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# RECOVERY OF BARYTES FROM DUMPS AT ORAPARINNA.

By

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# RECOVERY OF BARYTES FROM DUMPS AT ORAPARINNA.

## OBJECT.

Laboratory tests were made to establish whether barytes of a market grade could be recovered from dumps of discarded material accumulating from barytes mining at Oraparinna.

## SUMMARY.


1. A parcel of ore representing the dumps consisted of 81 per cent barytes, mixed with shale.
2. Heavy Media Separation at one inch recovered 69 per cent of total barytes in a product assaying 95.2 per cent barytes.
3. Jigging at 1/4 inch recovered 79 per cent of total barytes in a product assaying 96.4 per cent barytes. This could be applied to the portion of ore not sent to Heavy Media Separation, increasing total recovery to 92 per cent.
4. Tabling the fines not treated in the jig gave a recovery of 52.6 per cent in a product assaying 98.3 per cent. This would raise recovery by jigging and tabling to 87.5 per cent.
5. Flotation is not recommended.
6. Material produced from these dumps is suitable for second grade paint filler or oil drilling mud. It could be produced in Adelaide for £21 per ton without capital expenditure by sale of dumps for treatment in existing plants.
7. It is suggested that run-of-mine ore could probably be treated at the mine, and the product delivered in Adelaide for £20 per ton. Half the product would be first grade paint material, and half suitable for oil drilling mud.

## SAMPLE.

### Location.

Mineral leases 2933 and 2936 are held by S. A. Barytes Ltd. in the county of Taunton about 66 miles north northeast of Hawker. The dumps consist of fines rejected prior to hand sorting and coarse material rejected by hand sorting. It is estimated by the Assistant State Mining Engineer that 15,300 tons are available with an average content of 78.4 per cent barytes.

### Ore Parcel.

A bagged parcel weighing approximately six tons was supplied by S. A. Barytes Ltd. on 19th November 1953. It was stated to contain coarse and fine material in similar proportions to the size of the respective dumps.

| | |
|---|---|
| Part 1 - Old top stope spill | 22 bags |
| Part 2 - Dump leading off grizzly | 26 bags |
| Part 3 - Three dumps East | 29 bags |
| Part 4 - Present dump, fine | 27 bags |
| Part 5 - Present dump, coarse | 22 bags |

The bags were bulked for testing. A sample taken after thorough mixing assayed 81 per cent $BaSO_4$.

### Mineral Content.

The ore appeared to contain both clean and discoloured barytes in a gangue of shale. The Departmental Petrologist was asked to describe the impurities in the massive barytes and determine whether the shale carried fine barytes. Following is part of the report:-

"Each of the massive barite samples is of the same mineralogical composition, predominantly barite with a percentage of gangue minerals. The clay minerals only represent a small percentage of the rock, and they are ultra-fine particles distributed throughout the barite, in places concentrated along the fracture and cleavage planes. The reddish ferruginous material is confined to the shear and cleavage planes and the intergranular crystal faces. This can be removed by washing a crushed sample in a warm dilute solution of hydrochloric acid.

The specimens of shale are a finely granular laminated quartz mica shale. The minerals present are muscovite, chlorite, quartz, hematite, calcite and clay minerals. A limonitic film is coating the fracture planes and cleavage planes. No barite mineralisation is to be found in these shales."

## MINERAL DRESSING.

Barytes was first recovered by gravity methods, using Heavy Media Separation, jigging or tabling with ore crushed to the appropriate size ranges. Further cleaning of the product was attempted by electrostatic and magnetic separation of the iron stained particles.

Some preliminary tests were made with separation by flotation. This alternative does not appear attractive at present.

### Heavy Media Separation.

Approximately half a ton of ore was crushed to minus 3/4 inch and screened at ten mesh. The coarse portion was treated in a 20-inch cone Heavy Media Separator using ferrosilicon as the medium. Tests were run with the cone specific gravity at 2.80 and 2.95. In the first case the recovery was 97.9 per cent and the grade of product was 95.2 per cent $BaSO_4$ from that part of the ore treated in the H.M.S. unit. This is equal to 69.3 per cent of the barytes in the ore. At the higher bath density the recovery dropped to 86.8 per cent.

TABLE No. 1.

Distribution of Barytes in Ore.

| Size Range | % Weight | % $BaSO_4$ | % Distribution |
|---|---|---|---|
| -3/4 inch, +10 mesh | 71.1 | 81.3 | 70.8 |
| -10 mesh | 28.9 | 82.7 | 29.2 |
| Ore | 100 | 81.8 | 100 |

TABLE No. 2.

Heavy Media Separation with Conc. S.G. = 2.80.

| Product | % Weight | % $BaSO_4$ | % Distribution |
|---|---|---|---|
| Concentrate | 83.6 | 95.2 | 97.9 |
| Tailing | 16.4 | 10.6 | 2.1 |
| Feed to unit | 100 | 81.3 | 100 |

The distribution of the barytes in the various size fractions of the concentrates indicates that efficiency is lower with material less than 1/4 inch in size.

TABLE No. 3.

Barytes Distribution in H.M.S. Concentrate.

| Size Range | % Weight | % $BaSO_4$ | Recovery for fraction |
|---|---|---|---|
| plus 1/2 inch | 23.0 | 94.9 | 99.0 |
| 1/2 in. to 3/8 in. | 14.0 | 94.8 | 99.0 |
| 3/8 in. to 1/4 in. | 24.3 | 95.3 | 98.9 |
| 1/4 in. to 8 mesh | 28.6 | 96.5 | 96.8 |
| minus 8 mesh | 9.7 | 94.1 | 94.8 |
| Whole Concentrate | 100 | 95.2 | 97.9 |

The properties of the concentrate are as follows:-

| | | | | |
|---|---|---|---|---|
| 1. | Specific gravity | | 4.22 | |
| 2. | Colour | reflectance | 86 | per cent |
| 3. | Analysis | $BaSO_4$ | 95.2 | per cent |
| | | $Fe_2O_3$ | 0.61 | per cent |
| | | $SiO_2$ | 2.0 | per cent |

Jigging.

A parcel of ore was crushed to give 98 per cent through a screen with 1/4 inch openings. The plus 20 mesh portion was separated on a screen and the fines were treated in a hydrosizer using a rising water velocity of ten feet per minute. This gave a separation at close to 100 mesh. The coarse material from the screen and the sand from the hydrosizer was combined as jig feed. As will be seen from the test results, it appeared later that simple screening on 20 mesh would have been better because jig efficiency was low on material less than 40 mesh in size.

Jigging was carried out in a Denver Duplex machine, each hopper 12 inch by 9 inch in area. The feed rate was 0.20 tons per hour, with a circulating load of 0.07 tons per hour from the second hutch. Water required (most of which is recoverable) is 8,500 gallons per ton. Barytes between 1/2 inch and 1/4 inch was used as ragging to a depth of one inch on a 1/4 inch opening grid. The recovery was 94.5 per cent and the grade of product 96.4 per cent $BaSO_4$ from that part of the ore treated in the jig. This is equal to 79.3 per cent of the barytes in the ore.

TABLE No. 4.

Distribution of Barytes in Ore.

| Product | % Weight | % $BaSO_4$ | % Distribution |
|---|---|---|---|
| Jig Feed | 83.1 | 82.5 | 83.9 |
| Hydrosizer fines | 16.9 | 78.3 | 16.1 |
| Ore | 100 | 81.8 | 100 |

TABLE No. 5.

Results of Jigging.

| Product | % Weight | % $BaSO_4$ | % Distribution |
|---|---|---|---|
| Concentrate | 81.0 | 96.4 | 94.5 |
| Tailing | 19.0 | 23.8 | 5.5 |
| Feed to Unit | 100 | 82.5 | 100 |

TABLE No. 6.

Barytes Distribution in Jig Concentrate.

| Size Range | %. Weight | % $BaSO_4$ | Recovery for fraction |
|---|---|---|---|
| Plus 1/4 inch | 4.2 | 96.1 | 98.3 |
| 1/4 in. to 8 mesh | 37.5 | 96.2 | 98.7 |
| 8 mesh to 22 mesh | 34.3 | 96.3 | 98.7 |
| 22 mesh to 44 mesh | 13.8 | 97.0 | 95.7 |
| 44 mesh to 60 mesh | 6.8 | 97.4 | 76.7 |
| 60 mesh to 100 mesh | 2.0 | 97.6 | 61.9 |
| Minus 100 mesh | 1.4 | 95.1 | 60.8 |
| Jig Concentrate | 100 | 96.4 | 96.4 |

The properties of the concentrate are as follows:-

| | | | |
|---|---|---|---|
| 1. | Specific gravity | | 4.42 |
| 2. | Colour | reflectance | 87 per cent |
| 3. | Analysis | $BaSO_4$ | 96.6 per cent |
| | | $Fe_2O_3$ | 0.26 per cent |
| | | $SiO_2$ | 2.42 per cent |

Tabling.

Because good recoveries were obtained with gravity separation at 3/4 inch and 1/4 inch sizing it was not considered necessary to reduce all the ore to table size. It was hoped that tabling would be a suitable method for treating the hydrosizer slimes which were separated from the ore before jigging.

The small standard laboratory Wilfley deck was used for the tests. Three passes were given. This is estimated to reproduce closely one pass on a full size deck. The tailing from the first pass was retreated, and concentrates from these two runs was cleaned once. The results show an excellent grade of concentrate with a recovery of 52.6 per cent. This is equal to 8.5 per cent recovery on the original ore, to add to a recovery in jigs of 79.3 per cent.

TABLE No. 7.

Results of Tabling Hydrosizer Slime.

| Product | % Weight | % $BaSO_4$ | % Distribution |
|---|---|---|---|
| Concentrate | 42.0 | 98.3 | 52.6 |
| Middling | 1.85 | 86.5 | 2.0 |
| Tailing | 56.2 | 63.2 | 45.4 |
| Table Feed | 100 | 78.4 | 100 |

It was observed that the tailing from the jig section contained a considerable percentage of barytes in the fraction below 20 mesh sizing. This fraction was treated by four passes on the laboratory table. Although it contained only 70.7 per cent barytes, a concentrate of 95.7 per cent barytes was obtained, with a recovery of 85.7 per cent.

The properties of the table concentrate from the Hydrosizer Slime are as follows:-

| | | | |
|---|---|---|---|
| 1. Specific Gravity | | 4.3 | |
| 2. Colour | reflectance | 81 | per cent |
| 3. Analysis | $BaSO_4$ | 98.3 | per cent |
| | $Fe_2O_3$ | 0.34 | per cent |

Colour Improvement.

It was noticed that grains of iron bearing minerals accumulated in the Wilfley table concentrate, so removal was attempted by magnetic and electrostatic separation. A Stearns laboratory disc and belt magnetic machine was used which gives greatest selectivity but not the greatest field strength. The barytes must be dry before treatment, and free from dust.

After four passes the properties of the concentrate were as follows:-

| | | | |
|---|---|---|---|
| 1. Specific Gravity | | 4.3 | |
| 2. Colour | reflectance | 87 | per cent |
| 3. Analysis | $BaSO_4$ | 98.6 | per cent |
| | $Fe_2O_3$ | 0.18 | per cent |
| | $SiO_2$ | 0.78 | per cent |

Approximately half the iron was removed with an improvement from 81 to 87 per cent in the reflectance. An EEL photo-electric reflectometer was used to measure reflectance against a standard block of magnesium carbonate.

The jig concentrate did not contain an appreciable amount of free iron minerals and was too coarse for magnetic treatment. Crushing finer introduced a portion of dust which prevented useful cleaning. The colour was improved from 87 to 88 per cent reflectance.

A preliminary examination of the electrostatic method of separation was made as it was noticed that the iron stains were mostly associated with the particle surfaces. Neither the plate type nor the straight feeder type electrode was successful.

<u>Flotation</u>.

Tests were made on 500 grm. lots in a laboratory Fagergren flotation machine, with a variety of reagents which had been reported useful in practice. The best recovery was 68 per cent in a concentrate of 93.8 per cent $BaSO_4$.

The results obtained are given in detail for the sake of completing the record. The reagents are described first, and the concentrate assays are tabulated. The sizing after grinding was 79 per cent minus 200 mesh. The head assay was 86.2 per cent $BaSO_4$, 8.3 per cent $SiO_2$ and 0.64 per cent Fe.

Test 1. Reagents added to the Ball Mill.

1 lb. sodium silicate per ton.
1 lb. sodium oleate.
1 lb. Calgon.
1 lb. sodium hydroxide.

Added to flotation cell.

2 lb. sodium oleate.
2 lb. oleic acid.

Added to rougher concentrate.

0.5 lb. sodium silicate.
1 lb. sodium oleate.
1 lb. starch.

Test 2. Reagents added to the Ball Mill.

1 lb. Calgon per ton.

Added to flotation cell.

6 lb. R723
5 lb. Fuel oil.

Test 3. Reagents added to the Ball Mill.

1 lb. Calgon per ton.

Added to flotation cell.

0.5 lb. sodium lauryl sulphate.
0.2 lb. oleic acid.

Test 4. Reagents added to the flotation cell

0.2 lb. sodium lauryl sulphate per ton.
0.05 lb. oleic acid.

Test 5. Reagents added to the flotation cell.

1 lb. starch per ton.
1 lb. sodium silicate.
0.2 lb. sodium lauryl sulphate.
0.05 lb. oleic acid.

Added to rougher concentrate.

0.5 lb. starch.
0.5 lb. sodium silicate.

Test 6. Reagents added to the flotation cell.

2 lb. Twitchell 7200 per ton.
0.5 lb. oleic acid.

Added to rougher concentrate.

2.0 lb. Twitchell 7200.
0.5 lb. oleic acid.
1 lb. starch.

TABLE No. 8.

Flotation of Barytes.

| Test | Product | % Weight | Barytes % | Recovery % |
|---|---|---|---|---|
| 1 | Concentrate | 31.0 | 98.2 | 35.7 |
| | Middling | 23.2 | 93.3 | 25.4 |
| | Tailing | 45.8 | 72.6 | 38.9 |
| 2 | Concentrate | 49.6 | 96.3 | 57.6 |
| | Middling | 12.4 | 78.5 | 11.8 |
| | Tailing | 38.0 | 66.8 | 30.6 |
| 3 | Concentrate | 59.4 | 93.8 | 67.7 |
| | Middling | 13.0 | 66.1 | 10.5 |
| | Tailing | 27.6 | 65.2 | 21.8 |
| 4 | Concentrate | 24.9 | 97.3 | 29.3 |
| | Middling | 21.9 | 85.5 | 22.6 |
| | Tailing | 53.2 | 74.6 | 48.1 |
| 5 | Concentrate | 24.8 | 97.9 | 28.6 |
| | Middling | 13.6 | 87.5 | 14.0 |
| | Tailing | 61.6 | 78.8 | 57.4 |
| 6 | Concentrate | 39.6 | 95.3 | 45.6 |
| | Middling | 14.4 | 83.5 | 14.6 |
| | Tailing | 46.0 | 71.6 | 39.8 |

COST OF TREATMENT.

The estimates of treatment costs are divided into two parts. First the treatment of the dumps is considered and second the treatment of run-of-mine ore. These must be regarded as preliminary estimates only.

The tonnage available from dumps at present, added to current accumulation, would provide 10,000 tons a year for two years. The installation of a mobile concentrating plant, and its return, plus the provision of temporary housing, water supply etc., is estimated to cost £20,000 over and above actual cost of the plant. This is equal to £1 per ton of ore.

Additional cost would also occur due to higher power charges, water charges and subsidy of accommodation for the labour. This is estimated to cost a further £2/10/0 per ton of ore.

Transport of concentrate to Adelaide would cost £7 per ton, and transport of ore would cost £9/10/0 per ton of final concentrate. The difference of £2/10/0 is less than extra power, water etc., charges at the mine, added to the cost of moving in a plant for a short period.

The first estimate envisages that plants suitable for separation and grinding are available in Adelaide. The only capital required is that sufficient to finance transport and treatment charges for approximately six months.

Operating costs would be as follows for each ton of ore:-

| | |
|---|---|
| Contractor loading and carting dump to Hawker | £ 2/10/0 |
| Rail freight to Adelaide | 4/10/0 |
| Transport to treatment plants | 10/0 |
| Hire of treatment plant | 1/ 0/0 |
| Operation of treatment plant | 2/10/0 |
| | £11/ 0/0 |

From each 100 tons shipped only 75 tons of concentrate would be obtained, so that to this stage, each ton of concentrate would cost £14/13/4. The concentrate produced in this plant would then be reduced to a powder in a standard milling plant for £5 per ton, making total operating costs of £19/13/4. The total working capital necessary would be approximately £100,000 and interest on this would add 10 shillings per ton of product. Selling price would therefore be at least £21 per ton in Adelaide. The experiments indicate that this would be suitable for second grade paint, or oil drilling muds.

The economic picture is much more satisfactory if long term mining is considered. In this case capital expenditure at the mine is justified. A community of somewhat larger size must be established for mining anyway, and both community and erection charges can be written off over ten years.

The final grinding can also be carried out at the mine, without incurring separate management fees and profits. Advantage can be taken of freight saving by elimination of waste at the mine, and further savings made in sales in Western Australia or New South Wales by shipment direct through Kalgoorlie or Broken Hill respectively.

The capital requirements for the milling of 10,000 tons per year are given below. This does not include capital for mining or establishment of the community. The cost of ore taken into the mill is estimated to include the mining portion of capital expenses.

Capital for Mill at Oraparinna.

Capacity 10,000 tons per year.

| | |
|---|---|
| Crushing Section | £15,000 |
| Heavy Media Section | 25,000 |
| Jig Section | 15,000 |
| Storage | 15,000 |
| Grinding Units | 25,000 |
| Power House | 30,000 |
| Water Supply | 25,000 |
| | £150,000 |

Treatment costs/ton product in the Mill.

| | |
|---|---|
| Crushing | £ 10/0 |
| Separation | 2/10/0 |
| Grinding | 3/ 0/0 |
| | £6/ 0/0 |

Final Cost/ton of Product in Adelaide.

| | |
|---|---|
| Mining (including capital redemption) - - - - - - | £ 2/10/0 |
| Community - - - - - - - - - - - - - - - - - - - - - | 1/ 0/0 |
| Milling - - - - - - - - - - - - - - - - - - - - - - | 6/ 0/0 |
| Transport - - - - - - - - - - - - - - - - - - - - - | 7/ 0/0 |
| Interest on Capital (including Working Capital) - | 2/ 0/0 |
| Depreciation of Mill Capital - - - - - - - - - - - | 1/10/0 |
| | £20/ 0/0 |

There is no experimental evidence to determine the standard of this barytes. The ore would contain the best white barytes at present hand sorted from the dumps. It has been observed that the encrusting discoloration on second grade barytes is soft when the material is first mined, and would therefore be largely washed off if immediate treatment were given. It can be reasonably assumed that half the product would be first grade paint barytes, and the remainder suitable either for second grade paint or oil drilling muds.

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