TR20_296_310

R.688. Concentration of tin from Arthur River gravels

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Five separate samples of gravels and sands taken from test pits in the Arthur River were submitted for determination of tin content and recoverable tin.

The samples were identified as follows:

Indentification	*Registered No.*
Test pit 1 (0-1.2 m)	741634
Test pit 1 (1.2-2.1 m)	741635
Test pit 2	741636
Test pit 3	741637
Test pit 4	741638

Messrs Fisher and Verbaan of Northern Developments (Tas.) Pty Ltd intimated that it was hoped to be able to produce a concentrate assaying about 5% Sn at the Arthur River. It was then proposed to transport this concentrate by road to Burnie where it would be processed in a concentrate dressing plant.

TIN DISTRIBUTION IN SAMPLES

Each sample was screened and the screen fractions were assayed for tin so that the tin distribution of each sample could be determined.

The results for each of the samples are shown in the following tables.

Sample 751634 - Test pit 1 (0-1.2 m)

Size Fraction	*% Mass*	*Cum. % Mass*	*% Sn*	*Sn Distribution* %	Cum. %
+76.2 mm	2.2	2.2	<0.01	0.1	0.1
+38.1 mm	11.9	14.1	0.15	4.4	4.5
+19.05 mm	12.4	26.5	0.02	0.6	5.1
+9.53 mm	10.7	37.2	0.02	0.5	5.6
+4.75 mm	8.5	45.7	0.03	0.6	6.2
+2.36 mm	1.8	47.5	0.03	0.1	6.3
+1.18 mm	4.1	51.6	0.13	1.3	7.6
+600 µm	15.5	67.1	0.45	17.0	24.6
+300 µm	12.4	79.5	1.40	42.3	66.9
+150 µm	4.2	83.7	2.0	20.5	87.4
+75 µm	2.1	85.8	1.1	5.6	93.0
+38 µm	1.4	87.2	0.49	1.7	94.7
C/S 1	0.1	87.3	5.8	0.7	95.4
C/S 2	0.3	87.6	0.36	0.3	95.7
C/S 3	0.7	88.3	0.23	0.4	96.1
C/S 4	0.8	89.1	0.19	0.4	96.5
C/S 5	0.5	89.6	0.20	0.2	96.7
O/F	10.4	100.0	(0.13)	3.3	100.0
Calculated Head	100.0		(0.41)	100.0	

Sample 741635 - Test pit 1 (1.2-2.1 m)

Size Fraction	*% Mass*	*Cum. % Mass*	*% Sn*	*Sn Distribution*	
				%	*Cum. %*
+76.2 mm	0.9	0.9	0.01	trace	trace
+38.1 mm	11.7	12.6	0.01	0.4	0.4
+19.05 mm	12.7	25.3	0.01	0.5	0.9
+9.53 mm	11.1	36.4	0.02	0.8	1.7
+4.75 mm	8.9	45.3	0.04	1.3	3.0
+2.36 mm	3.1	48.4	0.02	0.2	3.2
+1.18 mm	9.9	58.3	0.09	3.2	6.4
+600 µm	14.5	72.8	0.31	16.3	22.7
+300 µm	11.5	84.3	0.94	39.1	61.8
+150 µm	4.5	88.8	1.3	21.2	83.0
+75 µm	2.1	90.9	0.86	6.5	89.5
+38 µm	1.3	92.2	0.36	1.7	91.2
C/S 1	trace	92.2	7.06	0.8	92.0
C/S 2	0.2	92.4	0.43	0.3	92.3
C/S 3	0.6	93.0	0.15	0.3	92.6
C/S 4	0.8	93.8	0.13	0.4	93.0
C/S 5	0.5	94.3	0.17	0.3	93.3
O/F	5.7	100.0	(0.32)	6.7	100.0
Calculated Head	100.0		(0.28)	100.0	

Sample 741636 - Test pit 2

Size Fraction	*% Mass*	*Cum. % Mass*	*% Sn*	*Sn Distribution*	
				%	*Cum. %*
+76.2 mm	0.9	0.9	0.01	0.1	0.1
+38.1 mm	8.7	9.6	0.01	0.9	1.0
+19.05 mm	9.8	19.4	0.01	1.0	2.0
+9.53 mm	7.0	26.4	0.02	1.5	3.5
+4.75 mm	5.0	31.4	0.02	1.1	4.6
+2.36 mm	1.4	32.8	0.02	0.3	4.9
+1.18 mm	3.4	36.2	0.05	1.8	6.7
+600 µm	3.5	39.7	0.26	9.6	16.3
+300 µm	5.2	44.9	0.50	27.5	43.8
+150 µm	9.3	54.2	0.30	29.4	73.2
+75 µm	10.2	64.4	0.14	15.1	88.3
+38 µm	6.9	71.3	0.09	6.6	94.9
C/S 1	trace	71.3	6.00	1.9	96.8
C/S 2	0.3	71.6	0.25	0.8	97.6
C/S 3	2.6	74.2	0.04	1.1	98.7
C/S 4	2.9	77.1	0.03	0.9	99.6
C/S 5	2.0	79.1	0.02	0.4	100.0
O/F	20.9	100.0	*	*	*
Calculated Head	100.0		(0.10)	100.0	

*Unable to place a calculated assay because the total of the units in size fractions +2.36 mm and finer was greater than the units from the assay on the material -4.75 mm which was 0.12% Sn.

Sample 741637 - Test pit 3

Size Fraction	*% Mass*	*Cum. % Mass*	*% Sn*	*Sn Distribution*	
				%	*Cum. %*
+1.18 mm	0.2	0.2	0.06	0.1	0.1
+600 µm	5.0	5.2	0.06	1.4	1.5
+300 µm	37.9	43.1	0.17	29.3	30.8
+150 µm	32.2	75.3	0.24	35.2	66.0
+75 µm	9.6	84.9	0.44	19.2	85.2
+38 µm	2.2	87.1	0.73	7.3	92.5
C/S 1	0.1	87.2	8.6	3.9	96.4
C/S 2	0.2	87.4	0.82	0.7	97.1
C/S 3	0.7	88.1	0.23	0.7	97.8
C/S 4	0.8	88.9	0.19	0.7	98.5
C/S 5	0.6	89.5	0.53	1.5	100.0
O/F	10.5	100.0	*	*	*
Head	100.0		0.22		

*Unable to place a calculated assay because the total of units from individual size fractions was greater than units from the head assay.

Sample 741638 - Test pit 4

Size Fraction	*% Mass*	*Cum. % Mass*	*% Sn*	*Sn Distribution*	
				%	*Cum. %*
+2.36 mm	0.3	0.3	0.29	0.1	0.1
+1.18 mm	0.5	0.8	0.49	0.4	0.5
+600 µm	2.0	2.8	0.27	0.9	1.4
+300 µm	19.4	22.2	0.20	6.1	7.5
+150 µm	36.8	59.0	0.37	21.6	29.1
+75 µm	17.1	76.1	0.86	23.3	52.4
+38 µm	6.4	82.5	1.20	12.2	64.6
C/S 1	0.2	82.7	24.2	7.6	72.2
C/S 2	0.5	83.2	5.1	4.0	76.2
C/S 3	2.4	85.6	0.67	2.6	78.8
C/S 4	2.6	88.2	0.92	3.8	82.6
C/S 5	1.4	89.6	1.4	3.2	85.8
O/F	10.4	100.0	(0.86)	14.2	100.0
Head	100.0		0.63		

Although the tin distributions of the samples from test pits 2 and 3 are not completely correct (because the tin content of the cyclosizer O/F was not included) it can be said that the major portion of the tin present in all samples occurs in the -1.18 mm +75 µm size range, as shown below:

Product	%
Test pit 1 (0-1.2 m)	85.4
Test pit 1 (1.2-2.1 m)	83.1
Test pit 2	81 *
Test pit 3	85 *
Test pit 4	51.9

*Approximate only because of incomplete distribution. Actual result would be somewhat lower.

In test pit 4, 74.8% of the tin occurs in the size range -600 µm to C/S 2. The tin in this hole is finer than the other holes.

TEST WORK

Test N1

It was proposed to do test work on a composite of samples from test pit 1 (0-1.2 m and 1.2-2.1 m) and from test pit 2 because of the large quantity of these samples available.

Accordingly half of the -4.75 mm products from the sizing of each of these samples were composited and screened on a 0.45 m diameter Sweco screen fitted with 1.14 mm, 600 µm and 75 µm screens (fig. 105).

The -1.14 mm +600 µm fraction was passed over a 10 cm x 15 cm Denver laboratory jig. The jig concentrate was magnetically separated on the Rapid dry magnetic separator to give four magnetic products and a non-magnetic product.

The non-magnetic fraction was again passed over the jig to produce a concentrate and a tail. The concentrate so formed was again passed over the jig to produce a concentrate J6C and tail J6T. The jig bed was washed in a panning dish and the dish concentrate was included with the jig concentrate J6C and the dish tail was included with the jig tail J6T.

The -600 µm +75 µm fraction was passed over the 10 cm x 15 cm Denver laboratory jig to produce a concentrate and a tail J7T. The jig concentrate was magnetically separated to give four magnetic products and a non-magnetic product.

The non-magnetic product was floated in the Denver D1 laboratory flotation cell using 0.5 kg/t of sodium ethyl xanthate and sufficient Teric 401 to produce a stable froth. These conditions were used in all subsequent occasions where sulphide flotation was used in this report. The flotation concentrate was refloated to produce a concentrate F15C and a tail F15T. The rougher flotation tail was wet screened by hand on 355 µm, 212 µm and 125 µm screens. Each screen fraction so produced was concentrated by tabling on the Deister laboratory table to produce a concentrate and a tail.

Gold was found in, and hand-picked from, the gravity concentrates with the aid of a microscope, and the amount increased as the sizing of the concentrate increased. Accordingly, the -4.75 mm +1.14 mm screen fraction from screen 2 was subsequently jigged for gold recovery only, and the gold was hand-picked from the jig concentrate.

Results

The feed for the test was half of the -4.75 mm fractions from the initial size/tin distribution tests. The +4.75 mm fractions have been weighted and included in the results so that an overall assessment can be made.

The results of the screening (screen 1 and 2 in the flow sheet), are as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
+4.75 mm	41.9	(0.03)	5.5
-4.75 mm +1.14 mm	17.1	0.07	5.0
-1.14 mm +600 µm	14.9	(0.55)	34.0
-600 µm +75 µm	17.9	(0.65)	48.1
-75 µm	8.2	0.22	7.4
Head	100.0	(0.24)	100.0

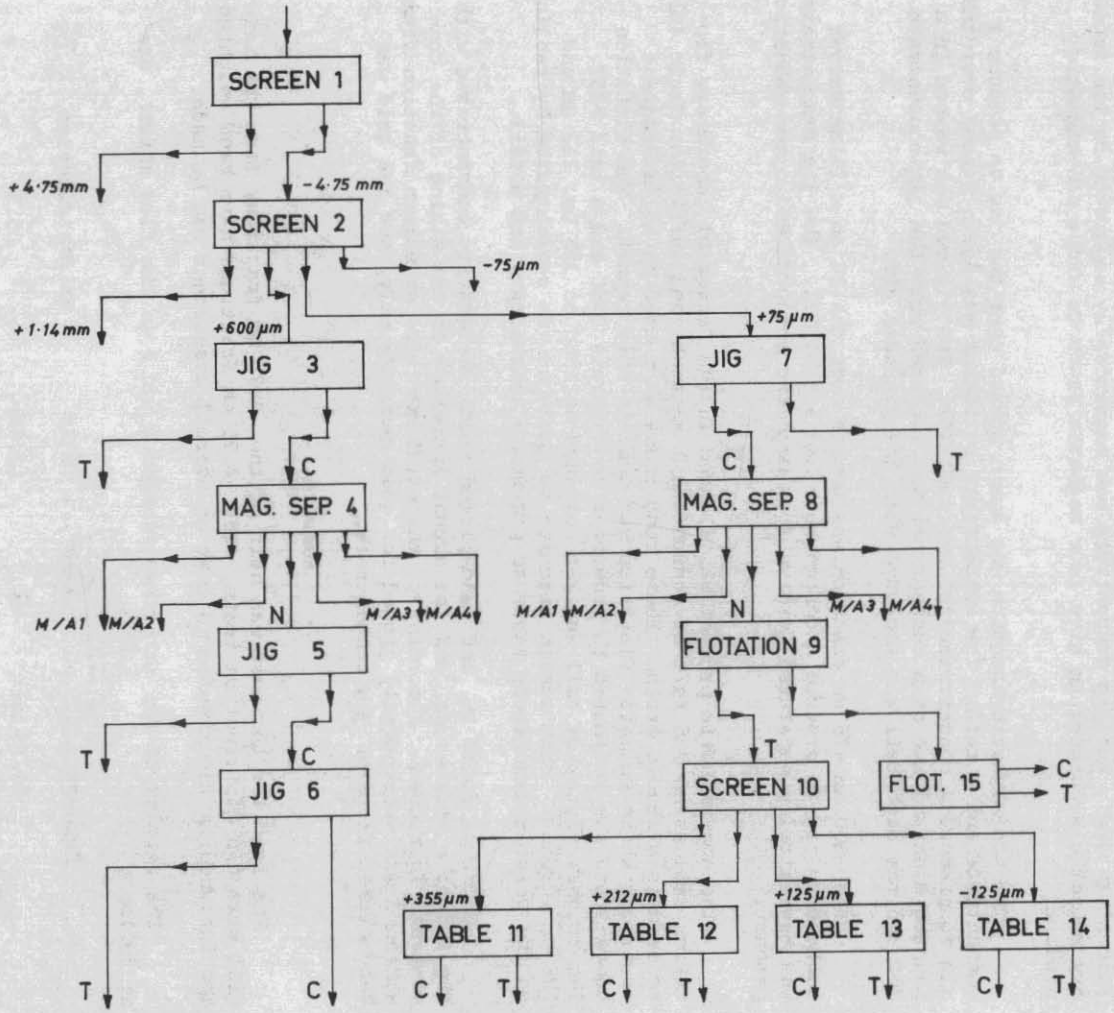


Figure 105. *R.688. Flow sheet, test N1.*

The results of jigging the -1.14 mm +600 µm screen fraction were:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J3C	4.9	(1.36)	27.4
J3T	10.0	0.16	6.6
-1.14 mm +600 µm	14.9	(0.55)	34.0

Magnetic separation of the -1.14 mm +600 µm jig concentrate gave the following results:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S4 M/A1	trace	0.36	trace
M/S4 M/A2	1.0	0.10	0.4
M/S4 M/A3	0.2	0.60	0.4
M/S4 M/A4	1.3	1.3	7.2
M/S4 N	2.4	1.9	19.4
J3C	4.9	(1.36)	27.4

Cleaning of the non-magnetic fraction by further jigging gave the following results:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J5T	2.0	1.0	8.3
J6T	0.3	3.35	4.0
J6C	0.1	16.0	7.1
M/S4 N	2.4	(1.95)	19.4

The results of jigging the -600 µm +75 µm screen fraction are as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J7C	3.0	3.2	40.7
J7T	14.9	0.12	7.4
-600 µm +75 µm	17.9	(0.65)	48.1

The -600 µm +75 µm jig concentrate was magnetically separated with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S8 M/A1	trace	0.30	trace
M/S8 M/A2	0.3	0.09	0.1
M/S8 M/A3	0.7	0.86	2.6
M/S8 M/A4	0.7	4.3	12.0
M/S8 N	1.3	4.9	26.0
J7C	3.0	(3.24)	40.7

The non-magnetics were then subjected to flotation to remove sulphides and the sulphide concentrate was cleaned with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
F9T	1.2	(5.59)	25.9
F15C	0.1	0.25	0.1
F15T	trace	0.52	trace
M/S8 N	1.3	(5.26)	26.0

The sulphide flotation tail (F9T) was screened on 355 µm, 212 µm, and 125 µm screens and the screen fractions were concentrated by tabling with the following result:

Product		*% Mass*	*% Sn*	*% Sn Distribution*
+355 µm	T11C	0.1	20.5	4.3
+355 µm	T11T	0.5	1.7	3.3
+212 µm	T12C	trace	45.6	7.1
+212 µm	T12T	0.4	3.2	4.3
+125 µm	T13C	trace	61.0	4.5
+125 µm	T13T	0.1	0.92	0.4
-125 µm	T14C	trace	63.7	1.3
-125 µm	T14T	0.1	1.5	0.7
	F9T	1.2	(5.59)	25.9

The weight of gold recovered and the size distribution was:

Size	*Mass of gold (mg)*
-4.75 mm +1.18 mm	9.58
-1.18 mm +600 µm	17.51
-600 µm +355 µm	2.11
-355 µm +212 µm	0.19
-212 µm +125 µm	0.06
-125 µm	nil
	29.45

This is equivalent to 0.0875 g of gold per tonne of feed. The largest pieces of gold were 2 mm across.

Discussion

It was hoped to produce, by jigging, a concentrate that would be suitable for transport to Burnie as outlined in the introduction. The combined result of the two jig concentrates J3C and J7C is:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J3C + J7C	7.9	(2.08)	68.1

On a feed rate of 500 tonnes per day, this would represent about 40 tonnes of concentrate to be transported.

Further treatment of the jig concentrates by magnetic separation, flotation, and additional gravity concentration (jigging and tabling) resulted in concentrates that gave a combined result as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J6C + T11C + T13C + T14C	0.2	(27.3)	24.3

Of these concentrates only two concentrates, T13C and T14C, were sufficiently high in grade to be regarded as final concentrate suitable for sale. When these are combined the result is as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
T13C + T14C	trace	(61.6)	5.8

A recovery of only 5.8% of the tin at a saleable grade from the jig concentrates that assayed 2.08% and carried 68.1% of the tin indicates that the cassiterate is very likely occurring in composite grains.

A second test was undertaken in which the grade of the primary jig concentrate was raised to a level more in line with what was proposed to be transported to Burnie, and incorporated grinding of products to effect liberation of the cassiterite.

Test N2

In this test the 0.45 m Sweco screen was fitted with a 2.41 mm aperture screen cloth and a 170 µm aperture screen cloth. The top screen was made larger in order that the coarse gold found present in test N1 could be recovered in the jig (fig. 106).

The -170 µm product from the screening was pumped to two 76 mm Warman cyclones in series - the underflow from the first cyclone being re-cycloned in the second cyclone. The purpose of cycloning the screen undersize was to collect any recoverable tin that was present in the screen undersize in test N1.

The -2.41 mm +170 µm fraction was jigged on the 10 cm x 15 cm Denver laboratory jig. The jig concentrate was hand screened with 710 µm, 425 µm and 250 µm screens. Each screen fraction was magnetically separated on the Rapid dry magnetic separator - operations M/S5 to M/S8 on the flow-sheet. The jig bedding was concentrated in a panning dish. The dish tail was put with the jig tail J3T. The dish concentrate was magnetically separated on the Rapid dry magnetic separator.

In test N1, the M/A4 magnetic fractions contained considerable quantities of tin. The M/A4 fractions from the above magnetic separations were bulked and subjected to five minutes grinding in the 203 mm diameter x 203 mm Warman laboratory ball mill, and then sulphide flotation in the Denver D1 laboratory flotation cell. The sulphide concentrate so produced was cleaned with two further stages of flotation.

The rougher flotation tail, F10T, was wet screened by hand using 425 µm, 250 µm, 150 µm and 38 µm screens. The screen fractions except the -38 µm fraction so produced, were dried and magnetically separated on the Rapid dry magnetic separator - operations M/S14 to M/S17 on the flow sheet (fig. 106.)

The non-magnetic fractions from the magnetic separations M/S7 and M/S8 were separately subjected to sulphide flotation in the Denver D1 flotation cell to produce sulphide concentrates (F22C and F23C). The flotation tails were separately tabled on the Deister laboratory table to produce final tin concentrates T24C and T25C and tailings. The table tailings were bulked with the non-magnetic fractions from magnetic separations M/S5 and M/S6 and ground in the 203 mm diameter x 203 mm Warman laboratory ball mill for five minutes.

The ball mill discharge was subjected to sulphide flotation to produce a sulphide concentrate F27C. The flotation tailing was wet screened by hand using 425 µm, 250 µm, 150 µm and 38 µm screens. The screen fractions so produced were bulked with the corresponding sized non-magmetic fractions from magnetic separations M/S14 to M/S17, and the -38 µm fraction was bulked with the -38 µm fraction from screen operation S13. There was insufficient +425 µm material and this was bulked with the +250 µm material for tabling but a satisfactory concentrate could not be made and this indicated that the material would require further grinding. The remaining fractions were tabled on the Deister table to produce final concentrates T19C to T21C and tailings.

The cyclone underflow C30 U/F was jigged on the 10 cm x 15 cm Denver

jig to produce a concentrate J31C and a tail. The bed for the jig was prepared by taking some of the dish tail that was produced by panning the jig bed from the jigging operation J3. Some of this oversize material found its way into the jig concentrate and was removed by hand screening the jig concentrate on a 180 µm screen. The oversize material was concentrated in a panning dish and the dish concentrate was included in the feed to B/M 27 for grinding and the dish tail was included with the coarse jig tail J3T. The jig bed from jigging the cyclone underflow J31 was hand screened on a 212 µm screen. The screen oversize was included in the coarse jig tail J3T, and the screen undersize was concentrated in a panning dish. The dish concentrate was put with the jig concentrate J31C and the dish tail was put with the jig tail J31T.

The jig concentrate J31C was subjected to sulphide flotation to produce a sulphide concentrate F32C, and the flotation tailing was concentrated by tabling on the Diester table to produce a concentrate and a tailing. The table concentrate T34C was magnetically separated to produce four magnetic products and a non-magnetic fraction. The non-magnetic fraction M/S 34N was concentrated by tabling on the Deister table to produce a final concentrate T35C and a tailing.

Gold was recovered from the gravity concentrates by hand-picking with the aid of the microscope.

Results

The feed for the test was the remaining half of the -4.75 mm fractions from the initial size/tin distribution tests. The +4.75 mm fractions have been weighted and included in the results so that an overall assessment can be made.

The results of the screening and the cycloning of the screen undersize in series (operations screen 1, screen 2, cyclone 29, and cyclone 30 in the flow sheet) are as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
+4.75 mm	41.9	(0.03)	4.9
-4.75 mm +2.41 mm	6.3	0.06	1.5
-2.41 mm +170 µm	40.3	(0.55)	86.7
-170 µm C30 U/F	6.5	(0.24)	6.1
-170 µm C30 O/F	3.1	0.03	0.4
-170 µm C29 O/F	1.9	0.06	0.4
Head	100.0	(0.26)	100.0

The results of the jigging of the -2.41 mm +170 µm screen fraction is as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J3C	1.0	(7.9)	31.5
J3T	39.3	0.36	54.8
Jig bed dish conc.	trace	(4.5)	0.4
-2.41 mm +170 µm	40.3	(0.55)	86.7

The jig concentrate was screened (screen 4) with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
+710 µm	0.4	(6.1)	9.5
-710 µm +425 µm	0.4	(7.5)	12.6
-425 µm +250 µm	0.2	(10.9)	6.7
-250 µm	trace	(22.8)	2.7
J3C	1.0	(7.9)	31.5

The screen fractions were then individually magnetically separated giving the following results:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S5 M/A1 + M/A2 + M/A3	0.2	0.84	0.7
M/S5 M/A4	trace	(7.4)	0.6
M/S5 N	0.2	(14.6)	8.2
+710 µm	0.4	(6.1)	9.5
M/S6 M/A1 + M/A2 + M/A3	0.2	1.0	0.6
M/S6 M/A4	0.1	(7.4)	4.0
M/S6 N	0.1	(14.6)	8.0
-710 µm +425 µm	0.4	(7.5)	12.6
M/S7 M/A1 + M/A2 + M/A3	0.1	2.5	0.5
M/S7 M/A4	0.1	(7.4)	2.0
M/S7 N	trace	(25.1)	4.2
-425 µm +250 µm	0.2	(10.9)	6.7
M/S8 M/A1 + M/A2 + M/A3	trace	2.0	trace
M/S8 M/A4	trace	(7.4)	0.3
M/S8 N	trace	(38.7)	2.4
-250 µm	trace	(22.8)	2.7

The jig bed dish concentrate was magnetically separated with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/A1 + M/A2 + M/A3	trace	0.97	trace
M/A4	trace	(7.4)	0.2
N	trace	6.2	0.2
Jig bed dish conc.	trade	(4.5)	0.4

The M/A4 fractions from magnetic separation M/S5 to M/S8 and from the magnetic separation of the jig bed dish concentrate were ground and then subjected to sulphide flotation. The sulphide concentrate was then refloated twice. The results of these flotation tests were as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
F10T	0.1	(13.6)	6.5
F11T	trace	3.9	0.4
F12T	trace	2.5	0.1
F12C	0.1	0.46	0.1
Total M/A4	0.2	(7.4)	7.1

The rougher flotation tail (F10T) was screened on 425 µm, 250 µm, 150 µm and 38 µm screens. Each screen fraction, except the -38 µm fraction, was magnetically separated. The +425 µm magnetic fractions were put with the -425 µm +250 µm magnetic fractions for assay purposes. The results were:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S14 + M/S15) M/A1 + M/A2)	trace	2.0	trace
M/S14 + M/S15) M/A 3 + M/A4)	trace	13.0	0.2
M/S14 + M/S15 N	trace	(17.5)	0.1
M/S16 M/A1 + M/A2	trace	2.1	0.1
M/S16 M/A3 + M/A4	trace	16.6	1.1
M/S16 N	trace	(17.5)	0.3
M/S17 M/A1 + M/A2	trace	1.8	0.1
M/S17 M/A3 + M/A4	trace	12.0	1.3
M/S17 N	trace	(17.5)	1.2
-38 µm	trace	(17.5)	2.1
F10T	0.1	(13.6)	6.5

The non-magnetics from magnetic separation 7 and 8 were each subjected to sulphide flotation (F22 and F23 respectively) with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
F22C	trace	0.38	trace
F22T	trace	(40.4)	4.2
M/S7 N	trace	(25.1)	4.2
F23C	trace	2.0	0.1
F23T	trace	(54.9)	2.3
M/S8 N	trace	(38.7)	2.4

The flotation tailings were then concentrated by tabling to give:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
T24C	trace	66.5	3.4
T24T	trace	(14.6)	0.8
F22T	trace	(40.4)	4.2
T25C	trace	72.5	2.1
T25T	trace	(14.6)	0.2
F23T	trace	(54.9)	2.3

The non-magnetics from magnetic separation 5 and 6 were combined with the above table tails T24T and T25T and ground in the ball mill and subjected to sulphide flotation with the following results:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
F27C	0.1	1.0	0.2
F27T	0.2	(17.5)	17.2
ball mill discharge	0.3	(14.6)	17.4

The flotation tail was screened on 425 µm, 250 µm, 150 µm and 38 µm screens (operation screen 28). The +425 µm and the +250 µm fractions were combined with the non-magnetics fraction from magnetic separations M/S14 and M/S15, but a satisfactory concentrate could not be produced by tabling, so the table products were recombined for assay purposes. The +150 µm and +38 µm screen fractions were combined with the corresponding non-magnetic fractions viz. M/S 16N and M/S17N respectively) and the -38µm fraction was

combined with the similar fraction from screen operation 13, and were individually concentrated by tabling to give final tin concentrates as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
+250 µm	trace	3.6	0.2
T19C	trace	58.3	3.4
T19T	0.1	6.7	1.8
T20C	trace	67.2	10.5
T20T	0.1	2.1	0.6
T21C	trace	58.2	2.9
T21T	0.1	4.5	1.5
	0.3	(17.5)	20.9

The cyclone underflow (C30 U/F) was jigged and gave the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J31C	0.3	(3.4)	4.1
J31T	6.2	0.07	1.7
Jig bed dish conc.	trace	(14.6)	0.3
C30 U/F	6.5	(0.24)	6.1

The jig concentrate was subjected to sulphide flotation to remove sulphides with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
F32C	trace	0.8	0.1
F32T	0.3	(3.8)	4.0
J31C	0.3	(3.4)	4.1

The flotation tail was concentrated by tabling with the following result:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
T33C	0.1	(18.8)	3.8
T33T	0.2	0.29	0.2
F32T	0.3	(3.8)	4.0

The table concentrate was then magnetically separated and gave the following results:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S34 M/A1 + M/A2	trace	1.0	0.1
M/S34 M/A3 + M/A4	trace	18.4	0.4
M/S34 N	trace	(41.6)	3.3
T33C	0.1	(18.8)	3.8

The non-magnetics were tabled to give a final tin concentrate of:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
T35C	trace	65.9	3.0
T35T	trace	9.7	0.3
M/S34 N	trace	(41.6)	3.3

The size distribution and weight of gold recovered was:

Size	*Weight of gold (mg)*
-2.41 mm +1.18 mm	102.46
-1.18 mm +600 µm	1.45
-600 µm +355 µm	7.49
-355 µm +212 µm	2.59
-212 µm +125 µm	0.33
-125 µm	nil
	114.32

This total weight is equivalent to 0.3073 g of gold per tonne of feed. The largest piece recovered measured approximately 2.5 mm x 3.75 mm. The number of gold particles recovered in this test was about the same as the previous test, but in this test the particles were generally bigger.

Discussion

A higher grade of primary jig concentrate which would be suitable for transport to Burnie, was produced in this test. The combined result of the two jig concentrates produced, J3C and J31C, is as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
J3C + J31C	1.3	(6.9)	35.7

On a feed rate of 500 tonnes per day, this would mean about 7 tonnes of jig concentrate to be transported.

Further treatment of the primary jig concentrates by magnetic separation, grinding, flotation, and further gravity separation yielded the following finished concentrates:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
T24C	trace	66.5	3.4
T25C	trace	72.5	2.1
T19C	trace	58.3	3.4
T20C	trace	67.2	10.5
T21C	trace	58.2	2.9
T25C	trace	65.9	3.0

The combined concentrate result in as follows:

Product	*% Mass*	*% Sn*	*% Sn Distribution*
Total tin cons.	0.1	(64.9)	25.3

The following products could be subjected to further grinding and concentration processes.

Product	*% Mass*	*% Sn*	*% Sn Distribution*
M/S14 + M/S15) M/A3 + M/A4)	trace	13.0	0.2
M/S16 M/A3 + M/A4	trace	16.6	1.1
M/S17 M/A3 + M/A4	trace	12.0	1.3
(S13 + S28) +250 µm	trace	3.6	0.2
M/S34 M/A3 + M/A4	trace	18.4	0.4
T19T	0.1	6.7	1.8
T20T	0.1	2.1	0.6
T21T	0.1	4.5	1.5
F11T	trace	3.9	0.4
F12T	trace	2.5	0.1
Total	0.3	(5.9)	7.6

About half of the tin in the above products could be expected to be recovered by further retreatment.

In test N1, 68.1% of the tin was recovered in the primary jig concentrate assaying 2% Sn, from 82.1% of the tin that was present in the jig feed. In test N2, 92.8% of the tin was present in the jig feed and if the jig had been operated as in test N1 to produce a 2% Sn concentrate, the recovery in the primary jig concentrate could be expected to be about 77%. In test N2, 25.3% of the tin was recovered as a saleable concentrate from 35.7% that was in the primary jig concentrate. If 77% was recovered in primary jig concentrate, then 55% could be expected to be recovered in a saleable concentrate. Retreatment of middlings would give further improvement to the recovery figure.

CONCLUSIONS

Screening of Arthur River gravels from test pits 1 and 2 and retaining the -1.14 mm +75 µm fraction recovered 82.1% of the tin in 32.8% of the mass.

Screening of the same material and retaining the -2.14 mm +170 µm fraction and cycloning the undersize recovered 92.8% of the tin in 46.8% of the mass.

In primary jigging, 68% of the tin can be recovered with a 2% concentrate grade, but recovery falls to 35% with a 7% concentrate grade.

In treatment of the primary jig concentrate with no grinding, only 24% of the tin can be recovered at a concentrate grade of 27% Sn, and only 6% can be recovered at a saleable grade - 61.6% Sn.

When grinding is utilised in the treatment of products derived from the primary jig concentrate carrying 35% of the total tin, a saleable grade concentrate assaying 64.9% Sn can be produced carrying 25% of the total tin.

It has been shown that 55% of the total tin could be expected to be recovered in a saleable grade of concentrate if the primary jig concentrate grade is maintained at about 2% Sn. This is more than double the recovery if primary jig concentrate is maintained at about 7% Sn.

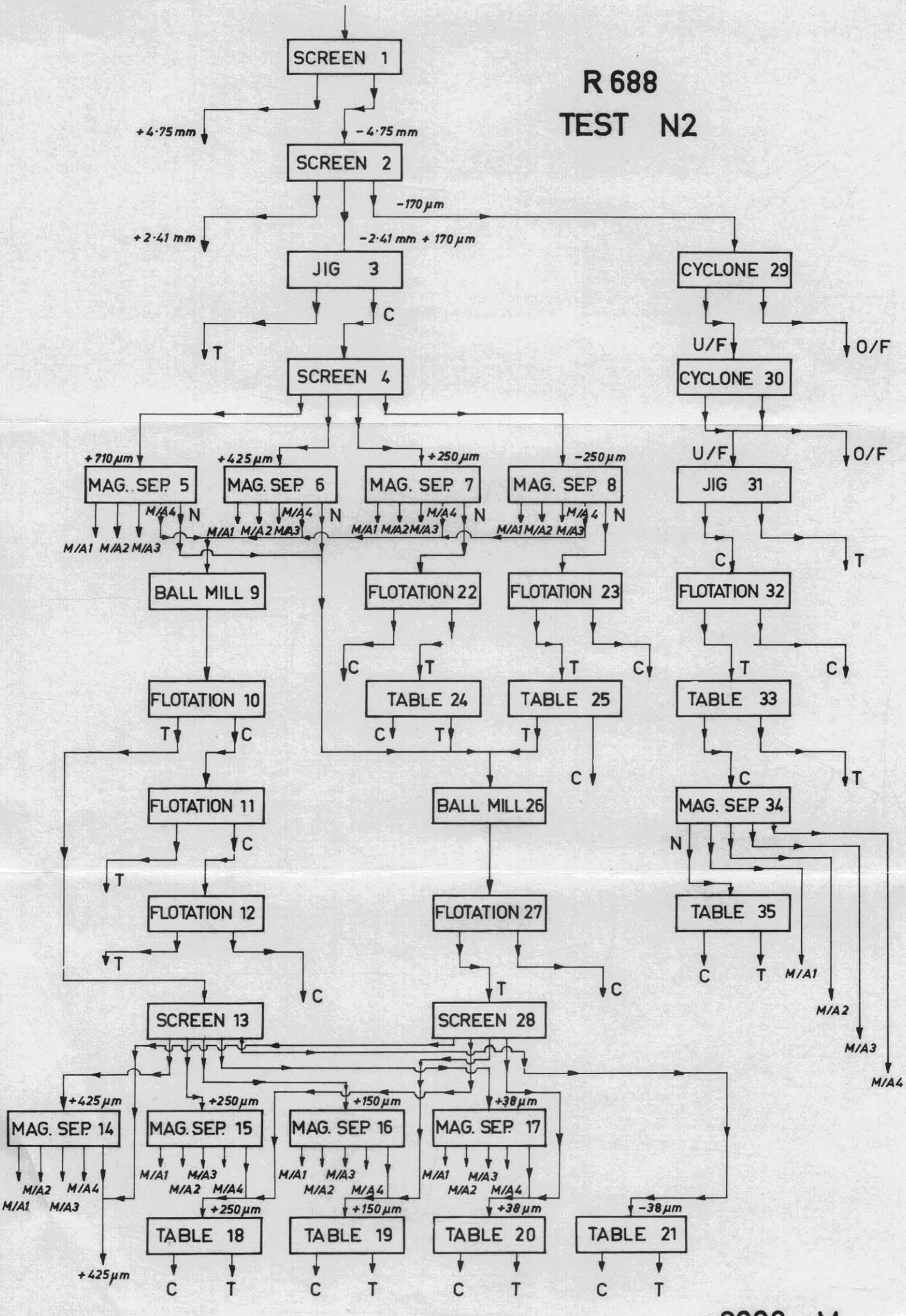
Production of a low grade primary jig concentrate may make the idea of transporting the primary jig concentrate to Burnie for concentrate dressing impractical. It is essential to produce a low grade primary jig concentrate to maximise tin recovery. It therefore seems advisable to produce a low grade primary jig concentrate for grinding and treatment on the site.

The flow sheet used in test N2 is fairly complicated, and with the knowledge gained from the test, future treatment of the primary jig concentrate should probably be as follows:- Ball mill grinding of the oversize material from a 400 µm screen which is in closed circuit with the ball mill. Screen undersize is floated to remove sulphides. The non-sulphide product is wet magnetically separated. The non-magnetics are sized and size fractions are concentrated by tabling. All middling products are reground in a separate regrind mill and recirculated. A vanner may be desirable to concentrate the very fine cassiterite that will be generated in the grinding.

The material recovered in the primary jig concentrate has the appearance of being a milled product, and is very likely derived from tailings from operations at Mt Bischoff.

Gold is present in comparatively coarse particles and should be readily recovered. The average of the gold contents for the two tests was 0.2 g/t.

[20 June 1975]



5 cm

FIGURE 106 TR20_296_310