John Convey, Director
Mines Branch



CANADA

DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA



MINES BRANCH INVESTIGATION REPORT IR 61-47



# BENEFICIATION OF A MANGANESE ORE RECEIVED FROM J. PAULOSKI, MASSETT, B.C.

by

R. S. KINASEVICH

MINERAL PROCESSING DIVISION



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# BENEFICIATION OF A MANGANESE ORE RECEIVED FROM J. PAULOSKI, MASSETT, B.C.

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R. S. Kinasevich*

## SUMMARY OF RESULTS

Manganese ore, assaying 23.47% Mn, was sized for concentration by gravity and flotation methods.

Heavy media concentration of -8+28 M feed, assaying an average of 25.35% Mn, gave a product of 52.8% Mn at a recovery of 79.8% (Table 8).

Tabling -8+48 M feed, assaying 24.00% Mn, gave a concentrate of 40.25% Mn at a recovery of 78.0% (Tables1 and 2).

Jigging the same type of feed yielded a concentrate assaying 50.6% Mn at a recovery of 63.0% (Table 3).

The best of three flotation tests performed on the -28 mesh fraction from the heavy media feed gave an overall product assaying 50.4% Mn at a recovery of 37.2% (Table 10). Combining these flotation results with those from the heavy media separation gave a product of 52.2% Mn with an overall recovery of 62.6% (Table 11).

Combining the gravity and flotation concentrates produced a total of about 40 lb of 50% Mn concentrate which was sent to the Extraction Metallurgy Division for pelletizing tests.

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* Scientific Officer, Mineral Processing Division, Mines Branch, Department of Mines and Technical Surveys, Ottawa, Canada.

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# INTRODUCTION

## Shipment

On May 15, 1960, the Mineral Processing Division's laboratories received 200 lb of manganese ore, which had been sent to the Extraction Metallurgy Division for pelletizing tests.

## Location of Property

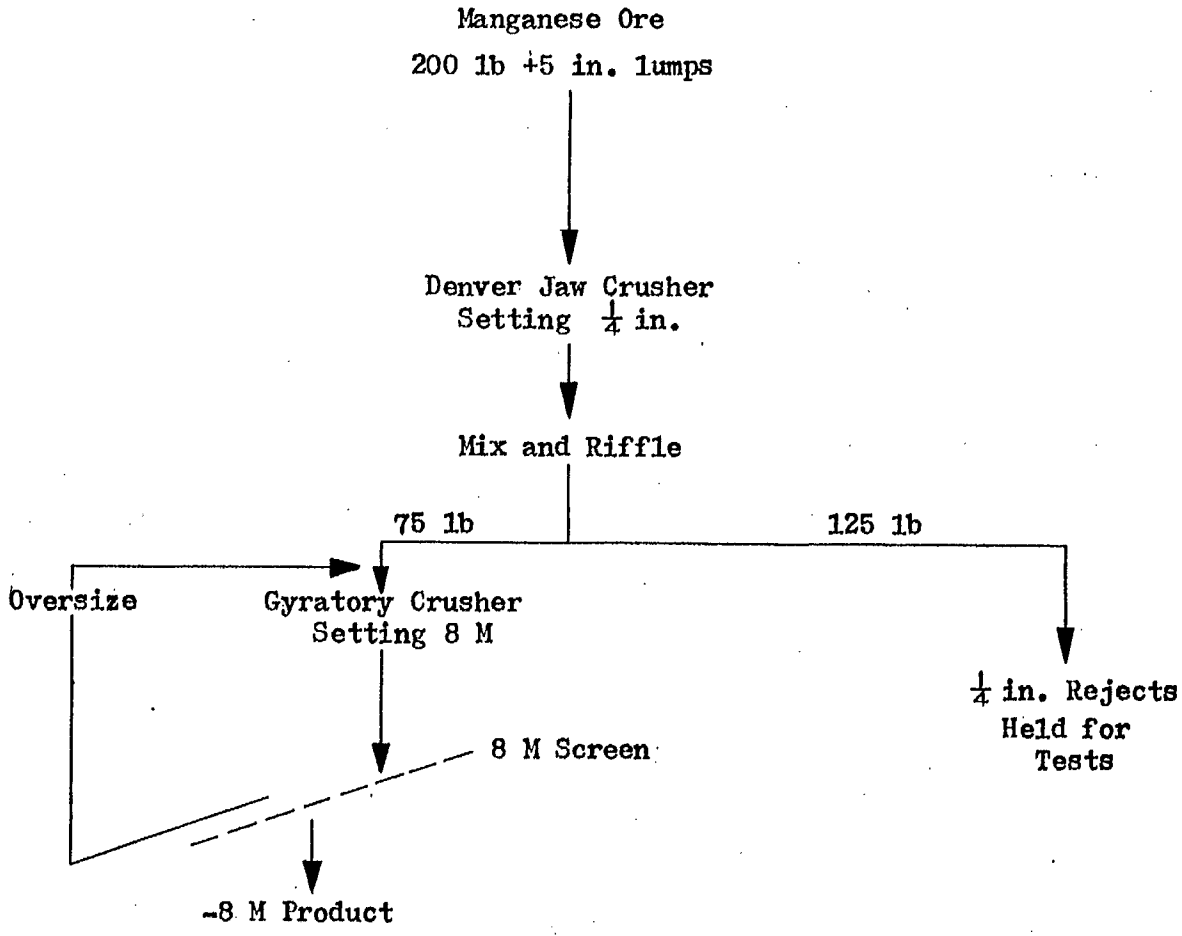
The shipment, sent by Mr. Joseph Pauloski, originated from a deposit at Klashwun Point, about 25 miles west of Massett on Graham Island, B.C.

## Purpose of Investigation

Initially, Mr. Pauloski corresponded with the Mines Branch and requested pelletizing tests on manganese concentrates. The correspondence and the problem was then forwarded to the Extraction Metallurgy Division. Since a shipment of ore was sent rather than concentrate, the problem became primarily one of ore beneficiation to obtain sufficient concentrate for later pelletizing. With his correspondence, Mr. Pauloski included a report of investigations done on a sample of the ore by Hudson Bay Mining and Smelting Co. Ltd., Flin Flon, Man.

## Sampling and Analysis

The following flowsheet illustrates the sampling procedure used on the shipment:




From the -8 M product, a 5 lb representative sample was taken and sent for chemical analysis and a semi-quantitative spectrographic analysis.

The elements detected by spectrographic means are listed below in decreasing order of abundance:

I. Mn, principal constituent

II. Si, Al, Fe (1.5%), Ca, Na.

III. Mg, Ti, Ba, Cu (0.04%), Ni.

IV. Zr, V, B, Mo, Cr

The chemical analysis of the head sample determined the following constituents:

| Constituent | Per Cent |
|---|---|
| Mn | 23.47 |
| Fe | 1.73 |
| P | 0.049 |
| $SiO_2$ | 34.38 |
| Insol | 46.34 |
| CaO | 1.67 |
| $Al_2O_3$ | 8.57 |

## MINERALOGICAL EXAMINATION*

Pyrolusite ($MnO_2$) and manganite (MnO(OH)) are the main manganese ore minerals present. They occur largely in botryoidal forms, with manganite forming the nuclei of the botryoidal sphere-like bodies, and pyrolusite, the concentric shells around them (Figs 1 and 2). They also form a complex mixture in the intervening areas between the sphere-like bodies. The pyrolusite is composed of tiny interlocking crystals that are about 0.01 mm to 0.06 mm in diameter, and the manganite is composed of interlocking crystals about one-tenth that size (Figs 1 and 2). The mode of occurrence of these two minerals suggests that pyrolusite is likely an alteration product of the manganite.

The gangue occurs as irregular shaped bodies and vug fillings in the manganese ore. The irregular shaped bodies are composed of a few grains of feldspar, quartz, and apatite embedded in a fine-grained opaque material.

* Internal Report, Mineral Sciences Division, MS-60-80, Sept. 6, 1960, by W. Petruk.

Partially filled vugs containing poorly cemented, crypto-crystalline dolomite and feldspar are present in the manganese ore. The dolomite reacts with HCl.

A few subhedral crystals of hausmannite ($MnMn_2O_4$) and jacobsite ($MnFe_2O_4$) are present in the gangue (Fig 3). They are about 0.1 mm in diameter.

The powdered manganese ore is slightly magnetic, and the hand-picked magnetic particles were identified as magnetite by X-ray diffraction. No magnetite was recognized in the polished sections so its textural relationship is unknown.

Summary

The manganese ore is composed of massive to botryoidal manganite and pyrolusite in a gangue of feldspar, dolomite, quartz, apatite and a fine-grained opaque material which is either a very fine-grained mixture of quartz and feldspar, or an alteration product of feldspar.

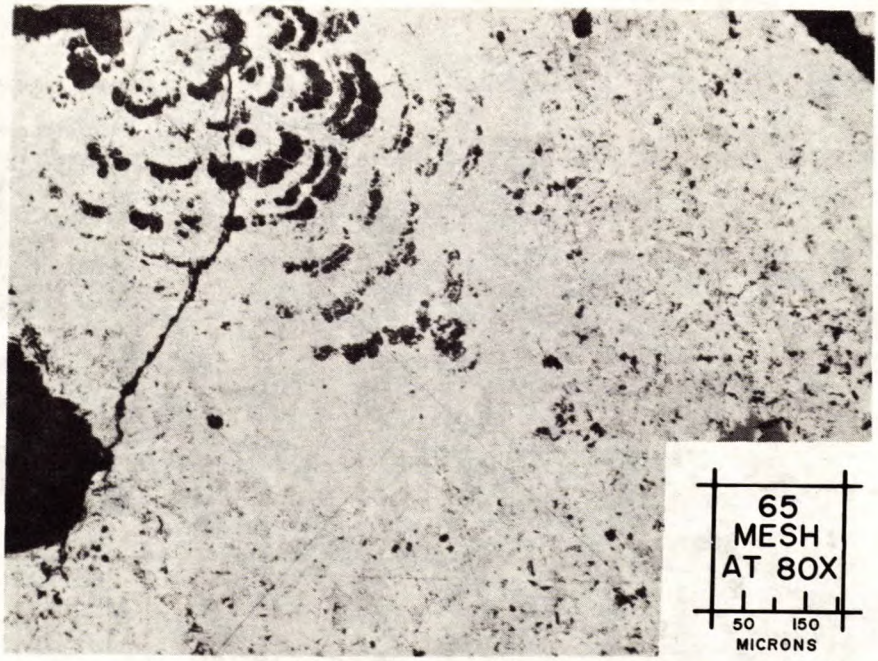



Figure 1. - Botryoidal manganite (upper left corner) surrounded by a coarser grained pyrolusite. Polished section in plain polarized light.

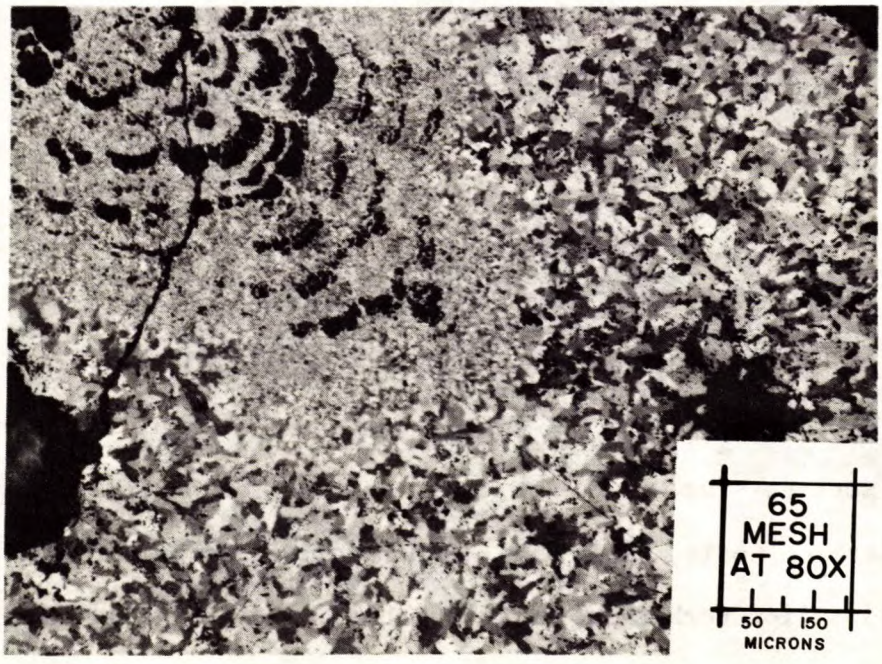



Figure 2. - Same section as Fig 1, with nicols crossed, showing grain size of manganite and pyrolusite.

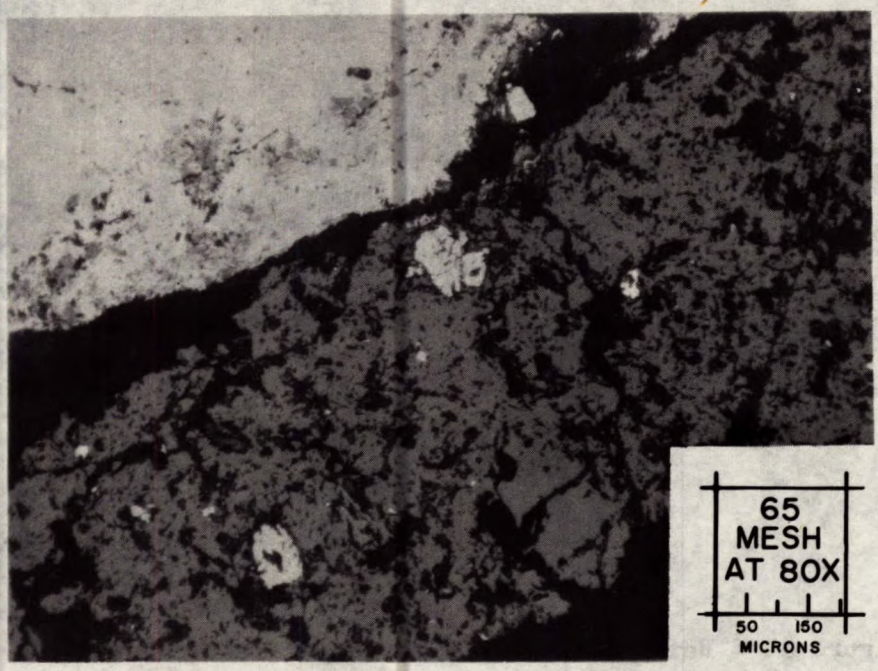



Figure 3. - Subhedral jacobsite crystals (white) in gangue (grey). Cloudy white mineral in the upper left corner is a mixture of manganite and pyrolusite. Polished section in plain polarized light.

## DETAILS OF INVESTIGATION

### Preliminary Gravity Concentration Tests

(a) Table Tests:

Initially, 22 lb of the -8 M head sample was screened on a 48 M screen. The -8+48 M portion was used for preliminary gravity concentration tests, while the -48 M fraction was held for possible later flotation work. The Wilfley table, used initially on a 2000 g sample, gave poor results.

A 4000 g sample was then treated on the Deister table

which yielded a rougher concentrate. This was cleaned on the Wilfley table. The cleaner middling and tailing were combined with the corresponding Deister table products.

(b) Jig Tests:

Jigging was done on another 2000 g sample to compare data with that from the table tests.

Tables 1 to 3 contain the results from the above tests.

TABLE 1

Size Analysis of Head Sample

| Product | Weight % | Assay, % | | | Distn, % | |
|---|---|---|---|---|---|---|
| | | Mn | Fe | Insol | Mn | Insol |
| -8+48 M | 74.1 | 24.00 | 1.57 | 46.04 | 74.5 | 73.9 |
| -48 M | 25.9 | 23.47 | 1.63 | 46.60 | 25.5 | 26.1 |
| Calcd Head | 100.0 | 23.89 | 1.60 | 46.19 | 100.0 | 100.0 |

TABLE 2

Results of Deister Table Concentration Tests

| Product | Weight % | Assay, % | | | Distn, % | |
|---|---|---|---|---|---|---|
| | | Mn | Fe | Insol | Mn | Insol |
| Table conc | 39.7 | 40.25 | 0.84 | 19.4 | 78.0 | 16.0 |
| " midd | 48.9 | 8.60 | 1.89 | 64.20 | 20.5 | 65.1 |
| " tail | 11.4 | 2.69 | 2.30 | 79.80 | 1.5 | 18.9 |
| -8+48 M* feed | 100.0 | 20.52 | 1.52 | 48.21 | 100.0 | 100.0 |

*calculated

Additional Assay - Table conc 3.50% $Al_2O_3$, 0.18% P.

TABLE 3

Results of Jig Tests

| Product | Weight % | Assay, % | | | Distn, % | |
|---|---|---|---|---|---|---|
| | | Mn | Fe | Insol | Mn | Insol |
| Jig conc | 27.1 | 50.60 | 0.76 | 3.88 | 63.0 | 2.3 |
| " tail | 64.5 | 9.67 | 1.94 | 61.84 | 28.7 | 89.0 |
| " bed | 8.4 | 21.68 | 1.62 | 46.28 | 8.3 | 8.7 |
| -8+48 M* feed | 100.0 | 21.77 | 1.59 | 44.84 | 100.0 | 100.0 |

*calculated

Additional Assay - Jig conc 0.67% $Al_2O_3$, 0.15% P.

(c) Heavy-Liquid Tests on -8+28 M Feed:

One of the tests done at the laboratories of the Hudson Bay Mining and Smelting Co. Ltd. was a small scale heavy-liquid separation using pure stannic bromide and a stannic bromide-carbon tetrachloride mixture. To determine if the sample sent to the Mines Branch was similar to that tested above, a heavy-liquid test, using tetrabromoethane, with a specific gravity of 2.965, was used for the test. The results of this work are shown in Table 4, and are similar to the results obtained by the H.B.M. & S. Co. tests.

TABLE 4

Results of Heavy-Liquid Test

| Product | Total Wt % | Mn, % | Total Mn Distn in Conc, % |
|---|---|---|---|
| -8+14 M Sink | 16.1 | 50.08 | 33.1 |
| Float | 17.0 | 1.99 | |
| Calcd Feed | 33.1 | 25.44 | |
| -14+28 M Sink | 12.4 | 49.63 | 25.2 |
| Float | 12.5 | 2.77 | |
| Calcd Feed | 24.9 | 26.1 | |
| -28 M, not treated | 42.0 | | |
| Total | 100.0 | | 58.3 |

Flotation Test No. 1

Flotation was done on 1000 g of the -28 M fraction, from the test above, which was conditioned at 33% solids with reagents. Subsequent to the rougher float, the rougher tailing was ground 10 min and floated separately. The flotation technique is described below:

| Operation | Reagents, lb/ton | | Time, min | pH |
|---|---|---|---|---|
| Conditioning | Sodium silicate | - 1.65 | | |
| | Oleic acid | - 0.56 | 5 | 7.5 |
| Rougher flotation | Pine oil | - 0.045 | 5½ | |
| Cleaner flotation | Sodium silicate | - 0.35 | | 7.8 |
| | Pine oil | - 0.03 | 4 | |

**Flotation of Rougher Tailing**

| Operation | Reagents, lb/ton | | Time, min | pH |
|---|---|---|---|---|
| Conditioning | Sodium silicate | - 0.5 | | |
| | Oleic acid | - 0.3 | 2 | -- |
| Rougher flotation | Pine oil | - 0.054 | $3\frac{1}{2}$ | |
| Cleaner flotation | Sodium silicate | - 0.25 | | |
| | Pine oil | - 0.04 | 2 | 8.7 |

The results of this flotation test are given in Table 5.

TABLE 5

Results from Flotation Test No. 1

| Product | Wt, % | Mn, % | Mn Units | Mn Distn, % |
|---|---|---|---|---|
| Cleaner conc | 29.2 | 36.00 | 1050 | 46.5 |
| Regrind cl conc | 25.3 | 26.14 | 662 | 29.3 |
| Cleaner tail | 9.2 | 22.84 | 211 | 9.4 |
| Regrind cl tail | 12.3 | 13.11 | 161 | 7.1 |
| " rougher tail | 24.0 | 7.26 | 174 | 7.7 |
| Calcd Feed | 100.0 | 22.58 | 2258 | 100.0 |

The flotation concentrates, when combined, averaged 31.4% Mn at a recovery of 75.8%, based on the flotation feed.

These flotation results are combined with those from the heavy-liquid test to give the overall results shown in Table 6.

TABLE 6

Combined Heavy-liquid and
No. 1 Flotation Test Results

| Product | Total Wt % | Mn, % | Overall Mn distribution in conc. % |
|---|---|---|---|
| -8+14 Sink | 16.1 | 50.08 | 33.1 |
| Float | 17.0 | 1.99 | |
| -14+28 Sink | 12.4 | 49.63 | 25.2 |
| Float | 12.5 | 2.77 | |
| -28 M flot conc | 22.9 | 31.4 | 29.5 |
| " " tail | 19.1 | 12.0 | |
| Calcd Head | 100.0 | 24.38 | 87.8 |

The overall Mn grade of the combined flotation concentrates and sink products was 41.7%. In the heavy-liquid test only, 95.4% of the manganese was recovered in the concentrate which had an average grade of 49.9% Mn.

The heavy-liquid and flotation tests showed that the weight distribution of the various fractions differed from those obtained from the sample investigated at the Hudson Bay Mining and Smelting laboratories. The assays of the products, however, were quite similar in both cases.

Heavy Media (Sink-Float) Test

Although the heavy-liquid method gave very encouraging results, this method is not a practical one for concentrating ore. Therefore, a practical heavy media method which used a suspension

of magnetite in water was carried out in a cyclone separator to obtain enough concentrate for the pelletizing tests requested. In this test, 153½ lb of raw material was screened into the following fractions:

| Mesh | Wt, lb | Wt, % |
|---|---|---|
| -8+14 | 57 | 37.2 |
| -14+28 | 37 | 24.0 |
| -28 | 59½ | 38.8 |
| Total | 153½ | 100.0 |

The -8+14 and -14+28 M fractions were separately treated in the cyclone. A total of 395 lb of dry magnetite with a specific gravity of 4.35 was pulped to a suspension having a specific gravity of 2.78. After the first fraction had been treated, the products were washed free of media which was then densified and re-used to treat the next fraction. The flow through the cyclone was regulated so that the specific gravities of the media in the cyclone overflow and underflow were 2.70 and 3.00, respectively.

The data from the heavy media test are in Table 7.

TABLE 7

Data from Heavy Media Tests

| Product | Weight | | Mn, % | Mn Units |
|---|---|---|---|---|
| | lb | % | | |
| -8+14 M Sink | 15½ | 16.5 | 53.10 | 876 |
| Float | 28½ | 30.3 | 7.79 | 236 |
| Rejects | 13 | 13.8 | 38.04(?) | 525 |
| -14+28 M Sink | 14 | 14.9 | 52.30 | 779 |
| Float | 18½ | 19.7 | 9.20 | 181 |
| Rejects | 4½ | 4.8 | 27.38 | 132 |
| Calcd Feed | 94 | 100.0 | 27.29 | 2729 |

(?) possibly in error, see below

The feed rejects consisted of material which had not been concentrated during the running period of the cyclone. No attempt was made to repass this material through the cyclone since the amount remaining was insufficient for a reliable test. The heavy media rejects were distributed between the sink and float products according to their weight proportion in each of the two fractions. The rejects can also be proportioned on the basis of the Mn distribution, but in using this method the calculations showed that the assay of the -8+14 M rejects was incorrect. The results of the heavy media test, after accounting for the rejects, are shown in Table 8.

TABLE 8

Results of Heavy Media Test

| Product | Wt, % | Mn, % | Mn, Units | Mn Distn in Conc, % |
|---|---|---|---|---|
| -8+14 M Sink | 21.4 | 53.10 | 1137 | 44.9 |
| Float | 39.3 | 7.79 | 306 | |
| -14+28 M Sink | 16.9 | 52.30 | 886 | 34.9 |
| Float | 22.4 | 9.20 | 206 | |
| Calcd Feed | 100.0 | 25.35 | 2535 | 79.8 |

Average grade of concentrates (sink) = 52.8% Mn

| Additional assays: | $Al_2O_3$ | P |
|---|---|---|
| -8+14 Sink | 1.32% | 0.16% |
| -14+28 Sink | 1.67 | 0.22 |

Flotation Test No. 2

A 2000 g sample of the -28 M fraction was ground with 1.0 lb/ton of sodium hydroxide to 68.5% -200 M. The flotation reagent used was a mixture of equal volumes of distilled oleic acid and Emulsol X-1. The rougher concentrate was cleaned once. A screen analysis of the -28 M flotation feed is given in Table 9, while Table 10 contains the flotation results.

TABLE 9

Screen Analysis of Feed for Flotation Tests

| Mesh | Wt, % | Mn, % | Mn Distn, % |
|---|---|---|---|
| +35 | 13.15 | 27.88 | 14.4 |
| +48 | 16.15 | 26.48 | 16.8 |
| +65 | 14.05 | 25.02 | 13.8 |
| +100 | 11.50 | 24.46 | 11.0 |
| +150 | 8.85 | 25.00 | 8.7 |
| +200 | 7.00 | 27.45 | 7.6 |
| -200 | 29.30 | 24.09 | 27.7 |
| Calcd Feed | 100.0 | 25.46 | 100.0 |

| Operation | Reagents - lb/ton | Time, min | pH |
|---|---|---|---|
| Grinding | Sodium hydroxide-1.04 | 15 | 9.6 |
| Conditioning | Collector mixture - 1.0 | 4 | |
| Rougher float | - | 12 | 9.1 |
| Conditioning | Sodium silicate - 1.0<br>10% Sulphuric acid - 0.94 | 2 | |
| Cleaner float | Collector mixture - 0.25 | 6 | 5.8 |

TABLE 10

Results from Flotation Test No. 2

| Product | Wt, % | Mn, % | Mn Units | Mn Distn % |
|---|---|---|---|---|
| Cleaner conc | 15.62 | 54.42 | 850 | 31.6 |
| " tail | 4.21 | 35.57 | 150 | 5.6 |
| Rougher tail | 80.17 | 21.10 | 1692 | 62.8 |
| Calcd Feed | 100.00 | 26.92 | 2692 | 100.0 |

Average grade of combined cleaner conc and tailing = 50.4%.

The results of the heavy media tests in Table 8, and No. 2 flotation test, Table 10, are combined in Table 11 to show the overall recovery.

TABLE 11

Combined Heavy Media and No. 2 Flotation Test Results

| Product | Wt, % | Mn, % | Mn Units | Mn Distn, % |
|---|---|---|---|---|
| -8+14 Sink | 13.1 | 53.10 | 696 | 26.8 |
| Float | 24.1 | 7.79 | 188 | |
| -14+28 Sink | 10.3 | 52.30 | 539 | 20.8 |
| Float | 13.7 | 9.20 | 126 | |
| Cleaner conc | 6.1 | 54.42 | 330 | 12.7 |
| " tail | 1.6 | 35.57 | 58 | 2.3 |
| Rougher tail | 31.1 | 21.10 | 656 | |
| Calcd Head | 100.0 | 25.92 | 2593 | 62.6 |

The concentrates and cleaner tailing, when combined, yield an average assay of 52.2% Mn.

Flotation Test No. 3

A different reagent combination was tried for this third flotation test. Since the flotation products from the other tests were difficult to filter, due to the presence of slimes, an attempt was made to reduce the latter by screening out and grinding the coarser fractions.

2000 g was screened on a 65 M screen. The oversize, weighing 1038 g, was ground 15 min. The ground sample and the undersize were mixed and sampled for a screen test, which showed that 46.7% -200 M was present.

The collector reagent was a water-fuel oil emulsion, stabilized with Petronate. A small amount of distilled oleic acid was also added.

Table 12 lists the proportion of emulsion constituents by weight and volume. The reagent consumption will be expressed in terms of the entire emulsion used. Table 12 can be used to express reagent consumption on the basis of the ingredients.

TABLE 12

Relative Proportions of Emulsion Ingredients

| Ingredient | Density, g/cm$^3$ (*) | Volume ml | Weight g | % by Volume | % by Weight |
|---|---|---|---|---|---|
| Distilled water | 0.998 | 90.0 | 89.82 | 88.53 | 90.19 |
| Fuel oil | 0.82 | 10.0 | 8.20 | 9.84 | 8.23 |
| Distilled Oleic Acid | 0.895 | 1.25 | 1.12 | 1.23 | 1.13 |
| Petronate | 1.11 | 0.41 | .45 | .40 | .45 |
| Total | 0.98 | 101.66 | 99.59 | 100.00 | 100.00 |

$$\text{Overall density of emulsion} = \frac{W_1 + W_2 + W_3 + W_4}{V_1 + V_2 + V_3 + V_4} \text{ g/ml}$$

$$= \frac{99.59}{101.66} = 0.98 \text{ g/ml}$$

For a 2000 g sample, 10 ml of emulsion = 9.8 lb/t of ore.

Following is the summary of the flotation procedure as well as the results in Table 13.

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(*) Handbook of Chemistry and Physics - 38th Edition - Chemical Rubber Publishing Co. (1956-1957).

| Operation | Reagents, lb/ton | Time, min | pH |
|---|---|---|---|
| Condition at 37% solids | Emulsion - 4.9 | 5 | 8.2 |
| Float | - | 2 | - |
| Conditioning | " - 9.8 | 4 | |
| Float | - | 5 | - |
| Conditioning | - 4.9 | 2 | |
| Float | - | 5 | 7.9 |
| Cleaner float | Sodium silicate - 1.0<br>10% Sulphuric acid - 0.94 | | |
| Conditioning | | 3 | |
| Float | - | 2 | 4.9 |
| Conditioning | Pine oil - 0.054 | 5 | |
| | Emulsion -14.7 | 2 | |
| Float | - - | 5 | - |

TABLE 13

Results of Flotation Test No. 3

| Product | Wt, % | Mn, % | Mn Unit | Mn Distn, % |
|---|---|---|---|---|
| Cleaner conc | 3.9 | 39.88 | 155.5 | 7.1 |
| Cleaner tail | 10.8 | 35.09 | 379.5 | 17.3 |
| Rougher tail | 85.3 | 19.48 | 1660.0 | 75.6 |
| Calcd Feed | 100.0 | 21.95 | 2195.0 | 100.0 |

Combining the cleaner concentrate and cleaner tailing gives a product assaying 36.4% Mn at a recovery of 24.4%.

The gravity and flotation concentrates were combined, making a total of about 40 lb of 50% Mn concentrate, which was sent to the Extraction Metallurgy Division for pelletizing tests. These results will be published in a separate report.

## CONCLUSIONS

The coarse fractions (-8+28 and -8+48 M) of the manganese ore are quite amenable to gravity concentration. Heavy media concentration produced the best metallurgical results in terms of grade and recovery, and appears more attractive than jigging or tabling due to its greater throughput capacity. However, the feed used for the jig and table tests was -8+48 M, which represented 74.1% of the total feed. The -8+28 M feed for the heavy media test represented 61.2% of the total feed, and assayed slightly higher in Mn. These factors may in part account for the superior results obtained by this method.

The present accepted specification for lump Mn concentrate is the following:

| ASSAY, % | | | |
|---|---|---|---|
| Mn | Fe max | Si + Al max | P max |
| 46-48 | 10 | 13 | 0.15 |

Although iron and insol content were not determined for the heavy media concentrates, their values would probably correspond quite closely to those obtained for the jig concentrate, and would

satisfy the above specifications. Phosphorus and alumina were also within the required limits. The spectrographic analysis of the head sample detected 0.04% Cu, and there is no reason to suspect that a significant amount exists in the concentrates.

The best Mn grade and recovery obtained from the flotation of the -28 M fraction resulted when the conditions of Test 2 were used. Tests 1 and 2 indicate that to obtain a suitable Mn grade, recovery would have to be sacrificed. Use of the emulsion as a collector in flotation Test 3 was unsuccessful in producing satisfactory concentrate grade and recovery of manganese.

Although the concentrates obtained satisfied the accepted chemical composition, agglomeration may be necessary in order to be competitive within the existing markets. Further information concerning possible markets for fine concentrates of this type may be obtained from the Mineral Resources Division of the Department of Mines and Technical Surveys, 588 Booth Street, Ottawa.

RSK:EBM