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R.674. Investigations on scheelite samples from Minefields Exploration N.L., Mt Mulgine, Western Australia.

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Two diamond drill hole samples were selected by Minefields Exploration N.L. from a batch of seven diamond drill hole samples for WO_3 recovery tests at a coarse size.

The samples chosen were from an area defined as the deep high grade, Central Western End. Their identification and WO_3 assays were as follows:

29 M	0.49% WO_3
106 M	0.18% WO_3

A composite sample was prepared by taking equal masses of the two drill hole samples and mixing them thoroughly by repeated coning and quartering. The two drill hole samples had been crushed to pass 9.53 mm for assay purposes and contained the reject fractions of the subsequent finer crushing stages taken to produce the assay samples. The sample therefore contained more fines than would have been the case if the sample had been simply crushed to pass 9.53 mm.

A head sample was taken for detailed analysis and a portion of this used for pressing for XRF analysis.

The results of the analyses were as follows:

	%		*g/t (ppm)*
WO_3	0.33	Cd	10
Insol.	79.1	Hg	2
Mo	0.01	Cu	160
As	0.01	Co	33
P	<0.01	Ni	560
S	2.8	Pb	37
Fe	1.8	Bi	14
Ca	0.46	Sn	12
Mg	0.92	Sb	35
F	1.4		
CO_2	0.2		
Zn	0.03		

The remainder was halved by riffling. One half was set aside for a concentration test. The other half was further reduced in quantity by riffling to provide a sample for a grindability test if required, a sample for heavy liquid separation tests and some remainder.

HEAVY LIQUID SEPARATION TEST N1

The sample for heavy liquid separation was sized on a 457 mm diameter Sweco screen fitted with 4.75 mm, 1.24 mm, and 653 µm aperture screen cloths.

The +4.75 mm, -4.75 mm +1.24 mm, and -1.24 mm +653 µm fractions were each subjected to heavy liquid separation at $\rho = 2.80\ t/m^3$ using tetra-bromoethane diluted with shellite.

Drill hole sample 29M contained a light coloured host rock consisting mainly of quartz and mica, and drill hole sample 106M contained a dark coloured host rock. Heavy liquid separation at a density of $2.80\ t/m^3$ caused the light coloured rock to float and the dark coloured rock to sink. A portion of the sink from the -4.75 mm +1.24 mm size fraction was examined under the microscope, and a number of free pieces of dark host rock were hand-picked and were identified as masses of small plates of biotite without any apparent orientation. The hand-picked pieces of biotite had a density of 2.89 t/m^3. It was seen under the microscope that most of the pieces of dark rock had a pyrite crystal embedded in it, with the dimension of one side usually lying within the range 1-2 mm.

The float from the +4.75 mm fraction was crushed to pass 4.75 mm and was hand screened on a 710 µm screen. The -4.75 mm +710 µm fraction was again subjected to heavy liquid separation.

All products obtained in this test were sampled and assayed and the results were as follows:

Size	*Product*		*% Mass*	*Assay* *% WO_3*	*% WO_3* *Distribution*
+4.75 mm	S/K		17.9	0.41	25.9
	F/T (crushed to pass 4.75 mm and screened on 710 µm)	+710 µm S/K	1.0	0.26	0.9
		+710 µm F/T	10.1	0.03	1.1
		-710 µm	3.5	0.07	0.9
	F/T (calculated)		(14.6)	(0.056)	(2.9)
	Size fraction (calculated)		(32.5)	(0.25)	(28.8)
-4.75 mm +1.24 mm	S/K		13.5	0.57	27.2
	F/T		14.9	0.02	1.0
	Size fraction (calculated)		(28.4)	(0.28)	(28.2)
-1.24 mm +653 µm	S/K		3.7	0.76	9.9
	F/T		4.4	0.02	0.3
	Size fraction (calculated)		(8.1)	(0.36)	(10.2)
-653 µm			31.0	0.30	32.8
	Head (calculated)		100.0	(0.28)	100.0

Discussion

Heavy liquid separation at a density of 2.80 t/m^3 has eliminated 33.9% by mass of the material for a loss of 4.2% of the WO_3. By crushing the float from the +4.75 mm size fraction to pass 4.75 mm and re-separating, the material eliminated in the float is reduced to 29.4% for a loss of 2.4% of the WO_3. The grade of the combined sink fractions is 0.47% WO_3 and with all the fines included the grade of the material is 0.38% WO_3.

The presence of the biotite (ρ = 2.89 t/m^3) may cause a problem if it is desired to eliminate more of this material by heavy liquid separation, as it mostly carries pyrite crystals, and if an attempt was made to eliminate these, then composites carrying scheelite would also be eliminated.

Float assays of 0.02% WO_3 in the fractions smaller than 4.75 mm are very satisfactory, and it is proposed in the subsequent gravity concentration test to crush to -4.75 mm before jigging is attempted.

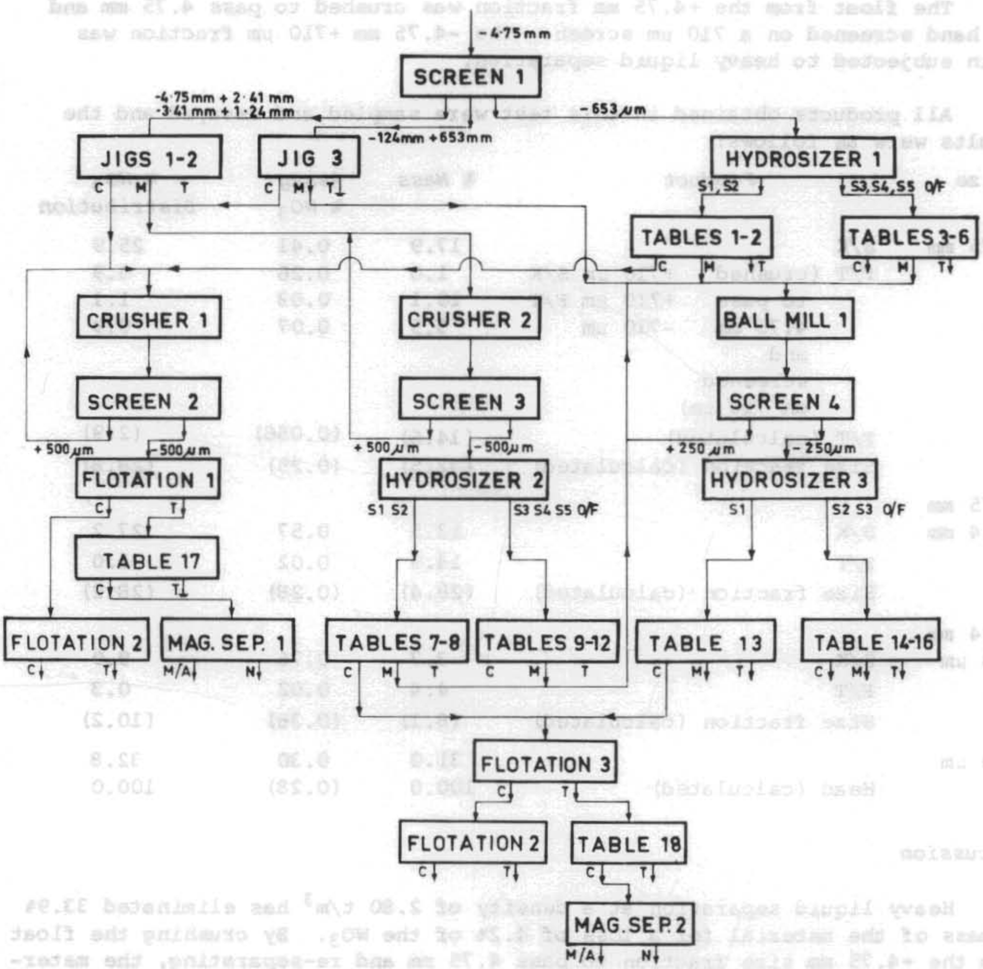

Figure 41. *R.674. Flow sheet, gravity concentration test N2.*

GRAVITY CONCENTRATION TEST N2

About 106.2 kg of ore was screened on a 4.75 mm screen and the oversize was crushed with a 152 mm x 254 mm diameter Denver rolls crusher. The crushed product was again screened and the oversize returned to the rolls until all the ore passed the 4.75 mm screen.

The ore was then wet screened on an 457 mm Sweco screen fitted with 2.41 mm, 1.24 mm, and 653 µm aperture screen cloths to produce the following size fractions, -4.75 mm +2.41 mm, -2.41 mm +1.24 mm, -1.24 mm +653 µm, and -653 µm.

Primary concentration

The three coarse screen fractions were concentrated by jigging in a 152 mm x 102 mm Denver jig to produce a jig concentrate and a tailing. The jig tailing was passed over the jig twice more to simulate treatment in a three-compartment jig. The concentrate from the second and third passes was regarded as jig middling, while the tailing from the third pass was final jig tailing. The jig concentrate from the first pass was further concentrated in a 50 mm x 38 mm Denver jig, to produce a final concentrate, and a tailing which was included with the jig middling.

The -653 µm fraction was passed through a three-spigot Geco hydrosizer using the three small water orifices to control the teeter water for the three spigots. When completed the orifice plates for No. 2 and No. 3 spigots were removed and the orifice plate from No. 1 spigot was transferred to No. 3 spigot, and larger orifices were placed in No. 1 and No. 2 spigots. The No. 1 spigot product from the first pass was returned through the hydrosizer again to form three spigot products - S1, S2, and S3. The overflow from this separation was bulked with the second spigot product of the first pass to give the overall fourth spigot product S4. The third spigot product from the first pass became the overall fifth spigot product S5, and the overflow from the first pass was considered the hydrosizer overflow. The operation was equivalent to using a five spigot hydrosizer.

Each hydrosizer fraction was separately concentrated on the Deister table, with the slime deck being used for the hydrosizer overflow. For each fraction, a concentrate, a middling and a tailing were produced.

Primary concentrate cleaning

Cleaning of the primary concentrates was achieved by taking the three jig concentrates and the table concentrates from the treatment of S1 and S2 hydrosizer spigots, and crushing with a pestle and mortar to pass a 500 µm screen, and then floating batchwise with a Denver D-1 laboratory flotation cell. Reagents used were 0.5 kg/t of potassium amyl xanthate and a few drops of Teric 401 as frother. The sulphide concentrate was refloated to give a clean sulphide concentrate. The non-sulphide product from the rougher flotation was tabled to form a concentrate and a tailing. The table concentrate was subjected to magnetic separation to remove magnetic minerals, and the non-magnetic fraction was the final scheelite concentrate. There was insufficient table concentrate from hydrosizer spigots S3, S4, S5, and the hydrosizer overflow to be included in the concentrate cleaning.

Jig middling retreatment

The jig middling from the two coarse screen fractions were crushed with the 152 mm x 254 mm diameter Denver rolls crusher and screened on a 500 µm screen. Screen oversize was again crushed until all the material passed through the screen. The material was treated in the three-spigot Geco hydrosizer to produce five spigot products and an overflow product as described in the primary treatment. Each hydrosizer fraction was concentrated by tabling in a similar manner to the primary treatment.

Fine jig and table middling retreatment

The table middlings produced from the jig middling retreatment were combined with the table middlings from the primary treatment together with the fine jig middlings, and were batch ground in an 203 mm diameter x 203 mm Warman laboratory ball mill. Ball mill discharge was screened on a 250 µm screen, and the oversize was included in the next batch, until all the material passed through the screen. Screen undersize was passed through the Geco hydrosizer fitted with the three smallest orifices to produce three-spigot products and an overflow product. The hydrosizer products were separately tabled to produce a concentrate, a middling, and a tailing.

Middling retreatment concentrate cleaning

Concentrates from the first two spigots in the jig middling retreatment and from the first spigot in the fine jig and table middling retreatment were combined and subjected to flotation using the same conditions as the cleaning of the primary concentrates.

The rougher sulphide concentrate was refloated to produce a cleaner sulphide concentrate and a cleaner tail. The rougher tail was tabled to produce a concentrate and a tailing. The concentrate was magnetically separated to remove the magnetic minerals, and the non-magnetic fraction was the final scheelite concentrate.

There was insufficient table concentrate from the hydrosizer spigots S3, S4, S5 and hydrosizer overflow from jig middling retreatment and from hydrosizer spigots S2, S3 and hydrosizer overflow from the table middling retreatment to be included in the concentrate cleaning.

Results

The results of the various treatments are shown on the following pages.

Primary concentration

Product		*% Mass*	*% WO_3*	*% WO_3 Distribution*
-4.75 mm +2.41 mm	J1C	0.27	6.6	5.94
	J1M	4.47	(0.27)	4.02
	J1T	14.92	0.11	5.47
-2.41 mm +1.24 mm	J2C	0.98	3.4	11.06
	J2M	9.01	(0.27)	8.12
	J2T	17.79	0.06	3.56
-1.24 mm +653 µm	J3C	1.41	2.46	11.55
	J3M	6.72	(0.12)	2.63
	J3T	2.04	0.04	0.27
S1	T1C	1.25	5.7	23.71
	T1M	3.61	0.28	3.37
	T1T	6.05	0.05	1.01
S2	T2C	0.34	5.32	6.13
	T2M	1.48	0.03	0.15
	T2T	7.56	0.01	0.25
S3	T3C	0.08	12.12	3.38
	T3M	0.46	0.05	0.08
	T3T	4.79	0.01	0.16
S4	T4C	0.07	11.16	2.65
	T4M	1.24	0.10	0.41
	T4T	7.54	0.01	0.25
S5	T5C	0.04	10.58	1.40
	T5M	1.05	0.35	1.22
	T5T	4.96	0.06	0.99
O/F	T6C	0.01	3.44	0.10
	T6M	0.21	0.29	0.20
	T6T	1.65	0.35	1.92
Calculated Head		100.0	(0.30)	100.0

Summary of primary concentration

Product	*% Mass*	*% WO_3*	*% WO_3 Distribution*
Total concentrates	4.45	4.44	65.92
Total middlings	28.25	0.21	20.20
Total tailings	67.30	0.06	13.88
Calculated head	100.0	(0.30)	100.0

Primary concentrate cleaning

				% WO_3 Distribution	
Product		*% Mass*	*% WO_3*	*Overall*	*Sectional*
F2C		2.87	0.04	0.38	0.7
F2T		0.32	0.43	0.46	0.8
F1T	T17T	0.78	0.80	2.09	3.7
	T17C M/A	0.04	4.9	0.58	1.0
	T17C N	0.24	66.0	53.43	93.8
Calculated head - derived products		4.25	(4.02)	56.94	100.0
Calculated head - products in feed			(4.12)		

Coarse jig middling retreatment

				% WO_3 Distribution	
Product		*% Mass*	*% WO_3*	*Overall*	*Sectional*
S1	T7C	0.48	3.5	5.58	45.8
	T7M	1.82	0.17	1.03	8.5
	T7T	1.52	0.08	0.41	3.3
S2	T8C	0.16	3.8	2.03	16.7
	T8M	0.62	0.07	0.15	1.2
	T8T	3.18	0.01*	0.11	0.9
S3	T9C	0.08	3.8	0.99	8.2
	T9M	0.37	0.01*	0.01	0.1
	T9T	1.62	0.01*	0.05	0.4
S4	T10C	0.03	3.0	0.26	2.1
	T10M	0.86	0.06	0.17	1.4
	T10T	1.50	0.01*	0.05	0.4
S5	T11C	0.01	7.0	0.32	2.6
	T11M	0.08	0.08	0.02	0.2
	T11T	0.55	0.05	0.10	0.8
O/F	T12C	0.01	6.9	0.26	2.1
	T12M	0.25	0.28	0.23	1.9
	T12T	0.34	0.36	0.41	3.4
Calculated Head		13.49	(0.27)	12.18	100.0

* These assays were reported as less than 0.01% but were taken as 0.01% in calculating the metal balance.

Summary of coarse jig middling retreatment

			% WO_3 Distribution	
Product	*% Mass*	*% WO_3*	*Overall*	*Sectional*
Total concentrates	0.77	3.69	9.44	77.5
Total middlings	4.00	0.12	1.62	13.3
Total tailings	8.72	0.04	1.12	9.2
Calculated head	13.49	(0.27)	12.18	100.0

Fine jig and table middling retreatment

				% WO_3 Distribution	
	Product	% Mass	% WO_3	Overall	Sectional
S1	T13C	0.44	3.8	5.55	57.4
	T13M	0.47	0.13	0.20	2.1
	T13T	7.67	0.01*	0.26	2.6
S2	T14C	0.03	6.1	0.73	7.6
	T14M	0.40	0.22	0.29	3.0
	T14T	4.35	0.01*	0.15	1.5
S3	T15C	0.05	5.1	0.81	8.4
	T15M	1.06	0.16	0.57	5.9
	T15T	2.46	0.01*	0.09	0.9
O/F	T16C	0.01	2.1	0.05	0.5
	T16M	0.52	0.21	0.36	3.8
	T16T	1.30	0.14	0.61	6.3
Calculated Head		18.76	(0.15)	9.67	100.0

* These assays were reported as less than 0.01% but were taken as 0.01% in calculating the metal balance.

Summary of fine jig and table middling retreatment

			% WO_3 Distribution	
Product	% Mass	% WO_3	Overall	Sectional
Total concentrates	0.53	4.05	7.14	73.9
Total middlings	2.45	0.17	1.42	14.8
Total tailings	15.78	0.02	1.11	11.3
Calculated head	18.76	(0.15)	9.67	100.0

Middling retreatment concentrate cleaning

				% WO_3 Distribution	
Product		% Mass	% WO_3	Overall	Sectional
F4C		0.83	0.01	0.03	0.3
F4T		0.01	7.4	0.31	3.2
F3T	T18T	0.18	0.69	0.42	4.3
	T18C M/A	0.02	2.3	0.12	1.2
	T18C N	0.04	69.4	8.91	91.0
Calculated head - derived products		1.08	(2.73)	9.79	100.0
Calculated head - products in feed			(3.66)		

Agreement between the calculated heads is not good. The amount of feed in this section was 709 g. There was a mass loss of 19.5 g during the testing and a mass loss of 7.5 g in the product T18C N would be sufficient to make the variation.

Discussion

These results show that cleaning of primary concentrates yielded a scheelite concentrate assaying 66.0% WO_3 with an overall recovery of 53.4%. Of the scheelite entering the cleaning section 93.8% was recovered in the final concentrate. If it is assumed that this recovery would also apply to the primary concentrates that were not sufficient in quantity to be cleaned, then the overall recovery is increased by 7.1% to 60.5%.

Cleaning of the concentrates derived from retreatment of the middlings yielded a scheelite concentrate assaying 69.4% WO_3 for an additional recovery of 8.9%. Of the scheelite entering the middlings retreatment cleaning section 91.0% was recovered in the concentrate. If it is assumed that this recovery would also apply to the middling retreatment concentrates that were not sufficient in quantity to be cleaned then overall recovery is increased by a further 3.1%.

It is therefore reasonable to assume that an overall recovery of 72.5% would be achieved. Concentrate grade would be 66.6% WO_3.

The concentrate cleaning results for both primary and middling retreatment show that the WO_3 assays of the cleaner sulphide concentrates are exceptionally low in WO_3 (0.04% and 0.01% respectively). This would indicate that the scheelite does not occur as composites with the pyrite.

Further treatment of the middlings from the retreatment of fine jig and table middlings, plus the sulphide cleaner flotation tail, the table tail, and the magnetics from the cleaning sections, would recover some of the 5.4% of WO_3 that is present.

The jig tailings contain 9.3% of the scheelite. Most of this could be recovered if it was crushed to pass a 500 µm screen as was done to the jig middlings.

Examination of the results of primary concentration, coarse jig middling retreatment, and fine jig and table middling retreatment show that tailings as low as 0.01% or less were achieved consistently. The following table shows at what hydrosizer spigot product they occurred. They are arranged in columns of corresponding size.

Primary concentration	S2	S3		
Coarse jig middling retreatment	S2	S3	S4	
Fine jig and table middling retreatment		S1	S2	S3

Following on from this it is useful to look at the distribution obtained in the gravity concentrations in primary treatment, coarse jig middling retreatment, and fine jig and table middling retreatment. These are shown in the following table with columns corresponding to approximately equal size ranges.

	J1	J2	J3	T1	T2	T3	T4	T5	T6	T7	T8	T9	T10	T11	T12	T13	T14	T15	T16
C	38.5	48.7	80.0	84.4	93.9	93.5	80.0	38.7	4.5	79.5	88.9	93.9	53.7	73.7	28.7	92.3	62.4	55.6	5.2
M	26.1	35.7	18.2	12.0	2.2	2.1	12.5	33.9	9.1	14.7	6.4	1.1	36.1	5.2	25.4	3.4	25.1	38.7	35.4
T	35.4	15.6	1.8	3.6	3.9	4.4	7.5	27.4	86.4	5.8	4.7	5.0	10.2	21.1	45.9	4.3	12.5	5.7	59.4

The above table shows that exceptionally good recovery can be obtained by table concentration of the middle size ranges. Recovery falls right away in the slime size range, and shows that it is essential to recover the scheelite as coarse as possible in primary concentration and in subsequent retreatment operations.

CONCLUSIONS

Scheelite concentrate of a saleable grade (66.6% WO_3) can be produced from ore with a head value of 0.30% WO_3 with a 72% recovery.

The scheelite is relatively coarse. Only 4.1% of the scheelite appears in the hydrosizer overflows in primary concentration, coarse jig middling retreatment, and fine jig and table middling retreatment. Of this only 0.4% is recovered. This recovery may be improved by cycloning the hydrosizer overflows, and tabling the cyclone underflow.

To prevent slime losses, the scheelite should be recovered as coarse as possible in primary concentration. It is suggested that the ore should be crushed to -4.75 mm before concentration of the coarse size fractions by jigging, and that the jig middlings and jig tailings should be reduced to -500 µm to liberate the composited scheelite present for recovery by subsequent table concentration. In effect it would mean concentration on single compartment jigs with tailing being ground in a rod mill to pass 500 µm, instead of trying to produce a middling fraction from a three compartment jig and rejecting a tailing as was done in this test.

The pyrite concentrates produced are exceptionally low in WO_3 and indicate that pyrite-scheelite composites are practically non-existent.

Overall recovery could be improved by an estimated 8% by retreatment of jig tailings as suggested above, and by retreatment of middlings from fine jig and table middlings retreatment, and by retreatment of the flotation cleaner tail, the table tail, and the magnetics from the concentrate cleaning sections.

ADDITIONAL DATA

The following additional work has been done.

Density determinations

Sample	ρ *(t/m^3)*
R.664 head assay sample	2.67
29M assay sample	2.60
106M assay sample	2.74

R.674 was conducted on an equal mass mixture of 29M and 106M.

Large drill core pieces:

29M	2.81
106M	2.90
116M yellow-white ore	2.72
116M greenish ore	3.05
120M (porous)	2.47
123M	2.74
130M	2.81
149M (porous)	2.51

Sizing

The only hydrosizer spigot products available from primary concentration were the table tailings from each spigot and the hydrosizer overflow. The mass of each table tailing was taken in proportion to the mass of the individual hydrosizer product (i.e. concentrate + middling + tailing for the hydrosizer product), and were then composited and a screen sizing carried out.

Results were as follows:

	Hydrosizer feed		*Overall*	
	% Mass	*Cum. % Mass*	*% Mass*	*Cum. % Mass*
+653 µm			57.6	57.6
+600 µm	1.6	1.6	0.7	58.3
+425 µm	11.9	13.5	5.0	63.3
+300 µm	9.6	23.1	4.1	67.4
+212 µm	9.9	33.0	4.2	71.6
+150 µm	10.8	43.8	4.6	76.2
+75 µm	26.2	70.0	11.1	87.3
+38 µm	16.8	86.8	7.1	94.4
-38 µm	13.2	100.0	5.6	100.0

Analysis of composite final scheelite concentrate.

	%		%
As	0.39	Bi	<0.02
Mo	0.01	S	0.4
P	0.11	Mn	0.02
Sn	0.3	Pb	0.01
Cu	<0.01	Zn	0.01
Sb	<0.01	Fe	0.4
F	2.1	SiO_2	4.5

The weighted composite assayed above has a calculated WO_3 content of 66.6%.

[25 January-23 October 1974]