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# A Mineral Dressing Study of the Benefication of Philipsburg Manganese Carbonate Ore

Frank M. Antonioli

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A MINERAL DRESSING STUDY OF THE BENEFICIATION  
OF PHILIPSBURG MANGANESE CARBONATE ORE

by

FRANK M. ANTONIOLI

A Thesis

Submitted to the Department of Mineral Dressing  
in Partial Fulfillment of the  
Requirements for the Degree of  
Bachelor of Science in Metallurgical Engineering  
Mineral Dressing Option

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Butte, Montana

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## INTRODUCTION

The beneficiation of rhodochrosite ore to the extent that it is suitable for nodulizing and subsequent use in the steel industry is intriguing. Such ores from certain mines is relatively easy to beneficiate, but the rhodochrosite ore from several deposits of the Philipsburg district has never been successfully up-graded by any of the several known beneficiating methods. Many investigators have attempted to develop a process which would produce a satisfactory raw material for nodulizing.

In this investigation a study was made to at least outline some of the factors which have prevented the utilization of the Philipsburg ore. In this study the previously employed methods considered for up-grading were investigated to some extent. In addition the affect of temperature on the selectivity of certain flotation reagents for the carbonate minerals was given a preliminary survey. Temperature coupled with careful selection of reagents holds some interesting possibilities and might, if thoroughly studied, bring about a satisfactory method of beneficiating the Philipsburg rhodochrosite ores.

## HISTORY

### Industrial Importance of Manganese

Manganese, one of the more abundant elements in nat-

ure, is of strategic importance to the industrial security and development of the United States. In 1949, the consumption of manganese ore in the United States was in excess of 1,330,000 short tons of which less than 10 per cent was produced from domestic sources; stocks of manganese ore at plants (including bonded warehouses) are sufficient to supply industry for about six months<sup>1</sup>. If the United States is going to become reasonably self-sufficient in respect to manganese, the importance of making available to industry of domestic manganese ores cannot be over emphasized. Large domestic deposits of low grade manganese ore offer a potential reserve of manganese, if new and improved beneficiating methods can be developed.

Manganese was first discovered in 1774; however, it wasn't until 1856, when Mushet discovered its value in the steelmaking industry that it assumed much importance in industry. Without manganese, as a deoxidizing agent in the manufacture of steel, the steelmaking industry would be very seriously handicapped<sup>2</sup>. Montana, in 1949, supplied 97 per cent of all the domestic shipments, mainly from production of rhodochrosite ores from the Butte district by the Anaconda Copper Mining Company<sup>1</sup>.

#### History of the Philipsburg District

Manganese in the Philipsburg district was first considered to be of commercial importance in 1900; however,

no large production was attained until 1916, and from 1916-1939, 477,000 long tons of over 35 per cent manganese was produced.<sup>3</sup> The manganese deposits in this district consist of replacement deposits in limestone and occur as north-south bedding planes having a dip of 45 degrees and lay adjacent to east-west fissure veins. The ore is friable and easily crushed. Substantial reserves are known to exist in this district, and it is with this fact in mind that a study has been undertaken to selectively treat the type of ore which exists in the Philipsburg district.

The major portion of the manganese production from this district has been the dioxide ore, commonly referred to as pyrolusite, and was used for battery purposes.<sup>4</sup> It wasn't until 1943 that rhodochrosite ore was mined, but during the second World War over 40,000 tons were mined and later processed by the Domestic Manganese and Development Company of Butte, Montana.

#### Previous Beneficiation Studies

Concentration and nodulizing of rhodochrosite ore from the Philipsburg district by the Domestic Manganese and Development Company was successful in meeting specified requirements; but, recovery of manganese from the ore was not entirely satisfactory. It is evident from a study of this problem that manganese can not be produced from the rhodochrosite ores in the Philipsburg district by known processes and achieve a high degree of recovery.

Considerable work has been conducted on the rhodochrosite ore from this district, by the Bureau of Mines and by George W. Griswold, engineer for the Domestic Manganese and Development Co. The method<sup>5</sup> as developed by the Bureau of Mines entailed stage grinding to -48 mesh. Flotation of the calcite and dolomite at a pH of 9 to 10, using sodium silicate as slime dispersant and silica depressant, and oleic acid as collector and frother. Pulp density was maintained at 20-25 per cent solids, and a pulp temperature of 18 degrees centigrade. In the subsequent manganese flotation step, the pH was lowered to 6.5 and additional oleic acid was added. The manganese flotation concentrate assaying 32 per cent manganese was nodulized. On an ore having a manganese content of 26.2 per cent, the Bureau of Mines made a 48 per cent manganese product with a recovery of 70.6 per cent.

Treatment method<sup>6</sup> employed by the Domestic Manganese and Development Company as developed by Griswold, consisted of grinding to approximately -200 mesh. From the ground pulp a bulk sulfide float was made after conditioning with  $\text{CuSO}_4$ , amyl xanthate, and cresylic acid. The sulfide tails made up the feed for the manganese circuit in which a manganese float was made after conditioning with soda ash, sodium silicate, oleic acid, and pine oil; the pH was 9.3 and the manganese float was cleaned twice.



## Manganese Specifications

There are both chemical and size specifications<sup>5</sup> governing metallurgical manganese products. As of May, 1943, the schedules were simplified by eliminating all ore classifications and placing the specifications on a base of 48 per cent manganese, 6 per cent iron, and 11 per cent silica plus alumina. Premiums and penalties are levied relative to manganese grade and iron and insoluble contents. The size limit specifications must be met by all ores and concentrates. This specification requires all material be minus 12 inches and at least 75 per cent plus 20 mesh. Thus, all material produced by tabling, flotation, and magnetic separation must be nodulized to meet the size requirements; also nodulizing is a necessary step in processing carbonate ores in order to meet manganese grade requirement by driving off the chemically combined  $\text{CO}_2$ , because the pure mineral rhodochrosite ( $\text{MnCO}_3$ ) contains only 47.8 per cent manganese.

## EXPERIMENTAL PROCEDURE

### Preparation of Sample

A two hundred and fifty pound sample of ore was received from the Trout Mine near Philipsburg. Several pieces of ore were chosen for mineralogical studies and

the remainder was used for the several concentration investigations made. The sample was first crushed in a laboratory jaw crusher to  $-1\frac{1}{4}$  inches, after which one-quarter of the sample was cut for the sink and float investigation. The remaining three-quarters of the sample was stage crushed to all -10 mesh for flotation testing.

#### Assay of the Ore

A representative portion of the ore was cut, from the sample of the -10 mesh material for flotation testing, by standard sampling procedure, and the assay\*of this representative portion is given in the following table (Table I).

TABLE I. Assay of the Ore

Mn	21.1	per cent
Insol	19.6	per cent
CaO	9.7	per cent
MgO	5.1	per cent
Fe	1.3	per cent
Zn	1.7**	per cent
Pb	0.2	per cent
Au	0.10	oz/ton
Ag	2.8	oz/ton

\*C. J. Bartzen, Analyst, Montana Bureau of Mines and Geology, Montana School of Mines, Butte, Montana

\*\*Zinc analysis made by assay office, Anaconda Copper Mining Co.

The analysis of the ore indicates that the ore is not very siliceous, and that the sulfides are present in such quantities as to require their removal from any manganese product. The CaO is present partially as free calcite as observed in the hand specimens, but the major portion of the CaO must be combined with the MgO as dolomite because no discernible magnesite was present.

The ore which this study has been conducted on was purposely taken, not to be a representative sample of the ore of the district\*, but ore which would contain a larger proportion of CaO and MgO. The reason for this selection was that the study as undertaken, was primarily to investigate the differential separation of the rhodochrosite minerals from the calcite and dolomite gangue minerals.

#### Mineralogical Studies

Megascope Examination. A visual analysis of the ore indicated that the rhodochrosite minerals were present in a gangue of quartz and calcite. Also present were sulfides of iron, lead, and zinc. Tests with concentrated HCl indicated the presence of dolomite in the hand specimens.

Microscopic Study. Screen sized fractions from the 10 minute pebble mill grinding test were mounted in lucite briquettes and polished for microscopic studies, in order to ascertain minerals present in the ore and to

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\*Ore processed by the Domestic Manganese and Development Co. assayed 22.2% Mn, 5.9% CaO, and 2.5% MgO.

determine the size of liberation. This study revealed the presence of manganese oxide minerals, limonite (as stain on calcite particles), and hematite in very small amounts, besides the minerals that were recognized in the megascopic study. The manganese oxide minerals comprise less than 0.1 per cent of the sample. The sulfides of iron, lead, and zinc were present as locked particles with the quartz and carbonate minerals in plus 150 mesh material; but, in the -200 mesh fraction only a very small per cent of the sulfides were present as locked particles. It was impossible to distinguish between the calcite, dolomite, and rhodochrosite minerals of any sized fraction below 48 mesh. In the -28 +35 mesh and -35 +48 mesh fractions, a few mineral particles could be identified as rhodochrosite because of their characteristic pinkish color. The number of these particles identified as rhodochrosite were few in relation to actual quantity of manganese present in the sample. Therefore, it was apparent that a reliable method of identifying and distinguishing the calcite, dolomite, and rhodochrosite minerals must be developed and employed in order to determine the chemical and physical relationships between the carbonate minerals.

Differential Staining. The sized samples from the grinding tests which had been briquetted and polished were used for the differential staining tests. The dolomite was stained first by immersing the sample in a solution of

1/100 N HCl plus a few drops of freshly prepared potassium ferricyanide. Because of the iron content in dolomite, the potassium ferricyanide reacts with the ferrous salts giving a blue stain (Turnbull's blue) to the dolomite particles.<sup>7</sup> After the dolomite particles were stained, the briquetted samples were then immersed for two minutes in Lemberg solution and the calcite particles were given a yellowish stain. This method of differential staining could be employed under certain conditions to permit a microscopic count<sup>8</sup> to be made of free particles, binary particles, and ternary particles in order to determine the liberation of the rhodochrosite minerals and to determine what chemical association exists between the carbonate minerals. No attempt was made to determine the quantitative mineragraphy on the briquetted samples, because it was difficult to determine particles exposed and particles lying below the lucite surface, due to their transparent nature. This undesirable condition could be corrected by mounting the samples in bakelite.

Lemberg Staining Solution.<sup>7</sup> The standard Lemberg solution used for staining the calcite mineral was prepared by boiling for 20 minutes a mixture of 4 grams of  $\text{AlCl}_3$ , 6 grams extract of logwood, and 60 grams of  $\text{H}_2\text{O}$ , with constant stirring. Water was added to make up for evaporation. The solution was then filtered and bottled for use. Calcite, when immersed in the standard Lemberg solution, becomes coated with  $\text{Al}(\text{OH})_3$ , which serves as a mordant, by absorbing the dye (extract of logwood).

X-ray Analysis. The powder method<sup>9</sup> of X-ray analysis, developed by Debye and Scherr, was employed in this investigation. This investigation was carried out on several +28 mesh particles, selected because of their very pinkish color which was assumed to be evidence of their being pure rhodochrosite. These particles were pulverized and the sample was tested by Chester Freshour, graduate student in the department of Mineral Dressing, and Orlano B. Ruste, graduate student in the department of Metallurgy.

The purpose of this test was to determine if calcite or dolomite existed in a solid solution with rhodochrosite. Several unsuccessful tests were made, but after the desirable adjustments were made in operating conditions, it was found that an exposure of two hours at 25 milliampers and 40 kv produced seven faint lines on the film. Four of the lines were identified as rhodochrosite, but the other three lines could not be identified, as they were too faint. It appeared that something was present to disperse the X-rays and to prevent a more pronounced picture to be made.

#### Liberation Studies

The microscopic tests conducted indicated that satisfactory liberation of the carbonate minerals exists at 28 mesh. Grinds of -10 mesh, 600 gram samples were made utilizing both pebble mill and rod mill, and using 5 kilograms of pebbles and steel rods respectively. The products from

the mills were wet screened for the -200 +325 mesh and -325 mesh fractions, and dry screened to obtain the other sized fractions. The results of the grinding tests are given in the following table (Table II).

TABLE II.

Sieve analyses of 600-gram samples of -10 mesh material ground for 10 minutes in both laboratory pebble mill and rod mill.

Mesh	Pebble	Mill	Rod	Mill
	% Weight	Cumulative Weight %	% Weight	Cumulative Weight %
+ 28	20.66	20.66		
+ 35	7.39	28.05		
+ 48	7.75	35.80		
+ 65	6.82	42.62		
+100	7.55	50.17	0.1	0.1
+150	8.43	58.60	1.9	2.0
+200	6.62	65.22	8.4	10.4
+325	6.29	71.51	28.8	39.2
-325	28.49	100.00	60.8	100.00

These grinding tests indicated that to achieve flotation samples of -48 mesh using the pebble mill, that stage grinding would have to be utilized in order to prevent the

production of excess slimes. Satisfactory grinding was attained by grinding in the mill for 15 minutes followed by wet screen sizing at 48 mesh and regrinding the oversize for 10 minutes. This grinding gave samples for flotation testing in which 99.9 per cent was -48 mesh. The grinding tests clearly show that the ore is very friable and that considerable fines (-325 mesh) are produced.

### GRAVITY CONCENTRATION

A sink and float test was conducted not only to determine the feasibility of applying this method as one step in the beneficiation of Philipsburg rhodochrosite ore, but also to arrive at an understanding to what degree a separation could be easily made between the rhodochrosite and calcite or dolomite minerals, and still obtain a reasonable recovery of the manganese.

The -10 mesh material was removed from the sample cut for the sink and float testing, and the  $-1\frac{1}{4}$  inch plus 10 mesh material was coned and quartered twice. This representative sample was submitted to the sink and float testing, using acetylene tetrabromide (sp. g. =2.96) and carbon tetrachloride (sp. g. =1.59) to dilute the acetylene tetrabromide and give solutions of lower specific gravity. With the use of the specific gravity balance, the specific gravities of the gangue and rhodochrosite particles were found to average 2.78-3.34 respectively. The results of the sink and float



test are given in the following table (Table III, page 14), and this shows an up-grading in the first sink product (sp. g. =2.96) of 8 per cent in the manganese content over the feed to the sink and float operation with a lowering of the CaO content of 3.7 per cent, and a recovery of 78.2 per cent of the manganese.

A visual examination of the sink and float products disclosed that many of the particles ranging in size  $-1\frac{1}{4}$   $+1\frac{1}{2}$  inch could be more satisfactorily concentrated if the feed to the sink and float operation was all  $-1\frac{1}{2}$  inch, because many of the  $+1\frac{1}{2}$  inch particles in the sink products had attached pieces of gangue which could be separated from the manganese minerals by a slightly finer crushing. Many of the  $+1\frac{1}{2}$  inch particles in the float product contained considerable manganese minerals which also would be freed by finer crushing.

In any proposed operation for beneficiating Philipsburg rhodochrosite ore, a thorough investigation of utilizing sink and float concentration as one step in the operation should be made. Sink and float could not be the only concentrating step employed, because the sulfides that would pass into the sink product would have to be removed before the manganese concentrate could meet the necessary chemical specifications; also, this sulfide product might be of sufficient worth to be considered as a secondary product. The results of the sink and float testing are a



means of determining a minimum goal for flotation testing, as it is clearly shown that a flotation product is possible in which the manganese content is 30.4 per cent, CaO content 6.1 per cent, and a recovery of 70.9 per cent.

## FLOTATION METHODS

Tabulated results of all the flotation tests conducted in this investigation are given in the appendix to this report. All of the flotation methods tried, consisted first of a bulk sulfide float using copper sulfate, amyl xanthate, and cresylic acid. Previous work<sup>6</sup> in flotation testing indicated that the reagents used to make the bulk sulfide float did not affect the manganese flotation circuit. Consequently, the sulfides were removed in all instances in the first flotation step.

The first approach to the flotation of the manganese was similar to the actual process employed by the Domestic Manganese and Development Company in treating rhodochrosite ore. This first method consisted of making a manganese float using sodium silicate, soda ash, oleic acid, and pine oil. The use of pine oil was beneficial in controlling the froth produced by the oleic acid. The purpose of this method was to investigate the affect of temperature on the process employed by the Domestic Manganese and Development Company.

A variation of this method was the stage addition of oleic acid using quebracho<sup>9, 10</sup> as a calcite and dolomite depressant. A preliminary flotation test was conducted on a

quartz and calcite sample using quebracho as a depressant; this test indicated that quebracho would partially depress the calcite as approximately 90 per cent of the flotation sample remained in the float tails. In the flotation tests the quebracho depressed the manganese minerals as well as the gangue carbonate minerals. Temperature studies applied to the first method resulted in only a slight increase in the grade of the manganese product with increased recovery in the first rougher manganese concentrate; these studies did not disclose any degree of selectivity between the manganese carbonate and the gangue carbonate minerals.

The second method of flotation was made with the use of lauryl amine as collector. This reagent did not display any collecting power for the carbonate minerals at room temperature or at elevated temperatures.

The third method employed was with stearic acid as a collector in the manganese circuit, but this reagent did not show any collecting power at the pulp temperatures employed. The pulp temperature used was only up to 40 degrees centigrade, which is considerably below the melting point of stearic acid (m.p. =69.4°C.). The lack of success of this reagent to collect might have been due to the temperature conditions, as Gaudin<sup>11</sup> gave evidence in the use of capric acid (m.p. =31°C.) the collecting power of the reagent was unsatisfactory when employed under flotation conditions in which the pulp temperature was below the melting point temperature of the reagent.

The fourth method employed in flotation testing involved grinding the -10 mesh samples in a rod mill for 10 minutes to produce a sample which would be -150 mesh; the sample was conditioned for 10 minutes using oleic acid as a collector, sodium silicate, and soda ash, and making a calcite float followed by an addition of oleic acid and flotation of a rougher manganese concentrate. The results of this method were similiar to those attained by the other methods.

#### CONCLUSIONS

Gravity concentration by sink and float showed that this method of concentration gave far better results than those attained by flotation, nevertheless, the flotation tests reported herein are only preliminary to the many other flotation methods that should be tested. Oleic acid was used in most of the tests conducted and no reagent was found to replace oleic acid in its role of flotation of the carbonate minerals; but, oleic acid did not display any selectivity in the flotation of manganese carbonate, and flotated the gangue carbonate minerals as easily, even at various elevated temperatures and under changed pH conditions. The employment of elevated temperatures in the consideration of any proposed flotation process should be thoroughly and systematically investigated, as this application of elevated temperature in the flotation tests conducted in this study is not conclusive evidence of its impracticability.

## RECOMMENDATIONS

Future research work on this problem must include a thorough study to determine the degree of solid solution which exists between the calcite, dolomite, and rhodochrosite, in order to discover the actual recovery that would be possible and still maintain the grade of product required by specifications.

Future research in flotation should be made using fatty acids other than oleic acid, at flotation temperatures above the melting points of the fatty acids.<sup>11</sup>

Any future research on Philipsburg rhodochrosite ore should include a sink and float test on a representative sample of the ore. The sample for this test should be crushed and screened to produce a feed for the sink and float test which would be  $-\frac{1}{2}$  inch +14 mesh. This size of feed to the sink and float test should account for a higher grade manganese product, and should give an increase in the manganese recovery unless the -14 mesh produced by the finer crushing would be much greater than that given when the sample was crushed to  $-1\frac{1}{4}$  inches for the sink and float test conducted in this study.

APPENDIX

TABLE IV

Flotation Test No. 1

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	4.6	10.3	11.1	2.1	5.9
Rougher Mn C	73.4	24.5	9.4	81.8	78.5
" Tails	22.0	16.1	6.2	16.1	15.6
Comp. (Calc)		22.0	8.8		

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
amyl xanthate	0.1				
CuSO <sub>4</sub>	0.3				
cresylic acid	0.1				
soda ash		5.0			
sodium silicate		1.5			
oleic acid		1.3			
pine oil		0.1			

Remarks: pH in sulfide circuit natural pH of the ore 7.6. The Mn rougher float was made at pH 10.2. Froth very wild. Feed to flotation -48 mesh material from pebble mill grind.



TABLE V

Flotation Test No. 2

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.2	12.8	6.1	3.7	4.4
Rougher Mn C	56.7	24.9	9.7	65.7	63.9
Rougher Tails	37.1	17.8	7.4	30.6	31.7
Comp. (Calc)		21.4	8.3		

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
amyl xanthate	0.1				
CuSO <sub>4</sub>	0.5				
oresylic acid	0.1				
soda ash		2.5			
sodium silicate		1.5			
oleic acid		0.6			
pine oil		0.06			

Remarks: pH of manganese float 9.8. Voluminous and very difficult to control. Feed -48 mesh.

TABLE VI

Flotation Test No. 3

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	5.8	13.6	6.5	3.7	4.6
Rougher Mn C	64.0	24.2	9.1	72.7	70.5
Rougher Tails	30.2	16.8	6.8	23.6	24.9
Comp. (Calc)		21.4	8.3		

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
amyl xanthate	0.1				
CuSO <sub>4</sub>	0.5				
creylic acid	0.1				
soda ash		0.5			
sodium silicate		1.5			
oleic acid		0.6			
pine oil		0.1			

Remarks: pH of rougher Mn float 8.6. Froth easily controlled with pine oil. Froth had good carrying power. Feed -48 mesh.

TABLE VII

Flotation Test No. 4

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	5.7	12.3	6.1	3.3	4.1
Rougher Mn C	52.8	26.2	9.0	64.3	55.5
Rougher Tails	41.5	16.8	8.3	32.4	40.4
Comp. (Calc)		22.4	8.9		

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
creylic acid	0.1				
soda ash		0.5			
quebracho		0.5			
sodium silicate		1.5			
oleic acid		1.0			

Remarks: pH of rougher Mn float 8.8. Froth did not have the carrying power as in the other previous tests. All the carbonate mine als are depressed. Feed -48 mesh.

TABLE VIII

Flotation Test No. 5

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.3	15.7	7.0	4.6	5.1
Rougher Mn C-1	26.7	27.6	9.5	34.2	29.6
" " C-2	47.1	22.8	9.1	49.8	50.0
" Tails	19.9	12.5	6.6	11.4	15.3
Comp. (Calc)		21.6	8.6		

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
cresylic acid	0.1				
soda ash		1.0			
quebracho		0.5			
sodium silicate		1.5			
oleic acid		1.0			
pine oil		0.1			

Remarks: pH of the rougher Mn float 9.0. Feed -48 mesh.



TABLE X

Flotation Test No. 7

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.4	14.1	6.5	4.4	4.9
Rougher Mn C-1	29.6	28.2	8.2	40.5	28.0
" " C-2	24.0	24.2	9.7	28.1	26.9
" Tails	40.0	13.9	8.7	27.0	40.2
Comp. (Calc)		20.6	8.7		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	to C-2 rough		
CUSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
cresylic acid	0.1				
soda ash		0.5			
quebracho		0.5			
sodium silicate		1.5			
oleic acid		0.3	0.3		

Remarks: Froth easily controlled by stage addition of oleic acid. Manganese circuit maintained at 30 degrees C. and pH of 9.0. Results show slight selectivity in the 1st conc of the calcite and rhodochrosite minerals. Feed -48 mesh.

TABLE XI

Flotation Test No. 8

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk, Sulf Conc	6.9	12.5	5.9	3.9	4.8
Rougher Mn C-1	29.7	24.8	10.1	36.0	34.1
" " C-2	35.8	22.8	8.8	40.0	35.9
" " C-3	11.2	19.1	9.3	10.5	11.8
" Tails	16.4	12.1	7.1	9.6	13.4
Comp. (Calc)		20.4	8.8		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	C-2	C-3	
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
cresylic acid	0.1				
soda ash		0.5			
Lemberg soln		0.2			
sodium silicate		1.5			
oleic acid		0.2	0.2	0.2	

Remarks: pH of rougher Mn float 8.8. Lemberg soln did not give and beneficial conditions as to selectivity. Stage addition of oleic acid very desirable in controlling the froth. Feed -48 mesh.

TABLE XII

Flotation Test No. 9

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	5.2	13.0	6.3	3.3	3.8
Rougher Mn C-1	38.8	25.4	9.5	47.8	42.0
" " C-2	16.9	24.8	9.5	20.3	18.2
" " C-3	7.8	21.7	9.5	8.2	8.4
" " C-4	11.0	17.8	9.3	9.5	11.6
" Tails	20.3	11.0	6.9	10.9	16.0
Comp. (Calc)		20.6	8.8		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	C-2	C-3	C-4
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
creylic acid	0.1				
soda ash		1.0			
quebracho		0.5			
sodium silicate		1.5			
oleic acid		0.1	0.1	0.1	0.2

Remarks: The manganese circuit was raised to 40 degrees C. where the stage addition of oleic acid was made and pH was 8.9. At the elevated temperatures the manganese circuit consumed less oleic acid than in similiar tests conducted at room temperature. Froth very desirable and easily controlled when stage addition of oleic acid is employed. Feed -48 mesh.



TABLE XIII

Flotation Test No. 10

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.0	-----	-----	-----	-----
Rougher Mn C	84.4	23.0	9.0	-----	-----
" Tails	9.6	-----	-----	-----	-----

Reagents in pounds per ton:

Reagent	to sulfide	to rough			
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
creylic acid	0.1				
soda ash		0.5			
sodium silicate		1.5			
oleic acid		1.3			
pine oil		0.5			

Remarks: Mn circuit conditioned at 30 degrees C. for 5 minutes and then cooled and floated at 19 degrees C. No selectivity and froth very wild and not controlled by addition of pine oil. pH Mn circuit 8.4. Feed -48 mesh.







TABLE XVII

Flotation Test No. 14

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.1	12.8	6.0	3.6	4.2
Rougher Mn C-1	17.2	24.5	10.1	19.4	19.2
" " C-2	28.2	26.5	10.5	34.5	33.2
" Tails	48.5	18.7	8.0	42.5	43.4
Comp. (Calc)		21.7	8.9		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	C-2		
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
creylic acid	0.1				
soda ash		0.2	0.3		
sodium silicate		1.5			
oleic acid		0.3	0.2		

Remarks: The rougher Mn circuit was conditioned for 10 minutes at 22 degrees C. before the 1st rougher conc was made, and then the circuit was conditioned for 1 minute after the addition of 0.2 pounds per ton of oleic. Conditioning the pulp for 10 minutes instead of the 2 minutes used in the previous tests did not show any differential selectivity in the flotation of the carbonate minerals. pH of 1st rougher conc was 8.0 and the pH of the circuit for C-2 was 8.8. Feed -48 mesh.



TABLE XIX

Flotation Test No. 16

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	6.6	13.1	6.0	4.0	4.7
Rougher Mn C-1	4.0	18.4	7.5	3.4	3.5
" " C-2	3.7	19.5	9.2	3.3	4.0
" " C-3	3.3	19.3	8.4	3.0	3.3
" " C-4	7.2	20.4	9.1	6.8	7.6
" " C-5	33.8	25.2	10.3	39.4	40.5
" T Froth	5.9	20.1	7.4	5.5	5.1
" Tails	35.5	21.1	7.6	34.6	31.3
Comp. (Calc)		21.6	8.6		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	C-2	C-3	C-4	C-5
CuSO <sub>4</sub>	0.3					
amyl xanthate	0.1					
cresylic acid	0.1					
sodium silicate		1.5				
soda ash		1.0				
stearic acid		0.1	0.2		0.1	
oleic acid						0.5

Remarks: pH Mn circuit 9.0. Pulp conditioned for 5 minutes at a temperature of 25 degrees centigrade. Good froth but no carrying power. Oleic acid gives good collection but no selectivity of carbonate minerals. Feed -48 mesh.

TABLE XX

Flotation Test No. 17

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	5.5	11.9	5.6	3.1	3.5
Rougher Mn C-1	39.8	25.5	10.7	47.2	48.6
" " C-2	20.6	22.9	9.0	22.0	21.2
" " C-3	10.0	22.6	9.8	10.4	11.2
" Tails	24.1	15.4	5.6	17.3	15.5
Comp. (Calc)		21.5	8.8		

Reagents in pounds per ton:

Reagent	to sulfide	to C-1 rough	C-2	C-3	
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
cresylic acid	0.1				
soda ash		2.5	0.5		
sodium silicate		1.5			
oleic acid		0.3	0.2	0.2	

Remarks: Feed was rod mill grind consisting of -150 mesh material. Sulfide circuit pH was natural pH of the ore 7.0 which was less than the natural pH of the ore when the flotation feed was -48 mesh material. pH of circuit during flotation of C-1 and C-2 was 9.4 and the pulp was conditioned for 2 minutes at a temperature of 26 degrees C.



TABLE XXI

Flotation Test No. 18

Products	% Weight	Assay		% Distribution	
		Mn	CaO	Mn	CaO
Bulk Sulf Conc	7.4	14.4	6.5	4.9	5.4
Rougher Ca C-1	21.2	23.6	10.1	23.0	24.1
" " C-2	13.8	21.9	9.2	13.8	14.3
" Mn C-1	25.2	24.5	9.9	28.5	28.2
" " C-2	12.9	22.9	9.4	13.6	13.6
" Tails	19.5	18.1	6.5	16.2	14.4
Comp. (Calc)		21.7	8.9		

Reagents in pounds per ton:

Reagent	to sulfide	to Ca-1 rough	Ca-2	Mn-1	Mn-2
CuSO <sub>4</sub>	0.3				
amyl xanthate	0.1				
cresylic acid	0.1				
sodium silicate		1.5			
soda ash		0.2		1.8	
oleic acid		0.1	0.1	0.1	0.1

Remarks: pH of circuit for Ca-1 and Ca-2 was 8.0. pH for Mn-1 and Mn-2 was 9.4. The conditioning time does not give any degree of selectivity in the floating of the carbonate minerals. Feed -150 mesh.

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