TRB-158-179

R. 421

21. CONCENTRATION OF A BULK SAMPLE FROM MOUNT CLEVELAND, TASMANIA

Summary

Concentration tests on a bulk sample from Mount Cleveland, Tasmania, indicate recoveries of the order of 65-68 percent of the tin from an ore assaying 0.99 per cent tin.

Results of four individual tests can be summarized as—

Test No.	Concentrate Per Cent Weight	Concentrate Per Cent Tin	Per Cent Tin Distribution
R.421/5	1.149	57.0	67.8
R.421/8	1.175	55.8	68.9
R.421/9	1.441	48.8*	66.2
R.421/6A	1.202	53.7	65.5

* Lower grade concentrate in test R.421/9 was due to difficulty in visually distinguishing the cassiterite from the gangue minerals after heating.

The bulk sample represents approximately 950 feet of drilling from 17 bores, as follows:—

Battery Lode: 6 bores, 187 feet of drilling.
Hall's Lode: 6 bores, 326 feet of drilling.
Henry's Lode: 5 bores, 332 feet of drilling.
Waterhole Lode: 2 bores, 106 feet of drilling.

A mineragraphic report by Mr. G. Everard has been supplied separately to Aberfoyle Tin Development Partnership. Concentration tests were carried out to determine the recovery of cassiterite by several methods, following more or less standard methods of concentrations of cassiterite from sulphide tin ores. These methods included—

(a) desulphidization of the ground ore by flotation, followed by gravity concentration of the desulphidized ore (tests R.421/5 and R.421/8).
(b) gravity concentration of the ground ore to a sulphide-cassiterite concentrate, with desulphidization of the concentrate by flotation (test R.421/6A).
(c) as (b) above, but sulphides destroyed by roasting (test R.421/6B).
(d) heating the ore in inert atmosphere, quenching and then following procedure (a) above (test R.421/9).

With some allowance for experimental error there is little to choose from in the results of tests R.421/5, 8, 9 and 6A quoted above. Results of commercial scale treatment of the ore should approximate these, provided the concentration plant is effectively designed and provides for similar treatments.

Gravity concentration of the ground ore to a sulphide-cassiterite concentrate was carried out in two tests.

Test No.	Concentrate Per Cent Weight	Concentrate Assay: Per Cent Tin	Recovery of Tin in Concentrate: Per Cent
R.421/4	24.77	3.38	83.5
R.421/6	11.862	6.64	79.5

These two tests are in reasonable agreement, making allowance for the difference in concentrate grade and variation in procedure. Recovery of the cassiterite after destruction of the sulphides by roasting gave similar overall results (see below), but these are markedly inferior to those obtained when sulphides were removed by flotation.

Test No.	Concentrate Per Cent Weight	Concentrate Assay: Per Cent Tin	Recovery of Tin in Concentrate: Per Cent
R.421/4	1.060	53.1	56.3
R.421/6B	0.937	61.2	57.9

Duplicate vanning assays were carried out on a total of 36 composite samples from the bores. On a weighted average (footage basis) a recovery of 66.1 per cent of the tin in a high grade concentrate is indicated. There are, however, considerable variations in the recoveries from samples of similar head values, probably due to differences in the cassiterite grain size and association. It is thus impossible to predict recoveries from any particular section of the deposit without prior testing.

The cassiterite is intimately associated with the gangue and sulphide minerals, and it is necessary to eventually grind the whole ore very finely. It is not practicable to discard tailings at comparatively coarse sizings without loss of recoverable cassiterite.

Sample

Tin determinations on samples from current diamond drilling operations at Mount Cleveland, Tasmania, have been carried out in the Department of Mines Metallurgical Laboratories, Launceston.

The samples received for assay sample preparation are split diamond drill cores. During assay sample preparation portions of these samples were retained at a relatively coarse size (generally at minus 10 mesh size). Composite samples were later made up from these coarse fractions, and vanning assays were carried out in duplicate on each composite. These vanning determinations are detailed later in this report.

The second halves of the split diamond drill cores were later received from the company for ore dressing research. These samples were received from Aberfoyle Tin Development Partnership as 38 composite samples, with compositions stated as in the tabulation below. Each of these composites was weighed and then combined to form the bulk sample which is the basis of the present investigation. An exception to the above procedure was six composites from Battery Lode (Bores C1-5). These six composites were first crushed minus 5 mesh and a small portion removed for examination by vanning, and the remainder was then added to the bulk sample.

The quantities of the various composites added to produce the bulk sample are detailed below. It will be noted that the whole of the available samples has been used, and that there is only approximate correlation between the length of drill core and weight of sample utilized.

The bulk sample was assayed with the following results:—

	%
Tin	0.99
Sulphur	9.8
Iron	21.8
Arsenic	0.56
Antimony	Nil
Bismuth	0.01
Copper	0.34
Zinc	0.13
Lead	Nil

No attempt was made to concentrate the copper content of the ore. The bulk sample was crushed minus 8 mesh, and this minus 8 mesh material was used as the basis for the research work detailed below. The minus 8 mesh material had the following sizing:—

Mesh	Per Cent Weight
Plus 10	8.1
Plus 22	45.2
Plus 44	14.3
Plus 60	5.6
Plus 85	4.4
Plus 100	1.8
Plus 120	2.1
Plus 150	2.0
Plus 200	2.0
Minus 200	14.5
	100.0

Total weight of the bulk sample was approximately 340 pounds.

Composition of bulk sample

Lode	Bore and Composite No.	Inter-section Footage	Core length Feet	Weight of Composite: Grams	Assay % Sn	Assay % S
Battery—	C. 1/ 1– 2	0 – 6	6	300	0.79	8.5
	C. 1/ 5– 6	96 –102	6	902	1.12	7.5
	C. 2/ 8– 15	43¾– 69¾	26	4,247	0.60	5.6
	C. 3/ 16– 29	85¾–126	40¼	6,292	0.80	8.0
	C. 4/ 30– 40	58½– 89½	31	5,095	0.57	5.5
	C. 5/ 41– 66	60½–138½	78	7,904	0.96	10.3
Hall's—	C. 9/ 71– 73	69¼– 77¼	8	1,110	0.95	8.3
	C.10/ 75– 76	53 –58¾	5¾	896	2.04	15.6
	C.10/ 77– 78	70 – 74½	4½	737	1.76	9.3
	C.10/ 79– 88	77 –106	29	5,592	1.25	11.5
	C.11/ 89–119	0 – 95	95	17,892	1.30	11.2
	C.12/120–134	0 – 44	44	7,694	1.17	9.4
	C.12/135–143	46½– 72	25½	4,550	1.00	9.8
	C.12/144–145	80 – 86	6	915	0.26	1.2
	C.12/146	111 –114	3	521	2.91	7.1
	C.18/314–330	187 –238	51	9,150	1.16	9.0
	C.23/334–351	65 –119	54	9,505	0.72	7.1
Henry's	C.14/239–244	105 –123	18	3,270	0.77	6.4
	C.14/245–260	130 –178	48	8,850	0.97	10.2
	C.15/167–171	0 – 15	15	2,809	3.10	17.1
	C.15/172	17½– 20	2½	408	0.80	6.8
	C.16/173–188	0 – 48	48	7,947	0.80	11.5
	C.16/189–190	48 – 54	6	1,297	0.075	2.6
	C.16/191–193	54 – 63	9	1,400	1.02	14.4
	C.16/194–212	63 –120	57	11,060	1.88	12.3
	C.16/213–214	120 –126	6	1,235	0.57	10.2
	C.16/215–219	132 –147	15	1,960	1.06	7.7
	C.20/270–278	0 – 27	27	5,082	1.01	9.0
	C.20/280	39 –42	3	436	1.24	3.2
	C.20/281–284	78 – 90	12	2,117	0.58	5.8
	C.20/286–292	100 –121	21	3,322	0.27	3.0
	C.20/297–311	133 –178	45	7,970	0.98	7.2
Waterhole–	C.13/147	58 – 61	3	571	1.82	12.1
	C.13/148–153	65 – 84	19	3,405	0.79	9.5
	C.13/154–161	89½–113½	24	4,000	0.81	9.9
	C.17/221–235	55 –115	60	7,637	0.87	11.5

Investigation

Aberfoyle Tin Development Partnership requested the investigation to obtain data relative to the commercial treatment of the deposit.

Tin ores containing appreciable sulphides can be concentrated in several different ways. The present investigation has briefly examined some of the variations most likely to be used commercially. The limitations of the present investigation must be stressed. This investigation must be regarded simply as a preliminary investigation to indicate probable recovery of tin in concentrates.

The quantity of the bulk sample available was only 340 pounds, and due to the number of variables which have had to be investigated, the quantity of sample used for each test has had to be kept at a minimum. Consequently the quantities of concentrates produced have been very small and there has been insufficient available to allow proper assessment of concentrate upgrading.

No responsibility is accepted for the results of this investigation, except in so far as they apply to the samples tested.

It will be noted that weight percentages are generally quoted to several places of decimals. No claim to such precision is intended, but such figures are necessary to show sensibly the extent of some of the small fractions produced.

Method of Treatment

Preliminary examination of the samples confirmed previous findings that much of the cassiterite in Mount Cleveland ores is very fine, and that all of the ore should be ground to the order of 200 mesh before being discarded as tailing. This has naturally made the concentration method rather elaborate.

There are several possible methods used to concentrate tin ores containing appreciable sulphides. The three methods most likely to be used commercially are briefly discussed below.

Method 1

The ground ore is desulphidized by flotation, and the desulphidized ore then concentrated by tabling, &c. Renison Bell is an example of this method of treatment. The advantage of this method is the increased gravity plant capacity due to prior removal of heavy sulphides. Disadvantages include:—

(a) Continuous flotation of the sulphides is unlikely to give complete removal of sulphides, so that batch flotation of sulphides from table concentrates may be required to produce high quality sale grade concentrate.

(b) Cost of flotation reagents will be comparatively high, unless slime gangue and sulphide minerals are removed prior to flotation.

Method 2

The ground ore is gravity concentrated to a sulphide-cassiterite concentrate, with desulphidization of the concentrate by flotation. The plant of Aberfoyle Tin N.L. at Rossarden treats ore in a manner similar to this method, although it should be noted that the sulphide content of the Aberfoyle ore is much less than that of Mount Cleveland ores in general. An advantage is the decreased flotation reagent cost, as only the sulphides retained by gravity concentration are floated, and the reagent consuming slime gangue and sulphide minerals do not appear in the flotation feed. Major disadvantage is the decreased gravity plant capacity, due to the presence of the heavy sulphides.

Method 3

A bulk sulphide-cassiterite concentrate is produced as in Method 2, but the sulphide-cassiterite concentrate is roasted, and then upgraded by further gravity concentration and regrind steps.

This method of treatment was used for many years for the sulphide ores at Mount Bischoff, Waratah.

There are two other possible methods of concentration of the ore, neither of which is at present in use commercially, as far as is known. These are:—

(a) Flotation of the cassiterite. Much work has been, and is being, done on this problem, without commercial application to date.
(b) Chemical methods of extraction of the tin. This is being currently investigated by the Chemical Engineering Division of the C.S.I.R.O.

Reference: Whitehead, A. B.—" Chemical Methods of Tin Extraction from West Tasmanian Ores."

Report No. CRL/CE/R-6,
Chemical Engineering Division,
C.S.I.R.O. Chemical Research Laboratories,
Fishermens Bend, Victoria.

Previous Investigations

Previous investigations relating to the treatment of Mount Cleveland ores carried out in the Department of Mines Metallurgical Laboratories, Launceston, are—

Ore Dressing Investigation R. 10, November, 1937.
R.110, April, 1944.
R.113, April, 1944.
R.132, May, 1945.

C.S.I.R.O. Mineragraphic Investigations 305 and 309.

Sizings

Screen sizes throughout refer to British Standard screens.

The minus 200 mesh fractions were further sized by a Haultain infrasizer. Estimated particle sizes of quartz and cassiterite segregated by the Haultain infrasizer are:—

Infrasizer Fraction	Quartz	Cassiterite
1	+56 microns	+45 microns
2	40—50 microns	30—45 microns
3	28—40 microns	20—30 microns
4	20—28 microns	14—20 microns
5	14—20 microns	10—14 microns
6	10—14 microns	8—10 microns
7	—10 microns	— 8 microns

Degree of Grind

Previous investigations (listed above) and measurement of the grain size of the cassiterite in the ore (see later) all indicated that there was comparatively little coarse cassiterite, and that fine grinding of the order of 200 mesh would be necessary for maximum recovery of the cassiterite.

Test R.421/4 detailed below was designed to determine quantitatively the effect of regrinding upon recovery of cassiterite in a sulphide-cassiterite concentrate.

Procedure was briefly:—

(1) The minus 8 mesh ore was batch ball mill ground to minus 36 mesh using short time intervals to minimize production of fines.

(2) The ground ore was then sized by screening on 60, 100 and 200 mesh screens. The four products were tabled separately, to produce sulphide-cassiterite concentrates and tailings.

(3) The plus 60 mesh tailing was batch ball mill ground minus 60 mesh, sized by screening on 100 and 200 mesh screens, and the three products tabled separately.

(4) The two plus 100 mesh tailings were batch ball mill ground separately to minus 100 mesh, sized by screening on a 200 mesh screen, and the products tabled separately.

(5) The several plus 200 mesh tailings were ground separately through a 200 mesh screen and tabled.

All tabling steps were carefully carried out, with retabling of a large middling.

The above procedure gave a total of 15 sulphide-cassiterite concentrates and 8 minus 200 mesh tailing products. The proportions of the various concentrates obtained from each stage of the procedure are:—

Derivation of Product	Product	Weight Percent	Assay Percent Sn	Distribution Percent Sn
Original Grind—	+ 60 mesh conc. A	7.57	1.85	14.0
	+100 ,, ,, A	5.62	2.75	15.4
	+200 ,, ,, A	6.31	2.64	16.6
	−200 ,, ,, A	2.47	8.67	21.4
	Composite Conc.	21.97	3.07	67.4
Primary Regrind	+100 mesh conc. B	0.03	2.77	1.7
	+200 ,, ,, B	0.55	4.42	2.4
	−200 ,, ,, B	0.22	11.09	2.5
	+200 ,, ,, E	0.38	4.99	1.9
	−200 ,, ,, E	0.17	15.16	2.6
	−200 ,, ,, G	0.17	9.47	1.6
	Composite Conc.	2.12	5.99	12.7
Secondary Regrind—	+200 mesh conc. C	0.24	2.93	0.7
	−200 ,, ,, C	0.08	9.97	0.8
	−200 ,, ,, D	0.09	7.52	0.7
	−200 ,, ,, H	0.20	4.48	0.9
	Composite Conc.	0.61	5.04	3.1
Tertiary Regrind—	—200 mesh conc. F	0.07	4.80	0.3
All Stages	Overall Concentrate	24.77	3.38	83.5
	Combined Tailings	75.23	0.22	16.5
	Calculated Head	100.00	1.00	100.0

The feed to each of the concentration steps after each grinding stage, and the recoveries in concentrates are:—

Derivation of Product	Product	Weight Percent	Assay Percent Tin	Distribution Percent Tin
Original Grind—	Concentrates ..	21.97	3.07	67.4
	Tailings	78.03	0.42	32.6
	Calculated Head ..	100.00	1.00	100.0
Primary Regrind—	Concentrates ..	2.12	5.99	12.7
	Tailings	43.18	0.23	10.1
	Calculated Head ..	45.30	0.50	22.8
Secondary Regrind–	Concentrates ..	0.61	5.04	3.1
	Tailings	18.29	0.16	2.9
	Calculated Head ..	18.90	0.32	6.0
Tertiary Regrind—	Concentrate	0.07	4.80	0.3
	Tailing	3.39	0.11	0.4
	Calculated Head ..	3.46	0.20	0.7

The above tabulations show that with each stage of regrinding approximately half of the tin contained in the product ground is recovered in the sulphide-cassiterite concentrate. This indicates that recoverable tin is contained in all comparatively coarse tailings, and for optimum recovery, the whole of the ore should be finely ground by stages, with intermediate concentration steps. It is most unlikely that any tailings can be discarded at comparatively coarse sizes without loss of recoverable tin.

During tabling, no free cassiterite was noted in the plus 60 or plus 100 mesh products. On tabling finely ground products, four rather distinct bands are obvious.

(1) A thin brown band of cassiterite.
(2) A medium band of green-olive pyrite.
(3) A medium, but rather indefinite, band of blackish pyrrhotite.
(4) A coarse band of green to olive-white gangue.

On the basis of the above results, it was decided to grind all of the ore through 200 mesh in later tests before discarding it into the tailings.

In the test work in this investigation, screens have been used throughout to define the degree of grind. Commercially this would be practicable to about 60 or 80 mesh using a sieve-bend, or electrically vibrated screen, but finer sizing would have to be carried out hydraulically. The present investigation is regarded as a preliminary investigation, and screens were used so that the separation sizes could be definitely established. On larger scale tests, hydraulic sizings would be used for grinding finer than say 80 mesh size.

Desulphidization by Flotation of Whole Ore

Several preliminary flotation tests (not included in this report) indicated that about one quarter of the ore could be removed by flotation as a sulphide concentrate, with a relatively low tin content. These tests indicated that copper sulphate, a mixture of ethyl and amyl xanthates and methyl isobutyl carbinol were an effective reagent combination. Reagent usage in the various tests is given below, but no real effort was made to determine optimum flotation conditions, and it is possible that the usages quoted would be in excess of that required on a commercial and continuous basis.

Two rather extended tests were carried out to determine the recovery obtainable by desulphidization of the ore by flotation, followed by gravity concentration of the desulphidized ore.

Test R.421/5

The procedure was—

(1) The minus 8 mesh ore was batch ball mill ground to minus 60 mesh. This size was chosen as it is difficult to float sulphides coarser than about this size, and also because there is little, if any, cassiterite coarser than this size.

(2) Sulphides were removed by flotation with copper sulphate: 1.1 lb/ton of ore. Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.27 lb./ton of ore.
Sodium amyl xanthate: 0.27 lb./ton of ore.
Methyl isobutyl carbinol: 0.20 lb./ton of ore.
Flotation time: 7 minutes.
pH of tailings: 6.4

(3) The sulphides were cleaned once, the bulk of the sulphides being removed without further reagents, but adding towards the end of flotation:—
Sodium ethyl xanthate: 0.04 lb./ton of ore.
Methyl isobutyl carbinol: 0.2 lb./ton of ore.
The cleaner tailing was retained as a separate product to allow assessment of the product.

(4) A sample of cleaner sulphide float was sized as follows:—

Mesh	Per Cent Weight	Tin	Tin Distribution
Plus 85	12.7	0.64	18.3
Plus 100	8.4	0.66	12.5
Plus 120	11.2	0.53	13.4
Plus 150	9.7	0.52	11.3
Plus 200	11.4	0.47	12.1
Plus 350	18.2	0.34	13.9
Minus 350	28.4	0.29	18.5
Calculated Head	100.0	0.45	100.0

This sizing indicates that some of the sulphides coarser than about 200 mesh are apparently composited with cassiterite. (Sizings of sulphide concentrates from preliminary flotation tests were similar to the above, but are not quoted in this report.)

A vanning assay on this sulphide concentrate gave a recovery of 50.6 per cent of tin in a concentrate assaying 55.3 per cent tin.

(5) On the basis of these data, the sulphides were batch ball mill ground to minus 200 mesh, and then floated using a similar reagent combination as used during original desulphidization of the ore, viz.:—

Copper sulphate: 1.1 lb./ton of ore.
Conditioning time: 5 minutes.

Sodium ethyl xanthate: 0.22 lb./ton of ore.
Sodium amyl xanthate: 0.22 lb./ton of ore.
Methyl isobutyl carbinol: 0.20 lb./ton of ore.
Flotation time: 7 minutes.

A sizing of the minus 200 mesh sulphide flotation concentrate gave:—

Infrasizer Fraction	Per Cent Weight
1	14.2
2	19.9
3	16.2
4	12.8
5	8.8
6	6.7
7	21.4
	100.0

(6) There was more minus 200 mesh flotation sink than expected, and this was probably due in part to the oxidation of pyrrhotite in the time elapsing during the sizing and assaying of the coarse sulphides, and in regrinding. The flotation sink was then tabled to produce a concentrate and a tailing, the tailing being added to the minus 200 mesh flotation sulphide concentrate.

(7) The desulphidized ore was then sized on 100 and 200 mesh screens, and the three products tabled separately. The plus 100 mesh tailings were ground minus 100 mesh, sized on a 200 mesh screen, and the products tabled. The two plus 200 mesh tailings were ground minus 200 mesh and tabled.

Overall results were:—

	Per Cent		
Product	Weight	Tin	Tin Distribution
Plus 100 mesh gravity concentrate	0.122	58.5	7.4
Plus 200 mesh gravity concentrate	0.325	57.2	19.2
Minus 200 mesh gravity concentrate	0.628	57.6	37.5
Concentrate from sulphide retreatment	0.074	48.7	3.7
Combined concentrates	1.149	57.0	67.8

Product	Per Cent Weight	Tin	Tin Distribution
Gravity tailings	71.016	0.29	21.6
Minus 200 mesh sulphides	26.217	0.33	9.1
Coarse sulphide cleaner tailing	1.618	0.92	1.5
Calculated Head	100.000	0.97	100.0

The overall recovery of 67.8 per cent of the tin in a concentrate assaying 57.0 per cent tin can be regarded as quite satisfactory. Concentrate grade is a little low, but this is primarily due to the comparatively small quantity of concentrates available to upgrade, rather than to any inherent difficulty in obtaining higher grade concentrates.

The comparatively small quantity of concentrate coarser than 100 mesh is interesting, and confirms earlier observations relating to the fine nature of the cassiterite in the ore.

The quantity of tin discarded in the cleaner tailings amounted to 1.6 per cent of the total tin in the ore. At this stage flotation cleaning of the coarse sulphide concentrate does not appear warranted, as the contained tin is comparatively small and in any case would be mostly recoverable after grinding the coarse sulphides.

By reconstitution of the several concentrates the proportions of recoverable cassiterite liberated by the initial grinding, and by primary and secondary regrinding, can be calculated.

Product	Per Cent Weight	Tin	Tin Distribution
Concentrates from primary grind	0.756	61.5	48.1
Concentrates from primary regrind	0.243	48.7	12.3
Concentrates from secondary regrind	0.076	47.5	3.7
Concentrates from sulphide retreatment	0.074	48.7	3.7
	1.149	57.0	67.8

In the above tests, results relating to the treatment of the sulphides should be regarded as indicative only. There was a fairly long time interval during the retreatment, and it is tolerably certain that there was some oxidation of the sulphides, particularly pyrrhotite.

It should be emphasized that the table concentration results in the above test are probably superior to what could be obtained commercially, as the table feed rates were very regular and low, while a large middling was produced during each tabling step and retabled carefully before grinding, &c., for the next stage of concentration. These remarks also refer to the other tests reported below.

The floated sulphides contained 34.5 per cent sulphur, and the desulphidized ore contained 2.1 per cent sulphur. The floated sulphides contain 85.4 per cent of the sulphur in the ore.

Test R.421/8

This test is similar to test R.421/5, and was designed to check the basic recovery data obtained in that test. Some procedural details have been modified on the basis of data obtained from the earlier tests.

Procedure was—

(1) The minus 8 mesh ore was batch ball mill ground minus 60 mesh.

(2) Sulphides were removed by flotation with:
Copper sulphate: 1.10 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.16 lb./ton of ore.
Sodium amyl xanthate: 0.16 lb./ton of ore.
Methyl isobutyl carbinol: 0.18 lb./ton of ore.
Flotation time: 7 minutes.

(3) The sulphides were batch ball mill ground minus 200 mesh and floated using:
Copper sulphate: 0.28 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0:06 lb./ton of ore.
Sodium amyl xanthate: 0:06 lb./ton of ore.
Methyl isobutyl carbinol: 0.05 lb./ton of ore.
Flotation time: 6 minutes.

(4) The sulphides were then recleaned, initially without further reagent addition, but adding towards the end of flotation:
Sodium ethyl xanthate: 0.01 lb./ton of ore.
Sodium amyl xanthate: 0.01 lb./ton of ore.

(5) Total reagents used were thus:

	Pounds per Ton of Ore			
	Rougher	Cleaner	Recleaner	Total
Copper sulphate	1.10	0.28	—	1.38
Sodium ethyl xanthate	0.16	0.06	0.01	0.23
Sodium amyl xanthate	0.16	0.06	0.01	0.23
Methyl isobutyl carbinol	0.18	0.05	....	0.23

(6) Flotation sink from the cleaner and recleaner flotation of the sulphides were combined and tabled to give a concentrate and a tailing. The tailing was returned to the desulphidized ore for sizing, concentration and regrinding, &c.

(7) The desulphidized ore was sized on 100 and 200 mesh screens, and the three products tabled. The plus 100 mesh tailings were ground minus 100 mesh, sized on a 200 mesh screen and tabled. The two plus 200 mesh tailings were ground minus 200 mesh and tabled.

(8) The combined minus 200 mesh table tailings were classified hydraulically, and the sands were batch ball mill ground until the tailings were minus approximately 40 microns cassiterite equivalent. Finer grinding was not considered warranted, as the cassiterite becomes increasingly difficult to recover by gravity below about 20-30 microns, and only minor amounts are normally recoverable in the 10-20 micron range.

(9) The finely ground bulked tailings were deslimed in the 3-inch Warman cyclone at a pressure of 30 pounds per square inch. Density of the pulp was approximately 5 per cent solids. Cyclone underflow was diluted to about 5 per cent solids and recycloned.

(10) The deslimed material was then concentrated on a Buckman tilting deck, and the tilting deck concentrate retreated once on the same deck.

(11) The tilting deck concentrate was upgraded by careful tabling on the slime table. The table tailing was retained separately as a middling, and is termed "tilting deck middling" in the tabulation below:—

Product	Per Cent Weight	Per Cent Tin	Per Cent Tin Distribution
Combined table concentrates	1.027	58.1	62.7
Tilting deck concentrates	0.064	44.1	3.0
Concentrate from sulphide retreatment	0.084	36.1	3.2
Combined concentrates	1.175	55.8	68.9
Tilting deck middling	2.985	1.10	3.5
Tilting deck tailing	49.30	0.32	16.6
Slimes	25.94	0.24	6.56
Sulphides	20.60	0.21	4.5
Calculated Head	100.000	0.95	100.0

Agreement between the results of tests R.421/5 and R.421/8 is very close, thus:

Test No.	Combined Concentrates Per Cent Weight	Tin	Tin Distribution
R.421/5	1.149	57.0	67.8
R.421/8	1.175	55.8	68.9

The sulphides from R.421/8 contain 0.21 per cent tin as against 0.33 per cent tin for R.421/5. However, this difference is probably more apparent than real, as in R.421/5 the gravity tailing from concentration of the flotation sinks was returned to the sulphides. In R.421/8 these tailings were returned to the desulphidized ore.

An infrasizer analysis of the sulphides R.421/8 gave—

Infrasizer Fraction	Per Cent Weight	Per Cent Tin	Per Cent Tin Distribution
1	27.9	0.36	48.6
2	21.1	0.21	21.5
3	19.2	0.13	12.1
4	13.2	0.16	9.5
5	6.0	0.09	2.6
6	3.8	0.08	1.5
7	9.7	0.09	4.2
Calculated Head	100.0	0.21	100.0

This infrasizing indicates that better liberation of the cassiterite may be obtained by finer grinding still, but it is doubtful if much of the cassiterite would be of sufficient size to economically recover and upgrade by gravity concentration. A vanning assay on a sample of the sulphides gave a recovery of 33 per cent of the contained tin in a concentrate assaying 46.2 per cent tin. As the total sulphides contain only 4.5 per cent of the tin in the ore, this indicates a maximum additional recovery of about 1.5 per cent of the tin in the total ore. On a commercial scale, involving grinding of the sulphides rather than acid dissolution, the probable recovery would be considerably less than this 1.5 per cent.

The floated sulphides contained 39.6 per cent sulphur, and the desulphidized ore 2.9 per cent sulphur. The floated sulphides contained 84.2 per cent of the sulphur in the ore.

The tilting deck middling contains 1.10 per cent tin, or some 3.5 per cent of the tin in the ore. A vanning assay on this product gave a recovery of 31.5 per cent of this tin in a concentrate assaying 46.3 per cent tin. This represents an overall recovery of approximately 1 per cent of the tin in the ore, and would approximate the maximum additional recovery to be expected by retreatment of this product by vanners, &c.

An infrasizer analysis of the tilting deck tailing gave—

Infrasizer Fraction	Per Cent Weight		Tin	Tin Distribution
1	6.6		0.51	9.7
2	27.1		0.24	18.8
3	23.3		0.23	15.4
4	17.4		0.31	15.5
5	13.1		0.51	19.3
6	7.8	12.5	0.59	21.3
7	4.7			
Calculated Head	100.0		0.35	100.0

The slimes (cyclone overflow) were also infrasized as follows:—

Infrasizer Fraction	Per Cent Weight		Tin	Tin Distribution
1	0.9			
2	0.4			
3	0.6	6.2	0.28	8.2
4	0.6			
5	3.7			
6	30.9		0.20	29.2
7	62.9		0.21	62.5
Calculated Head	100.0		0.21	100.0

Desulphidization of Table Concentrate

An extended test was carried out to determine recovery obtainable by treatment similar to that used on some low sulphide tin ores. This involves gravity concentration of the ground ore to produce a sulphide-cassiterite concentrate and desulphidization of this concentrate by flotation.

Test R.421/6A

The procedure was—

(1) The minus 8 mesh ore was batch ball mill ground to minus 60 mesh.

(2) The ground ore was sized on 100 and 200 mesh screens, and the three products tabled separately. The plus 100 mesh tailings were ground minus 100 mesh, sized on a 200 mesh screen and the products tabled. The two plus 200 mesh tailings from the above operations were ground through a 200 mesh screen and tabled. A sulphide-cassiterite concentrate was removed in all cases.

(3) The seven sulphide-cassiterite concentrates were bulked and accurately halved on a rotary distributor. One half of the concentrate was set aside for test work involving roasting—this work is detailed later.

(4) The sulphide-cassiterite concentrate was desulphidized by flotation, using—

Copper sulphate: 0.55 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.07 lb./ton of ore.
Sodium amyl xanthate: 0.07 lb./ton of ore.
Methyl isobutyl carbinol: 0.013 lb./ton of ore.
Flotation time: 5 minutes.

(5) The sulphide float was cleaned by reflotation with—
Sodium ethyl xanthate: 0.02 lb./ton of ore.
Sodium amyl xanthate: 0.02 lb./ton of ore.
Methyl isobutyl carbinol: 0.003 lb./ton of ore.
Flotation time: 5 minutes.

(6) The sinks from the rougher and cleaner flotation steps were combined and carefully tabled to produce a primary concentrate and a table tailing. The tailing was added to the floated sulphides and the combined product was ground to minus 200 mesh.

(7) The sulphides from the minus 200 mesh product were removed by flotation using—

Copper sulphate: 0.22 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.05 lb./ton of ore.
Sodium amyl xanthate: 0.05 lb./ton of ore.
Methyl isobutyl carbinol: 0.005 lb./ton of ore.
Flotation time: 5 minutes.

(8) The floated sulphides were then cleaned using—
Sodium ethyl xanthate: 0.01 lb./ton of ore.
Methyl isobutyl carbinol: 0.005 lb./ton of ore.

(9) Flotation sinks from the rougher and cleaner flotation of the minus 200 mesh sulphides were combined and tabled to produce a secondary concentrate and a tailing which has been designated "sulphide middling".

Final products were—

Product	Per Cent Weight	Tin	Tin Distribution
Primary concentrate	0.813	52.2	43.1
Secondary concentrate	0.389	56.9	22.4
Combined concentrate	1.202	53.7	65.5
Sulphide middling	1.593	2.13	3.5
Sulphide float	9.067	1.14	10.5
Sulphide-cassiterite concentrate	11.862	6.64	79.5
Gravity tailing	88.138	0.23	20.5
Calculated Head	100.000	0.99	100.0

Total reagents used for the stages of flotation were—

	Pounds per Ton of Ore				
	—60 mesh concentrate		—200 mesh sulphide		
	Rougher	Cleaner	Rougher	Cleaner	Total
Copper sulphate	0.55	—	0.22	—	0.77
Sodium ethyl xanthate	0.07	0.02	0.05	0.01	0.15
Sodium amyl xanthate	0.07	0.02	0.05	—	0.14
Methyl isobutyl carbinol	0.013	0.003	0.005	0.005	0.026

It is possible that reagent usage was in excess of consumption. It will be noted that the quantities of reagents used by this procedure are considerably less than that used in tests R.421/5 and R.421/8 (when the sulphides were floated from the ore). There are several reasons for this, including—

(*a*) flotation feed did not contain reagent consuming slime gangue minerals nor did it contain the very fine sulphides, which would consume proportionately more reagents:

(*b*) total sulphides removed by flotation amounted to 11.9 per cent by weight of the ore in R.421/6, compared with 20.6 per cent by weight of the ore in R.421/8.

The sulphides, even though they have been floated a total of four times, assay 1.14 per cent tin, and contain a total of 10.5 per cent of the tin in the original ore. A vanning assay carried out on this product gave a recovery of 60.7 per cent of the tin in a concentrate assaying 57.7 per cent tin.

An infrasizer analysis of the sulphides gave—

Infrasizer Fraction	Per Cent Weight	Tin	Tin Distribution
1	41.7	0.80	29.3
2	20.8	0.96	17.5
3	13.2	1.95	22.6
4	7.0	2.55	15.6
5	4.4	1.47	5.7
6	3.9	1.17	4.0
7	9.0	0.67	5.3
Calculated Head	100.0	1.14	100.0

The high tin content of the sulphides and the manner of distribution of the tin in the sizing were unexpected. The product contains some 10.5 per cent of the tin in the ore, and the vanning assay indicates that about 60 per cent of this (equivalent to about 6 per cent of the tin in the ore), is potentially recoverable.

Further work along the lines of this test, with more attention to the floated sulphides, is necessary.

Desulphidization by Roasting

For many years the concentration of sulphide-cassiterite ores at Mount Bischoff involved gravity concentration of the ground ore to a sulphide-cassiterite concentrate. This concentrate was then roasted and upgraded to sale grade concentrates by further gravity concentration and regrinding steps.

Test R.421/4

In test R.421/4 reported above, a sulphide-cassiterite concentrate was made with the following overall result:—

Product	Per Cent Weight	Tin	Tin Distribution
Combined concentrates	24.77	3.38	83.5
Combined tailings	75.23	0.22	16.5
Calculated Head	100.00	1.00	100.0

The combined concentrates actually consisted of some 15 individual concentrates. These were all roasted separately to a good red heat with intermittent stirring. The roasted concentrates were then upgraded by panning and regrinding. Overall results were—

Product	Per Cent Weight	Tin	Tin Distribution
Roasted final concentrates	1.060	53.1	56.3

This overall result represents a recovery of 67.4 per cent of the tin in the sulphide-cassiterite concentrate. Due to the very small quantities of most of the individual concentrates it was not possible to duplicate the roasting procedures. Unfortunately in the case of three sulphide-cassiterite concentrates (containing a total of 4.87 per cent of the tin in the original ore), roasting temperatures were apparently excessive, and the material fritted, giving comparatively low recoveries from these three products. It is possible that the overall recovery of the cassiterite in the roasted concentrate would have increased by say 1-2 per cent (over the 55.7 per cent actually obtained) if roasting conditions had been more correctly maintained.

Test R.421/6B

As mentioned previously, in test R.421/6A, the sulphide-cassiterite concentrate was accurately halved in a rotary distributor. One half of this concentrate was set aside for concentration tests involving roasting, &c.

Overall results to the sulphide-cassiterite concentrate stage were—

Product	Per Cent Weight	Tin	Tin Distribution
Sulphide-cassiterite concentrate	11.862	6.64	79.5
Combined tailings	88.138	0.23	20.5
Calculated Head	100.000	0.99	100.0

The composite sulphide-cassiterite concentrate was carefully roasted with constant stirring to a sweet roast. The roasted material was then upgraded by panning, with intermediate grinding, to a high grade concentrate. This procedure was carried out in duplicate, and the results quoted are the average of the duplicate.

Final result is given below, together with the result of test R.421/4 for comparison.

Product	Per Cent Weight	Tin	Tin Distribution
Roasted concentrate R.421/6B	0.937	61.2	57.9
Roasted concentrate R.421/4	1.060	53.1	56.3

These results are in reasonable agreement, considering that three small concentrates in test R.421/4 were incorrectly roasted.

Duplicate vanning assay determinations were also carried out on the bulk sulphide-cassiterite concentrate, and the average of these duplicates were—

Product	Per Cent Weight	Tin	Tin Distribution
Vanned concentrate R.421/6B	1.074	61.3	66.5

This can be regarded as close to the optimum recovery obtainable from the product by gravity concentration.

A comparison of the various results is of interest—

Product	Per Cent Weight	Tin	Tin Distribution
R.421/6 Sulphide - cassiterite concentrate	11.826	6.64	79.5
R.421/6B Vanned concentrate	1.074	61.3	66.5
R.421/6B Roasted concentrate	0.937	61.2	57.9
R.421/6A Concentrate after desulphidization by flotation	1.202	53.7	65.5

It will be noted that the recovery in R.421/6A at 65.5 per cent is close to the recovery in the vanned concentrate at 66.5 per cent of the tin. However, concentrate grade is appreciably lower at 53.7 per cent tin as compared with 61.3 per cent in the vanned concentrate.

Overall it appears that with this particular ore, roasting of a sulphide-cassiterite concentrate will give lower recoveries than desulphidization by flotation.

Disintegration of Pyrrhotite: Test R.421/9.

In the "Chemical Methods of Tin Extraction from West Tasmanian Ores," Part 1, Whitehead mentioned that pyrrhotite "when heated to 550°C in the absence of air it looses strength and disintegrates to a powder which is more magnetic than the original ore".

The Mount Cleveland bulk sample contains about 25 per cent by weight of sulphides, of which about half appears to be pyrrhotite. It was therefore decided to heat a sample of the ore to determine if the disintegration of the pyrrhotite would assist concentration of the cassiterite.

A sample of minus 8 mesh ore was heated to 550-560°C in an electric furnace for six hours, after being brought up to temperature overnight. The ore container excluded air. At the end of the heating period, the ore was quenched in cold water. There was negligable dust loss on quenching and only a very small amount of SO_2 evolved during the transfer to the quench bath. The quench was violent, with considerable evolution of steam.

The quenched ore weighed 91.0 per cent of the original weight taken. A sizing of the ore before and after heating and quenching gave—

Mesh	Before Heating	After Quenching
Plus 10	8.1	7.8
Plus 22	45.2	44.6
Plus 44	14.3	13.7
Plus 60	5.6	5.4
Plus 85	4.4	5.3
Plus 100	1.8	1.9
Plus 120	2.1	2.2
Plus 150	2.0	2.0
Plus 200	2.0	2.4
Minus 200	14.5	14.7
	100.0	100.0

The sizings show only minor disintegration due to the heating and quenching of the ore. The quenched ore generally was coloured black.

The quenched ore was batch ball mill ground to minus 60 mesh, and sulphides were floated with—

Copper sulphate: 1.10 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.22 lb./ton of ore.
Sodium amyl xanthate: 0.22 lb./ton of ore.
Methyl isobutyl carbinol: 0.08 lb./ton of ore.
Flotation time: 7 minutes.

The sulphides were then batch ground minus 200 mesh and refloated using—

Copper sulphate: 0.44 lb./ton of ore.
Conditioning time: 5 minutes.
Sodium ethyl xanthate: 0.22 lb./ton of ore.
Sodium amyl xanthate: 0.22 lb./ton of ore.
Methyl isobutyl carbinol: 0.09 lb./ton of ore.
Flotation time: 7 minutes.

The rougher and cleaner flotation sinks were bulked and sized on 100 and 200 mesh screens and the products tabled. The tailings were progressively ground through 100 and 200 mesh screens as outlined in previous tests, and tabled.

During tabling it was very difficult to distinguish tin from some of the gangue minerals, as the bulk of the sample was black or greyish-black coloured.

Overall results are tabulated below. These results are calculated on the basis of the quenched ore, which in turn is 91.0 per cent by weight of the original ore.

Product	Per Cent Weight	Per Cent Tin	Per Cent Tin Distribution
Concentrates	0.975	65.0	59.6
Middling	0.466	15.0	6.6
Combined conc. and middling	1.441	48.8	66.2
Sulphides	10.699	0.49	4.9
Gravity tailing	87.86	0.35	28.9
Calculated Head	100.000	1.06	100.0

The calculated head at 1.06 per cent tin on the heated ore is equivalent to 0.97 per cent tin on the original ore.

A middling product was made during the tabling of the minus 200 mesh material because it was extremely difficult to distinguish between the cassiterite and the heavy gangue minerals in the heated ore.

Overall recovery at 66.2 per cent is similar to that obtained in tests R.421/5, R.421/6A and R.421/8. Concentrate grade at 48.8 per cent tin is lower than the 53.7 per cent tin in the other tests. Increase of the grade to this level would possibly decrease the recovery of the tin by less than 1 per cent.

Heating and then grinding the ore does not appear to offer any particular advantage over the normal concentration methods. Details of this test are included mainly because it reinforces the position of the overall recoveries obtained in tests R.421/5, R.421/6A and R.421/8.

The above test must be regarded as preliminary only, and no other tests were carried out at different temperatures.

Determination of the Grain Size of the Cassiterite

The grain size range of the cassiterite in the ore was determined by Mr. C. J. Penman, Senior Chemist, as follows:—

This method was developed in 1940 for examination of Renison Bell ores (R.48) and the treatments result in the decomposition of most associated minerals such as sulphides, carbonates and quartz leaving the cassiterite unattacked in the residue. Sizing of this residue is a valuable guide to the recoverability of the cassiterite by gravity concentration. The method has an advantage over microscopical examinations of polished specimens in so far that reasonably accurate quantitative data are obtainable which is hardly practicable by examination of polished specimens.

Sample

Head sample ground to minus 22 mesh B.S.S.

Treatment

To remove gangue and liberate all composite cassiterite particles, the sample was subjected to the following treatment:—

1. Nitric acid was added to decompose sulphides and the solution finally heated to oxidize sulphur particles.
2. The solution was diluted and solids allowed to settle out overnight. The liquid was then removed by siphoning, and the residue washed by diluting, settling out for at least eight hours followed by siphoning off the liquid and repeating twice.
3. The residue was then completely washed into a polyethylene beaker, the solids allowed to settle as in 2 and the bulk of the liquid siphoned off, and the remainder evaporated.
4. Silica and silicates were then largely removed by adding hydrofluoric acid and evaporating over a water bath to volatilize silica as the tetrafluoride. This requires a number of additions of hydrofluoric acid and several days are required for this stage.
5. The dried residue was then washed into a glass beaker and boiled with nitric acid to dissolve fluoride, &c., then diluted, settled and washed as in 2.
6. The residue was then transferred to a platinum basin and given a final treatment with hydrofluoric acid to remove small remaining amounts of silica, &c., finally evaporating to dryness.
7. The residue was then heated with nitric acid, diluted and washed as in 2.
8. The final settled solids, essentially cassiterite, were dried and weighed.

Determination of the Grain Size

The dried cassiterite was then screened as shown below, and that portion passing through a 350 mesh screen was elutriated in a Bardwell Tube to determine the finer sizes.

Note that the minus 200 mesh plus 350 mesh fraction corresponds to elutriator fraction 1. For this fraction the screen was preferred.

The results were as follows:—

Fraction	Mean Diam. Microns	Tin Distribution Per Cent	Tin Distribution Per Cent Cumulative
+100 mesh	—152	16.0	16.0
—100 +150 mesh	—152 +104	11.1	27.1
—150 +200 mesh	—104 + 76	18.1	45.2
—200 +350 mesh	— 76 + 43	19.6	64.8
Elutriator F/2	38	10.6	75.4
Elutriator F/3	27	10.9	86.3
Elutriator F/4	20	9.0	95.3
Elutriator F/5	10	2.7	98.0
Elutriator F/6	— 10	2.0	100.0

Composite Head:—0.93 per cent tin.

This sizing indicates that only a small proportion of the cassiterite exists in the ore in sizes which would be irrecoverable by gravity concentration. However, there will naturally be depreciation of the cassiterite grain size during crushing and grinding operation prior to concentration.

REFERENCE

WHITEHEAD, A. B.—Chemical methods of tin extraction from West Tasmanian ores. *Coun. sci. ind. Res. Org.* Rep. No. CRL/CE/R-6.

26. RENISON ASSOCIATED TIN MINES