

TR18 197-200

R.664. Recovery test on scheelite ore from Mt Mulgine, Western Australia.

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A 23 kg sample of scheelite ore from Mt Mulgine, Western Australia, was forwarded by G.M. Sainsbury, on behalf of Minefields Exploration N.L., for recovery tests.

The sample was said to be made up of core sample rejects, crushed to -3.18 mm, combined in proportions to represent the grade of the presently inferred ore reserves, approximately 0.5% WO₃.

It was requested that the liberation characteristics of the tungsten mineral be determined together with the optimum concentrating procedure, reporting WO₃ recovery and concentrate grade.

TEST WORK

The sample was cut down by riffing to produce a head sample for assay purposes, and a second sample for a sizing analysis. It was observed that some pieces of ore were present up to 9.5 mm in size. The head sample assayed 0.49% WO₃, <0.01% Mo.

A qualitative scan of the head sample by XRF showed that tungsten was the only economic element present.

The sample for a sizing analysis was wet screened on a 38 µm screen. The oversize was dried and screened on 2.36 mm, 1.18 mm, 600 µm, 300 µm, 150 µm, 75 µm and 38 µm screens.

All size fractions except the -38 µm fraction were subjected to heavy liquid separation using tetra-bromo-ethane (density 2920 kg/m³). The sinks and the floats for each size fraction and the -38 µm were assayed for WO₃.

The results are as follows:

| Size Fraction | % Mass | Sink | | | Float | | |
|-----------------|--------|--------|-------------------|-------------------------|--------|-------------------|-------------------------|
| | | % Mass | % WO ₃ | WO ₃ % Distn | % Mass | % WO ₃ | WO ₃ % Distn |
| +2.36 mm | 8.0 | 1.1 | 1.3 | 2.8 | 6.9 | 0.07 | 0.9 |
| +1.18 mm | 7.0 | 1.0 | 0.79 | 1.6 | 6.0 | 0.05 | 0.6 |
| +600 µm | 12.6 | 1.6 | 3.5 | 11.0 | 11.0 | 0.05 | 1.1 |
| +300 µm | 16.7 | 1.7 | 5.1 | 17.0 | 15.0 | 0.03 | 0.9 |
| +150 µm | 17.2 | 1.5 | 6.1 | 18.0 | 15.7 | 0.03 | 0.9 |
| +75 µm | 16.9 | 1.2 | 6.8 | 16.0 | 15.7 | 0.02 | 0.6 |
| +38 µm | 11.0 | 0.8 | 7.5 | 11.8 | 10.2 | 0.02 | 0.4 |
| -38 µm | 10.6 | | 0.79 | 16.4 | | | |
| Calculated Head | 100.0 | | 0.51 | | | | |

Before pulverising for assay, the sink from the +300 µm size fraction was subjected to skin flotation in a dish to remove the sulphides, and the non-sulphides were then magnetically separated. Each product obtained was then examined under the microscope. The sulphide fraction appeared to be all pyrite, the magnetic fraction consisted of ferromagnesian minerals and there appeared to be fluorite and scheelite in the non-magnetic fraction. The +300 µm size fraction was then reconstituted for assay.

In the above table, summation of the WO₃ distribution in the sink

products gave 78.2%, and it was felt that this value would be a guide to what recovery could be expected.

An examination of the assays in the table shows that the WO₃ content of the sink products increased fairly sharply up to the +300 μm fraction after which the increase in WO₃ content was not so great. The WO₃ content of the float products decreased to the +300 μm fraction, after which the assay values of the float products levelled out. It appeared, therefore, that the liberation of the scheelite occurred at about 600 μm.

Accordingly, the sample as received was screened on an 18-inch (457 mm) Sweco screen fitted with 653 μm, 315 μm, 170 μm and 76 μm screens. The +653 μm material was roll crushed and re-screened. This process was repeated until all the material eventually passed through the 653 μm screen.

Each of the size fractions obtained was table concentrated with the +315 μm and +170 μm fractions using the sand deck, and the +76 μm and -76 μm fractions using the slime deck of the Deister table.

The -76 μm fraction was riffled into two parts, one for table concentration and the other for a flotation test.

In each of the tabling operations, three products were made, a table concentrate, a table middling and table tailings. In each case, the cut on the table to produce the concentrate was made in the middle of the sulphide band with the aim of eliminating the less dense heavy minerals such as fluorite and the ferromagnesian minerals. The table concentrate was then floated in a flotation cell using potassium amyl xanthate, sulphuric acid and methyl iso-butyl carbinol (MIBC) to remove the sulphides. The non-sulphide product was dried and passed over a Rapid magnetic separator. The non-magnetic fraction was the final scheelite concentrate. There were only a few grams of magnetic minerals from each size fraction and these were bulked for assay purposes.

The results were as follows:

| Size Range (μm) | Product | % Mass | % WO ₃ | % WO ₃ Distn |
|-----------------|------------|--------|-------------------|-------------------------|
| -653 +315 | N | 0.24 | 60.0 | 29.6 |
| | FC | 1.03 | 0.17 | 0.3 |
| | TM | 5.6 | 0.16 | 1.8 |
| | TT | 30.3 | 0.05 | 3.1 |
| -315 +170 | N | 0.23 | 63.8 | 30.6 |
| | FC | 0.86 | 0.21 | 0.4 |
| | TM | 1.8 | 0.85 | 3.1 |
| | TT | 37.6 | 0.09 | 7.0 |
| -170 +76 | N | 0.13 | 51.5 | 13.7 |
| | FC | 0.31 | 0.30 | 0.2 |
| | TM | 0.8 | 0.29 | 0.5 |
| | TT | 16.6 | 0.07 | 2.4 |
| -76 | N | 0.02 | 43.0 | 2.1 |
| | FC | 0.05 | 4.6 | 0.5 |
| | TM | 1.8 | 0.29 | 1.1 |
| | TT | 2.6 | 0.31 | 1.7 |
| | Total M/A | 0.06 | 14.5 | 1.9 |
| | Calculated | | | |
| | Head | 100.0 | 0.48 | 100.0 |

A summary of these results is as follows:

| Product | % Mass | % WO ₃ | % WO ₃ Distn |
|-----------------|--------|-------------------|-------------------------|
| Total N | 0.6 | 59.0 | 76.0 |
| Total M/A | 0.1 | 14.5 | 1.9 |
| Total FC | 2.2 | 0.30 | 1.4 |
| Total TM | 10.0 | 0.32 | 6.5 |
| Total TT | 87.1 | 0.08 | 14.2 |
| Calculated Head | 100.0 | 0.48 | 100.0 |

The composite concentrate was assayed for impurities with the following result.

| | | | |
|----|--------|----|-------|
| Mo | 0.01% | S | 0.14% |
| As | <0.01% | Fe | 0.16% |
| P | 0.25% | Cu | Nil |

In the flotation test, sulphides were first floated off using potassium amyl xanthate, sulphuric acid, and MIBC, and then an attempt was made to float the scheelite using oleic acid and sodium carbonate. Some material floated which appeared to be talc and at the end of flotation the concentrate was examined under ultra-violet light and scheelite was seen in the concentrate, and was also seen in the flotation tail. The assays showed that the test was unsuccessful. The results of the test were as follows:

| Product | % Mass | % WO ₃ | % WO ₃ Distn |
|---------|--------|-------------------|-------------------------|
| F1C | 14.3 | 0.50 | 11.2 |
| F2C | 3.3 | 0.62 | 3.1 |
| FT | 82.4 | 0.67 | 85.7 |
| | 100.0 | | 100.0 |

Note: The per cent mass and per cent distribution figures in the above table are with respect to the flotation test only.

DISCUSSION

This recovery test can only be regarded as a preliminary test to give an idea of what recovery can be expected with this ore, and should not be used as the basis for designing a plant to treat the ore.

A second test on a much larger sample should be contemplated. A larger test will allow mineralogical examinations of the concentrate and middling fractions to determine the diluent minerals and the predominance of scheelite-bearing composites.

The size distribution in the gravity concentration of the various size fractions differs from the size distribution as shown in the WO₃ distribution determined by heavy liquid separation. This is due to the fact that the material for the gravity concentration tests was dry screened. It is recommended that in subsequent test work, sizing for tabling be by wet screening or by hydrosizing.

The summary table indicates that treatment of the middling presents no great problem. It is possible that the WO₃ present in the middling and in the tailing would be reduced either by wet screening or by hydrosizing the table feed. The WO₃ could be present in these products as fine free scheelite. If this is the case, then it could be that there is no middling

problem at all, and a middling regrind circuit would not be warranted. This information would be provided by a larger test.

The table concentrate was made by taking a cut at the point where clean sulphide material was being produced, with the aim of eliminating the less dense heavy minerals. The concentrate was then subjected to sulphide flotation and then magnetic separation. A small amount of magnetic ferromagnesian material was removed, suggesting that some non-magnetic material like fluorite could still be present in the concentrate. It is therefore recommended that in further test work, the table concentrate after sulphide flotation be tabled again to produce a final concentrate.

The scheelite flotation test was unsuccessful. It is difficult to judge how well scheelite is floating, as it is necessary to wait on assay results before the success of a flotation test can be determined. A large quantity of material would be necessary so that a number of flotation tests with various reagent combinations could be made before any conclusion could be reached regarding the desirability of using flotation. The table tail on the -76 μm fraction was 0.31% WO_3 , and it is possible that this value could be lowered by the use of vanners in plant practice. If this is the case, then flotation for the recovery of fine scheelite may not be necessary.

There is an abundance of mica present and it is possible that this could be a profitable by-product.

CONCLUSIONS

The ore is easy to treat. A recovery of 76% of the WO_3 was made with a grade of 59% WO_3 .

The scheelite is present as comparatively coarse particles and it appears that below 600 μm composite scheelite particles present no problem to concentrate. Pyrite is the main gangue heavy mineral with smaller amounts of fluorite and ferromagnesian minerals present.

Further test work on a larger sample with alterations in procedure as outlined above should result in improvement in both grade and recovery.

[30 July 1973]