

TR16-288-290

R.629. Tin recovery tests on ore from the Queen mine, Zeehan.

Six pulverised assay samples were submitted by Gippsland Minerals N.L. The samples were identified G1, G2, G3, G4, G5 and G6. The samples were to be used to make up two composite samples.

Composite Sample 1 was made as follows:

Samples	Weight in Composite
G3, G4 and G5	420 g each
G2 and G6	210 g each
G1	320 g

Composite Sample 2 was made by bulking all the remaining material that was unused in the preparation of Composite Sample 1.

It was required to make a bulk sulphide concentrate from the Composite Sample 1, and then recover the tin in flotation tailing. The Composite Sample 2 was to be used to find conditions necessary to give satisfactory flotation of the sulphides.

TEST WORK

Composite Sample 1 was made as shown above. The remainder was made into Composite Sample 2 containing 7.897 kg of material. It was found that Samples G1, G3 and G6 contained numerous lumps. The lumps appeared to be the pulverised sample that had agglomerated by partial oxidation. The lumps were broken up by forcing the material through a fine screen.

A head sample was cut from Composite Sample 1 for assay purposes.

Four flotation tests (F1) were conducted on 500 g samples from Composite Sample 2 with varying reagent conditions. Response to flotation was satisfactory and was somewhat similar in each test.

The Composite Sample 1 was riffled into four equal fractions and the sulphides were floated using the following conditions on each of the four fractions. The flotation concentrates and flotation tails were individually bulked.

Flotation Conditions

1 lb/ton copper sulphate	2 minutes conditioning
1 lb/ton potassium amyl xanthate	2 minutes conditioning
0.1 lb/ton Teric 401	5 minutes flotation
0.1 lb/ton Teric 401	10 minutes flotation

The bulked flotation tails were wet screened on a 53 μ m screen (300#). The screen oversize was dried and screened on 355 μ m, 250 μ m, 180 μ m, 90 μ m and 53 μ m screens (44, 60, 85, 170 and 300# respectively). Each fraction coarser than 180 μ m was subjected to skin flotation to remove any coarse sulphides remaining. The non-floating material was subjected to heavy liquid separation at S.G. 2.9. The heavy liquid sink in each size fraction was magnetically separated. The non-magnetic fractions were bulked and set aside as final concentrate. The magnetic fractions, heavy liquid floats and the sulphides from skin flotation in each size fraction were collectively bulked for assay purposes.

The -180 +90 μ m fraction was tabled (T1) to give a concentrate, a

middling and a tail. The concentrate was subjected to sulphide flotation (F2). The sink from this flotation was dried and magnetically separated. The non-magnetic fraction was set aside as final concentrate. The magnetic fraction, the flotation concentrate and the table middling were bulked for assay purposes.

The -90 +53 μ m fraction was tabled (T2) to give a concentrate and a tail. The concentrate was subjected to sulphide flotation (F3). The sink from this flotation was dried and magnetically separated. The non-magnetic fraction was set aside as final concentrate. The magnetic fraction and the flotation concentrate were bulked for assay purposes.

The -53 μ m fraction was tabled (T3) and the concentrate floated (F4) in the same way as the -90 +53 μ m fraction.

Tabling of this material proved to be fairly difficult as it was hard to distinguish between the gangue material, the middling material which is probably siderite, and the cassiterite on the table, as the colours were very similar.

TEST RESULTS

The Composite Sample 1 and the flotation concentrate made from it were assayed with the following results:

Product	T.Fe %	S.Fe %	Pb %	Cu %
F/D	26.6	10.7	3.2	0.20
FlC	34.7	9.7	5.2	0.34

Product	Ag (oz/ton)	Bi %	Cd %	Zn %	Sn %
F/D	5.0	Trace	Trace	0.38	1.2
FlC	8.0	Trace	Trace	0.62	0.66

The recovery of these metals in the sulphide concentrate is as follows:

Product	% Wt	% Recovery			
		T.Fe	S.Fe	Pb	Cu
FlC	52.3	68.2	47.4	84.9	88.8

Product	Ag	% Recovery			
		Bi	Cd	Zn	Sn
FlC	83.6	-	-	85.3	26.7

The HCl soluble iron (S.Fe) in the sulphide concentrate is an approximate indication of the iron present in the sulphide concentrate as pyrrhotite. The remaining iron present is an approximate indication of the amount of iron occurring in the sulphide concentrate as pyrite.

Siderite present in the ore accounts for the higher HCl soluble iron assay of the feed.

The results of the gravity concentration tests are as follows:

Product	% Wt	Assay	
		% Sn	% Sn Distribution
F1C	52.45	0.66	26.7
F1T +180 µm H/L S/K N	0.03	23.5	0.5
+180 µm H/L S/K M/A	1.28	2.8	2.8
+180 µm H/L F/T + skin			
flotation sulphides			
+90 µm T1C F2T N	0.16	26.3	3.3
+90 µm T1C F2T M/A	2.07	2.8	4.5
+90 µm T1C F2C			
+90 µm T1M			
+90 µm T1T	2.90	0.41	0.9
+53 µm T2C F3T N	0.12	25.5	2.4
+53 µm T2C F3T M/A	0.97	4.4	3.3
+53 µm T2C F3C			
+53 µm T2T			
-53 µm T3C F4T N	0.53	37.5	15.4
-53 µm T3C F4T M/A	2.24	4.6	8.0
-53 µm T3C F4C	33.09	1.15	29.7
-53 µm T3T			
F/D	100.0	1.29	100.0

The test results may be summarised as follows:

Product	% Wt	Assay	
		% Sn	% Sn Distribution
Sulphide Concentrate	52.5	0.66	26.7
Tin Concentrate	0.8	33.1	21.6
Middlings	6.6	3.65	18.6
Tailings	40.1	1.06	33.1

Despite the initial sulphide flotation and subsequent sulphide flotation of the gravity concentrates, some sulphide material was still found in the final tin concentrates and consequently affected the grade of the final concentrate. The poor response of this sulphide to flotation may have been due to the partial oxidation of some of the sample as already noted. Sulphide flotation from freshly ground ore should be more successful than in the present test.

CONCLUSIONS

Because of the nature and size of the sample, this investigation can only be regarded as being of a preliminary nature. The sample was made from reject portions of samples that had been pulverised for assay purposes. It is likely, therefore, that the cassiterite present has been ground finer than is necessary for its recovery. Indeed, it is possible that some of the cassiterite has been overground to a point where it could not be recovered.

The ore contains siderite which reports in the gravity concentrate and makes the production of a high grade concentrate difficult, as it is difficult to distinguish between the cassiterite and the siderite on the table.

The tin recovery of 21.6% at a concentrate grade of 33.1% indicates that the ore is difficult to treat.

It is recommended that further investigation be carried out on a large sample of freshly mined ore so that a more accurate idea of the recovery can be determined.