7R20_357_367

R.701. Tin recovery tests on Khaki ore from Cleveland mine.

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Experience in milling Khaki ore in the concentrator at Cleveland Tin N.L., has shown that a decline in tin recovery occurs by comparison with similar treatment of ore from Hall's lode.

Khaki ore is known to contain considerably more siderite than Hall's ore and a cassiterite grain size analysis done on a grab sample of Khaki ore (R.672) indicated a somewhat finer grain size.

Metallurgical studies were requested by the company to establish an initial mesh of grind for sensible liberation of cassiterite for the primary concentration stage and to assess the probable recovery of cassiterite using the procedures currently in use at the mill.

SAMPLES

A sample of approximately 15 tonnes of -50 mm Khaki ore was submitted. This ore was stated to have been 'spotted' from the fine ore heap on 10 April 1975. It was considered that the sample obtained was reasonably representative of the fine ore heap, and this in turn was generally representative of Khaki ore being mined at the time.

The bulk sample was flattened into a roughly rectangular outline and samples of 350-400 kg taken from each corner and the centre. A sizing was done on one of these samples.

Each sample was then individually jaw and roll crushed to -4.75 mm and assay samples cut out. These samples were assayed for copper, soluble and total tin to establish the homogeneity of the stockpile.

The assay and sizing results are given below:

	Assays			
Sample No.	*Cu %*	*S. Sn %*	*C. Sn %*	*T. Sn %*
1	0.20	0.02	0.84	0.86
2	0.21	0.02	0.89	0.91
3	0.22	0.04	0.86	0.90
4	0.20	0.02	0.84	0.86
5	0.21	0.02	0.85	0.87

Sizing: Sample No. 4

Fraction (mm)	*% Mass*
+38.1	4.5
+19.05	67.4
+9.53	17.2
+4.75	4.9
-4.75	6.0

An equal weight composite of the five samples was then made up. After thorough mixing samples for analysis and test work were cut out by coning and quartering and riffling. The head sample was analysed with the following results:

Analysis of the head sample

	%		%
CaO	4.1	S	3.6
MgO	1.7	Cu	0.20
Fe	13.9	S. Sn	0.04
CO_2	11.3	C. Sn	0.84
		T. Sn	0.88

The iron, CO_2, sulphur, copper and soluble tin assays indicate approximately mineral contents of pyrrhotite (9%), siderite (30%) and chalcopyrite-stannite (0.7%).

INVESTIGATIONS AND RESULTS

CASSITERITE GRAIN SIZE ANALYSIS

This was determined on ore roll crushed to pass 4.75 mm. Decomposition of the ore was effected by repeated attack with hydrochloric, nitric, hydrofluoric and sulphuric acids. Cassiterite is relatively unaffected by this treatment and remains in its naturally occurring grain size when the decomposition of gangue minerals is completed.

The residue, containing the free cassiterite was sized by screening and cyclosizing, the fractions assayed for tin and the tin distribution calculated.

Results

Fraction	*% Sn*	*Cum. % Sn*
+1.0 mm -1.0 mm +600 µm	0.5	0.5
-600 µm +300 µm	3.0	3.5
-300 µm +212 µm	4.9	8.4
-212 µm +150 µm	7.4	15.8
-150 µm +106 µm	18.3	34.1
-106 µm +75 µm	13.5	47.6
-75 µm +38 µm	30.0	77.6
-38 µm +26 µm	10.8	88.4
-26 µm +19 µm	3.0	91.4
-19 µm +14 µm	3.8	95.2
-14 µm +9 µm	2.4	97.6
-9 µm +7 µm	0.8	98.4
-7 µm	1.6	100.0

Comments

The above analysis closely parallels a similar analysis done on a composite sample obtained from 290 metres of drilling in 1962 and reported in Investigation R.421.

That composite was made up from:

Battery lode	57.0 m	Henry's lode	101.2 m
Hall's lode	99.4 m	Waterhole lode	32.3 m

The head value of this composite was 0.97% tin, similar to the Khaki sample being investigated.

This sample showed 75.4% of the cassiterite as being coarser than 38 microns compared with 77.6% +38 µm in the current sample of Khaki ore.

Both samples show a coarser tin distribution than that determined on the grab sample of Khaki obtained in 1973 and reported in R.672. In this sample 52% of the cassiterite was coarser than 38 µm. The present 15 tonne sample should be more reliable than the few kilograms involved in the grab sample.

METALLURGICAL TESTS

Only 3.5% of the total tin occurs in sizes above 300 µm, according to the grain size analysis discussed in the previous section while some 44% of the tin occurs between 300 and 75 µm. Closed circuit grinding to -300 µm should effect significant liberation of the cassiterite without much risk of overgrinding.

In addition to these considerations 300 µm approximates the mesh of grind theoretically obtainable by grinding through a 0.5 mm DSM screen, which size is currently in use in the plant grinding circuit.

For these reasons 300 µm was chosen as the mesh of grind for primary treatment.

Primary treatment

Roll crushed -4.75 mm ore was ground continuously at about 75% solids in a 12 in. x 12 in. Denver ball mill closed by a Sweco vibrating screen fitted with a 315 µm aperture stainless steel screen.

Time was taken to allow the build up of an adequate circulating load, during which time the screen undersize was discarded. When conditions were apparently stable, approximately 11 kg of ground ore was transferred to the 10 kg Galligher flotation cell and sulphides removed by cleaner froth flotation.

Flotation conditions were as follows:

		Usage
H_2SO_4	(pH 4.5)	2 kg/t
Copper sulphate		100 g/t
Sodium ethyl xanthate		130 g/t
Potassium amyl xanthate		130 g/t
MIBC		sufficient for froth
Conditioning time		3 minutes
Flotation times:	Rougher	10 minutes
	Cleaner	7 minutes

All reagents were added in rougher stage.

The combined rougher and cleaner flotation tailings were fed to a Geco 3 spigot hydrosizer and the products separately tabled, with retreatment of middling. Crude concentrates were retreated to maintain grades as high as possible, consistent with reasonable recoveries of the tin. The concentrate, middling and tailings from each spigot and the overflow were separately collected, weighed and assayed for tin and the total tin distribution, including the sulphides was calculated.

Sizing analyses were carried out on each of the table tailings.

Results: Primary treatment

Product	% Mass	% C. Sn	% C. Sn Distn
S1 TC	0.5	49.0	24.6
M	2.4	2.5	6.0
T	22.1	0.52	11.6
S1 Composite	25.0	1.68	42.2
S2 TC	0.3	36.3	11.0
M	2.3	1.0	2.3
T	13.9	0.29	4.0
S2 Composite	16.5	1.04	17.3
S3 TC	0.2	38.8	7.9
M	2.5	0.47	1.2
T	7.8	0.22	1.7
S3 Composite	10.5	1.02	10.8
O/F TC	0.3	20.6	6.2
M	2.0	0.50	1.0
T	30.1	0.41	12.4
O/F Composite	32.4	0.60	19.6
F2C	15.6	0.65	10.1
Composite Head	100.0	1.00	100.0

Total primary concentrate

Product	% Mass	% C. Sn	% C. Sn Distn
S1C	0.5	49.0	24.6
S2C	0.3	36.3	11.0
S3C	0.2	38.8	7.9
O/F C	0.3	20.6	6.2
Composite Primary C	1.3	38.0	49.7

A total of 49.7% of the total tin has been recovered in the primary operations after grinding to -315 μm. Of this, about half was recovered from the first spigot, the sizing of which was about 70% +150 μm.

Sizings: Table tailings

These were sized by wet and dry screening to 38 μm and cyclosizing below this range.

Product and Fraction		% Mass	% C. Sn	% C. Sn Distn
S1T	+300 μm	1.7	0.54	1.7
	+150 μm	67.8	0.53	69.0
	+75 μm	25.2	0.51	24.8
	+38 μm	4.5	0.31	2.8
	-38 μm	0.8	1.12	1.7
S1T	Composite	100.0	0.52	100.0
S2T	+300 μm	nil	-	-
	+150 μm	25.5	0.34	29.8
	+75 μm	51.6	0.28	49.3
	+38 μm	19.6	0.22	14.7
	-38 μm	3.3	0.54	6.2
S2T	Composite	100.0	0.29	100.0

Product and Fraction	*% Mass*	*% C. Sn*	*% C. Sn Distn*
S3T +150 µm	1.5	0.21	33.3
+75 µm	30.6		
+38 µm	58.6	0.16	46.8
-38 µm	9.3	0.43	19.9
S3T Composite	100.0	0.20	100.0
O/F T +75 µm	trace	0.25	1.5
+38 µm	2.4		
C/S 1	trace	0.66	0.7
C/S 2	0.4		
C/S 3	8.7	0.35	7.3
C/S 4	12.1	0.59	17.3
C/S 5	7.7	0.66	12.5
O/F	68.7	0.36	60.7
O/F T Composite	100.0	0.41	100.0

The temperature of the C/S water was 6°C.

Comments on tailing sizing

The sizings show that classification in the Geco hydrosizer has been quite precise. It is most unlikely that tin losses occurring in the coarse end of any product would be other than composite.

It follows that significant amounts of composite tin are present in the +75 µm sizes in spigots 1 and 2 and that these should be the subject of regrinding operations.

Spigot No. 3 tailing is substantially -75 µm and the overall loss in the product is only 1.7% of the total tin.

The hydrosizer overflow tailing is practically all -38 µm. It would be expected that all tin was free and that losses occurring are a result of the extreme fineness of the cassiterite. In fact, 61% of the tin in this tailing is finer than 7 µm.

Sizings: Ball mill feed and screen undersize (flotation feed)

	Ball mill feed		*315 µm screen U/S*	
Fractions	*% Mass*	*Cum. % Mass*	*% Mass*	*Cum. % Mass*
+2.36 mm	28.5	28.5		
+1.18 mm	26.4	54.9		
+600 µm	14.7	69.6		
+300 µm	8.9	78.5	0.4	0.4
+212 µm	3.9	82.4	13.4	13.8
+150 µm	2.4	84.8	11.9	25.7
+106 µm			12.8	38.5
+75 µm	4.8	89.6	8.7	47.2
+38 µm	3.4	93.0	13.5	60.7
-38 µm	7.0	100.0	39.3	100.0

Secondary grinding and concentration

The primary No. 1 spigot tailing sizing shows that 67.8% of the mass and 69.0% of the contained tin are present above 150 µm. The tin is certain to be present as composite particles and probably some composites are present in the -150 µm +75 µm fraction also.

This tailing and the associated middling contain 17.6% of the total tin in the initial feed.

For secondary treatment the primary first spigot middling and tailing were stage ground to -150 μm using wet screening by hand to close the grinding. The ground product was classified in the Geco hydrosizer and the fractions separately tabled similarly to the primary operations.

Results: Secondary treatment

Product		*% Mass*	*% C. Sn*	*% C. Sn Distn*
S1	TC	0.2	34.8	6.5
	T	7.6	0.52	3.7
S1	Composite	7.8	1.40	10.2
S2	TC	0.1	16.2	1.5
	T	5.1	0.28	1.3
S2	Composite	5.2	0.59	2.8
S3	TC	0.1	16.5	1.5
	T	5.5	0.21	1.1
S3	Composite	5.6	0.50	2.6
O/F	TC	0.1	6.5	0.6
	T	5.8	0.26	1.4
O/F	Composite	5.9	0.37	2.0
Comp. Regrind F/D		24.5	0.78	17.6

Total secondary concentrates

Product	*% Mass*	*% C. Sn*	*% C. Sn Distn*
S1C	0.2	34.8	6.5
S2C	0.1	16.2	1.5
S3C	0.1	16.5	1.5
O/F C	0.1	6.5	0.6
Composite Sec. C.	0.5	21.8	10.1

The tin recovery within this stage amounted to 57.4%, equivalent to an increase in overall recovery of 10.1%.

Tertiary grinding and concentration

The primary No. 2 spigot middling and tailing contain 6.3% of the total tin and 16.2% of the total mass. The sizing analysis of the tailing indicates that most of this is present as composite particles above 75 μm.

The secondary No. 1 spigot tailing (-150 μm) contains a further 3.7% of the total tin, also probably present as composite grains, and 7.6% of the total feed mass.

Further stage grinding of these products was undertaken followed by classification and tabling as before.

The screen size chosen was 106 μm, grinding through this screen should liberate all composites coarser than 150 μm and many of those between 150 μm and 75 μm with minimum risk of overgrinding.

Results: Tertiary treatment

Product	% Mass	% C. Sn	% C. Sn Distn
S1 TC	0.1	19.9	2.1
T	3.5	0.28	1.0
S1 Composite	3.6	0.82	3.1
S2 TC	0.1	8.4	0.9
T	4.0	0.18	0.8
S2 Composite	4.1	0.38	1.7
S3 TC	0.2	9.0	1.9
T	5.9	0.14	0.9
S3 Composite	6.1	0.43	2.8
O/F TC	0.1	4.4	0.5
T	9.9	0.18	1.9
O/F Composite	10.0	0.22	2.4
Composite Tert. F/D	23.8	0.42	10.0

Total tertiary concentrate

Product	% Mass	% C. Sn	% C. Sn Distn
S1C	0.1	19.9	2.1
S2C	0.1	8.4	0.9
S3C	0.2	9.0	1.9
O/F C	0.1	4.4	0.5
Composite Tert. C.	0.5	10.1	5.4

The tin recovery within this stage amounted to 54%, equivalent to a further 5.4% in overall recovery.

The comparatively low concentrate grade (10.1% Sn) is not unexpected, considering the overall fineness of the feed and its relatively low tin content. Two or three stages of vanner treatment would yield similar recoveries with enhanced grades.

Retreatment of sulphides

The cleaner sulphides flotation concentrate contained 10.1% of the total cassiterite tin in the feed. Sizing of the product showed 50% of its mass to be coarser than 106 μm.

On the assumption that tin coarser than this was present as composite particles, the sulphides were stage ground to pass a 106 μm screen, subjected to a further stage of flotation, and the flotation tailings concentrated by panning.

Results: Sulphide retreatment

Product	% Mass	% C. Sn	% C. Sn Distn
F3C	12.1	0.18	2.1
F3T PC	0.1	47.9	4.8
F3T PT	3.4	0.94	3.2
Composite F2C	15.6	0.65	10.1

The above operations resulted in a recovery of 47.5% of the tin in the pan concentrate or a further 4.8% in terms of overall recovery.

The pan tailing was unexpectedly high in tin at 0.94% and contained 3.2% of the total tin. This would remain in circuit in commercial milling and some further significant recovery from such a product could be expected.

The -106 µm recleaner sulphides assay: Cu 1.80%; S. Sn 0.13%.

Final gravity tailing

The composite gravity tailing from the primary, secondary and tertiary operations assays 0.26% C. Sn. This value is considered to be satisfactory, considering that tabling only has been employed. Vanners could be expected to give somewhat better performance on the very fine fractions (-38 µm) of the feed with a consequently lower overall tailing assay.

The sizing analysis of the composite tailing is given below. It would be of interest to compare this with a similar product from current milling operations.

Sizing: Composite final gravity tailing

Fraction	*% Mass*	*Cum. % Mass*	*% C. Sn*	*% C. Sn Distn*	*Cum. % C. Sn Distn*
+300 µm	trace				
+212 µm	trace				
+150 µm	0.4	0.4	0.26	3.8	3.8
+106 µm	3.5	3.9			
+75 µm	10.1	14.0	0.23	8.9	12.7
+38 µm	23.9	37.9	0.19	17.4	30.1
C/S 1	0.2	38.1	0.88	7.4	37.5
C/S 2	2.0	40.1			
C/S 3	9.6	49.7	0.28	10.3	47.8
C/S 4	9.6	59.3	0.40	14.7	62.5
C/S 5	5.8	65.1	0.43	9.5	72.0
O/F	34.9	100.0	0.21	28.0	100.0
Comp. T	100.0		0.26	100.0	

The temperature of the C/S water was 7°C.

Gravity concentrates: Overall recovery

Product	*% Mass*	*% C. Sn*	*% C. Sn Distn*
C Primary	1.3	38.0	49.7
C Secondary	0.5	21.8	10.1
C Tertiary	0.5	10.1	5.4
C Ex Sulphides	0.1	47.9	4.8
Comp. gravity C	2.4	29.3	70.0

The above tabulation shows that 71% of the total recovery obtained derives from the primary operations (-315 µm).

The composite gravity concentrate contains 13.2% CO_2, equivalent to 35% siderite, and should be equivalent to the Plant Gill magnetic separator feed.

Some assessment of plant performance on current ore could perhaps be gained by a comparison with the similar plant products.

Concentrate cleaning by magnetic separation

The quantity of concentrate available precluded the use of wet magnetic separation. However, dry high intensity separation should yield somewhat similar results.

A weighted composite of all gravity concentrate was made up and separated on Rapid high intensity dry magnetic separator, with one retreatment of magnetics.

The magnetics were then wet ground by hand in a wedgwood mortar to approximately -106 µm.

The ground magnetics were concentrated by panning and the pan concentrate dried and magnetically separated and assayed for tin and CO_2.

Results: Concentrate cleaning

M/S1

Product	*% Mass*	*% C. Sn*	*% CO_2*	*% C. Sn Distn*
M/A	1.20	6.67	22.8	8.1
N (Tin conc.)	1.20	56.5	3.7	61.9
Comp. gravity C	2.40	31.6	13.2	70.0

Grinding, panning and M/S2

Product	*% Mass*	*% C. Sn*	*% CO_2*	*% C. Sn Distn*
PT	0.49	1.97	23.4	0.9
PC M/A	0.53	3.6	22.7	1.7
PC N (Tin conc.)	0.18	33.5	12.8	5.5
Comp. M/S1 M/A	1.20	6.67	21.5	8.1

Composite final concentrate: M/S1 N + M/S2 PC N

% Mass	1.38	Grade: % C. Sn	53.5
% C. Sn recovery	66.9	% CO_2	4.8

The CO_2 content is equivalent to a siderite content of 12.7%.

SUMMARY

Cassiterite grain size analysis

The grain size analysis indicated that only a relatively small amount of the tin would be irrecoverable by gravity concentration, *i.e.* the approximate 10% -20 µm.

However some depreciation in grain size must occur during crushing and grinding, and the analysis, based on past experiences would suggest that a recovery of about 70% could be expected. This is reinforced by the fact that the sizing closely parallels that of a previous sample, from which both test work and commercial milling in the Cleveland concentrator have yielded recoveries of this order.

Metallurgical result

The test work outlined in this report has produced a result which would be expected in view of the cassiterite sizing.

This result is an overall recovery of cassiterite tin of 66.9% in concentrate assaying 53.5% Sn.

Initial mesh of grind

More than two thirds of the total tin recovery was obtained from the primary sizing and tabling after closed circuit grinding through 315 µm. The grade of this portion of the concentrate (38% Sn) is considered satisfactory in view of the high siderite content of the ore, and shows that a substantial degree of liberation was obtained.

It is unfortunate that rather more -38 µm material was produced than was desirable, indicating that some degree of overgrinding had occurred. However, some 47% of the ground product lies between 300 µm and 75 µm and it can be assumed that commercial grinding through the same mesh, although not duplicating the sizing would nevertheless effect a substantial degree of liberation of cassiterite.

The test work gives no indication that an initial grind of less than 300 µm is warranted.

Regrind of primary gravity tailings

About 48% of the total feed mass was subjected to regrinding in two separate operations, determined by the sizing analyses of the primary table tailings. A little more than half of this emanated from the first spigot product and the sizing analysis indicated that grinding to -150 µm would be necessary for significant liberation of tin to take place.

Table concentration of products from regrinding of primary spigot No. 1 tailing and middling to -150 µm yielded a recovery of 10.1% of the total tin.

The sizing of the primary No. 2 spigot tailing indicated that grinding to -75 µm would be necessary to effect near total liberation of tin from composites. A mesh of grind of 106 µm was used as a compromise between the ideal of 75 µm below which probably few composites exist and the 150 µm above which most of the tin in the product is certainly composite.

The probable adverse economics of fine grinding (to -75 µm) of a relatively low grade product were taken into account in deciding on 106 µm in this particular instance.

The No. 1 spigot tailing from the secondary operations (-150 µm) assayed 0.51% Sn and this was also included in this grind, bringing the total feed to 23.7% of the original feed mass.

Table concentration of the products from this grind yielded a further tin recovery of 5.4%.

Sulphide retreatment

Grinding of the cleaner bulk sulphides to -106 µm followed by reflotation and gravity concentration of flotation tailings gave a recovery of 4.2% overall in a concentrate assaying 47.9% Sn. This represented a recovery of 47.5% of the tin in the product and is in line with results of previous investigations of Cleveland ores.

The final bulk sulphide concentrate (F3C) contained 2.1% of the total cassiterite tin in the ore, and it is unlikely that this loss could be reduced any further.

Concentrate cleaning

The composite rougher concentrate proved to be amenable to upgrading by magnetic separation to remove siderite followed by grinding and gravity concentration of the magnetic product.

About 95% of the tin in rougher concentrate was recovered in a final concentrate assaying 53.5% Sn.

Final concentrate grade is a little low; this is probably related to the small quantity of material available for upgrading but it should not be overlooked that some further tin loss could occur at this stage if the grade were taken much above 60% Sn.

CONCLUSIONS

The test work has shown that recoveries of 65-70% can be obtained from gravity concentration of this particular sample of Khaki ore. It should be possible to maintain this recovery with concentrate grades of about 60% Sn.

It has been reasonably well established that adequate liberation of cassiterite occurs with an initial mesh of grind of about 300 µm. This approximates the size delivered as underflow by a 0.5 mm DSM screen.

Some 48% of the total feed mass appears to require regrinding, assuming that any tailing over 0.30% Sn should be reground and retreated.

Grinding the bulk sulphides to -106 µm results in near total release of cassiterite. It is not known whether finer grinding would be necessary for the release of stannite and chalcopyrite from the bulk sulphides in this particular sample.

The sulphide concentrate amounted to about 12% of the total feed mass by comparison with the 25% generally encountered in Hall's lode material. This means that a similar milling rate for Khaki ore would impose a 20% greater loading on the gravity section of the plant than when milling Hall's lode. This factor, together with the higher siderite content could be a significant influence in reducing concentrator recovery.

[12 November 1975]