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R.594. Recovery of molybdenite from King Island scheelite ore. Part 2. Further test work.

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A further four bench flotation tests and a pilot plant scale test using flotation and gravity concentration for recovery of molybdenite and scheelite respectively was done on the ore sample crushed to 3 mm as described in Part 1 (Technical Report No. 16, p.244).

TEST WORK

Four 1000 g samples were ground in the 200 mm diameter x 200 mm Warman laboratory ball mill. Two samples were ground to 10% +180 µm and two samples were ground to 10% +125 µm.

Flotation tests on these samples were conducted in the Denver D1 laboratory flotation cell.

The details of the flotation tests conditions are:

Condition		*Test N11*	*N12*	*N13*	*N14*
10% retained on screen	(µm)	125	180	180	125
Pine oil	(drops)	3	4	4	4
Sodium aerofloat	(grams/tonne)	-	-	50	50
Flotation time	(minutes)	4	3	3	3
Sodium ethyl xanthate	(grams/tonne)	100	250	250	250
Pine oil	(drops)	-	-	2	-
Flotation time	(minutes)	8	8	6	6

In the pilot plant run (test N15) the ore was ground in a 300 mm diameter x 300 mm Denver ball mill in closed circuit with a Hummer screen fitted with a 250 µm screen cloth. Only one-half of the screen surface was utilised, the actual screen area being 750 mm x 210 mm.

Screen undersize gravitated to a bank of no. 5 Denver sub-A flotation cells. Initially one cell was used for molybdenite flotation and three cells were used for pyrite flotation. However the molybdenite flotation was slow and a second cell for molybdenite flotation was introduced into the circuit. Pine oil was used for molybdenite flotation, and sodium ethyl xanthate was used for pyrite flotation.

The flotation tail was classified by means of a three spigot Geco hydrosizer. Each spigot product was separately concentrated on the Deister laboratory table. The concentrates from the two coarser spigot products were dried and magnetically separated on the Rapid laboratory magnetic separator.

The non-magnetics from each spigot fraction were then floated individually in the Denver D1 flotation cell to produce a molybdenite concentrate using pine oil and a pyrite concentrate using sodium ethyl xanthate. The two molybdenite concentrates so produced were bulked for assay purposes, as were the two pyrite concentrates. The flotation tail in each case was the final scheelite concentrate.

The magnetics were put with the table middlings from the first two spigots and were reground in the Warman laboratory ball mill. Molybdenite and pyrite concentrates were then produced from the reground middling by flotation in the Denver D1 flotation cell using pine oil, and sodium ethyl xanthate

respectively. The flotation tail was classified in the Geco hydrosizer. The two coarser spigot products were individually tabled. The table concentrates were magnetically separated and the non-magnetics were the final scheelite concentrate.

The third Geco spigot product from the middlings retreatment was put with the third Geco spigot product from primary treatment, and was tabled to produce a final concentrate.

The table middlings in this case joined the magnetics from the magnetic separation in middling retreatment, and was further reground in the Warman laboratory ball mill. The reground product joined the overflow from the Geco hydrosizer from primary treatment and from middlings retreatment and was cycloned in series in two 76 mm Warman cyclones. The underflow from the first cyclone was cycloned in the second cyclone.

The underflow from the second cyclone was designated as sand, and the overflows from the two cyclones were bulked and designated as slime. The sand fraction and the slime fraction were separately subjected to flotation to produce scheelite concentrates.

Flotation conditions were the same for both fractions and were as follows:

Sodium carbonate	4.7 kg/t
Sodium silicate	1.9 kg/t
Conditioning time	5 minutes
Cyanamid reagent 710	0.28 kg/t
Flotation time	7 minutes
Cyanamid reagent 710, after 4 minutes flotation	0.28 kg/t

Two stages of cleaning were used on the sand fraction and three stages of cleaning were used on the slime fraction. The results of the four batch tests are as follows:

Test	*Product*	*% Mass*	*Assay %*			*% Distribution*		
			WO_3	*Mo*	*S*	WO_3	*Mo*	*S*
N11	FC1	0.79	1.14	20.7	18.6	0.5	60.8	27.6
	FC2	1.56	1.22	1.09	20.9	1.1	6.4	61.3
	FT	97.65	1.70	0.09	0.06	98.4	32.8	11.1
	Head	100.0	1.69	0.27	0.53	100.0	100.0	100.0
N12	FC1	0.75	1.47	20.8	18.8	0.6	56.3	25.7
	FC2	2.82	1.59	1.20	13.1	2.7	12.3	67.2
	FT	96.43	1.70	0.09	0.04	96.7	31.4	7.1
	Head	100.0	1.70	0.28	0.55	100.0	100.0	100.0
N13	FC1	0.94	1.49	15.9	13.7	0.9	54.7	25.0
	FC2	1.98	1.41	2.32	17.6	1.7	16.8	67.4
	FT	97.08	1.60	0.08	0.04	97.4	28.5	7.6
	Head	100.0	1.60	0.27	0.52	100.0	100.0	100.0
N14	FC1	0.72	0.98	23.1	21.6	0.4	64.6	28.7
	FC2	1.67	1.49	1.37	20.7	1.5	8.9	64.1
	FT	97.61	1.70	0.07	0.04	98.1	26.5	7.2
	Head	100.0	1.69	0.26	0.54	100.0	100.0	100.0

These results show that pine oil is an adequate collector for molybdenite, and enables selective flotation of the molybdenite from the pyrite. The addition of a little sodium aerofloat may improve molybdenite flotation. Recovery of the molybdenite was better with a grind of 10% +125 µm than with a grind of 10% +180 µm.

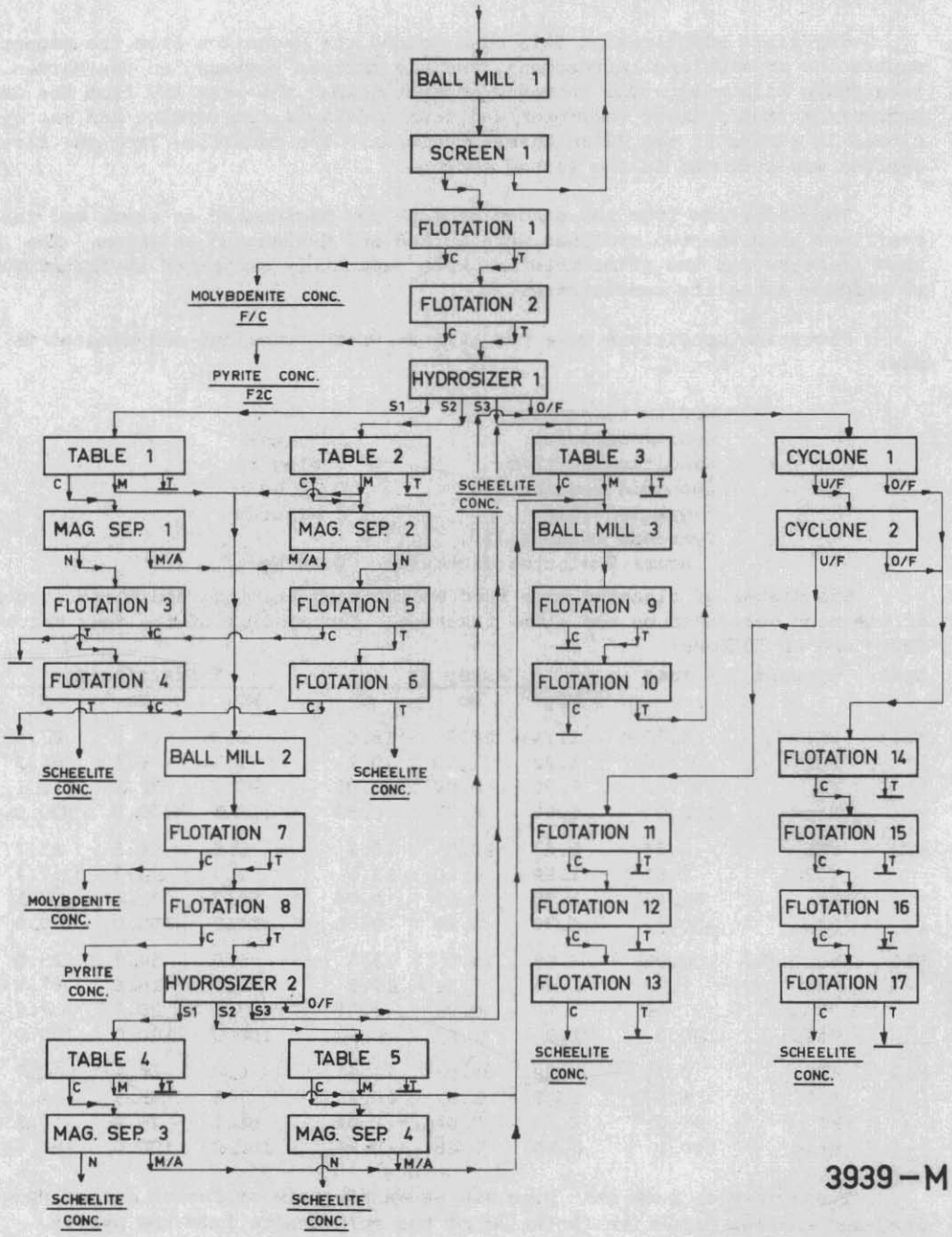

Figure 104. *R.594. Flow sheet, test N15.*

In the pilot plant test (N15), it became apparent that one cell was not adequate in floating all the molybdenite, as it could be seen floating with the pyrite concentrate. A second cell was thus included in the circuit. In spite of the additional cell molybdenite could still be seen to be floating with the pyrite concentrate. The addition of some sodium aerofloat, or additional cells would certainly have recovered some of this molybdenite. The result of the pilot plant run is:

Product	*% Mass*	*Assay %*		*% Distribution*	
		WO_3	*Mo*	*WO_3*	*Mo*
F1C	0.16	0.54	45.0	0.1	30.2
F2C	1.09	0.44	6.7	0.4	31.0
F3C + F5C	trace	11.1	*	trace	*
F4C + F6C	trace	13.6	*	trace	*
F4T	0.89	71.7	2.7	51.9	10.2
F6T	0.18	73.9	2.3	10.9	1.8
T3C	0.11	66.5	2.2	6.1	1.1
T1T	24.18	0.07	0.07	1.4	7.2
T2T	20.49	0.06	0.02	1.0	1.7
T3T	17.37	0.12	0.01	1.7	0.7
F7C	0.03	0.31	47.0	trace	5.9
F8C	0.04	0.32	1.6	trace	0.3
M/S3N	0.03	74.4	2.4	1.7	0.3
T4T	3.10	0.03	<0.01	0.1	0.1
M/S4N	0.01	72.8	2.4	0.7	0.1
T5T	3.14	0.05	<0.01	0.1	0.1
F9C	0.01	1.5	4.9	trace	0.2
F10C	0.01	0.87	0.43	trace	trace
F11T	14.74	0.08	0.01	1.0	0.6
F12T	0.54	0.56	0.06	0.2	0.1
F13T	0.08	1.4	0.11	0.1	trace
F13C	2.06	7.7	0.30	12.9	2.6
F14T	10.25	0.76	0.09	6.4	3.9
F15T	0.75	1.4	0.18	0.9	0.6
F16T	0.11	2.0	0.26	0.2	0.1
F17T	0.10	3.4	0.29	0.3	0.1
F17C	0.53	4.5	0.49	1.9	1.1
Head	100.0	1.22	0.24	100.0	100.0

*Insufficient sample to do Mo assays.

These results are summarised below:

	% Mass	*Assay %*		*% Distribution*	
		WO_3	*Mo*	*WO_3*	*Mo*
Scheelite gravity conc.	1.22	71.6	2.59	71.3	13.4
Scheelite flotation conc.	2.58	7.05	0.34	14.9	3.7
Molybdenite concentrate	0.19	0.50	45.3	0.1	36.0
Pyrite concentrate	1.13	0.44	6.53	0.4	31.2
Middlings	1.61	1.33	0.17	1.7	1.2
Tailings	93.27	0.15	0.04	11.6	14.5
Head	100.0	1.22	0.24	100.0	100.0

The grade of the scheelite concentrate recovered by gravity means was satisfactory at 71.6% WO_3 for a recovery of 71.3%. The molybdenum content of the gravity concentrate was 2.59% Mo. A further 14.9% of the scheelite was recovered in a flotation concentrate assaying 7.05% WO_3 and 0.34% Mo.

The grade of the molybdenite concentrate obtained was 45.3% Mo equivalent to about 75% MoS_2 which is a satisfactory result. Cleaning of this concentrate should produce molybdenite of a saleable grade. Scheelite loss in this concentrate was 0.1%.

From the results of the batch tests and from the molybdenite recovered in the molybdenite plus pyrite concentrates, it would appear a recovery around 60% should be possible, providing the correct conditions for molybdenite flotation prevail.

The molybdenite concentrate was given a sizing analysis with the following result:

Size (μm)	*% Mass*	*Cum. % Mass*
+212	0.2	0.2
-212 +150	2.5	2.7
-150 +106	8.5	11.2
-106 +75	11.9	23.1
-75 +53	13.6	36.7
-53 +38	15.5	52.2
-38	47.8	100.0

The Table tailings from the primary concentration of the three hydrosizer spigot products were given sizing analyses and the fractions were assayed for WO_3 and Mo to determine the respective distributions. The results were as follows:

Spigot 1 table tail

		Assay %		*% Distribution*	
Size (μm)	*% Mass*	*WO_3*	*Mo*	*WO_3*	*Mo*
+212	18.0	0.05	0.08	15.3	26.4
-212 +150	33.3	0.04	0.08	22.6	49.0
-150 +106	26.8	0.03	0.04	13.7	19.7
-106 +75	11.2	0.05	0.01	9.5	2.0
-75 +53	5.8	0.10	0.01	9.8	1.1
-53	4.9	0.35	0.02	29.1	1.8
	100.0	0.06	0.05	100.0	100.0

Spigot 2 table tail

		Assay %		*% Distribution*	
Size (μm)	*% Mass*	*WO_3*	*Mo*	*WO_3*	*Mo*
+212	3.6	0.04	0.06	3.0	10.6
-212 +150	12.7	0.04	0.05	10.6	31.5
-150 +106	32.8	0.01	0.02	6.9	32.6
-106 +75	17.2	0.02	0.01	7.2	8.5
-75 +53	16.4	0.02	0.01	6.9	8.2
-53	17.3	0.18	0.01	65.4	8.6
	100.0	0.05	0.02	100.0	100.0

Spigot 3 table tail

Size (μm)	% Mass	Assay %		% Distribution	
		WO_3	Mo	WO_3	Mo
+150	1.3	0.58	0.21	6.2	21.7
-150 +106	4.5	0.02	0.01	0.7	3.6
-106 +75	8.4	<0.01	<0.01	0.7	6.6
-75 +53	17.0	<0.01	<0.01	1.4	13.5
-53 +38	36.1	0.02	0.01	5.9	28.6
-38	32.7	0.32	0.01	85.1	26.0
		0.12	0.01	100.0	100.0

These results show that molybdenum losses occur mainly in the coarser fractions of each spigot product. During tabling, molybdenite flakes were seen to be floating across the surface of the table. Scheelite losses on the other hand occur mainly in the slime fractions and illustrates the need for careful classification of table feed.

At the time of tabling the hydrosizer spigot 3 products, the grade of the table concentrate appeared to be sufficiently high and magnetic separation was not used on the table concentrate. However the assay of the concentrate (T3C) was only 66.5% WO_3, and magnetic separation would have raised the grade to over 70% WO_3.

All the magnetic fractions from magnetic separation were put with the table middlings and reground, and eventually found its way to scheelite flotation. Higher scheelite flotation concentrate grades would very likely have been obtained if the magnetic fractions had not been included in the middling regrind.

The test was designed to keep untreated middling products down to a minimum. In a larger scale of treatment, the following untreated middling products would need to be recirculated for further treatment:-

(*a*) F3C + F5C, F4C + F6C, being respectively molybdenite and pyrite concentrates from flotation of the non-magnetic fractions from the magnetic separation of the table concentrates T1C and T2C.
(*b*) F9C and F10C, molybdenite and pyrite concentrates respectively from flotation of reground middlings from tabling of hydrosizer spigot 3 products.
(*c*) F12T, F13T and F15T, F16T, F17T, being successive cleaner tails from scheelite flotation of the sand and slime products.

The total amount of scheelite present in all these products amounted to only 1.7% of the total scheelite in the feed.

CONCLUSIONS

A high grade molybdenite concentrate can be produced by flotation prior to gravity concentration.

High grade scheelite concentrates can be produced by subsequent gravity concentration. The weighted average concentrate so produced in this test assayed 71.6% WO_3 and 2.59% Mo.

Slime scheelite proved to be difficult to recover in both gravity and flotation methods.