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R. 413

CONCENTRATION OF A BULK SAMPLE FROM WHITE CRYSTAL AND WILD CHERRY AREAS, ARDLETHAN, N.S.W.

Summary

Concentration tests have been carried out on a bulk sample from current wagon drilling at Ardlethan, N.S.W. The bulk sample contained 0.42 per cent tin, and represents a total of approximately 2,470 feet of drilling from 12 White Crystal and 7 Wild Cherry bores.

Overall concentration results were—

Product	Weight %	Assay % Sn	Distribution % Sn
Combined concentrates ..	0.436	60.9	63.3
Combined tailings	99.564	0.16	36.7
Composite head	100.000	0.42	100.0

The method of treatment was—

(1) The bulk sample was ground to minus 25 mesh by rod-milling in closed circuit with a Hummer screen.

(2) The ground ore was hydraulically classified and cycloned to give 5 products, of which 4 were concentrated. The slimes were not concentrated.

(3) The three coarsest sized fractions were concentrated by tabling, desulphidized by flotation and then upgraded by tabling and magnetic separation.

(4) Middlings from these fractions were ground and concentrates were obtained by tabling and upgrading in a similar manner.

(5) The fine fraction of the ground ore was concentrated on a Buckman tilting deck and upgraded by desulphidizing and tabling.

(6) A partial analysis of the final concentrate is shown, but there was insufficient quantity of concentrate produced to investigate the removal of residual impurities.

All treatments were carried out on pilot scale models of commercial machines. Results of commercial scale treatment of the ore should approximate those summarized above, provided the concentration plant is effectively designed and provides for all treatments reported herein.

Introduction

Tin determination on samples from current wagon drilling operations at Ardlethan, N.S.W., have been carried out in the Department of Mines, Metallurgical Laboratories, Launceston, Tasmania.

In general, each drill hole is about 150 feet deep, and 16 samples were received per hole. These samples consist of 15 samples, each representing 10 feet of drilling, and a sample representing the total hole. This last sample was composed at Ardlethan.

Sample

The wagon drill samples received from Ardlethan were reduced in bulk by riffling, and a sample removed for assay sample preparation. The remainder of the sample was stored for possible future requirements.

A bulk sample of approximately 350 pounds weight was made by combining the samples from 19 wagon drill holes, specified by Mr. H. K. Turner of Aberfoyle Tin N.L. Of the 19 drill holes, 12 are in the White Crystal area and 7 are in the Wild Cherry area.

The bulk sample consisted of 255 individual samples, representing approximately 2,470 feet of drilling. The weights of the 255 individual samples added to the bulk sample are available from this laboratory, if required. The whole of each available sample was used. In general the weights of the individual samples were rather uniform, the weight of the smallest sample being about 70 per cent of the weight of the largest sample.

The samples representing the total hole as received from Ardlethan were not incorporated in the bulk sample.

Data relating to the samples from the various wagon drill holes are:—

Lab. Reg. Nos.	Area	Bore No.	Numerical Average of Assays % Sn	Approx. Footage of Hole	Total No. of Individual Samples
752- 760	White Crystal	Line NA. 5	0.51	73	8
975- 990		NB. 5	0.42	147	15
1007-1021		NK. 2	0.62	135	14
2052-2058		NK. 4	0.32	65	6
1199-1213		NL. 4	0.36	145	15
991-1006		NM. 2	0.34	147	15
1247-1261		CR. WA. 11	0.37	145	15
1215-1229		CR. WA. 13	0.31	145	15
1263-1277		CR. WA. 14	0.44	145	15
1420-1434		CR. WA. 15	0.37	146	15
1436-1446		CR. WA. 16	0.31	101	11
1834-1844		CR. WA. 28	0.49	107	11
2178-2192	Wild Cherry	Line EA. 5	0.52	135	14
2162-2177		EA. 6	0.31	145	15
514- 529		EF. 6	0.42	147	15
482- 497		EF. 7	0.43	147	15
326- 338		EH. 1	0.40	115	12
435- 449		EL. 1	0.25	132	14
606- 621		EL. 4	0.35	147	15
Total		19 Bores	....	2469	255

The bulk sample assayed 0.42 per cent Sn and 0.35 per cent S. Specific gravity of the sample was 2.79.

Sizing analysis of the bulk sample:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
+ 5 mesh	1.7	9.4	0.35	8.1
10	7.7			
25	20.5		0.32	16.1
36	9.2		0.37	8.3
44	4.6		0.31	3.5
60	8.2		0.29	5.8
85	9.2	12.3	0.45	13.6
100	3.1			
120	4.6			
150	3.4	12.0	0.48	14.1
200	4.0			
— 200	23.8		—	—
Infrasizer fraction 1	1.9		2.48	11.5
2	5.5		0.50	6.7
3	3.7		0.42	3.8
4	2.4		0.50	2.9
5	1.8		0.41	1.8
6	2.7		0.18	1.2
7	5.8		0.18	2.6
Composite Head	100.0		0.41	100.0

It will be noted that the material as received from Ardlethan contains appreciable fines.

The sample tested represents basically the surface 150 feet of the deposit. From previous ore dressing investigations (R.390, R.392, R.394-R.396) on diamond drill core to 250 feet depth, it is known that the sulphide content of the ore increases generally with depth to about 5 per cent by weight in the 200-250 feet region. This is illustrated by the quantities of sulphides floated in the above mentioned investigations.

Approx. Depth Feet	Weight per cent of Sulphides Floated				
	Hole DP-1 R.390	Hole DP-2 R.392	Hole NC-1 R.394	Hole DP-3 R.395	Hole DP-4 R.396
0- 50	negligible	negligible	negligible	negligible	*
50-100	negligible	negligible	negligible	negligible	3.65
100-150	negligible	0.39	0.81	1.05	1.08
150-200	1.68	1.94	—	4.01	2.96
200-250	2.87	5.44	—	5.52	4.76

* The top 30 feet of this hole was very low grade and was not used in the samples for the ore dressing investigation.

When mixing the bulk sample, it was very obvious that there were wide variations in the colour of the samples. As a general rule, the surface samples were much redder than samples from near the bottom of the hole. These differences would be due to iron oxides near the surface, while at depth the pyrite had not been oxidized.

The bulk sample assayed 0.35 per cent S, which is equivalent to about 0.7 per cent sulphides in the ore. This is, of course, considerably less than the sulphide content of ore in the 200-500 feet region.

Investigation

Aberfoyle Tin N.L. (Tin Development Partnership) requested the investigation to obtain data relating to the commercial treatment of the deposit.

During the investigation the following factors were considered:

(1) All treatments should approximate commercial practice, and would be carried out on pilot scale models of commercial machines.

(2) Middlings from each section were to be treated separately in order to more fully assess any problems inherent in middlings treatment.

(3) Overall concentration results should show separation into concentrate and tailing only.

(4) Final concentrates should be high grade, and should be low penalty sale grade concentrate.

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

It will be noted that weight and distributional 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 very small fractions produced during the investigation.

Previous investigations

Five more or less preliminary investigations on diamond drill cores NC-1 and DP-1 to DP-4 inclusive have been carried out previously (See R.390, R.392, R.394 in Tech. Rep. 6 and R.395, R.396 in this volume.) Excluding the first investigation (R.390) in which middlings were not reground and retreated, the results of the other investigations were similar, with the exception that concentrate grade was increased somewhat in the later investigations.

Investigation	Hole No.	Assays: % Sn Head	Conc.	Tail.	Recovery % Sn
R.392	DP-2	0.34	9.9	0.13	64.9
R.394	NC-1	0.51	25.7	0.17	67.7
R.395	DP-3	0.58	26.7	0.17	69.9
R.396	DP-4	0.56	33.7	0.22	60.0
Arithmetical Average		0.50	24.0	0.17	65.6

In these investigations, the ore samples were roll crushed to minus 22 mesh and then ball mill ground to minus 44 mesh. The ground ore was screened into four sized fractions and tabled separately. Tabling showed that the minus 44 plus 60 mesh fraction gave satisfactory tailings, provided a reasonable quantity (say 5-10 per cent by weight) of the material was taken as a middling for regrinding and retreatment.

On samples of the five drill cores crushed through 5 mesh, heavy liquid separations were carried out at densities of about 2.9. The tin content of the minus 5 plus 10 mesh, and of the minus 10 plus 30 mesh float products were—

Investigation	Assay: % Sn Minus 5 plus 10 mesh	Minus 10 plus 30 mesh
R.390	0.25	0.15
R.392	0.21	0.10
R.394	0.22	0.13
R.395	0.26	0.10
R.396	0.20	0.14
Average	0.23	0.12

These figures clearly show that it is most unlikely that a gravity tailing of low content could be produced at sizes coarser than 10 mesh. The tin content of the minus 10 plus 30 mesh float products are comparatively low. As the tin content of the plus 10 mesh float product is roughly twice that of the minus 10 plus 30 mesh float product, it is highly probable that the tin content of the minus 10 mesh plus 30 mesh float product decreases significantly with size within the fraction.

Following examinations of the results of previous investigations, it was decided to grind to a little coarser than 30 mesh, and a ton-cap screen of approximately 25 mesh was chosen.

Examinations of the various sink products in the previous investigations had shown very minor amounts of free cassiterite. From these tests it appeared unlikely that any significant quantity of free cassiterite would be obtained at comparatively coarse sizes by (say) jigging the rod mill circulating load. However, this is a factor that would be best assessed from examination of actual mill products.

Method of Treatment

The method of treatment followed in the present investigation follows modern gravity concentration practice. A number of rather arbitrary decisions regarding some aspects had to be made. In this we have been guided by data obtained from other investigations and mill surveys of various gravity mills. In particular, data from the gravity plants at Rossarden (Aberfoyle Tin N.L.), Storey's Creek, Mount Bischoff, Renison Bell and King Island were considered.

Crushing

As the sample as received was virtually minus 5 mesh, no crushing was required.

Degree of Grind

As discussed above, a grind to minus 25 mesh was chosen on the basis of data from previous investigations on diamond drill core from the deposit.

Method of Grinding

Rod milling was an obvious choice to give a finished product from (say) minus $\frac{5}{8}$ or $\frac{3}{4}$ inch ore. An alternative was roll crushing to about $\frac{1}{4}$ inch, followed by ball mill or rod mill grinding. There does not appear to be any advantage in a two stage reduction, as it is not possible to reject tailings of low tin content at sizings coarser than 10 mesh (and probably rather finer than this).

Screens or Classifiers

Use of a screen to define the degree of grind is based on the fact that any pulp classification to control the grinding circulating load will result in serious overgrinding of cassiterite. At Mount Bischoff it was definitely shown that replacement of the classifier by Hummer screens gave an increase of about 15 per cent in the recovery of tin by reducing overgrinding of the cassiterite.

Due to the comparatively fine nature of the bulk sample, it was decided to screen the material before rod milling

Hydraulic Classification and Number of Spigots

Concentrating tables operate most efficiently when handling a small size range. In gravity plants hydraulic classifiers often have up to about 10 spigots. As an example Aberfoyle have modified their classifier to give 7 spigot products, each feeding a jig or a table. In the present investigation there had to be some compromise between the desire for small size ranges for each concentration stage, and the need to simplify to avoid a multiplicity of products. As a result a three spigot classifier was used to give three separate spigot products for concentration by tabling. On a commercial scale it is, of course, probable that more than three sized products would be made by the hydraulic classifier.

Concentration of Plus 60 Mesh Material

Material in the size range 25 mesh to about 60 mesh can be concentrated efficiently by either jigging or tabling. At Aberfoyle and Storeys Creek, three hutch 36 inch Harz jigs handle material in this size range at the rate of about 0.8 t.p.h. at Aberfoyle, and about 1.2 t.p.h. at Storeys Creek. Similarly at Aberfoyle, each of sand tables 1, 2 and 3 handle about 0.8 to 1.2 t.p.h. of material in the 25 mesh to 100 mesh size range.

After rod milling, the screen undersize contained about 35 per cent by weight of plus 60 mesh material. This material could be concentrated either by jigging or tabling. On Aberfoyle and Storeys Creek data, there does not appear to be much difference in performance between a three hutch Harz jig and a sand table. For the sake of simplicity, tabling only was used in the present investigation.

Desliming Fines

The decision to deslime the fines before concentration was based mainly upon several investigations (of which R.379 is the most important) into recovery of tin from the slime table tailings at Aberfoyle. In this investigation it was shown that the true slimes (infrasizer 6 and 7 fractions) could be discarded by cycloning without loss of recoverble tin.

Tilting Deck Concentration of Fines

The decision to use Buckman tilting decks for preconcentration of the fines was based on the results of milling at Kimberley, Canada, and on our mill experience with tilting decks at Mount Bischoff.

Pilot Plant Equipment

To minimize repetition of detail in the body of the report, a brief summary of equipment used and some detail of operating conditions is given below.

Rod Mill

Denver laboratory rod mill, 3 feet by 1 foot diameter. Rod charge was:—

Rod Diameter	Weight: pounds
1 inch	58.2
7/8 inch	22.0
3/4 inch	32.8
5/8 inch	36.7
Total	149.7

Screen

Twenty inch by thirty inch type "38" Junior Hummer screen, fitted with No. 434 ton cap screen. Screen aperture is 0.023 by 3/16 inches. The short dimension is approximately equivalent to 25 mesh British standard screen.

Hydraulic Classifier

Three spigot Geco hydraulic classifier. Flow rates set to give an overflow approximately minus 40 microns (quartz equivalent).

Concentrating Table

Deister No. 15-5 Super Duty Diagonal deck laboratory table. Two interchangeable decks for "sands" and "slimes" respectively.

Cyclone

A three inch rubber lined series "A" Warman cyclone, fed by a 1/3/4 inch Warman split case pump with variable speed drive.

Cyclone operating pressure was 25 pounds per square inches. Spigot diameter 9/16 inch. Vortex finder diameter 5/8 inch.

Tilting Deck

Deck 12 inches wide, 6 feet long.
Slope: 2 inches per foot.
Feed rate: 1 gallon per minute/foot width.
Pulp density of feed: 20 per cent solids.
Feed cycle: 5 minutes.

Magnetic Separator

Two disc Rapid type "O" electro-magnetic separator.

Flotation Cell

Model D1 Denver "Sub A" laboratory flotation machine. The following reagents were used and appeared to be effective.

Copper sulphate, sulphuric acid, sodium ethyl xanthate, sodium amyl xanthate and cresylic acid.

pH of pulps: 6 approximately.

No attempt was made to determine optimum flotation conditions. In an operating plant, it is unlikely that sulphuric acid would be required.

Sizing Screens

Screen sizes throughout refer to British Standard Screens.

Infrasizer

Minus 200 mesh fractions were further sized by a Haultain infrasizer.

The estimated particle sizes of quartz and cassiterite segregated by the Haultain infrasizer are:—

Infrasizer Fraction	Quartz	Cassiterite
1	+56 microns	+45 microns
2	40—56 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

Concentration of the Sample

Grinding and Classification

The bulk sample was pulped and screened on the Hummer screen. Screen oversize was ground in the rod mill operating in closed circuit with the screen. Circulating load was estimated at about 100 per cent of new rod mill feed. A sizing of the ground ore is tabulated below.

Due to the comparatively fine nature of the bulk sample and from consideration of data from previous investigations, no attempt was made to assess the possibility of recovering free cassiterite from the screen oversize.

The screen undersize was sized into four products in the three spigot Geco hydraulic classifier.

Classifier overflow was deslimed in the 3 inch Warman cyclone. Cyclone underflow was diluted to about five per cent solids and recycloned.

From the sizing and cyclone operations, five products were obtained, namely:—

(1) No. 1 spigot product.
(2) No. 2 spigot product.
(3) No. 3 spigot product.
(4) Deslimed classifier product.
(5) Slimes.

The first four products were concentrated separately. The slimes were not concentrated.

The fractions were:—

Product	Weight %	Assay % Sn	Distribution % Sn
No. 1 Spigot product	39.35	0.61	57.52
No. 2 Spigot product	24.07	0.30	17.31
No. 3 Spigot product	16.33	0.34	13.31
Deslimed classifier overflow ..	7.68	0.35	6.44
Slimes	12.57	0.18	5.42
Composite head	100.00	0.42	100.00

Sizings of the various products were:—

Size Fraction	Ground Ore	No. 1 Spigot Product	No. 2 Spigot Product	No. 3 Spigot Product	Classifier Overflow	Slimes
	Weight Per Cent					
+ 25 mesh ..	3.3	6.7				
+ 36 mesh ..	11.5	27.2				
+ 44 mesh ..	9.1	13.5	8.6	1.1		
+ 60 mesh ..	12.8	26.0	13.7	2.5		
+ 85 mesh ..	14.3	18.4	28.1	6.2		
+100 mesh ..	3.4	3.1	10.8	3.6		
+120 mesh ..	5.9	2.8	16.8	9.1		
+150 mesh ..	3.5	1.0	9.4	10.0		
+200 mesh ..	4.7	0.8	7.3	23.5	Trace	Trace
(—200) mesh	(31.5)	0.5	5.3	(44.0)		
I.F. 1	4.7			18.1	0.8	1.4
I.F. 2	6.2			21.0	9.1	1.0
I.F. 3	4.0			3.3	34.4	1.2
I.F. 4	2.6			0.7	24.5	3.0
I.F. 5	2.6			0.5	15.8	6.8
I.F. 6	2.6			0.3	8.3	13.3
I.F. 7	8.8			0.1	7.1	73.3
Composite	100.0	100.0	100.0	100.0	100.0	100.0

The sizings for Nos. 1, 2 and 3 spigot products are sizings of the table tailings. As these tailings represent over 93 per cent of the weight of the feed in each case, it is unlikely that there is any significant difference between sizings of the table feed, and the respective table tailing.

Concentration of No. 1 Spigot Product

No. 1 spigot product was tabled on the Deister table to give—

(*a*) a concentrate.
(*b*) a primary table middling.
(*c*) a primary table tailing.

The table concentrate was upgraded by—

(1) removal of sulphides by flotation. The sulphides were refloated once.
(2) retabling to give an upgraded concentrate and a table middling.
(3) removal of magnetics by dry magnetic separation.

The table middling from upgrading the concentrate and the magnetics were added to the primary table middling. The combined middlings were progressively batch ball mill ground successively through 60, 100, 150 and 200 screens, with intermediate tabling of the coarsest fractions to produce a concentrate, a middling and a tailing. The middling was then batch ball mill ground through the next screen and the process continued. At each tabling, about half of the material was rejected as a tailing. No minus 200 mesh middling was made.

This procedure was adopted to simulate open circuit grinding of middlings, followed by hydraulic classification and tabling, as would probably be done on a commercial scale.

The combined table concentrates were desulphidized by flotation, and the magnetic minerals removed by magnetic separation.

A summary of the results of the concentration of No. 1 spigot product is given below.

To allow a better assessment of the table, the weights and distributions have been calculated on the basis of the whole bulk sample, and on the basis of No. 1 spigot product only.

Product	Weight Percent Whole Sample	Weight Percent No. 1 Spigot	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn No. 1 Spigot
Primary concentrate	0.221	0.561	65.1	34.46	59.90
Middling concentrate	0.067	0.171	46.6	7.50	13.05
Combined concentrates ..	0.288	0.732	60.8	41.96	72.95
Combined sulphides	0.369	0.937	0.43	0.38	0.66
Middling—					
— 60 mesh tailing	0.689	1.752	0.12	0.20	0.34
—100 mesh tailing	0.169	0.430	0.78	0.32	0.55
—150 mesh tailing	0.134	0.341	0.58	0.18	0.32
—200 mesh tailing	0.871	2.214	1.86	3.88	6.75
Middling magnetics	0.002	0.004	1.24	0.01	0.01
Primary table tailings ..	36.828	93.59	0.12	10.59	18.42
Combined tailings	39.062	99.268	0.17	15.56	27.05
Calculated head	39.35	100.000	0.61	57.52	100.00

A sizing analysis of No. 1 spigot primary table tailing gave:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
+ 25 mesh	6.7	22.0	0.17	30.0
+ 30 mesh	15.3			....
+ 36 mesh	11.9		0.15	14.3
+ 44 mesh	13.5		0.11	11.9
+ 60 mesh	26.0		0.08	16.7
+ 85 mesh	18.4		0.11	16.2
+100 mesh	3.1		0.09	2.2
+120 mesh	2.8		0.11	2.5
+150 mesh	1.0		....	....
+200 mesh	0.8	2.3	0.34	6.2
—200 mesh	0.5		....	....
Composite	100.0		0.12	100.0

The tabulation below shows the relative quantities and grades of the three middling products from the primary concentration of No. 1 spigot.

Product	Weight % Whole Sample	Weight % No. 1 Spigot	Assay % Sn	% Sn Distribution Whole Sample	% Sn Distribution No. 1 Spigot
Primary table middling	1.912	4.859	1.77	8.12	14.11
Middling from up-grading	0.050	0.127	33.3	3.98	6.93
Magnetics	0.001	0.002	8.9	0.02	0.03
Combined middlings	1.963	4.988	2.58	12.12	21.07

The specific gravity of the combined middlings was 3.35.

Concentration of No. 2 Spigot Product

No. 2 spigot product was concentrated in a manner similar to No. 1 spigot product, i.e.—

(1) tabled to give concentrate, middling and tailing;
(2) concentrate desulphidized by flotation;
(3) retabled to upgrade concentrate;
(4) magnetics removed by dry magnetic separation.

The three middling fractions were batch ball mill ground through 150 mesh, and tabled. The tailings were then batch ground through 200 mesh, and retabled. The combined middling concentrates were then upgraded by desulphidizing, retabling and dry magnetic separation.

A summary of the results of the concentration of No. 2 spigot product is given below. Weights and distributions have been calculated on the basis of the whole bulk sample, and on the basis of No. 2 spigot product only.

Product	Weight % Whole Sample	Weight % No. 2 Spigot	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn No. 2 Spigot
Primary concentrate	0.061	0.254	62.3	9.09	52.52
Middling concentrate	0.017	0.068	53.1	2.07	11.98
Combined concentrates	0.078	0.322	60.4	11.16	64.50
Combined sulphides	0.086	0.356	0.55	0.11	0.65
Middling —200 mesh tailing	0.862	3.581	0.67	1.38	7.96
Tailings from upgrading middling concentrate	0.008	0.034	11.1	0.22	1.25
Middling magnetics	0.002	0.007	10.0	0.04	0.23
Primary table tailing	23.034	95.70	0.08	4.40	25.41
Combined tailings	23.992	99.678	0.11	6.15	35.50
Calculated head	24.07	100.000	0.30	17.31	100.00

A sizing analysis of No. 2 spigot primary table tailing gave:—

Size Fraction	Weight %	Assay % Sn	Distribution % Sn
+ 44 mesh	8.6	0.05	5.7
+ 60 mesh	13.7	0.05	9.0
+ 85 mesh	28.1	0.06	22.2
+100 mesh	10.8	0.06	8.5
+120 mesh	16.8	0.07	15.5
+150 mesh	9.4	0.07	8.7
+200 mesh	7.3	0.09	8.7
—200 mesh	5.3	0.31	21.7
Composite	100.0	0.08	100.0

The following tabulation shows the relative quantities and grades of the three middling products from the primary concentration of No. 2 spigot.

Product	Weight % Whole Sample	Weight % No. 2 Spigot	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn No. 2 Spigot
Primary table middling	0.917	3.810	1.64	3.59	20.77
Middling from upgrading	0.004	0.017	13.4	0.13	0.75
Magnetics	0.002	0.009	5.6	0.03	0.17
Combined middlings	0.923	3.836	1.70	3.75	21.69

Concentration of No. 3 Spigot Product

No. 3 spigot product was concentrated in a manner similar to Nos. 1 and 2 spigot products, i.e.—

(1) tabled to give concentrate, middling and tailing;

(2) concentrate desulphidized by flotation;

(3) retabled to upgrade concentrate;

(4) magnetics removed by dry magnetic separation.

The middling fractions were batch ball mill ground through 200 mesh and tabled. The concentrate was upgraded by desulphidizing, retabling and dry magnetic separation.

A summary of the results of the concentration of No. 3 spigot product is given below.

Product	Weight % Whole Sample	Weight % No. 3 Spigot	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn No. 3 Spigot
Primary concentrate	0.052	0.316	65.3	8.01	60.18
Middling concentrate	0.003	0.020	38.3	0.29	2.23
Combined concentrates	0.055	0.336	63.7	8.30	62.41
Combined sulphides	0.043	0.265	0.45	0.05	0.35
Middling —200 mesh tailing	0.395	2.419	0.57	0.53	4.02
Tailings from upgrading middling concentrate	0.015	0.090	8.2	0.29	2.15
Middling magnetics	Trace	Trace	....	....	....
Primary table tailing	15.822	96.89	0.11	4.14	31.07
Combined tailings	16.275	99.664	0.13	5.01	37.59
Calculated head	16.33	100.00	0.34	13.31	100.00

A sizing analysis of No. 3 spigot primary table tailing gave:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
+ 44 mesh	1.1			
+ 60 mesh	2.5	9.8	0.11	9.1
+ 85 mesh	6.2			
+100 mesh	3.6	12.7	0.08	8.6
+120 mesh	9.1			
+150 mesh	10.0		0.06	5.1
+200 mesh	23.5		0.06	11.9
Infrasizer fraction 1	18.1		0.16	24.5
Infrasizer fraction 2	21.0		0.15	26.7
Infrasizer fraction 3	3.3			
Infrasizer fraction 4	0.7			
Infrasizer fraction 5	0.5	4.9	0.34	14.1
Infrasizer fraction 6	0.3			
Infrasizer fraction 7	0.1			
Composite	100.0		0.12	100.0

The relative quantities and grades of the three middling products from the primary concentration of No. 3 spigot are shown below:—

Product	Weight % Whole Sample	Weight % No. 3 Spigot	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn No. 3 Spigot
Primary table tailing	0.628	2.610	1.04	1.37	7.90
Middling from upgrading	0.003	0.013	8.9	0.06	0.34
Magnetics	0.003	0.011	9.3	0.05	0.30
Combined middlings	0.634	2.634	1.11	1.48	8.54

Concentration of Deslimed Classifier Overflow

The deslimed classifier overflow was concentrated on the Buckman tilting deck. Tilting deck concentrate was desulphidized by flotation, and then upgraded by repeated tabling on a Deister table, using a slime deck.

Weights and distributions of products have been calculated on the basis of the whole sample, and on the basis of the deslimed classifier overflow.

Product	Weight % Whole Sample	Weight % Classifier Overflow	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn Classifier Overflow
Concentrate	0.015	0.189	54.9	1.92	29.83
Tilting deck tailing	6.905	89.91	0.17	2.83	43.94
Sulphides	0.028	0.36	0.69	0.05	0.71
Slime table tailing 1	0.589	7.67	0.46	0.65	10.14
Slime table tailing 2	0.070	0.91	0.76	0.13	1.99
Slime table tailing 3	0.058	0.76	4.10	0.58	8.96
Slime table tailing 4	0.013	0.175	6.95	0.22	3.50
Slime table tailing 5	0.002	0.026	12.4	0.06	0.93
Combined tailings	7.665	99.811	0.24	4.52	70.17
Calculated head	7.680	100.00	0.35	6.44	100.00

Sizing analysis of the feed to the tilting deck was:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
+200 mesh	Trace	9.9	0.35	10.2
Infrasizer fraction 1	0.8			
Infrasizer fraction 2	9.1			
Infrasizer fraction 3	34.4		0.26	26.5
Infrasizer fraction 4	24.5		0.36	26.3
Infrasizer fraction 5	15.8		0.41	19.2
Infrasizer fraction 6	8.3	15.4	0.39	17.8
Infrasizer fraction 7	7.1			
Composite	100.0		0.34	100.0

Sizing analysis of the tailings from the tilting deck was:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
Infrasizer fraction 1	0.4	10.9	0.04	2.5
Infrasizer fraction 2	10.5			
Infrasizer fraction 3	34.1		0.06	11.9
Infrasizer fraction 4	24.6		0.14	20.1
Infrasizer fraction 5	15.9		0.35	32.5
Infrasizer fraction 6	9.1	14.5	0.39	33.0
Infrasizer fraction 7	5.4			
Composite	100.0		0.17	100.0

Slimes

Slimes produced by cycloning the hydraulic classifier overflow contained 0.18 per cent tin. These slimes were not concentrated.

The slimes had the following sizing analysis:—

Size Fraction	Weight %		Assay % Sn	Distribution % Sn
+200 mesh	Trace			
Infrasizer fraction 1	1.4	3.6	0.50	9.5
Infrasizer fraction 2	1.0			
Infrasizer fraction 3	1.2			
Infrasizer fraction 4	3.0		0.24	3.8
Infrasizer fraction 5	6.8		0.29	10.5
Infrasizer fraction 6	13.3		0.25	17.9
Infrasizer fraction 7	73.3		0.15	58.3
Composite	100.0		0.19	100.0

The slimes amounted to 12.57 per cent by weight of the total bulk sample, and contained 5.42 per cent of the tin.

The majority of the cassiterite in the slimes would not be economically concentrated by established gravity processes.

Summary of Concentration Results

The various fractions concentrated separately were:—

Product	Weight %	Assay % Sn	Distribution % Sn
No. 1 Spigot product	39.35	0.61	57.52
No. 2 Spigot product	24.07	0.30	17.31
No. 3 Spigot product	16.33	0.34	13.31
Deslimed classifier overflow	7.68	0.35	6.44
Slimes	12.57	0.18	5.42
Composite bulk sample	100.00	0.42	100.00

Concentration results for the various fractions can be summarized as:—

Product	Weight % Whole Sample	Weight % Individual Fraction	Assay % Sn	Distribution % Sn Whole Sample	Distribution % Sn Individual Fraction
No. 1 Spigot:					
Concentrate	0.288	0.732	60.8	41.96	72.95
Tailing	39.062	99.268	0.17	15.56	27.05
Composite	39.350	100.000	0.61	57.72	100.00
No. 2 Spigot:					
Concentrate	0.078	0.322	60.4	11.16	64.50
Tailing	23.992	99.678	0.11	6.15	35.50
Composite	24.070	100.000	0.30	17.31	100.00
No. 3 Spigot:					
Concentrate	0.055	0.336	63.7	8.30	62.41
Tailing	16.275	99.664	0.13	5.01	37.59
Composite	16.330	100.000	0.34	13.31	100.00
Deslimed Class. O'flow:					
Concentrate	0.015	0.189	54.9	1.92	29.83
Tailing	7.665	99.811	0.24	4.52	70.17
Composite	7.680	100.000	0.35	6.44	100.00
Slimes	12.570	100.000	0.18	5.42	100.00
Composite bulk sample	100.000		0.42	100.00	

Combining the various concentrates and tailings we get the overall result of the concentration test for the bulk sample.

Product	Weight %	Assay % Sn	Distribution % Sn
Concentrate:			
No. 1 Spigot	0.288	60.8	41.96
No. 2 Spigot	0.078	60.4	11.16
No. 3 Spigot	0.055	63.7	8.30
Class. O'flow	0.015	54.9	1.92
Total concentrate	0.436	60.9	63.34
Tailings:			
No. 1 Spigot	39.062	0.17	15.56
No. 2 Spigot	23.992	0.11	6.15
No. 3 Spigot	16.275	0.13	5.01
Class. O'flow	7.665	0.24	4.52
Slimes	12.570	0.18	5.42
Total tailings	99.564	0.16	36.66
Composite bulk sample	100.000	0.42	100.00

Thus the overall recovery was 63.34 per cent of the tin, in a concentrate assaying 60.9 per cent tin.

Discussion of Results

Overall results of the present investigation were more or less as anticipated from the examination of results of the previous investigations on diamond drill core samples. Treatment of similar ore on a commercial scale should approximate these results.

For some aspects, it is probable that conditions obtained in the investigation were superior to those likely to be attained in a commercial plant, and vice versa. In particular, it is considered that the very regular and low feed rates and continued attention, would give enhanced tabling performance in the investigation. On the other hand, many of the steps in the investigation were intermittent, and it is probable that better results could be obtained on a continuous basis. For instance, it was necessary to allocate several small fractions containing up to 10 per cent tin in the tailings, whereas continuous plant operations would automatically further grind and reconcentrate these products to give some sensible recovery from them.

Tabling

During tabling, five rather distinct bands were obvious:—

(1) A thin purple-brown band of cassiterite.
(2) A medium band of green-olive pyrite.
(3) A medium band of purple-white topaz.
(4) A rather indefinite medium band of brown iron oxides.
(5) A broad band of colourless-white quartz, &c.

A very thin band of galena was noted when treating spigot No. 2 product.

It has been stated that the deposit contains traces of gold. No gold was noted during tabling.

No. 1 Spigot Product.

Overall results for the fraction were satisfactory. The tin content of the fraction, at 0.61 per cent tin, was significantly higher than the bulk sample content of 0.42 per cent tin. This, of course, is the usual result of hydraulic classification of an ore ground in closed circuit with a screen.

The primary table tailing at 0.12 per cent tin, and the overall tailing at 0.17 per cent tin, are significantly higher than the corresponding values for No. 2 spigot tailings at 0.08 and 0.11 per cent tin respectively.

The sizing analysis of the primary table tailing shows that the tin content of the plus 36 mesh fractions are rather higher than the fine fractions; thus:—

Fraction	Assay: % Sn
+ 30 mesh	0.17
+ 36 mesh	0.15
+ 44 mesh	0.11
+ 60 mesh	0.08
+ 85 mesh	0.11
+100 mesh	0.09
+120 mesh	0.11

These data are open to interpretation. It may be that slightly better overall recovery of tin would be obtained if the ore was ground a little finer, to say minus 30 or 36 mesh, instead of through minus 25 mesh, as in the investigation. The assumed decrease in the overall table tailings would, of course, have to be balanced against the probable increase in the loss of tin caused by finer of products.

Alternatively, it may be that No. 1 spigot product contains too wide a sizing distribution, and that better overall tabling results would be obtainable by separating the fraction into several more closely sized fractions and tabling these separately. Commercially this would be quite easy to arrange. In this investigation the number of fractions was limited to avoid multiplicity of product.

At this stage the data indicate that a grind to approximately 25 mesh is required.

The total primary middling fractions amounted to approximately 5 per cent by weight of the fraction, and contained a little over 21 per cent of the tin in the fraction. Retreatment of the middling recovered about 60 per cent of this tin (equivalent to about 13 per cent overall from the fraction). Several of the middling fractions which have been calculated into the combined tailings have comparatively high tin contents. Thus the minus 200 mesh retreatment tailing contains 1.86 per cent, amounting to over 6 per cent of the total tin in No. 1 spigot product. Commercially it should be possible to recover some of this (and other fractions) tin by further grinding and concentration on tilting decks, &c. This was not possible in the investigation, due to the relatively small quantities of the products involved.

No. 2 Spigot Product

Concentration results from this product were good. The total tailing was only 0.11 per cent tin, and the primary table tailing was 0.08 per cent tin. With the exception of the minus 200 mesh fraction, the tin contents of all sizing fractions were uniformly low.

The combined tailings contain two fractions containing respectively 10.0 and 11.1 per cent tin. On a commerical scale it should be possible to recover some of this tin in high grade concentrate. However, the quantity of tin contained in these fractions is less than 1.5 per cent of the tin in the whole spigot product.

As with No. 1 spigot product, there is a possibility of further tin recovery from the middlings retreat tailing, which assays 0.67 per cent tin, and contains 7.96 per cent of the tin from the spigot product.

No. 3 Spigot Product

Concentration results were good. Total tailing was 0.13 per cent tin, and the primary table tailing was 0.11 per cent tin.

As with Nos. 1 and 2 spigot products, there is a possibility of a slight increase in recovery by retreatment of several of the fractions that have been calculated into the combined tailings.

Deslimed Classifier Overflow

This product was first concentrated on a tilting deck, and then upgraded by successive tablings. Comparison of the sizing analyses of the feed and tailings of the tilting deck show that good recoveries of tin were made from fractions coarser than infrasizer fraction 4. There was low recovery from infrasizer 5, and negligible recovery from fractions 6 and 7.

During concentrate upgrading on the slime table, there was appreciable loss of tin, amounting in all to about one quarter of the total tin in the classifier overflow fraction. Much of this tin was very fine tin that could be recovered on the tilting deck, but was too fine to be recovered on the slime table. It should be possible to recover portion of this tin with vanners.

Sulphides

The floated sulphides generally contain about 0.5 per cent tin, and the tin content of the sulphides tends to increase with decrease in size.

Sulphides from	% Sn in Sulphides
No. 1 Spigot product	0.43
No. 2 Spigot product	0.55
No. 3 Spigot product	0.45
Deslimed classifier overflow	0.69

Overall loss of tin in the sulphides was about one-half per cent of the total tin in the bulk sample. In the present case this loss can be neglected. However, sample R.413 contains only about 0.7 per cent weight of sulphides, while the material in the 200-250 feet region contains about 5 per cent sulphides. Loss of tin in the sulphides may therefore be rather higher from this ore.

No attempt was made to assess the possibility of recovering further tin from the sulphides.

Grinding to minus 25 mesh will produce some sulphides coarser than 60 mesh. Froth flotation is not normally efficient on sulphides coarser than 60 mesh, and hence table flotation may have to be used if there is appreciable plus 60 mesh sulphides in table concentrates.

Magnetic Separation

Magnetic separation after desulphidization and gravity concentration gave a small quantity of magnetics. Owing to the very small quantity of these products available, it has not been possible to fully investigate their nature.

The magnetics can be roughly divided into strongly magnetic and weakly magnetic fractions.

The strongly magnetic fractions were included in the various middling fractions for regrinding and reconcentration. The proportions of these magnetic fractions calculated on the basis of the whole bulk sample were—

Product	Weight %	Assay % Sn	Distribution % Sn
No. 1 Spigot magnetics	0.001	8.9	0.03
No. 2 Spigot magnetics	0.002	5.6	0.03
No. 3 Spigot magnetics	0.003	9.3	0.05
Combined magnetics ..	0.006	8.0	0.11

Addition of these magnetics to the final concentrate would drop the concentrate grade by about 1.5 per cent tin, and increase recovery by about 0.1 per cent.

These magnetics appear to consist of iron oxides, with some composites of iron oxides and cassiterite. As the various magnetics were ground with the middlings, there was none available for examination.

The faintly magnetic fractions assay between about 20 and 45 per cent tin. During the investigation these fractions were included in the primary concentrates for Nos. 1, 2 and 3 spigots. The quantity of these magnetic fractions totalled about 4 per cent by weight of the primary concentrates, and contained about 1 per cent of the total tin in the bulk sample.

A small quantity of the faintly magnetic material from No. 1 spigot product was available for examination. This assayed—

Tin (Sn)	21.10 per cent
Tungstic Oxide (WO_3)	12.40 per cent
Titanium Oxide (TiO_2)	1.25 per cent
Iron (Fe)	29.60 per cent

Mr. G. B. Everard, Departmental Mineralogist and Petrologist, microscopically examined a sample of this product. His examination was hampered by opaque red iron oxides which cover most of the grains, especially those containing wolframite, and which are very difficult to remove. He found—

> "This sample contains wolframite and cassiterite with smaller quantities of quartz, tourmaline, topaz and zircon. Practically all the grains are composite.
> The wolframite grains are either masses of, or fragments covered by, minute crystals, while grains of cassiterite, quartz, tourmaline, topaz and zircon are coated in the same way by minute crystals of wolframite.
>
> The grains of cassiterite vary from 25 to 100 microns across.".

It is hoped to investigate the effect of the magnetics more fully at a later date, but a much larger quantity of concentrate will have to be prepared to produce significant quantities of the magnetics.

Composition of the Final Concentrate

A composite sample of final concentrate was made up from proportional weights of the various components. This concentrate had the following composition:—

Tin (Sn)	60.90 per cent
Tungstic Oxide (WO_3)	0.23 per cent
Sulphur (S)	0.27 per cent
Lead (Pb)	0.41 per cent
Copper (Cu)	0.03 per cent
Bismuth (Bi)	0.02 per cent
Zinc (Zn)	0.02 per cent
Arsenic (As)	0.12 per cent
Titanium Oxide (TiO_2)	3.09 per cent

On the basis of current schedules, the penalties for such concentrates can be estimated.

1. Penalty for low tin content (taken as 61 per cent Sn)—

5 units @ 5 pence	=	2/1	per unit
4 units @ 6 pence	=	2/-	per unit
Total		4/1	

Unit price 198/1 — 4/1 = 194/- per unit.
61 units @ 194/- = £591 14s. 0d. per ton of concentrate.
(A bonus of 3/- per unit is payable for lots in excess of 5 tons.)

2. Penalty for WO_3 content—

Below 0.5 per cent WO_3—no penalty. Therefore no penalty.

3. Penalty for sulphur (S) content—

Nil to 0.15 per cent S. Penalty for excess.
£2 10s. 0d. per ton of concentrate. Therefore penalty £2 10s. 0d. per ton of concentrate.

4. Penalty for arsenic (As)—

Nil to 0.1 per cent As. Penalty 5s. per ton for each 0.1 per cent excess. Therefore penalty 5s. per ton concentrate.

5. Penalty for combined metals Pb, plus Cu, plus Bi, plus Zn, plus Sb, &c.—

Penalty 12s. 6d. per ton per each 0.1 per cent, with minimum £3 15s. 0d. Therefore penalty £3 15s. 0d. per ton of concentrate.

6. Value of the 61 per cent tin concentrate produced is thus—

£591 14s. 0d. per ton of concentrate.
Less £2 10s. 0d. as sulphur penalty.
Less 5s. 0d. as arsenic penalty
Less £3 15s. 0d. as combined metal penalty. Therefore value is—

£585 4s. 0d. per ton concentrate.

Based on a schedule of 70 per cent tin, penalties can be considered as—

(*a*) Penalty for low grade 4s. 1d. per unit =	£12 9s. 1d.
(*b*) Penalty for impurities	£6 10s. 0d.
Total penalty	£18 19s. 1d.

This penalty amounts to approximately 3 per cent of total value of tin.

On a commercial basis, it is certain that the concentrate grade would be increased to the region of say 67 per cent tin, with the same recovery as attained in the present investigation. This would approximately halve the penalty.

Pyromorphite has been stated to occur in the deposit, and it has been assumed that the 0.41 per cent of lead is present as pyromorphite, or other oxide lead minerals. It is unlikely that the combined metal impurities could be removed totally, and as the minimum penalty for combined metals is £3 15s. 0d. per ton of concentrate there is no increase in penalty charge until combined impurities exceed about 0.6 per cent. It is hoped to carry out further work at a later date to establish if, in part, the lead is present as pyromorphite.

The arsenic content of the concentrate, at 0.12 per cent As, is only slightly over the free limit of 0.10 per cent As, and it is probable that commercially the limit would be bettered. However, the arsenic penalty at 5s. per ton concentrate is negligible.

The free limit for no sulphur penalty is 0.15 per cent sulphur. This limit may be attained when treating ore from the upper levels of the deposit, but it is unlikely to be attained when treating ore from the lower levels, which may contain over 5 per cent of sulphides.

The titanium oxide content of the final concentrate, at 3.9 per cent TiO_2, was higher than anticipated. Most likely minerals are ilmenite, rutile, or possibly leucoxene. On a commercial scale, separation of the tin from any of these minerals by gravity (or magnetic separation for ilmenite) would be comparatively simple, provided the grains are free.

Portion of the primary table concentrate from No. 1 spigot product was sized and treated on the magnetic separator at maximum intensity. The products were assayed for titania, tungstic oxide and iron as follows:—

Product	Weight %	Assay % TiO_2	Assay % WO_3	Assay % Fe
Magnetics	8.8	2.01	3.18	12.7
Non-magnetics ..	91.2	3.49	0.19	1.6

As only about 6 per cent of the titanium oxide has been removed in the magnetics (probably as ilmenite), it appears that most of the titanium oxide occurs as rutile, or possibly leucoxene.

In this investigation we have been hampered in the examination of concentrate products by the very small quantities produced. A fuller examination would require much larger quantities of concentrates.

See also other investigations:

R.390, R.392, R.394: Technical Reports Department of Mines No. 6, pp. 229-253.

R.395, R.396. This vloume, pp. 137-150.