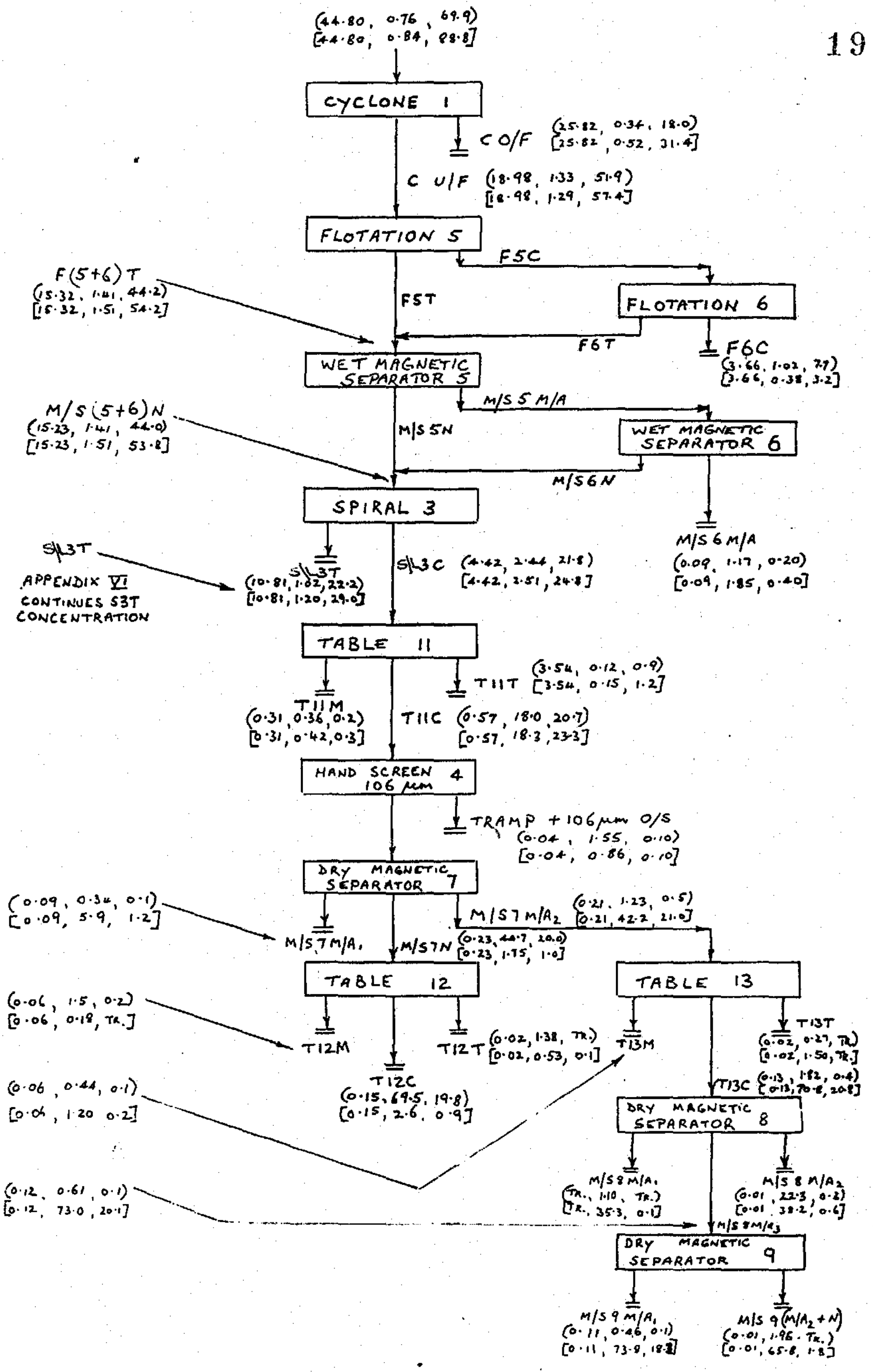
CONCENTRATION OF -894 μm +315 μm MATERIAL



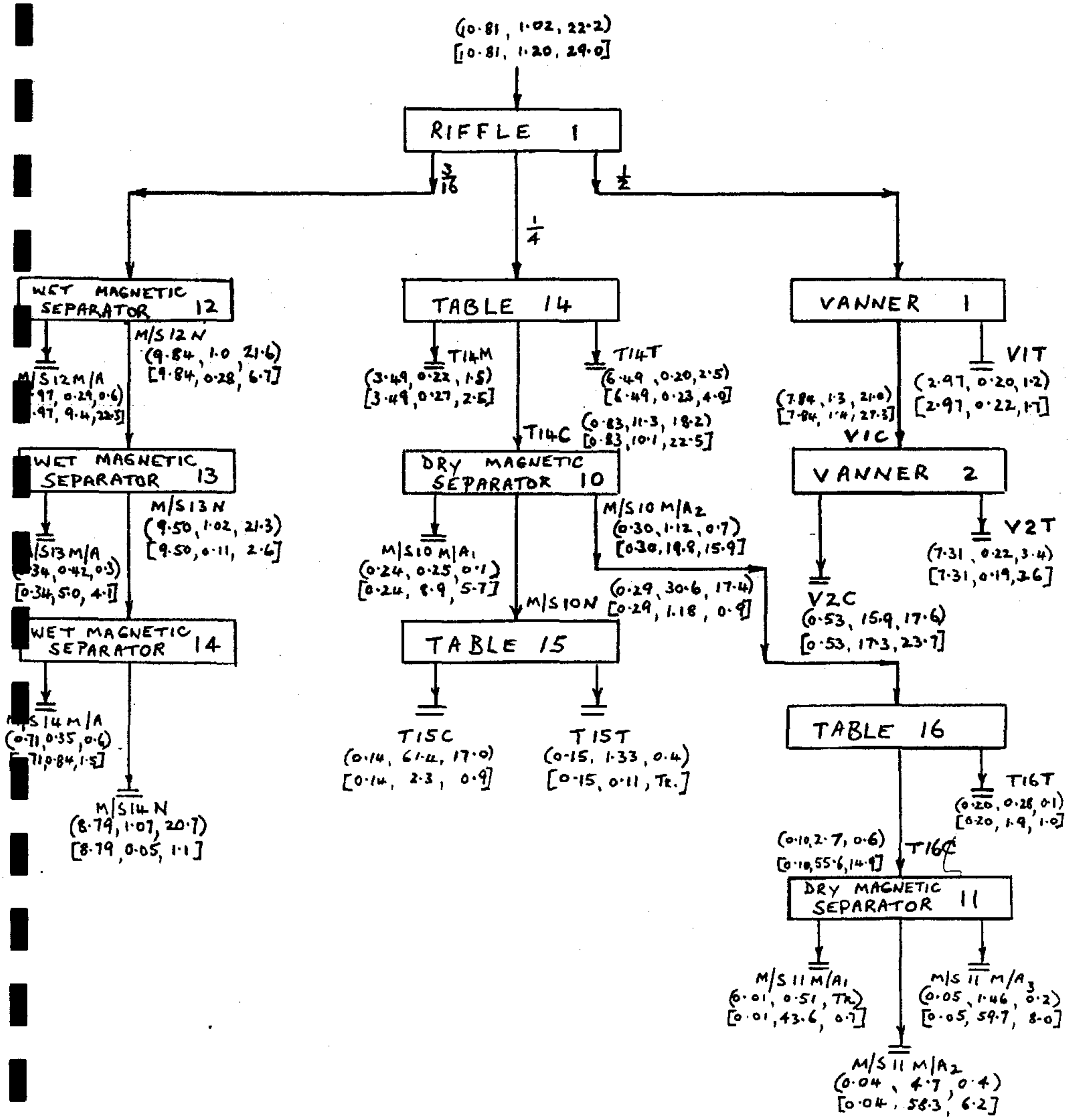
APPENDIX III CONCENTRATION OF −315 µm +108 µm MATERIAL



APPENDIX IV CONCENTRATION OF −108 µm MATERIAL



APPENDIX V CONCENTRATION OF S\L3T



056

Cunnenor Dam

Wheal Lutwyche   Reg. No 850279

Fraction µm	Per Cent Mass	Per Cent Mass Cum	Per Cent Sn	Per Cent WO_3	Per Cent Distribution Sn	Per Cent Distribution Sn Cum	Per Cent Distribution WO_3	Per Cent Distribution WO_3 Cum
+ 1.18 mm	2.3	2.3	0.05	0.04	0.3	0.3	0.2	0.2
+ 600 µm	1.9	4.2	0.18	0.02	1.0	1.3	0.1	0.3
+ 300 µm	3.2	7.4	0.13	0.04	1.2	2.5	0.3	0.6
+ 150 µm	6.6	14.0	0.13	0.04	2.5	5.0	0.7	1.3
+ 75 µm	10.1	24.1	0.10	0.04	3.0	8.0	1.0	2.3
+ 38 µm	13.3	37.4	0.28	0.20	10.9	18.9	6.5	8.8
- 38 µm			0.44	0.59				
C/S 1	2.5	39.9	3.5	4.4	25.4	44.3	26.8	35.6
C/S 2								
C/S 3	9.3	49.2	0.47	0.67	12.9	57.2	15.4	51.0
C/S 4	10.6	59.8	0.37	0.50	11.6	68.8	13.2	64.2
C/S 5	6.6	66.4	0.30	0.42	5.8	74.6	6.8	71.0
C/S O/F	33.6	100.0	(0.26)	(0.35)	25.4	100.0	29.0	100.0

Temperature of cyclosizing 20.5°C.

	Sn	WO_3
Head (calc)	(0.34)	(0.41)
Head Assay	0.32	0.38
Head (composite calc.)	(0.32)	(0.39)

(H. K. Wellington)
Chief Chemist & Metallurgist

Senior Metallurgist .... L G Rhodes

Fee $140.00

057

194058

L. Bollen Metallurgical Services,
10 Rosewarne Avenue,
Cheltenham. 3192
Victoria

31st January, 1985

App ②

Mr. H. Stacpoole,
1 Lindsay Street,
LAUNCESTON. 7250
Tasmania

Dear Harry,

Wheal Lutwyche Pty. Ltd.

I apologise for putting you to the trouble of locating a hand written table connected with dam grades and tonnages. It was located in my papers after the third attempt. I have enclosed a photocopy of same which is titled "Dam Resources" and from which you can readily see the overall significance of the "Current Dam" and hence my reason for recommending that it be accurately sampled.

The Report R833 dated 14th November 1984 has been received, studied and commented on in detail, later in this text. I would however like to stress that for the No. 2 Tailings Dam, previous accurate sampling and assaying produced significantly different analyses to that sample used for the testwork given in R833 (refer table below).

	Aberfoyle Sampling	Wheal Lutwyche Sampling
Sn%	0.39	0.49
WO_3%	0.42	0.42

It is stated in R833 (under Sample: Page 1) that:

"The sample, designated registered No. 830929, was stated to have been taken around the edges of the Rossarden No. 2 tailings dam by means of a backhoe".

This sampling procedure must be regarded as "suspect" and can be grossly misleading as regards sizing and metal values. Much money and time has been spent on this sample of the dam and results quoted in R833 must be regarded as guidelines only.

I earnestly hope that the "Current Dam" is sampled with great care by some appropriate acceptable method such as pattern auger drilling etc.

Detailed Comments on R833:

The aims of the test work, even though it was based on a "suspect" sample, were achieved.

Some shortcomings, I believe in the aims and laboratory procedures adopted for the testwork are; -

(1) The mass balance of products should have been reported more accurately.

(2) The cycloning procedures adopted on the -108µm material are unrealistic when related to standard laboratory procedures and plant practice. The results therefore are misleading (the adopted procedures gave excellent laboratory results especially in tabling-vanning etc.). Commercial cycloning will produce cyclone U/F's containing more "fines" than the testwork and hence tabling-vanning results would not be as good.

(3) Concentrates produced by the testwork, are said to be "marketable". The concentrates produced by the testwork, are of quite satisfactory grade, however impurities, both measured and unmeasured, would be <u>far</u> from good. ~ agree lots of apy for example

Tin concentrates (Page 33) of 64.3% Sn contain 2.3%WO_3 and an unmeasured sulphur content and many other penalisable elements (Cu, As, Fe etc.). Tungsten concentrates of 67.4% WO_3 contain 1.6% Sn and unmeasured Sulphur content. It must also be noted that these concentrates (Page 31) have been secured by complicated procedures of drying, with reprocessing of magnetics and non magnetics.

(4) This issue of prior wet magnetic separation, say on the -108µm sized fraction, appears to have escaped the technicians even though results (page 36) demonstrate that the dry magnetic separation results become increasingly poor as the particle size becomes finer (probably due to dried agglomerates of cassiterite and wolframite, fine particle adherence, dusting etc. etc.). These problems have shown up in the laboratory, where, I have no doubt, that screening (to break up fine agglomerates) prior to dry magnetic separation in the laboratory was much finer than would be tolerated in plant practice.

The report R833 highlights some noteworthy problems of true middlings, recommending some selected regrinding; and has further highlighted the Sn and WO_3 losses associated with the sulphides.

As a continuation of Report R833 it is recommended that:

(a) Composite samples of <u>each</u> of the
-108 µm tin concentrate
-108 µm wolframite concentrate
be assayed for S, As and Cu.

../3

(b) The sulphide cleaner concentrate sample F6C (page 22) be examined mineralogically to establish the reason why this product assays 1.02%Sn and 0.38%WO_3.

If this testwork is to be repeated on any sample, it is recommended that a testing procedure be adopted along the lines as indicated in my flowsheet dated 1st December 1984 and that the project be suitably supervised, thus ensuring that a future feasibility study is more adequately supported.

The flowsheets of 1st December 1984 do not demonstrate methods of upgrading crude concentrates coarser than 108µm into final sales concentrates, and this small recovery contribution needs to be discussed later as does any small regrinding operation which may be necessary. The flowsheet (1/12/84) embraces froth sulphide flotation "clean up" and scavenging from cleaner table concentrate of combined wolframite and cassiterite. I believe this is highly desirable to produce sales concentrates of acceptable impurity levels although the testwork has not demonstrated anything in this regard.

After sulphide clean up, the proposed flowsheet demonstrates wet high intensity magnetic separation of the desulphided high grade mixed cassiterite - wolframite concentrate. In Report R833 the results of Jones high intensity wet magnetic separation tests on a -108µm material (pages 28 and 29) are not commented on. Washings in this testwork were returned to feed for magnetic separation. The separation results produced by this testing are:

Electro Magnet Current	Separation Efficiency
10 AMPS	$\frac{21.6}{22.2}$ = 97.3% of the tin into Non Mags
	$\frac{22.3}{29.0}$ = 76.9% of the WO_3 into Mags
18 AMPS	$\frac{21.3}{22.2}$ = 95.9% of the tin into Non Mags
	$\frac{26.4}{29.0}$ = 91.0% of the WO_3 into Mags

The above results are excellent and clearly demonstrate that a separation of wolframite from cassiterite could be effectively conducted, prior to drying and subsequent dry magnetic separation to further separately produce clean cassiterite and wolframite concentrates. This procedure, I have no doubt, will produce better final sales concentrates with less recycle - return - retreatments than is indicated by the report R833 type treatments.

../4

Losses of Tin and Tungsten in Slime Fractions:

In the testwork, the only unconcentrated upgraded fraction (into sales type concentrates) of the sample representing the No. 2 Dam is the -108µm Cyclone Overflow fraction (page 21). In summary, this fraction represented 25.82% weight carrying 18.0% of the tin and 31.4% of the WO_3 in the original sample. This fraction assayed 0.36% Sn and 0.53% WO_3 and after appropriate treatments has potential for tin and tungsten flotation as would the table tailings generated if and when the flowsheet of 1st December 1984 is adopted.

Please do not hesitate to telephone me if anything is not "crystal clear" in this letter.

Yours faithfully,

Leonard Bollen

DAM RESOURCES

Dams No.	Dry Tonnes	Grade Sn %	WO$_3$%		
1	50,000	0.40	0.41	20,000	20,500
2	23,500 7,049	0.39	0.42	9,165	9,870
3	12,000	0.46	0.26	5,520	3,120
4	22,500	0.56	0.34	12,600	7,650
Current	53,000	0.40*	0.40*	21,200	21,200
Measured Feb. 85	(90,000)	(0.33)	(0.40)		
Overall	161,000 (198,000)	0.425	0.387	68,485	62,340

*Estimated

062

194063

Bartles

Bartles (Carn Brea) Limited
North Street
Redruth
Cornwall
England TR15 1HJ

Telephone 0209 215172
Cables Bartles
Redruth
Telex 45611

1909-1984
75
YEARS

Your ref

Our ref MFE/DP

Date 26th November, 1984

(3)

Wheal Lutwyche Pty Ltd.,
Registered Office,
1 Lindsay Street,
Launceston,
Tasmania 7250.

For the attention of: Mr. H. Stacpoole

Dear Harry,

Thank you for your Letter of 14th November, 1984 with the Mines Department Report attached. The results are certainly encouraging and we are sure that with a little more time we could have produced slightly better results if we had optimised the machines. However, having said that, we are extremely pleased with the results which clearly indicate the capabilities of our machines to recover tin and tungsten fines.

We have pleasure in including with this Letter our Detailed Report of the Testwork performed on your sample and hope that it is self-explanatory. The most important fact as far as we are concerned is that with current Tin Prices we calculate the payback period for our machines, as discussed, to be in weeks rather than in months as is the usual case. As was stated by the gentlemen that you met at Medway Tin and Kernow Tin during your visit, it is just as well that your dumps are not in Cornwall as they would all be fighting for a share of the action.

We have studied your suggestion of testing the minus 20 micron cyclone o/f product. In isolation this size fraction alone would have quite a high apparent viscosity which would reduce separation efficiency on our equipment. As we have no information regarding the size and value analysis of the minus 20 micron fraction, it is difficult to assess the potential recovery, or if further desliming would be necessary. However, we are confident that significant recoveries could be made in the 20-10 micron size fraction, and you may therefore consider that desliming at 10 microns may be more beneficial. If you require testwork on a further sample, we would require a 5kg sample for this work, or enough sample to ensure that 5kg would remain after desliming. This desliming work would be carried out by Richard Mozley Ltd. at a cost of £250.00 which would be additional to our charge of £250.00 for Preliminary Testwork. A copy of our Testwork Services sheet is attached for you to study in the event of you deciding to Air Freight a sample to us.

/Contd.

The success of our machines on the treatment of various size fractions of a typical material is suitably described in the paper which was recently given by Gary Player at the IMM Conference in China last month and a Photocopy is attached for your information. I am sure that this paper will also be of interest to Messrs. Wellington and Rhodes at the Mines Department.

There is very little else that I can add at this juncture except to hope that you will soon be in a position to commence a limited scale operation at your plant and that we will be able to provide you with the necessary equipment to ensure its unquestionable success and associated profit for your Company.

We hope that you and your family are all well and look forward to your reply to our proposals.

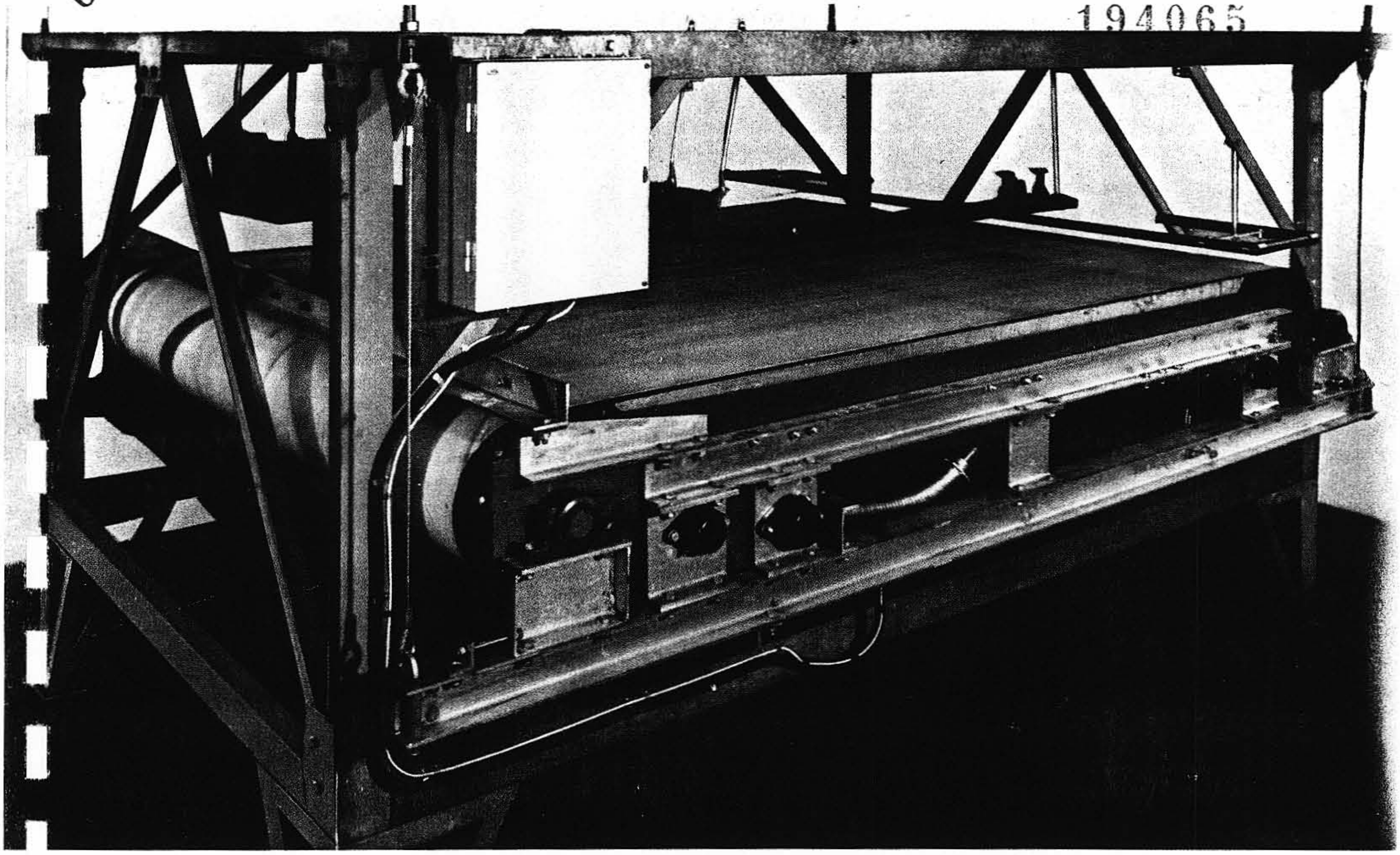
Yours sincerely,

M. F. Endean,
Sales Manager.

Encs.

064

194065



Crossbelt Concentrator

a high grade
slimes
gravity–
concentrator

XB
CONCENTRATOR
Bartles

M 1588

065

DEPARTMENT OF MINES—TASMANIA

194066

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

TELEPHONES:
Metallurgical Research .. .. } 44 2431-2 (2 lines)
Laboratory .. .. ..
Mines Inspection .. .. ..
Explosives & Inflammable Liquids

25th October 1984

Ap ③

Wheal Lutwytche Pty Ltd,
C/- Mr. H. Stacpoole,
1 Lindsay Street,
Invermay,
LAUNCESTON

Attention Mr. H. Stacpoole

R849: Rossarden Tailings: Bartles Crossbelt Test

Dear Sir,

Please find attached results of assays on samples obtained from this test and the metallurgical balance obtained from the results and the product masses supplied by Bartles.

While the results show superior metallurgical efficiency when compared with table and vanner results on the same sample one should evaluate the Bartles Crossbelt performance against its capital cost and maintenance cost in making a selection of equipment.

Yours faithfully,

(H. K. Wellington)
Chief Chemist & Metallurgist

Fee $92.00

M 1688

066

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research
Laboratory
Mines Inspection
Explosives & Inflammable Liquids
} 44 2431-2 (2 lines)

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

194067

25th October 1984

R849

EVALUATION OF A BARTLES CROSSBELT CONCENTRATOR

Introduction

In the investigation R833 on retreatment of tailings from Rossarden, the -108µm cyclone U/F, FT, N product was concentrated on a Vickers 5 turn CC series spiral. The spiral tail assayed 1.02% Sn and 1.20% WO_3.

The spiral tail was subsequently retreated in investigation R833 using on one portion the Deister table, and on another portion the Fieldhouse vanner. A third portion of the spiral tail was given to Mr. H. Stacpoole for testing on a Bartles Crossbelt concentrator in Cornwall, U.K. Samples of products of these tests were returned to the Mines Department for analysis. These were as follows:-

Reg. No	Identification.	
841203	4668	Con.
841204	4669	Mid.
841205	4670	Tail.
841206	4671	Tail.
841207	4672	Con.
841208	4673	Mid.
841209	4674	Tail.

Results

Bartles conducted two tests. In the first test a low grade concentrate and a tail was produced, and then the concentrate was cleaned to produce a high grade concentrate, middling and a tail. In the second test a high grade concentrate, a middling and a tail was produced in one operation.

The results were as follows:-

			Assay %		% Distribution	
Reg. No	**Product**	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
841203	V2C	2.26	31.6	37.4	75.2	81.6
841204	V2M	2.48	3.0	2.3	7.8	5.5
841205	V2T	24.17	0.46	0.26	11.7	6.1
841206	V1T	71.09	0.07	0.10	5.3	6.8
	Head	100.00	(0.95)	(1.04)	100.0	100.0
841207	VC	2.68	30.5	35.8	80.2	83.6
841208	VM	6.69	1.25	0.79	8.2	4.6
841209	VT	90.63	0.13	0.15	11.6	11.8
	Head	100.00	(1.02)	(1.15)	100.0	100.0

The results of the table test done on spiral tailings in R833 was as follows:-

		Assay %		% Distribution	
Product	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
T4C	7.68	(11.3)	(11.7)	82.0	79.9
T4M	32.24	0.22	0.27	6.7	7.8
T4T	60.08	0.20	0.23	11.3	12.3
Head	100.00	(1.06)	(1.12)	100.0	100.0

The result of the vanner test done on spiral tailings in R833 was as follows:-

		Assay %		% Distribution	
Product	% Mass	Sn	WO_3	Sn	WO_3
V2C	4.96	15.9	17.3	79.3	81.8
V2T	67.59	0.22	0.19	15.1	12.4
V1T	27.45	0.20	0.22	5.6	5.8
Head	100.00	(0.98)	(1.04)	100.0	100.0
Head Assay		1.02	1.20		

Conclusions

The Bartles crossbelt concentrator produced results that were considerably superior to results obtained by tabling and by use of the Fieldhouse vanner.

The unit should be seriously considered when planning a treatment plant for the retreatment of tailings at Rossarden.

Senior Metallurgist...............

(H. K. Wellington)
Chief Chemist & Metallurgist

L. Bollen Metallurgical Services,
10 Rosewarne Avenue,
Cheltenham.   3192
Victoria

14th September, 1984.

(H)

Mr. H.Stacpoole,
1 Lindsay Street,
LAUNCESTON.   7250
Tasmania

Dear Harry,

Tailings Retreatment

Attached are two copies of each of four sheets constituting the first draft of the preliminary Flow Diagram.

As generally discussed with you on our valuable discussion of 7th September 1984, one of the copies is marked up with some guesstimated sizes as discussed with you.

Please also find enclosed two technical reports, which must be returned in say two months, these reports may be of interest.

I will present to you, in about a fortnight, a revised Flow Diagram, because I can see some minor imperfections in the first draft.

To obtain the best performance out of any proposed plant for this job, the key units must be high quality and in the best condition possible.

Wilfley shaking tables are not acceptable for the tabling of these fine sizes.

Obviously pumps shown in the flowsheet may not be positioned correctly and this must be dependant on plant layout. Consideration must also be given to tailing pumps and combination of products to tailing dams. The capital cost of preparing new tailing dams should not be omitted.

I was delighted to hear that you have generated some interest with Taffy Perkins and I look forward to having long discussions with him, especially before purchase of any gear.

Kindest regards,

Leonard Bollen

P.S. On another tailings retreatment job of a slightly different nature, the 1983 costs are:

Excavation and carting to stockpile	$1.00 per ton
Treatment, labour, materials, maintenance, power.	5.00 " "
Total	$6.00

PRELIMINARY FLOWDIAGRAM

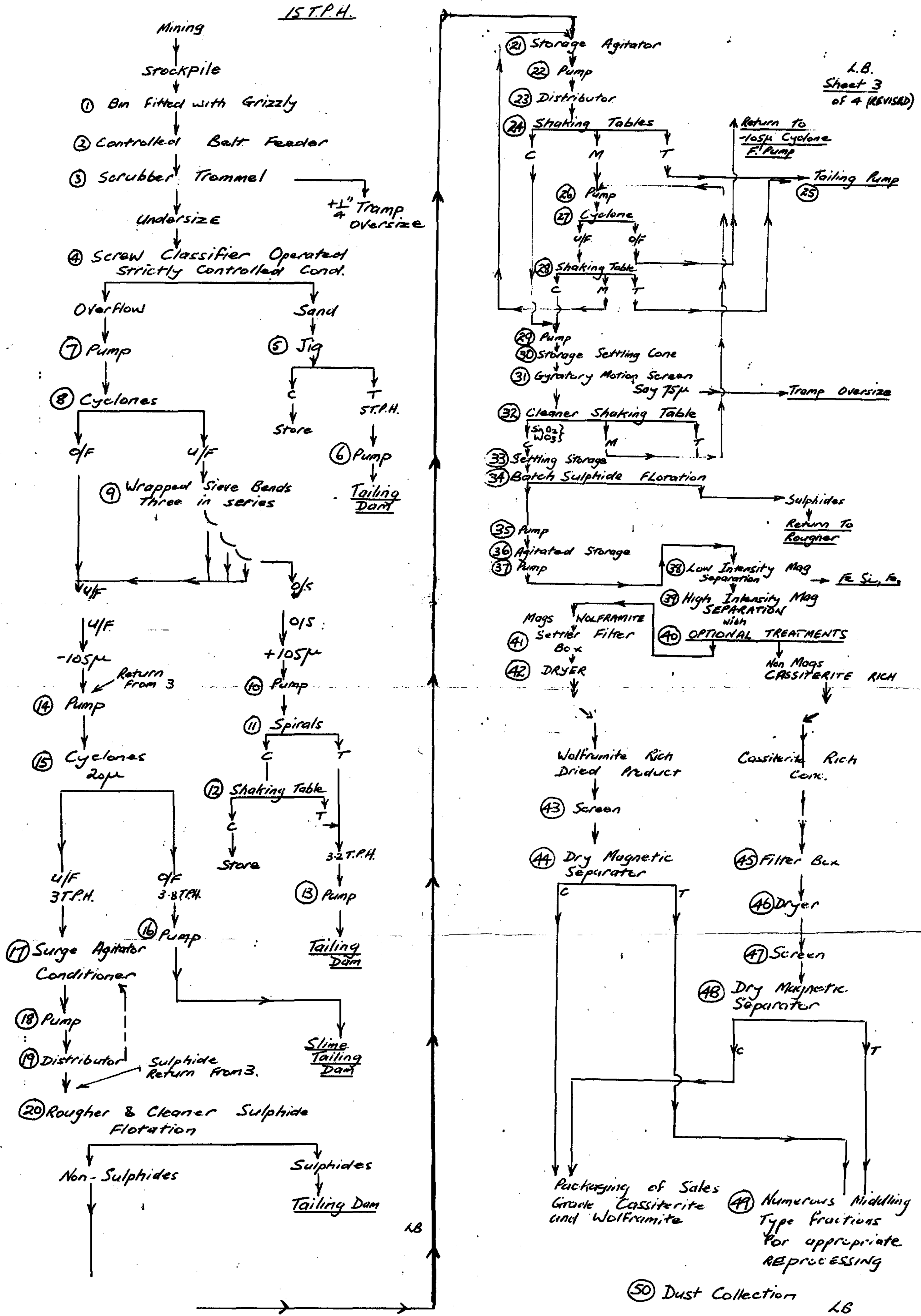
TAILING DAM RETREATMENT

15 T.P.H. (PROPOSAL)

ESTIMATED EQUIPMENT SIZE

Item No.	Equipment Details
1	
2	20" Wide belt fitted with slow vari-speed drive.
3	
4	36" Screw desliming classifier.
5	Existing 2 cell 42" x 42" Pan American jig
6	3/2 Warman pump.
7	3/2 Warman pump.
8	1 only 10" cylcone.
9	Wrapped sieve bends. Three feet wide. Three in series.
10	3/2 Warman pump.
11	Concentrating spirals (existing).
12	One shaking table (coarse)
13	3/2 Warman pump.
14	4/3" Warman pump.
15	4 off 6" dia. cyclones.
16	4/3 Warman pump.
17	Surge agitator conditioner 5' dia. x 5'.
18	2/1½ Warman pump.
19	Distributor.
20	6 cell No. 36 Galigher flotation cells. 4 roughers. 2 cleaners plus auxiliaries.
21	Agitated storage 4' dia. x 4'.
22	2/1½" Warman pump.
23	Distributor - 7 way.
24	Fine shaking tables - 7 off.
25	4/3 Warman pump.
26	1½/1 Warman pump
27	4" Cyclone - 1 off.
28	Middling shaking table - 1 off.
29	1½/1 Warman pump.
30	Settling cone 5' dia. x 5'.
31	Gyratory motion screen 30" dia.
32	Cleaner shaking table - 1 off.
33	Settling cone 5' dia . x 5'.
34	No. 36 Galigher Flotation units 2 cells.
35	1½/1 Warman pump.
36	Agitated storage 4' dia. x 4'.
37	1½/1 Warman pump.
38	Low intensity mag. separator 12" dia. x 12" drum.
39	Readings 4 pole high intensity magnetic separator.
40	3 off 1½/1 Warman pumps. 3 off agitated storages 4' dia. x 4'.
41	Settler filter box - wolframite.
42	Special dryer.
43	Screen to smash agglometes.
44	Weatherall type cross-belt magnetic separator in conjunction with Rapids mag separator (both existing).
45	Settler filter box - cassiterite.
46	Special dryer.
47	Screen to smash agglomerates.
48	Rapids magnetic separator (existing).
49	Containers - shifting, loading gear.
50	Dust collection.
Not Listed	Water pumps - return water pumps, "clean up" pumps, etc. etc.

PRELIMINARY FLOWDIAGRAM
TAILING DAM RETREATMENT
15 T.P.H.



㊿ Dust Collection

11/9/84 Sheet 3 Revised 1/12/84

LB
Sheet 4 of 4

072.

194073 ⑤

L. Bollen Metallurgical Services,
10 Rosewarne Avenue,
Cheltenham.   3192
Victoria.

8th October, 1984.

Mr. H. Stacpoole,
1 Lindsay Street,
LAUNCESTON.   7250
Tasmania

Dear Sir,

Further to your request I have carefully studied the two following reports:

"Retreatment of Jig Tailings at Aberfoyle Tin Limited"
by Aberfoyle Limited, 1980.

and

"R829-Tin and Tungsten Distribution in Coarse Tailings From the Aberfoyle Mine at Rossarden and a Recovery Test Incorporating Grinding of a Selected Size Range of The Tailings"
Dept. of Mines - Tasmania 27/5/1983.

Aberfoyle Proposal:

It has been estimated that the accumulated Aberfoyle H.M.S. float and 3/8 : 1/4 jig tailings, together with Storeys Creek jig tailings constitute some 570,000 tonnes grading 0.135% C.M.

Aberfoyle proposed screening this variable material at 12 M.M., with oversize rejection. Undersize was to be treated by Heavy Media Cycloning (to replace H.M.S. using an inefficient Atkins spiral unit). The H.M. Cyclone sink product was to be appropriately ground, treated on concentrating spirals and such concentrate was to be treated in an existing, operating fairly efficient concentrator.

The Aberfoyle proposal, estimated to cost $929,000 had much merit for the proposal was to utilize H.M.C. and this new unit would more efficiently treat the R.O.M. ores. For undisclosed reasons, this proposal was not proceeded with.

For the tailings retreatment, the estimated metal recovery into sales concentrates was 51%. A senior metallurgist of Aberfoyle believed that 35% recovery was more realistically based.

../2

Treatment by Aberfoyle of a 550 tonne sample of high grade (0.18% C.M.) Aberfoyle jig tailing, by grinding the total material to minus 1.5 M.M., with treatment by spirals. The spiral concentrate was separately treated in the Aberfoyle concentrator complex. The overall recovery achieved was 25.3% (excluding some middling fractions of doubtful metal recovery.) There were no doubts that this recovery could be significantly improved.

Some contributors to the report doubted the tonnage of readily reclaimable tailings and grade of material. The metallurgical interpretation of laboratory testing procedures, and accuracy of analysis raise further doubts in my mind.

R829 : Tas. Dept. of Mines report:

The proposal envisaged in this report was to screen the tailings feed materials, to produce and reject a relatively barren oversize for use as paving material (etc.). Undersize, say -4.75 MM, (and containing 63% of the tin and 64% of the tungsten) was to be appropriately ground, and concentrated in jigs and spirals. The jig and spiral concentrates in plant practice would then have to undergo similar type treatment as applied to the Aberfoyle flowsheet.

This report demonstrated that in securing final sales grade concentrates, that an overall tin recovery of 32.1% and an overall tungsten recovery of 34.4%, were achievable.

I have reservations as to the detailed interpretation of the sizing analysis quoted in this work and again with the accuracy of analysis. This testwork should have produced both calculated and assayed head values and only this would substantiate grades and recoveries stated.

Summary:

1. If the grade of these tailing materials is accepted at 0.135% C.M. and with appropriate treatment a 40% recovery is obtained, we have a recovered value of say $5.40 tonne. Recovered value of 570,000 tonnes = $3,078,000. The recovered value as estimated would not cover the capital and operating costs for ;

   a. Mining
   b. Feed preparation
      H.M. cyclone plant
   c.   or
      (Appropriate sized grinding mill with
      (jigging and spirals.
   d. Product disposal (waste) from b. and c. above.

The capital and operating costs to take the product from c. above (0.5% Sn to 3% Sn) and containing much sulphides etc. to saleable grade concentrates will require considerable amount of equipment and all unit processes are labour intensive (relatively).

2. From 1. above it can be seen that these coarse tailings materials cannot be profitably treated in the metallurgical treatment circuit as proposed for the much finer, much higher grade material in the fine tailing dams. The use of spirals and or jig may be common to both circuits but the similarity of circuits is finished at that point and these circuits are neither compatible nor comparable.

../3

3. These coarse tailing materials have a significant value for use in construction and concrete manufacture for any future major construction, road, building etc. in the area.

It is also believed that good quality building sand could be produced by appropriate segregation of tailings in the proposed treatment of the accumulated fine tailings dams (dumps).

4. The accumulated H.M. float and coarse jig tailings have a limited low profit potential (but nevertheless - a profit) in any future large operations that could eventuate for the treatment of "Open Cut" ores from the area. This could conceivabily apply if any future metallurgical circuit encompassed the use of heavy media cycloning in the appropriate size ranges.

General:

I hope this satisfactorily answers your question. The reports referred to, are safely held in my custody, however please do not hesitate to ask for them to be returned.

Yours faithfully,

Leonard Ballen

0-75

M 158F

194076

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research Laboratory
Mines Inspection
Explosives & Inflammable Liquids
44 2431-2 (2 lines)

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R829

TIN AND TUNGSTEN DISTRIBUTION IN COARSE TAILINGS FROM THE ABERFOYLE MINE AT ROSSARDEN AND A RECOVERY TEST INCORPORATING GRINDING OF A SELECTED SIZE RANGE OF THE TAILINGS.

Introduction

Research project R827 examined grab samples from the face from several locations around the coarse tailings dump at Rossarden.

The samples in this investigation were derived by drilling the tailings dump from top to bottom at selected localities. Holes 2 and 3 were reported to be very successful holes in which sample recovery was maintained from the top to the bottom of the hole. The remaining holes were less successful as sample recovery was not so good. The samples from these latter holes were bulked to provide the feed for rod milling and a recovery test.

The details of the samples which were submitted by Mr. H. Stacpoole are as follows:-

Registered Number	Description
821925	Hole 2
821926	Hole 3
821927	Hole 1, Hole 4, Hole 5, Bag 6, Bag 7, Hole 8 (to Hole 8, 8.5-15, Hole 9, Hole 10-11 metre

Tin and WO_3 Distributions

Samples 821925 (Hole 2) and 821926 (Hole 3) were screened and the size fractions were assayed for tin (Sn) and tungstic oxide (WO_3).

The sizings and distribution of Sn and WO_3 were as follows:-

Reg. No 821925 Hole 2.

Size Fraction	Mass		Assay %		% Distribution			
	%	% Cum	Sn	WO_3	Sn	Sn Cum	WO_3	WO_3 Cum
+ 12.70 mm	7.1	7.1	0.02	0.01	1.3	1.3	2.9	2.9
+ 9.53 mm	9.1	16.2	0.03	0.01	2.4	3.7	2.6	5.5
+ 5.6 mm	18.3	34.5	0.04	0.01	6.4	10.1	8.9	14.4
+ 4.75 mm	8.9	43.4	0.12	0.01	10.2	20.3	4.0	18.4
+ 4.0 mm	8.0	51.4	0.05	0.00	3.9	24.2	1.3	19.7
+ 3.35 mm	11.1	62.5	0.07	0.01	7.3	31.5	3.1	22.8
+ 2.8 mm	9.2	71.7	0.10	0.03	8.7	40.2	12.0	34.8
+ 2.36 mm	6.6	78.3	0.05	0.04	3.2	43.4	10.9	45.7
+ 1.4 mm	13.9	92.2	0.14	0.04	18.4	61.8	23.6	69.3
- 1.4 mm	7.8	100.0	0.53	0.10	38.2	100.0	30.7	100.0
Calc. Head	100.0		(0.11)	(0.02)	100.0		100.0	

Reg. No 821926 Hole 3

Size Fraction	Mass		Assay %		% Distribution			
	%	% Cum	Sn	WO_3	Sn	Sn Cum	WO_3	WO_3 Cum
+ 12.70 mm	17.5	17.5	0.04	0.02	6.6	6.6	9.2	9.2
+ 9.53 mm	9.7	27.2	0.04	0.02	3.4	10.0	5.1	14.3
+ 5.6 mm	20.2	47.4	0.05	0.01	9.7	19.7	7.8	22.1
+ 4.75 mm	9.2	56.6	0.16	0.02	13.5	33.2	7.7	29.8
+ 4.0 mm	8.0	64.6	0.11	0.04	8.5	41.7	11.8	41.6
+ 3.35 mm	9.3	73.9	0.11	0.01	9.2	50.9	4.2	45.8
+ 2.8 mm	6.8	80.7	0.06	0.02	3.9	54.8	3.8	49.6
+ 2.36 mm	4.8	85.5	0.13	0.08	5.5	60.3	13.6	63.2
+ 1.4 mm	9.5	95.0	0.23	0.06	20.0	80.3	20.7	83.9
- 1.4 mm	5.0	100.0	0.43	0.09	19.7	100.0	16.1	100.0
Calc. Head	100.0		(0.11)	(0.03)	100.0		100.0	

These results show that it would be desirable to screen the material on 5.6 mm to produce a coarse product for paving use, since the -5.6mm + 4.75mm fraction carries a considerable quantity of tin. Removal of the oversize material lifts the grade of the undersize material as follows:-

		Assay %		% ~~Recovery~~ Distribn	
-5.6 mm	% Mass	Sn	WO_3	Sn	WO_3
Hole 2	65.5	(0.15)	(0.03)	89.9	85.6
Hole 3	52.6	(0.17)	(0.04)	80.3	77.9

Inserting a 2.36 mm screen would give two products as follows:-

	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
Hole 2					
-5.6mm + 2.36mm	43.8	(0.08)	(0.02)	33.3	31.3
-2.36mm	21.7	(0.28)	(0.06)	56.6	54.3
Hole 3					
-5.6mm + 2.36mm	38.1	(0.12)	(0.03)	40.6	41.1
-2.36mm	14.5	(0.30)	(0.07)	39.7	36.8

Inserting a 1.4 mm screen would give two products as follows:-

	% Mass	Assay %		% Distribution	
		Sn	WO_3	Sn	WO_3
Hole 2					
-5.6mm + 1.4mm	57.7	(0.10)	(0.02)	51.7	54.9
-1.4mm	7.8	0.53	0.10	38.2	30.7
Hole 3					
-5.6mm + 1.4mm	47.6	(0.16)	(0.04)	60.6	61.8
-1.4mm	5.0	0.43	0.09	19.7	16.1

Recovery Test

The composite sample, Reg. No 821927 was wet screened on a 0.457 m. dia. Sweco screen fitted with a 4.75 mm and a 1.24 mm screen cloths. The -1.24 mm product was pumped to a Vickers 5 turn CC series spiral.

The spiral concentrate was sized by hand screening using 200 mm dia. laboratory test screens with 500µm and 250µm apertures. The +500µm fraction was further concentrated in the 60mm X 40mm Denver jig. This concentrate was magnetically separated with the dry Rapid high intensity magnetic separator and the non-magnetics were further concentrated by panning. Sulphides were skin floated out during the panning operation.

The -500µm + 250µm fraction was concentrated on the Deister table and the concentrate was further concentrated by panning and sulphides were removed by skin flotation during panning. The pan concentrate was magnetically separated with the dry Rapid high intensity magnetic separator.

The -250µm fraction was concentrated on the Deister table, and the table concentrate was subjected to low intensity magnetic separation using a hand magnet to remove highly magnetic material, possibly ferro-silicon. The non-magnetics were then floated in a flotation cell to remove sulphides. The flotation tail was further upgraded by panning and the pan concentrate was magnetically separated using the dry Rapid high intensity magnetic separator.

The results of this part of the recovery test were as follows:-

Spiral Concentrate

		Assay %		% Distribution	
Product	**% Mass**	**Sn**	**WO_3**	**Sn**	**WO_3**
-1.24mm S/L C					
+500µm JT	1.743	0.05	0.02	0.9	0.7
JC M/A	0.001		49.9		1.2
JC N PC.	0.001	68.7		0.9	
-500µm + 250µm TT	1.240	0.04	0.02	0.4	0.4
TC PT	0.059	0.65	0.18	0.4	0.2
PC M/A	0.003		49.9		2.8
PC N	0.004	72.4		2.9	
-250µm TT	1.498	0.04	0.02	0.6	0.6
TC M/A	0.006				
TC FC	0.022	1.21	0.21	0.3	0.1
FT PT	0.052	0.82	0.34	0.4	0.4
PC M/A	0.007		49.9		7.5
PC N	0.008	66.5		5.3	

Because of the poor recovery of tin and tungsten in the +500µm fraction, it was decided to retreat the spiral tailing.

The spiral tailing was wet screened by hand using 200mm dia. laboratory test screens with 500µm and 125µm apertures.

The.+500µm fraction and the -500µm + 125µm fractions were concentrated by panning. The pan concentrate in each case was magnetically separated using the dry Rapid high intensity magnetic separator. The -125µm fraction was table concentrated using the Deister table to give a concentrate T1C and a tailing T1T. The concentrate was retabled to give a concentrate T2C and a tailing T2T. This table concentrate T2C was magnetically separated with a low intensity hand magnet to remove highly magnetic material. The non-magnetics were then floated in a flotation cell to remove sulphides. The flotation tail was magnetically separated with the dry Rapid high intensity magnetic separator, and the non-magnetics were panned to produce a final concentrate.

The results of this retreatment of the spiral tailing were as follows:-

Spiral Tailing

Product		% Mass	Assay %		% Distribution	
			Sn	WO$_3$	Sn	WO$_3$
-1.24mm S/L T						
+500µm	PC M/A	0.002		50.8		2.0
	PC N	0.005	69.4		3.1	
	PT	1.066	0.15	0.05	1.5	1.1
-500µm+250µm						
	PC M/A	0.000		50.8		0.2
	PC N	0.000	51.2		0.2	
	PT	0.083	0.11	0.03	0.1	0.1
-125µm	T1T	3.101	0.08	0.09	2.4	5.8
	T2T	0.467	0.07	0.05	0.3	0.5
	T2C M/A1	0.003				
	FC	0.010	1.23	0.78	0.1	0.2
	M/A2	0.014	0.29	2.56	Trace	0.8
	PC	0.001	51.8		0.4	
	PT	0.011	0.88	0.08	0.1	Trace

The distribution in the two sets of results from the recovery test reported so far relate to the tin and tungsten in the original feed.

The actual recovery performance of the spiral in the -1.24mm +500µm size range in a final product was only 13.6% for tin and 23.5% for tungsten. A further 48.9% of the tin and 40.4% of the tungsten was recovered by panning this size fraction in the spiral tailing to give final concentrates.

The spiral did much better in the size ranges finer than 500µm. The actual recovery performance of the spiral in the size ranges finer than 500µm was 58.6% for tin and 52.4% for tungsten. Retreatment of these size fractions in the spiral tailing recovered a further 4.7% of the tin and 5.2% of the tungsten.

The -4.75mm + 1.24mm fraction of the original feed was ground in the 0.31m. dia. X 1.0m Denver rod mill in closed circuit with the 0.457m. dia. Sweco screen fitted with a 1.24mm screen cloth.

The screen undersize was concentrated with the 150mm X 100mm Denver jig. The jig concentrate was hand screened using 200mm. dia. laboratory test screens with 500µm and 125µm apertures.

The +500µm size fraction was then further concentrated using the 60mm X 40mm Denver jig. The jig concentrate and the jig bed were panned. A sulphide concentrate was removed by skin flotation. The pan concentrate was magnetically separated with the dry Rapid high intensity magnetic separator. The first three magnetic products were bulked for a wolfram concentrate and the fourth magnetic product was put with the non-magnetics to give a tin concentrate.

The -500µm + 125µm fraction was tabled to produce a concentrate and a tailing. The table concentrate was panned. The pan concentrate was skin floated to remove sulphides, and then was magnetically separated. The first three magnetic products were bulked for a wolfram concentrate and the fourth magnetic product was put with the non-magnetics to give a tin concentrate.

The -125µm fraction was floated in a flotation cell to remove sulphides. The floation tail was panned, and the pan concentrate was magnetically separated. The first three magnetic products were bulked for a wolfram concentrate, and the fourth magnetic product was put with the non-magnetics for a tin concentrate.

The results of cleaning the jig concentrate are as follows:-

Product			Mass	Assay %		% Distribution	
			%	Sn	WO_3	Sn	WO_3
J1C							
	+500µm	J2C PCN + M/A4	0.009	66.7		5.9	
		PC	0.025	4.69	0.60	1.1	0.3
		PT	0.048	1.16	0.34	0.5	0.4
		M/A1+M/A2+M/A3	0.004		49.0		4.8
		J2T	6.259	0.08	0.02	4.7	3.2
	-500µm+125µm	TC PCN + M/A4	0.014	65.3		8.6	
		PC	0.052	2.80	0.53	1.4	0.6
		PT	0.043	0.81	0.19	0.3	0.2
		M/A1+M/A2+M/A3	0.009		49.0		10.2
		TT	2.391	0.06	0.03	1.4	1.7
	-125µm	PCN + M/A4	0.002	66.6		1.4	
		M/A1+M/A2+M/A3	0.002		49.0		1.9
		PC	0.012	2.52	0.91	0.3	0.2
		PT	0.056	0.17	0.11	0.1	0.1

Because of the poor recovery of tin and tungsten in the -125µm fraction, it was decided to retreat the jig tailing. The jig tailing was screened on the 0.457 m. dia. Sweco screen fitted with a 152µm screen cloth. The +152µm material was not further treat The -152µm material was concentrated on the Deister table. The table concentrate was floated in the flotation cell to remove sulphides, and the flotation tailing was magnetically separated.

The third and fourth magnetic products and the non-magnetics were individually panned. The panned tails from each were combined with the first two magnetic products.

The results of the retreatment of the jig tailings were as follows:-

Product	Mass %	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
J1T + 152µm	15.562	0.05	0.02	7.2	5.2
- 152µm TT	10.114	0.06	0.02	5.5	5.4
TC FC	0.030	1.67	0.14	0.5	0.1
FT M/A1+M/A2 etc	0.019	2.42	3.14	0.4	1.3
M/A3 PC	0.002		54.3		2.9
M/A4 PC	0.003	34.1	15.9	0.8	0.9
N PC	0.005	59.2		2.6	

Retreatment of the -152µm fraction of the jig tailings increased the overall tin recovery from the grinding of the -4.75mm + 1.24mm fraction of the original feed from 37.2% to 45.2%. The wolfram recovery was increased from 43.0% to 52.7%.

A summary of the whole operation is as follows:-

Product		Mass %	Assay % Sn	Assay % WO_3	% Distribution Sn	% Distribution WO_3
+4.75mm		55.93	0.07	0.03	37.0	36.0
-4.75mm + 1.24mm	Tin Conc.	0.03	(62.5)		19.3	
	WO_3 Conc.	0.02		(45.6)		20.7
Combined Other Products		34.61	(0.07)	(0.02)	23.4	18.7
-1.24mm	Tin Conc.	0.02	(67.8)		12.8	
	WO_3 Conc.	0.01		(50.0)		13.7
Combined Other Products		9.38	(0.08)	(0.05)	7.5	10.9
		100.00			100.0	100.0

Disregarding the tin and tungsten in the +4.75mm fraction which is destined for sale as paving material, then the recoveries in each operation and the total recovery from the material that was treated is summarised as follows:-

	Recovery Sn	Recovery WO_3
-4.75mm + 1.24mm rod milled	45.2	52.5
-1.24mm fines treatment	63.1	55.7
Overall	51.0	53.8

Discussion

Examination of the tin and tungsten distributions in Holes 2 and 3 suggest that the rod mill feed should be -5.6mm instead of 4.75mm, as in both of these holes a considerable quantity of tin appears in the -5.6mm + 4.75mm fraction. Unfortunately this size fraction could not be included in the rod milling recovery test because there was no 5.6mm screen cloth to fit the Sweco screen. The tin and tungsten distributions also suggest a further variation from the size range used to feed the rod mill in the recovery test, and that is that the lower size of the rod mill feed be lifted to about 2.0mm, and thus increase the size range of the material for fines treatment.

Neither the spiral or the jig proved to be satisfactory as a rougher concentrator of the -1.24mm material. The spiral lost significant quantities of tin and wolfram in the coarse end of the size range, and the jig lost significant quantities of tin and wolfram in the fine end of the size range.

Probably if the -2.0mm material was screened on about 300µm, and the +300µm material was fed to a jig and the -300µm material was fed to spirals, the best recovery would be obtained in rougher concentration.

Conclusions

After screening coarse tailings from the old Aberfoyle Mine at Rossarden on a 4.75mm screen, 51% of the tin and 53% of the wolfram in the undersize can be recovered at a saleable grade.

The recovery obtained by rod milling the -4.75mm + 1.24mm fraction was 45% of the tin and 52% of the wolfram.

The recovery obtained by treatment of the -1.24mm fraction was 63% of the tin and 55% of the wolfram.

Senior Metallurgist........Rhodes........

Rhodes for

(H. K. Wellington)
Chief Chemist & Metallurgist

Composite holes 1, 4, 5, Bag 6 Bag 7, 8 top, 8 3.5 - 15.1, 9, 10, 11

Tin price on April 13th $140.11 a unit

Wolfram price Feb 25th $ 80.00 a unit (American)

April 13th $92.50 a unit April 13th exchange rate.

To treat 16 tph. in rod mill would need to screen 46.16 tph.

4.822864 units of tin per hour

2.155395 units of WO_3 per hour.

Rod milling would recover 0.930813 units of tin per hour.

$130.42

0.461255 units of WO_3 per hour.

$42.67

$173.09

Without grinding. would recover 0.617327 units of tin per hour $86.49

0.282357 units of WO_3 per hour $26.12

112.61

Rod milling plus fines treatment would net $285.70 per hour

Would produce 25.82 tons of oversize material.

and 4.34 tons of undersize material.

Therefore tons of material to be treated for tin recovery would be 20.34 tons per hour.

Total material 46.16 tons $6.19 per ton.

oversize $3 a ton.

084

194085

Hole 2

With a screening at 4·75 mm and rod mill treating 16 tons per hour

Sn Units in rod mill feed = 1·4736

∴ Sn units recovered = 0·664594 — \$93·12

WO_3 units in rod mill feed = 0·483568

WO_3 recovered 0·247587 — \$22·90

116·02

Fines treatment Sn units present 1·355410

Sn units recovered 0·853908 — 119·64

WO_3 units present 0·272754

WO_3 units recovered 0·155545 — \$14·39

134·03

Total treatment — ~~\$250·05~~ 250·05

~~Screening at 5·6 mm~~ 14·23 tons of oversize produced.

16 tons rod milled.

2·56 tons of fines treated. — \$13·47

32·79 tons per hour total. — \$7·93 per ton

Tons for tin recovery 18·56 tons per hour

Screening at 5·6 mm. and rod mill treating 16 tons per hour.

Sn units in rod mill feed = 1·553088

∴ Sn units recovered = 0·700443 — \$98·14

WO_3 units in rod mill feed = 0·443616

units recovered = 0·227131 — \$21·01

119·15

Fines treatment Sn units present 1·146343

Sn units recovered 0·722196 — \$101·19

WO_3 units present 0 235757 — \$12·17

\$113·36

Total Treatment — \$232·51

9·57 tons of oversize produced

16 tons of rod mill feed. — \$12·80

recovery 2·16 tons of fines total 27·73 tons — \$8·38 per ton

Hole 3

With a screening at 4.75 mm and rod milling 16 tons per hour

Sn units in rod mill feed = 2.124624

∴ Sn units recovered = 0.958205 $134.25

WO₃ units in rod mill feed = 0.751008

WO₃ units recovered = 0.324516 $ 35.57

$164 82

Fines treatment Sn units present 0.291667 -

Sn units recovered 0.561750 $ 72.71

WO₃ units present 0.222917

WO₃ units recovered 0.124387 11.51

$ 90.22

Total treatment ~~$260.04~~ ~~[illegible]~~ 260.04

23.58 tons of oversize produced

16 tons of rod mill feed.

2.08 tons of fines

41.66 tons per hour total $6.24 per ton.

$14.38

Screening at 5.6 mm and rod mill treating 16 tons per hour

Sn units in rod mill feed ~~20~~ 2.206656

∴ Sn units recovered 0.995202 $139.44

WO₃ units present 0.688976

WO₃ units recovered 0.352756 $ 32.63

$172.07

Fines treatment Sn units present 0.719328

Sn units recovered 0.453177 63.49

WO₃ units present 0.179832

WO₃ units recovered 0.100346 9.28

72.777

15.43 tons of oversize

16 tons rod mill feed.

Total treatment 244.84

App ⑥

ROSSARDEN DAMS RECOVERY PROJECT APRIL 1985

Revenue based on 40% recovery. Sn \$145.00 Unit. WO_3 \$114.00 Unit.

Dam	Head Grade	Gross Revenue		~~Volume~~ (Dry Tonnes)	Total Amount
1	.40 Sn .41 WO_3	\$23.20) \$18.70)	\$41.90	50,000	\$2,095,000.00
2	.39 Sn .42 WO_3	\$22.62) \$19.15)	\$41.77	23,500	981,595.00
3	.46 Sn .26 WO_3	\$26.68) \$11.86)	\$38.54	12,000	462,480.00
4	.56 Sn .34 WO_3	\$32.48) \$15.50)	\$47.98	22,500	1,079,550.00
Current Dam	.33 Sn .40 WO_3	\$19.14) \$18.24)	\$37.38	90,000	3,364,200.00
	Average:	\$40.31	Tonne	198,000	\$7,982,825.00

15 Tonnes per hour = 28,000 tonnes per year (@ 40% recovery) \$1,128,680.00.
90% Realization - Smelting = \$1,015,812.00 = \$36.27 tonne x 15 tonnes per hour = \$544.05 hour.
Mine life on proven reserves 7 years.

20 Tonnes per hour = 37,000 tonnes per year @ 40% recovery \$1,491,470.00.
90% Realization - Smelting = \$1,342,323.00 = \$36.27 tonne x 20 tonnes per hour = \$725.40 hour.
Mine life on proven reserves 5.3 years.

ROSSARDEN DAMS RECOVERY PROJECT APRIL 1985

Revenue based on 40% recovery. Sn \$150.00 Unit. WO_3 \$114.00 Unit.

Dam	Head Grade	Gross Revenue		~~Volume~~ (Dry Tonnes)	Total Amount
1	.40 Sn .41 WO_3	\$24.00) \$18.70)	\$42.70	50,000	\$2,135,000.00
2	.39 Sn .42 WO_3	\$23.40) \$19.15)	\$42.55	23,500	999,925.00
3	.46 Sn .26 WO_3	\$27.60) \$11.86)	\$39.46	12,000	473,520.00
4	.56 Sn .34 WO_3	\$33.60) \$15.50)	\$49.10	22,500	1,104,750.00
Current Dam	.33 Sn .40 WO_3	\$19.80) \$18.24)	\$38.04	90,000	3,423,600.00
	Average:	\$41.09	Tonne	198,000	\$8,136,795.00

15 Tonnes per hour = 28,000 tonnes per year @ \$41.09 tonne \$1,150,520.00. 90% Realization - Smelting = \$1,035,468.00 = \$36.98 tonne x 15 tonnes per hour = \$554.70 hour.

N.B. These figures can be taken as a guide to revenue.
Recovery will vary from dam to dam due to different head grades and particle size affecting recovery.

ROSSARDEN RETREATMENT PROJECT APRIL 1985.

ASSUMPTIONS.

S^N $145.00 UNIT WO^3 $114.00 UNIT. REVENUE AFTER SMELTING CHARGES (REALIZATION 90%) TAX AT 46% SALVAGE $100000

CASE 1. (000) CAPITOL $1.5m RECOVERY 40%. REVENUE $36.25 T

YEAR	CAPITOL	OPERATING COSTS	REVENUE	DEPRECIATION	TAXABLE INC.	TAX	CASH FLOW
1	1500						(1500)
2		420	1015	300	295	136	459
3		420	1015	300	295	136	459
4		420	1015	300	295	136	459
5		420	1015	300	295	136	459
6		420	1015	300	295	136	459
7		420	1015	-	595	274	321
8		420	1015	-	595	274	321 100

14.5%. NPV $362700.00 IRR. 22.4%.

CASE 2. CAPITOL $1.5m RECOVERY 35% REVENUE $31.74 T

1	1500						(1500)
2		420	889	300	169	77	392
3		420	889	300	169	77	392
4		420	889	300	169	77	392
5		420	889	300	169	77	392
6		420	889	300	169	77	392
7		420	889		469	216	253
8		420	889		469	216	253 100

14.5% NPV $78800.00 IRR 16.3%

[illegible]

CASE 3. CAPITOL $1·0M RECOVERY 35%. REVENUE $31·74 C

YEAR	CAPITOL	OPERATING COSTS	REVENUE	DEPRECIATION	TAXABLE INCOME	TAX	CASH FLOW
1	1·000						(1000)
2		420	889	200	269	124	345
3		420	889	200	269	124	345
4		420	889	200	269	124	345
5		420	889	200	269	124	345
6		420	889	200	269	124	345
7		420	889	-	469	216	253
8		420	889	-	469	216	253 100

14·5% NPV $419,400.00 IRR 27·7%

CASE 4 CAPITOL $1·0M RECOVERY 40%. REVENUE $36·25 C

1	1000						(1000)
2		420	1015	200	395	182	413
3		420	1015	200	395	182	413
4		420	1015	200	395	182	413
5		420	1015	200	395	182	413
6		420	1015	200	395	182	413
7		420	1015		595	216	321
8		420	1015		595	216	321 100

14·5% NPV $706600.00 IRR 36·0%

194091

The AUSTRALIAN CRANE COMPANY Pty. Ltd.
ERECTION ENGINEERS
385 Hobart Road, Launceston, TAS. 7250
Telephone: (003) 44 8233

9th September, 1984.

Wheal Lutwyche Pty. Ltd.,
1 Lindsay St.,
Launceston. 7250.

Attention: Mr. H. Stacpoole.

Dear Sir,

Tailings Retreatment, Rossarden.

We have examined the preliminary flow chart prepared by Mr. L. Bollen (Metallurgist), for a 15 t.p.h. throughput plant for the retreatment of tailings at Rossarden.

Over many years we have carried out a great deal of work at the Rossarden and Storeys Creek mines and mill, and we would make the following comments concerning your proposal.

1. Existing Concentrates Building.

This building is eminently suitable for the gravity plant that you propose, and could be commissioned for your purposes with only relatively minor modifications and reworks. It's value is estimated at $250,000.

2. On-Site Equipment.

An appreciable amount of the plant presently on site could be utilised for your project. Estimated cost for modifications and reworks would be $15,000.

3. Plant Installation.

A budget figure for the installation of all plant shown in the flow chart provided would be $50,000.

4. In-Plant Pipework.

We would estimate a figure of $85,000/$90,000 would be required for the supply and installation of all pipework, valves and associated equipment required for the process.

The above estimates are, of course, preliminary, but they are close enough to give an idea of the cost of installing and commissioning the plant, mechanically, to a dry-run situation.

If you decide to proceed further with your project we will give you all the assistance that we can, and we would be pleased to provide you with the benefit of our experience with the treatment plant at Rossarden.

Yours faithfully,
THE AUSTRALIAN CRANE COMPANY PTY LTD

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A.K. Smith
General Manager.