

(0290) 0000 0180

MMS# 15

Report on
Beneficiation of a
Magnetite-Pyrrhotite-Chalcopyrite Ore
for
WALKER-MARTEL MINING COMPANY
Project No. 260211 April 8, 1966

FOR GOVERNMENT USE ONLY

FOR GOVERNMENT USE ONLY

Report on

MMS# 15

Beneficiation of a Magnetite-Pyrrhotite-Chalcopyrite Ore

PROPRIETARY

Prepared for

WALKER-MARTEL MINING COMPANY
1080 PINE RIDGE DRIVE
RENO, NEVADA


By


Colorado School of Mines Research Foundation, Inc.
Golden, Colorado

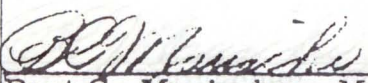
Project No. 260211

April 8, 1966

APPROVED:


E. H. Crabtree, Director


F. T. Davis
Director of Research


Burt C. Mariacher, Manager
Metallurgical Division

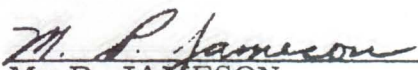

M. P. JAMESON
PROJECT ENGINEER
METALLURGICAL DIVISION

TABLE OF CONTENTS

	<u>Page</u>
Introduction	1
Objective and Scope	2
Summary of Results	3
Recommendations	5
Discussion	7
Sample Description	8
Davis Tube Tests	10
Flotation	15
Roasting	22
Bond Grindability	24
Technical Feasibility	25
Appendix	27
Exhibit 1, Sample Description and Preparation	28
Exhibit 2, Petrographic Analysis	30
Exhibit 3, Davis Tube Tests	34
Exhibit 4, Flotation Tests	37
Exhibit 5, Roasting Test	48
Exhibit 6, Screen Tests	49
Exhibit 7, Bond Grindability Test	57

INTRODUCTION

The Walker-Martel Mining Company, Reno, Nevada, is interested in developing a metallurgical process for a magnetite-pyrrhotite-chalcopyrite ore deposit in Nevada.

The Colorado School of Mines Research Foundation, Inc., was engaged to conduct preliminary beneficiation studies on a core sample from one drill hole to obtain information on the technical feasibility of treating the ore by conventional methods.

The project reported herein was proposed in a letter from Mr. F. T. Davis to Mr. Robert L. Redmond, General Manager, Walker-Martel Mining Company, dated February 8, 1966, and approved in a formal agreement signed by Mr. Redmond dated February 18, 1966.

OBJECTIVE AND SCOPE

The primary objective of this project was to determine the technical feasibility of producing a marketable magnetite and also a chalcopyrite product from the Nevada ore.

The scope of this project was to test the magnetite-pyrrhotite-chalcopyrite ore to determine its amenability to conventional beneficiation techniques such as flotation, magnetic separation, and roasting.

SUMMARY

The following is a summary of the beneficiation studies completed in the preliminary investigation of the NX Core sample submitted by the Sponsor.

1. Chemical analysis of a composite sample for major constituents and the important contaminants gave the following results:

<u>Fe</u> <u>%</u>	<u>S</u> <u>%</u>	<u>Cu</u> <u>%</u>	<u>P</u> <u>%</u>
46.8	3.08	0.059	0.03

These values may be slightly high because the manner of compositing the sample could have resulted in a minor deficiency in non-mineralized material.

2. Magnetic concentration of the magnetite using a Davis tube separator and feed ranging in size from nominal minus 35 to minus 200 mesh yielded products containing 62 to 69 per cent iron, less than 0.02 per cent copper, but high sulfur content from 1.9 to 2.5 per cent.

3. A nominal minus 100 mesh grind containing 56.6 per cent minus 325 mesh produced a magnetic concentrate as follows:

<u>Weight</u> <u>%</u>	<u>Chemical Analysis</u>			<u>Per Cent</u> <u>Distribution</u> <u>Fe</u>
	<u>Fe</u> <u>%</u>	<u>Cu</u> <u>%</u>	<u>S</u> <u>%</u>	
59.5	67.8	0.006	1.92	85.0

SUMMARY (continued)

4. When a pelletized magnetic concentrate containing 3.5 per cent sulfur was roasted at 2250°F, the sulfur content was reduced to less than 0.01 per cent.

5. The pellets had good resistance to spalling during firing and the roasted pellets appeared to have suitable strength properties.

6. Flotation of sulfide material from the magnetic concentrate gave a product containing 65.3 per cent iron, and 0.20 per cent sulfur with a recovery of 78.9 per cent of the iron based on the original feed.

7. High recovery of the chalcopyrite by flotation in a product containing 12.6 per cent copper was obtained. The small amount of concentrate produced limited reliable testing for ultimate grade.

8. The work index of the ore as determined by the Bond grindability system was 13.03, slightly lower than average taconite.

9. Magnetite, pyrrhotite, chalcopyrite, and pyrite are the minerals of major importance. In general, the composite sample had a 150 mesh particle size liberation for the magnetite mineral.

10. The ore represented by this NX Core sample from one drill hole had satisfactory beneficiation characteristics using techniques currently practiced in the iron ore industry.

RECOMMENDATIONS

The results of the preliminary test work conducted to date on the ore as represented by one drill hole indicate that further exploration of the deposit with laboratory studies conducted on the drill cores is warranted.

The areas of laboratory studies which should be investigated are listed as follows:

Magnetic Concentration

Studies are required to determine the extent to which dry cobbing and wet magnetic roughing stages can be used in the production of a magnetic concentrate. In conjunction with this, grindability tests would be required for the cobbing and roughing concentrates so the magnetic separation-grinding circuit can be designed.

Flotation

It is likely that additional drilling will show that the ore body contains a higher copper content than was present in this first drill core. If so, flotation studies should be made to develop information related to the economics of producing a chalcopyrite by-product. This study would include determining grinding requirements (substantial copper may be present in the nonmagnetics from coarse cobbing), copper recovery, and grade of concentrate.

Further work on flotation to reduce the sulfur content of the iron concentrate is not anticipated because it appears that this may be accomplished readily in the pellet firing operation.

RECOMMENDATIONS (continued)

Pelletizing and Pellet Firing

A study should be made of the factors which influence the strength of green and dried pellets. These factors would include the particle size distribution of a magnetic concentrate, and the type and amount of binder.

The time and temperature for firing the pellets should be investigated. These would be evaluated by determining their effect on sulfur removal, and on the pellet resistance to abrasion, compression, and spalling.

Economic Estimates

When sufficient data are available from drill cores to establish more reliably the ore body characteristics and test work has been completed on representative material, estimates can be undertaken to develop information on capital investment and operating costs.

DISCUSSION

This report covers the preliminary investigation of the beneficiation of a magnetite-pyrrhotite-pyrite-chalcopyrite ore from Nevada submitted by the Walker-Martel Mining Company.

Crushed NX Core sections from one drill hole were shipped to the Research Foundation where a portion of each sample was composited for use as feed material for various analytical and beneficiation tests as follows:

1. Chemical and mineralogical analyses were made of the composite sample. A petrographic examination of one magnetic concentrate was conducted.
2. Davis tube tests were employed as an instrument for determining the mesh of grind with relationship to magnetic concentrate grade and iron recovery.
3. Flotation tests were made on the composite sample and on products from magnetic separation to lower the sulfur content in the magnetite concentrate and to obtain a copper concentrate.
4. Roasting of pelletized magnetic concentrate was studied as an alternate means of reducing the sulfur content.
5. Work index for grinding the ore was determined by a Bond grindability study.

DESCRIPTION OF SAMPLE OF CALICO ORE SUBMITTED TO COLORADO SCHOOL OF MINES RESEARCH FOUNDATION, INC., FOR METALLURGICAL TESTING.

The basic sample for this work was obtained by a vertical drill test, designated drill hole CA-1, located on the magnetic anomaly on the Calico Prospect. This hole was drilled by rotary techniques from the surface to 1332', and was continued from 1332' to 2640' with NX wireline core drilling techniques. Core recovery was in excess of 95% throughout the hole. The core was placed in core boxes, and the mineralized portions were split longitudinally with a hand core splitter, and one-half of the core was placed in sample bags, each identified with a unique number, and the other half returned to the core box. The portion in the sample bags was sent to Union Assay Office, P. O. Box 1528, Salt Lake City, Utah, for iron and copper assays. After assaying, the rejects from these samples were then returned to Walker-Martel Mining Co. in Reno.

For purposes of this metallurgical testing, the entire bags of rejects from samples 1013R to 1050 R, inclusive, and 1101R to 1120R, inclusive, representing the drill hole interval 1766.0' to 2286.0', a distance of 520', were shipped to Colorado School of Mines Research Foundation, Inc., together with instructions as to the type of testing desired. Colorado School of Mines Research Foundation is still in possession of approximately one-half of each bag of rejects, the balance having been used in the testing. Walker-Martel has possession of the assay pulps and the other one-half of the raw core, stored in the core boxes, which is available for further testing or examination.

All the above work was done in accordance with standard practices used in the mining industry, by technical personnel of Walker-Martel's staff, and it is felt that this sample fairly represents the interval in the drill hole from which it was taken.

DISCUSSION (continued)

The composite sample was examined petrographically to determine the liberation size of the magnetite, chalcopyrite, and the gangue minerals. It was determined that the magnetite was almost completely liberated from the gangue at 150 mesh, but some gangue particles remained locked in magnetite at this size.

The chalcopyrite did not have sufficient concentration to permit a valid determination of liberation size; however, none of the crystals observed were larger than 150 mesh and they were mostly associated with the magnetite. Other sulfide minerals identified were pyrrhotite and pyrite. Principal gangue minerals were augite, calcite, and feldspar. Exhibit 2 of the Appendix presents a more detailed description of the petrographic analysis of the composite sample.

DISCUSSION (continued)

Davis Tube Tests

The Davis tube test will normally produce a higher grade magnetic concentrate than can be obtained in a commercial separator because the long period of washing in the Davis tube will remove a larger percentage of the gangue minerals from a sample. However, the test will give a good indication of what can be expected in a commercial operation with multiple stages of magnetic cleaning.

The feed for the Davis tube tests was produced by wet grinding in a laboratory ball mill with a minus 20 mesh charge of 260 grams.

Three tests were programmed where the grinding time was varied to produce different particle size distribution analysis. Size designation of the three tests may be summarized as follows:

Test No.	Nominal Designation	Mesh Size of Davis Tube Feed
		Weight % Passing 325M
1	-35M	28.9
2	-100M	56.6
3	-200M	85.6

Details of the grinding procedures are presented in Exhibit 3 of the Appendix, and Exhibit 6 of the Appendix contains the screen analyses of the ground material.

DISCUSSION (continued)

The iron concentrate grade and iron recovery produced in the Davis tube tests are plotted graphically, Figure 1, against mesh of grind designated by the minus 325 mesh fraction of the test feed.

As the graph illustrates, to obtain a concentrate containing 67 per cent iron, the feed material would necessarily have to contain 50 per cent minus 325 mesh size.

Iron recovery of about 85 per cent is reasonable for this type of ore containing nonmagnetic iron-bearing minerals such as pyrite and chalcopyrite. A part of the pyrrhotite also can be expected to report to the nonmagnetic product in these tests. The small increase in iron recovery in the finest grind as compared to the intermediate grind may be related to liberation of pyrrhotite. Particles attached to gangue may have been lost at the coarser grind but held in the concentrate when liberated at the finer grind.

Weight recovery in the magnetic concentrate is summarized as follows:

<u>Test No.</u>	<u>Concentrate Weight %</u>	<u>Ratio of Concentration</u>
1	67.8	1.47
2	59.5	1.68
3	59.2	1.69

The copper content of the concentrates produced in the Davis tube tests is presented in the following table.

1" x 10" INCHES
 MADE IN U.S.A.
 353
 KENNEL & ESSER CO.
 H: C TO

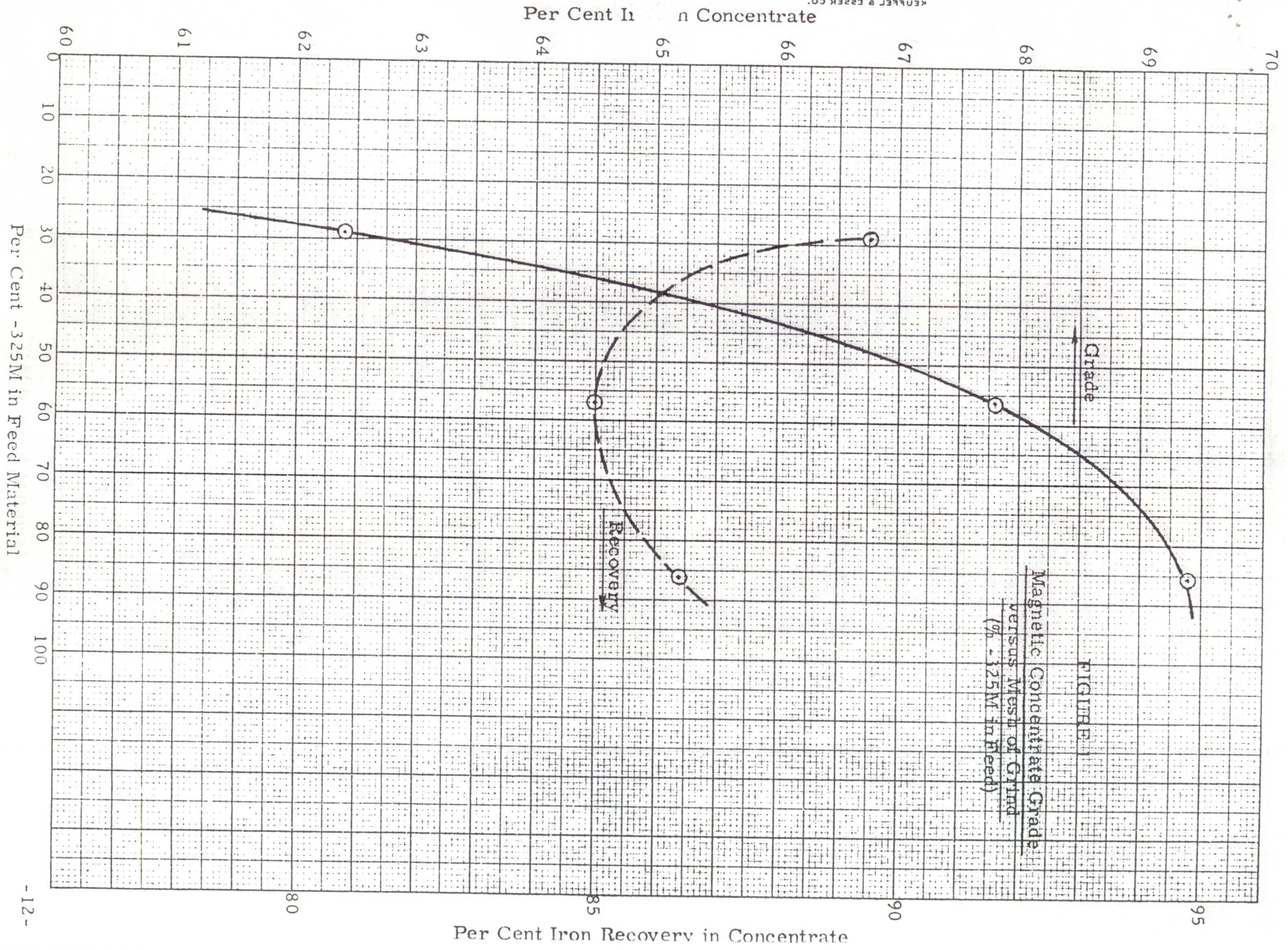


FIGURE 1
 Magnetic Concentrate Grade
 versus Mesh of Grind
 (% -325M in Feed)

Per Cent Iron Recovery in Concentrate

Per Cent -325M in Feed Material

DISCUSSION (continued)

Copper Occurrence in Magnetic Concentrate

<u>Davis Tube Test No.</u>	<u>Per Cent -325M in Feed</u>	<u>Chemical Analysis Cu %</u>
1	28.9	0.016
2	56.6	0.006
3	85.6	0.005

These results demonstrate that the copper content of a magnetic concentrate would be acceptably low at a grind of about 50 per cent minus 325 mesh which is necessary to obtain a high iron concentrate.

The per cent sulfur in the magnetic concentrates produced in the Davis tube tests versus the per cent minus 325 mesh is presented in the graph, Figure 2.

The apparent anomaly as shown by the graph in which a lower sulfur content was obtained for the intermediate grind possibly can be explained as an effect of the amount of liberated pyrrhotite. This is in agreement with iron recovery data as previously discussed.

The higher sulfur content of the fine grind material would indicate a liberation of the pyrrhotite which would then remain in the magnetic concentrate.

The magnetic concentrate of the three Davis tube tests had a sulfur content that probably would be unacceptable.

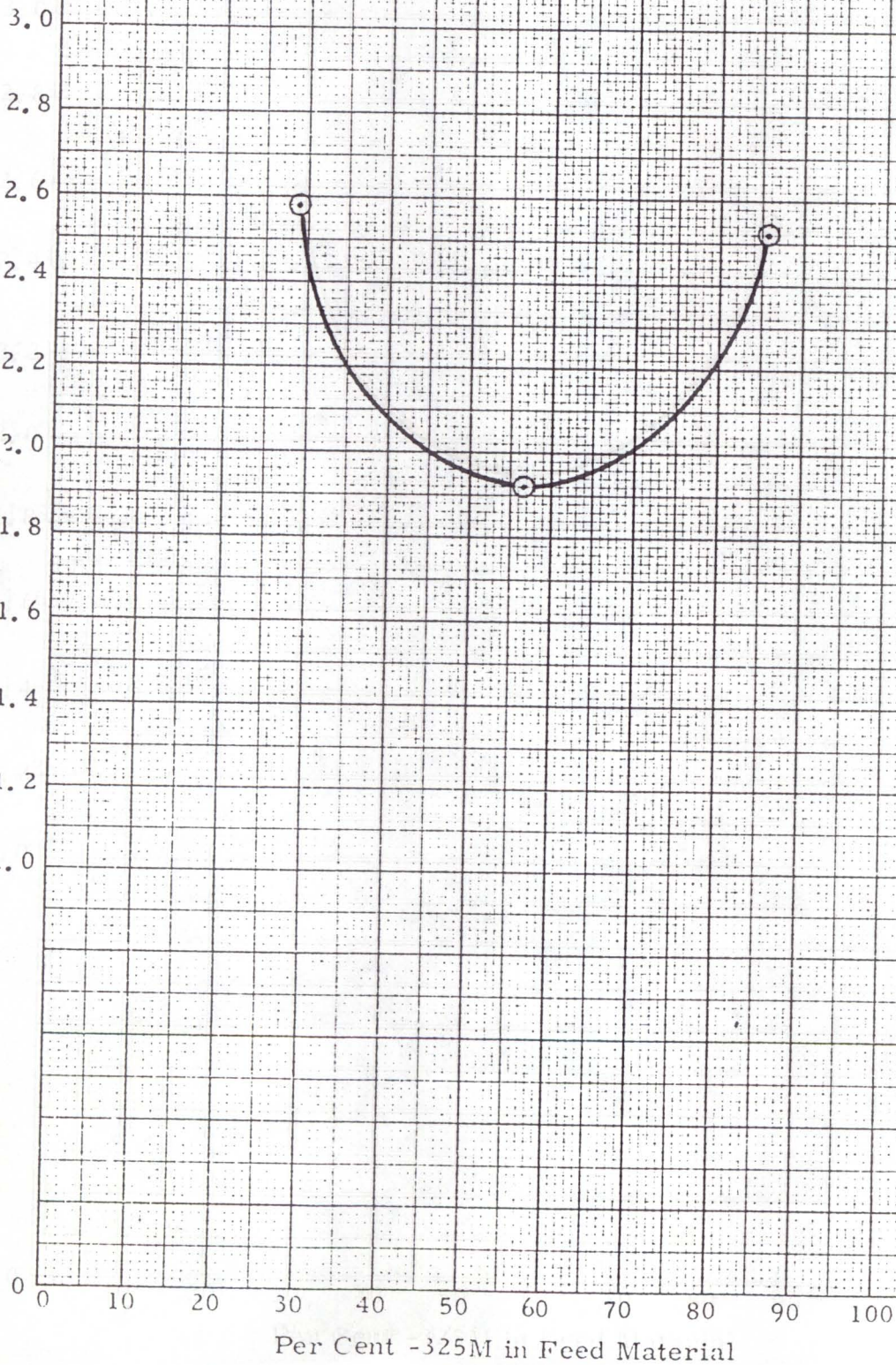
Complete details of the Davis tube tests are presented in Exhibit 3 of the Appendix.

856
H: 3 TO
KENNEL & ESSER CO.
3 X 10 INCHES
MAG. IN. 0.25"

Per Cent Sulfur in Magnetic Concentrate

FIGURE 2

Per Cent Sulfur in Magnetic Concentrate
versus Mesh of Grind
(% -325M in Feed)



DISCUSSION (continued)

Flotation

A series of flotation tests was conducted on Composite Sample No. 1 to attempt a reduction of the sulfur content in the iron concentrate. Flotation was also employed as a means of recovering the copper minerals present in the ore.

Magnetic separation was used to supplement the flotation tests as a beneficiation technique. A laboratory Sala wet drum magnetic separator was used. The Sala is a permanent magnet, wet drum separator with a drum face of 4-5/8 inches and a drum diameter of 7-7/8 inches. The drum was rotated at 64 rpm for all magnetic separation operations in the flotation studies.

Flotation Test No. 1 was made to study the possible removal of sulfides by flotation prior to a magnetic separation to obtain an iron concentrate of acceptable grade. The feed was ground to approximately 56 per cent minus 325 mesh and flotation was conducted in a laboratory Fagergren cell. MIBC (methyl isobutyl carbinol) was added as frother and Z-6 as the collector. A rougher float product was cleaned to produce a copper concentrate using lime to increase the pH to 10.1 and sodium cyanide to depress the pyrite and pyrrhotite.

The nonfloat product from the rougher flotation was subjected to magnetic separation and the magnetic concentrate cleaned by a second pass through the Sala separator.

DISCUSSION (continued)

The results of Flotation Test No. 1 with respect to final concentrates are presented in the following table. Complete details of the flotation testing, with flowsheets included, and the flotation results are presented in Exhibit 4 of the Appendix.

<u>Flotation Test No. 1</u>							
<u>Product</u>	<u>Weight %</u>	<u>Chemical Analysis</u>			<u>Per Cent Distribution</u>		
		<u>Fe %</u>	<u>S %</u>	<u>Cu %</u>	<u>Fe</u>	<u>S</u>	<u>Cu</u>
Feed (assayed)	--	46.8	3.08	0.059	--	--	--
Magnetic Concentrate	64.5	65.6	2.95	--	88.5	58.9	--
Copper Concentrate	0.8	41.9	25.7	4.65	0.7	6.2	62.0 ⁽¹⁾

1/ Calculated from head and product assays.

The above results show that a magnetic concentrate of satisfactory iron grade and recovery was obtained; however, the sulfur content of the magnetic concentrate was high and equal to that obtained from magnetic concentration alone. The copper grade and recovery were both poor. The small amount of material in the concentrate prevented upgrading by additional cleaning stages.

Flotation Test No. 2 was identical to Flotation Test No. 1 except a second sulfide flotation was conducted on the magnetic concentrate. The pH for primary flotation was reduced with sulfuric acid to 7.0, and the rougher float product was given one stage of cleaning at a pH of 10.0.

A magnetic separation was conducted on the rougher nonfloat product and the magnetic concentrate was cleaned by a second magnetic separation.

DISCUSSION (continued)

The cleaned magnetic concentrate was then subjected to sulfide flotation using sodium sulfide to activate the sulfide minerals and Z-6 as the collector. The results of Flotation Test No. 2 with respect to final concentrates are shown in the following table:

Product	Flotation Test No. 2						
	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)	--	46.8	3.08	0.059	--	--	--
Magnetic Concentrate	59.2	64.4	1.65	--	83.3	29.9	--
Copper Concentrate	1.3	42.4	25.4	3.62	1.2	10.4	82.6 ⁽¹⁾

^{1/} Calculated from head and product assays.

The results of this test show that the sulfur content of the magnetic concentrate was reduced approximately 50 per cent as compared with Flotation Test No. 1, but was still too high. The recovery of iron and the grade of the iron concentrate were lower than in Flotation Test No. 1.

The copper concentrate grade was still poor, but copper recovery was increased by approximately 20 per cent. The reduction of pH in the primary flotation might account for the increased copper recovery.

The magnetic concentrate produced in this test was analyzed petrographically to determine if the sulfide material was liberated or locked and also to determine the mineral source of the sulfur contamination. The examination indicated that less than 1 per cent of the weight of the sample was

DISCUSSION (continued)

locked pyrrhotite and there was no evidence of liberated pyrrhotite. The pyrite content of the sample was approximately 5 per cent of the weight of the sample of which approximately 2 per cent was liberated. The size of the pyrrhotite and the pyrite ranged from 14 to 56 microns. Accordingly, a regrind of the magnetic concentrate before refloatation would appear to be beneficial.

Magnetic separation was conducted prior to flotation in Flotation Test No. 3. The magnetic concentrate was cleaned in a second pass through the magnetic separator and then demagnetized. The demagnetized magnetic concentrate was then conditioned with CuSO_4 at a pH of 3.9. Z-6 was used as a collector in rougher and scavenger flotation.

The nonmagnetic material was conditioned at a pH of 9.2 with Z-6 and subjected to flotation. The rougher float product was cleaned twice with sodium cyanide added to the first cleaner. The pH of the two cleaner stages of flotation was 10.2 and 10.4 for the first and second stages, respectively.

The following table presents the results of Flotation Test No. 3 with respect to final concentrates.

DISCUSSION (continued)

Flotation Test No. 3

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)	--	46.8	3.08	0.059	--	--	--
Magnetic Concentrate	56.7	65.3	0.20	--	78.9	5.6	--
Copper Concentrate	0.5	30.4	21.8	12.6	0.3	5.1	100.0 ⁽¹⁾

^{1/} Calculated from feed and product assays.

The results of Flotation Test No. 3 show that a magnetic concentrate apparently can be treated by flotation to reduce the sulfur to an acceptable level. The per cent iron in the concentrate was satisfactory, but as a result of flotation, recovery of iron in the magnetic concentrate was reduced.

There are two possible explanations for the improved removal of sulfide minerals from the magnetic concentrate, alone or in conjunction. First, the copper sulfate and the low pH could cause a better activation of the sulfide minerals and secondly, the demagnetization may have caused a reduction in attraction forces between the magnetite particles and between the magnetite and the sulfides, thus allowing more sulfide surfaces to be collector coated.

The accountability of the sulfur in this test was very poor. The feed material assayed approximately 3 per cent sulfur and the calculated feed was only approximately 2 per cent. The sulfur assay on the magnetic concentrate was checked by two different methods and the computed sulfur content of the

DISCUSSION (continued)

magnetic concentrate was reasonable; however, the results should be verified by repeated tests.

The copper concentrate grade was much better in this test, but, again because of the small amount of material in the product, further upgrading was not attempted. The accountability of the copper in the test was also very poor. The small amount of weight of the products with the relatively high copper content makes it very difficult to obtain satisfactory accountability. This test indicates that the copper probably can be recovered with a satisfactory grade of concentrate.

Flotation Test No. 4 was conducted using sulfide flotation prior to magnetic separation as in Flotation Tests No. 1 and 2. The magnetic concentrate was then reground in an attempt to liberate the pyrite and pyrrhotite still locked in the magnetite as shown by the petrographic examination of the magnetic concentrate from Flotation Test No. 2. The sulfide minerals were then activated by CuSO_4 prior to conditioning with the collector and floating.

Primary bulk sulfide flotation, prior to magnetic separation, employed Z-6 as the collector and pine oil as the frother. The nonfloat product was then given a magnetic separation with the concentrate being cleaned with a second pass through the Sala. The cleaned magnetic concentrate was then reground to liberate the locked sulfides. The reground material which contained 99.3 per cent minus 325 mesh was then conditioned with CuSO_4 at a pH of 5.8 and then

DISCUSSION (continued)

conditioned with Z-6 and pine oil. The conditioned charge was then given a secondary sulfide flotation.

The results of Flotation Test No. 4 are tabulated below with respect to final concentrates.

<u>Flotation Test No. 4</u>							
<u>Product</u>	<u>Weight %</u>	<u>Chemical Analysis</u>			<u>Per Cent Distribution</u>		
		<u>Fe %</u>	<u>S %</u>	<u>Cu %</u>	<u>Fe</u>	<u>S</u>	<u>Cu</u>
Feed (assayed)	--	46.8	3.08	0.059	--	--	--
Magnetic Concentrate	56.0	65.6	2.93	--	77.6	45.1	--
Copper Concentrate	9.8	35.4	15.2	0.58	7.3	40.9	96.3 ⁽¹⁾

1/ Calculated from feed and product assays.

The sulfur content of the magnetic concentrate is unacceptable. The CuSO_4 activation apparently did not aid in the collector coating of the sulfide. The reasons for the failure of activation could be, first, the pH was not as acid as in Flotation Test No. 3, and, second, the magnetic concentrate was not demagnetized prior to flotation.

The copper recovery was very satisfactory but concentrate cleaning was not attempted for this test.

DISCUSSION (continued)

Roasting

Roasting Test No. 1 studied the effect of removing the sulfur from a magnetic concentrate by heating. Feed material for this work was Composite Sample No. 1 ground to approximately 72 per cent minus 325 mesh size. The magnetic concentrate was cleaned by a second pass through the separator. The cleaned magnetic concentrate which contained 72.1 per cent minus 325 mesh material was dried and then pelletized in a laboratory disc pelletizer. Bentonite at a concentration of 0.5 weight per cent was blended into the pellet feed material.

The pellets were dried at 250°F for 3.5 hours and then four dishes of pellets were placed into an electric furnace at 1500°F. The temperature was raised to 2250°F in 2 hours and the first dish was removed.

At intervals of 15 minutes, the three remaining dishes were removed from the furnace. Air was blown into the furnace for the last 15 minutes of the test to study the effect that a strong oxidizing atmosphere would have on the removal of sulfur from the pellets. The introduction of air lowered the temperature in the furnace from 2250°F to 2200°F in 15 minutes.

The results of Roasting Test No. 1 for the removal of sulfur from a magnetic concentrate are presented in the table on the following page. Complete test details are presented in Exhibit 5 of the Appendix.

DISCUSSION (continued)

Roasting Test No. 1

<u>Dish No.</u>	<u>Time at 2250°F min</u>	<u>Chemical Analysis S %</u>	<u>Per Cent Removal S</u>
Feed	--	3.50	--
No. 1	0	0.006	99.8
No. 2	15	0.006	99.8
No. 3	30	0.004	99.9
No. 4	45 ⁽¹⁾	0.004	99.9

1/ Temperature dropped from 2250°F to 2200°F in last 15 minutes.

As can be seen from the results, the sulfur was almost completely removed in the 2-hour heating process. Because of this, the variation of time at the roasting temperature and the increased oxidizing atmosphere of the last 15 minutes had little effect.

Observation of the pellets during their introduction into furnace, the entire roasting time, and the cooling time indicated that their resistance to thermal shock or spalling was very satisfactory.

DISCUSSION (continued)

Bond Grindability

The Bond grindability system using work index is a reliable method of designing the grinding circuit. At this stage of development, detailed design data of this nature are not important, but the work index is helpful for a preliminary rating of the grindability of this ore.

The work index was determined to be 13.03. The average work index of 83 magnetite ores is 10.2, and the average work index of 66 taconite ores is 14.9. Accordingly, it may be anticipated that the grinding circuit for this ore will be similar to conventional taconite circuits with the possibility of a little less power required.

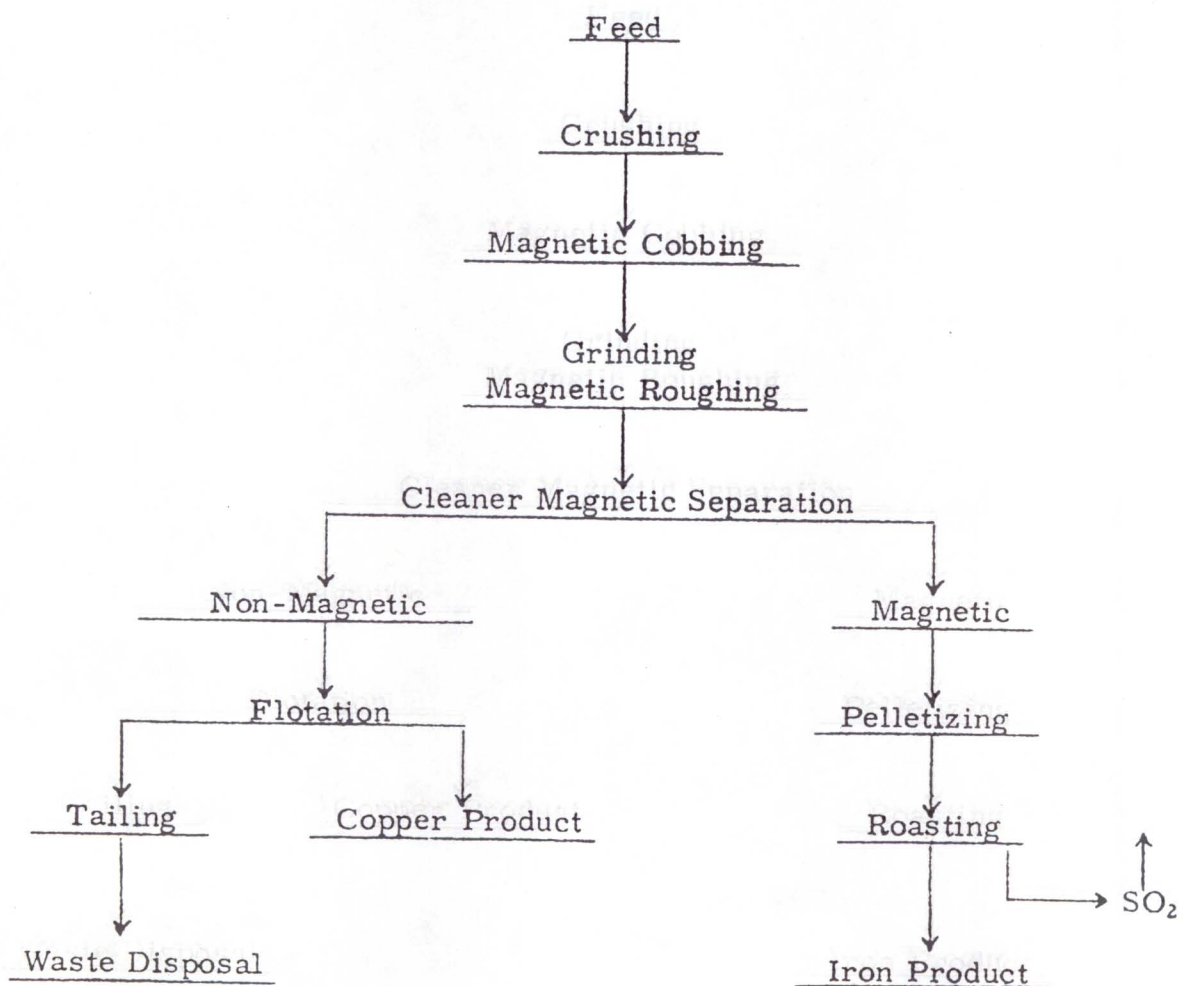
The details of the grindability tests are presented in Exhibit 7.

DISCUSSION (continued)

Technical Feasibility

The preliminary investigations of the beneficiation characteristics of the NX Core composite sample indicate that the ore would be amenable to treatment, which would produce marketable products, by conventional techniques.

A flowsheet of a likely process which appears suited at this time for beneficiating this ore is as follows:.



DISCUSSION (continued)

The feed to final magnetic cleaner separation should have a fineness of about 50 per cent minus 325 mesh. The magnetic concentrate may require regrinding for improved pelletizing.

The roasting operation may require an oxidizing atmosphere for oxidation of the sulfur to sulfur dioxide gas.

The flotation would include a bulk sulfide float followed by cleaning operations to concentrate the chalcopyrite. A pyrite by-product may be feasible if it occurs in sufficient quantity.

Report on
Beneficiation on a Magnetite-Pyrrhotite-Chalcopyrite Ore

APPENDIX

Project No. 260211

April 8, 1966

COLORADO SCHOOL OF MINES RESEARCH FOUNDATION

EXHIBIT 1

SAMPLE DESCRIPTION AND PREPARATION

Sponsor's Designation
of Sample: Nevada chalcopyrite-magnetite ore.

Date Received
at Foundation: Feb. 23, 1966.

Sample Weight: 1027 lb (gross).

Container: 10 burlap sacks with 57 canvas sacks of samples.

Sample Description: Sample numbers were 1013R to 1050R and 1101R to 1120R. The sample ranged from -1/2" down. Sample was gray-black to black, except Sample No. 1032R which was light gray in color.

Method of Preparation: Each individual sack was split using a Jones riffle. One half was replaced in the original sack and the remaining was composited into one sample (Composite No. 1). The composite was coned and quartered down to approx 50 lb, and this fraction was split using the Jones riffle to obtain samples for petrographic and chemical analysis.

The petrographic sample was approx 5 lb "as received". The chemical analysis sample, approx 5 lb, was pulverized to approx 48 to 65M. This was blended and split to 100 grams which was pulverized to -150M.

Crushing and Grinding
for Laboratory Test Work: The Composite Sample No. 1 was coned and quartered and a 50 lb sample was taken for feed material for the laboratory testing.

The sample was screened on a 20M screen and the oversize was stage crushed in a roll crusher until the total sample was -20M. The sample was blended in a tilted cylinder blender.

EXHIBIT 1

Sample Description and Preparation (continued)

The large -20M sample was then split by a Jones splitter into smaller charges which were used as feed to the wet laboratory ball mill where the charge was reduced to the desired particle size.

The inside dimensions of the ball mill were 7" in depth by 7-5/8" in diameter. The ball charge was made up as follows:

<u>Size in.</u>	<u>Ball Charge No.</u>	<u>Weight gm</u>
+1-1/2	5	1,154
-1-1/2 +1-1/4	10	1,573
-1-1/4 +1	31	2,392
Total	46	5,119

	<u>Chemical Analysis</u>			
	<u>Fe %</u>	<u>Cu %</u>	<u>S %</u>	<u>P %</u>
Head (Composite No. 1)	46.8	0.059	3.08	0.03

EXHIBIT 2

PETROGRAPHIC ANALYSIS

Composite Sample No. 1, "as received", was submitted for petrographic examination to determine the liberation size of the magnetite and chalcopyrite and the essential mineralogy of the gangue material found in the ore.

This composite sample was screened at 48 and 200M, and magnetic and heavy liquid separations were conducted on the resulting two fractions to facilitate the x-ray and petrographic studies on these fractions.

The -48+200M head sample was subjected to a heavy liquid separation with methylene iodide ($d = 3.3$). The lighter material consisted mainly of the gangue material. This was examined in a petrographic study, and for further verification a portion was submitted for determination by x-ray. A wet magnetic separation was made on the +3.3 sp gr fraction to separate the magnetite and pyrrhotite from this fraction. A polished section was then made of the non-magnetic material to determine if pyrite and chalcopyrite were the only metallic minerals present.

A wet magnetic separation was made on the -200M sample to separate the magnetite and pyrrhotite from this fraction. By elutriation, the nonmagnetic fraction was subsequently separated into two fractions; the light minerals resulting from this separation were submitted for analysis by x-ray diffraction.

A polished section of randomly picked +10M material was made for a liberation size study of the ore minerals. The petrographic study of the translucent minerals was done through oil immersion examination of crushed fragments.

Results

Study of the polished section showed the opaque mineralogy of the composite sample to consist of magnetite, chalcopyrite, pyrite and pyrrhotite.

A petrographic study of the -3.3 sp gr fraction showed the translucent minerals to be augite, diopside, calcite, feldspar (andesine), and a trace of quartz. The x-ray analysis of the elutriation "lights" showed essentially the same composition as the coarser fraction.

The liberation size study is based on a 317 grain count and is contained in the following tables:

EXHIBIT 2

Petrographic Analysis (continued)

Table 1

Size Distribution of Magnetite Particles in Gangue

<u>Mesh</u>	<u>A Number of Grains</u>	<u>B Units</u>	<u>A x B</u>	<u>Weight %</u>	<u>Cumulative %</u>
-14 +20	1	50	50	18.4	18
-20 +28	2	25	50	18.4	37
-28 +35	2	12.5	25	9.2	46
-35 +48	4	6.2	24.4	9.0	55
-48 +65	11	3.1	34.1	12.6	68
-65 +100	15	1.5	22.5	8.1	76
-100 +150	24	0.7	16.8	6.2	82
-150 +200	15	0.3	45.0	16.6	98.5
-200 +270	22	0.1	2.2	0.8	99.3
-270 +400	13	0.05	0.7	0.3	99.6
-400 +26 μ	32	0.02	0.6	0.2	99.8
-26 μ +18 μ	21	0.01	0.2	0.1	99.9
-18 μ	11	0.005	0.06	0.02	
			<u>271.56</u>		

EXHIBIT 2

Petrographic Analysis (continued)

Table 2

Size Distribution of Gangue Particles in Magnetite

<u>Mesh</u>	<u>A Number of Grains</u>	<u>B Units</u>	<u>A x B</u>	<u>Weight %</u>	<u>Cumulative %</u>
-35 +48	1	50.0	50.0	15.3	15
-48 +65	2	25.0	50.0	15.3	31
-65 +100	5	12.5	62.5	19.1	50
-100 +150	9	6.2	55.8	17.0	67
-150 +200	13	3.1	40.3	12.3	79
-200 +270	26	1.5	39.0	11.9	91
-270 +400	24	0.7	16.8	5.1	96
-400 +26 μ	36	0.3	10.8	3.2	99.2
-26 μ +18 μ	21	0.1	2.1	0.6	99.8
-18 μ	7	0.05	0.4	0.1	99.9
			<u>327.7</u>		

Discussion

Chalcopyrite generally was found occurring locked in the magnetite grains and not occurring in the gangue minerals. As observed from the tables, the ore minerals were found to have a coarser size distribution than the gangue minerals, but the relative abundance of the two categories was not determined. A point count was not made of the gangue and the ore minerals, as this would have required numerous polished sections to have been made and studied to have representative sampling.

Conclusions

Magnetite grains locked in the gangue are 98% liberated at -150M. The gangue minerals locked in the magnetite are only 79% liberated at -150M.

The chalcopyrite grains counted did not occur in sufficient amount to have a valid size distribution, however, it is significant to note that none larger than 104 μ or -150M were observed.

EXHIBIT 2

Petrographic Analysis (continued)

The essential mineralogy of this iron-copper ore appears to be magnetite, chalcopyrite, pyrite, pyrrhotite, augite, diopside, calcite, and andesine, a plagioclase feldspar.

A sample of the magnetic concentrate obtained from Flotation Test No. 2 was submitted for petrographic examination to determine the nature of the sulfide contaminant.

A mineralographic modal analysis of this sample gave the results as listed in the following table.

<u>Modal Analysis</u>	
<u>Mineral</u>	<u>Approximate Weight %</u>
Magnetite	91.5
Pyrrhotite (locked)	0.6
Pyrrhotite (liberated)	0.0
Pyrite (locked)	3.4
Pyrite (liberated)	2.1
Gangue	2.5

The locked pyrite and pyrrhotite ranged in size from 14 microns to 56 microns. The average size of the liberated pyrite was found to be approximately 14 microns.

EXHIBIT 3

DAVIS TUBE TESTS

Davis Tube Test No. 1

Purpose: To obtain the greatest concentrate grade of a magnetic concentrate.

Sample: Two 15-gram charges of Composite Sample No. 1.

Procedure: A charge of 260 grams of -20M Composite Sample No. 1 was placed into a laboratory ball mill and water was added to make a 50% solids pulp. The mill was turned at 74 rpm for 3 min. The charge was then filtered and dried at 150°F.

Two 15-gram samples of the ground material were placed into the Davis tube. The tube was operated at 100 strokes/min at a current of 1.6 amp. 0.32 gal./min of water was used to wash the nonmagnetic material free from the magnetic material. Each charge was washed for 10 min.

See Screen Test No. 1.

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	Cu %	S %	Fe	Cu	S
Feed (assayed)		46.8	0.059	3.08			
Feed (calculated)	100.0	47.3	0.061	3.27	100.0	100.0	100.0
Mag. Concentrate	67.8	62.4	0.016	2.58	89.5	17.7	53.4
Tail	32.2	15.4	0.157	4.73	10.5	82.3	46.6

EXHIBIT 3

Davis Tube Test No. 2

Purpose: To obtain the greatest concentrate grade of a magnetic concentrate.

Sample: Two 15-gram charges of Composite Sample No. 1.

Procedure: The procedure was the same as for Davis Tube Test No. 1 except that the laboratory ball mill was turned for 10 min.

See Screen Test No. 2.

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	Cu %	S %	Fe	Cu	S
Feed (assayed)		46.8	0.059	3.08			
Feed (calculated)	100.0	47.4	0.059	3.35	100.0	100.0	100.0
Mag. Concentrate	59.5	67.8	0.006	1.92	85.0	6.0	34.1
Tail	40.5	17.5	0.137	5.45	15.0	94.0	65.9

EXHIBIT 3

Davis Tube Test No. 3

Purpose: To obtain the greatest concentrate grade of a magnetic concentrate.

Sample: Two 15-gram charges of Composite Sample No. 1.

Procedure: The procedure was the same as for Davis Tube Test No. 1 except the ball mill was turned for 25 min.

See Screen Test No. 3.

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	Cu %	S %	Fe	Cu	S
Feed (assayed)		46.8	0.059	3.08			
Feed (calculated)	100.0	47.6	0.059	3.23	100.0	100.0	100.0
Mag. Concentrate	59.2	69.4	0.005	2.52	86.4	5.0	46.3
Tail	40.8	15.9	0.138	4.25	13.6	95.0	53.7

EXHIBIT 4

FLOTATION TESTS

Flotation Test No. 1

Purpose: To obtain a bulk sulfide float which would lower the sulfur content of the magnetic material in subsequent magnetic separation. Refer to flowsheet and Screen Test No. 4.

Sample: 500 grams of -20M Composite Sample No. 1.

Test Conditions:

	Time min	Solids %	pH		Reagents, Pounds per Ton of Ore		
			Start	Finish	Z-6	MIBC	NaCN
Grind	20	50					
Cond	2	20			0.02	0.006	
Flotn	7			8.7			
Clnr Cond	2		10.1			0.003	0.005
Clnr Flotn	6						
Ro Tail							
(Mag. Separation)							

Test Notes: Golden tap water was used. $\text{Ca}(\text{OH})_2$ was used for pH control.

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)		46.8	3.08	0.059			
Feed (calculated)	100.0	47.8	3.23		100.0	100.0	
Primary Flotn							
Clnr Float	0.8	41.9	25.7	4.65	0.7	6.2	62.0 ⁽¹⁾
Clnr Nonfloat	3.0	38.0	22.4		2.4	20.8	
Mag. Separation							
(Primary Flotn Nonfloat)							
Mag. Concentrate	64.5	65.6	2.95		88.5	58.9	
Mag. Tail	31.7	12.6	1.43		8.4	14.1	

^{1/} Calculated value.

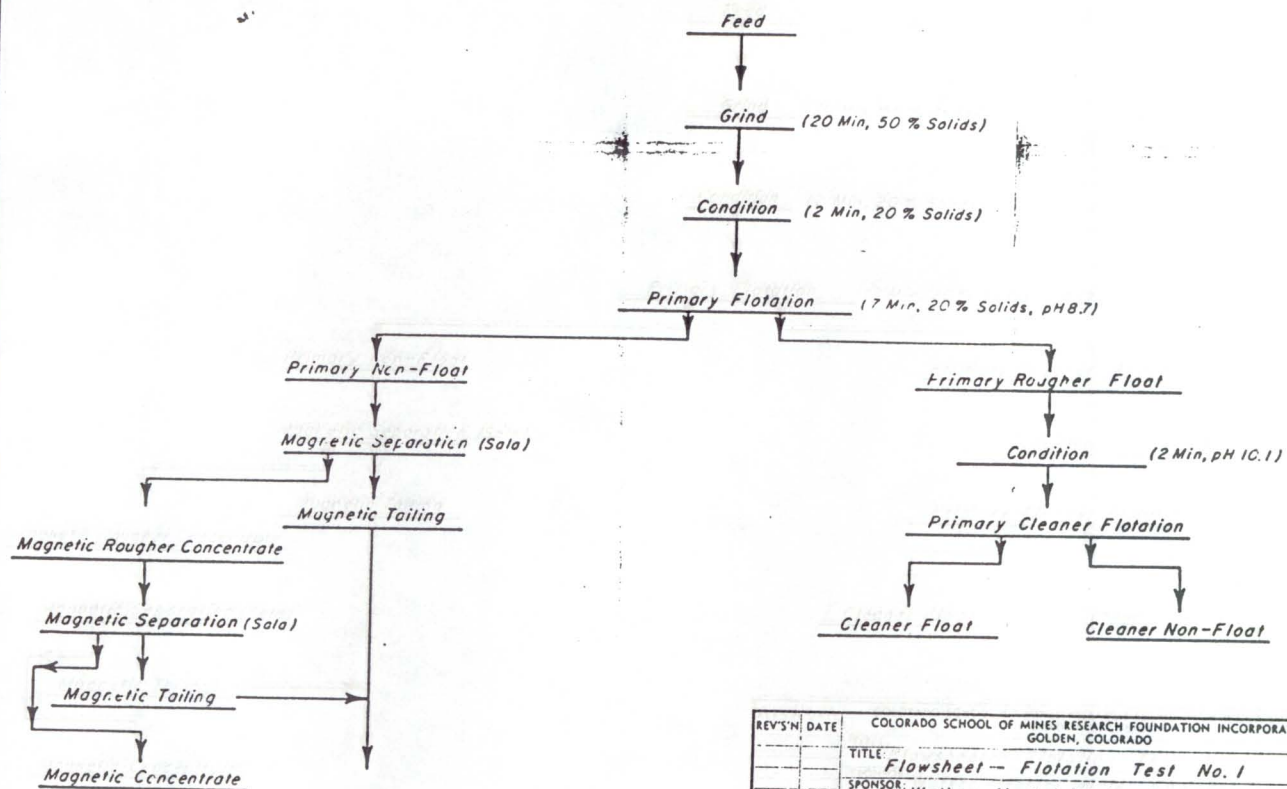
Observations: Grind approx 83% -200M.

A-

B-

C-

D-



REV'S N	DATE	COLORADO SCHOOL OF MINES RESEARCH FOUNDATION INCORPORATED GOLDEN, COLORADO	
		TITLE <i>Flowsheet -- Flotation Test No. 1</i>	
		SPONSOR: <i>Walker-Martel Mining Company</i>	
		SCALE: <i>None</i>	DATE: <i>4-5-66</i>
		DRAWN BY: <i>F.T.M.</i>	PROJECT NO.: <i>260211</i>
		CHECKED BY:	SHEET NO.: OF
		APPROVED BY:	DRW. NO.:

EXHIBIT 4

Flotation Test No. 2

Purpose: Similar to Flotation Test No. 1 except to reduce the pH in primary flotation to neutral for bulk sulfide flotation. The magnetic concentrate was also given a flotation step to remove sulfide material.

Refer to flowsheet.

Sample: 500 grams of -20M Composite Sample No. 1.

Test Conditions:

	Time min	Solids %	pH		Reagents, Pounds per Ton of Ore					Pine Oil
			Start	Finish	Z-6	MIBC	NaCN	Na ₂ S		
Grind	20	50								
Cond	2	20	7.0	7.6	0.02	0.006				
Flotn	5			8.5						
Clnr Cond	2		10.0			0.003	0.005			
Clnr Flotn	5			9.2						
Mag. Separation										
Mag. Concentrate Cond	2	20		9.9				0.5		
Cond	3	20	6.5		0.05					0.006
Flotn	5			8.0						
Clnr Cond	2		7.0		0.01					
Clnr Flotn	3			8.2						

Test Notes: Golden tap water was used. $\text{Ca}(\text{OH})_2$ and H_2SO_4 were used for pH control.

Results: See following page.

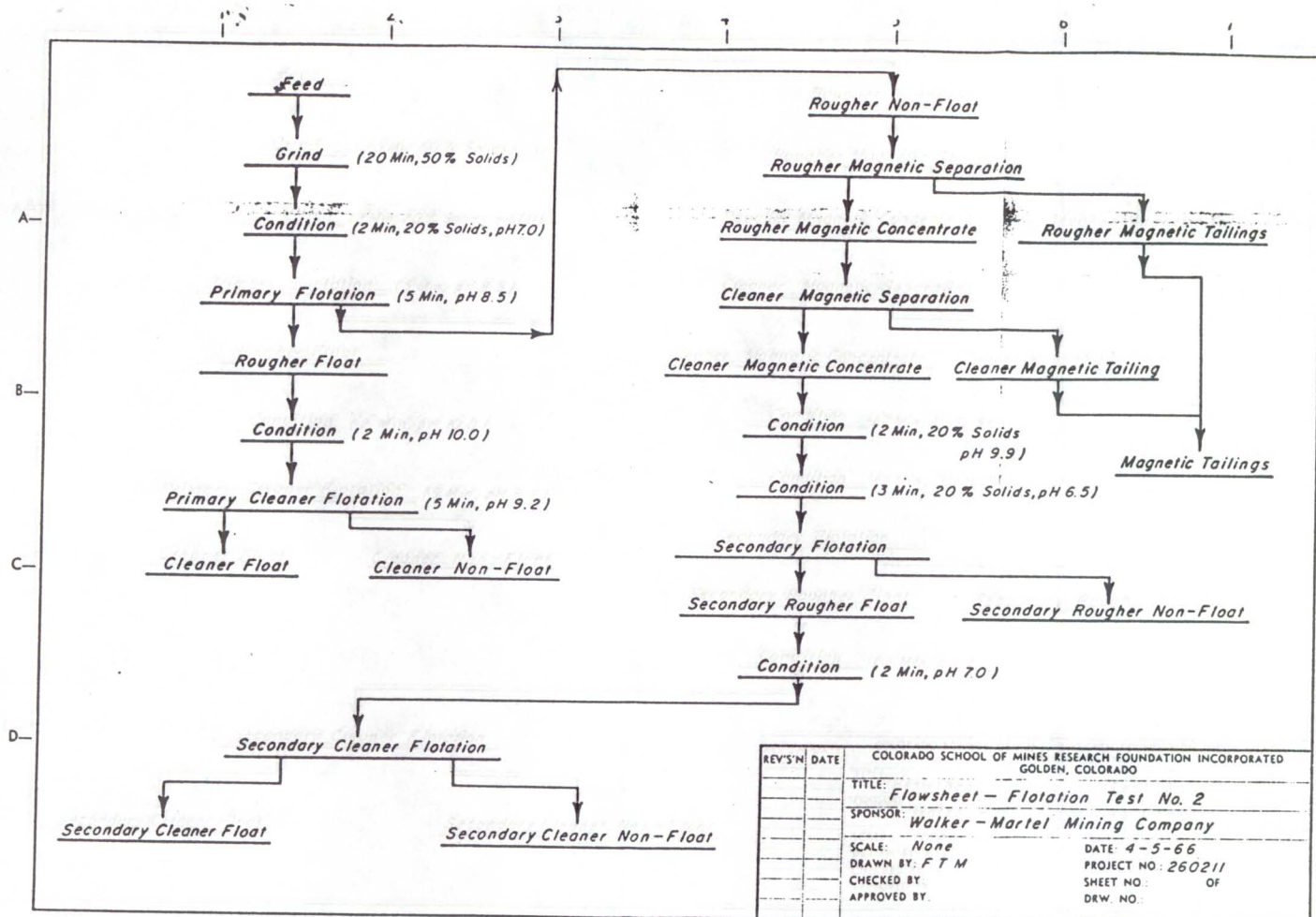
EXHIBIT 4

Flotation Test No. 2 (continued)

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)		46.8	3.08	0.059			
Feed (calculated)	100.0	45.8	3.27		100.0	100.0	
Primary Flotn							
Clnr Float	1.3	42.4	25.4	3.62	1.2	10.4	82.6 ⁽¹⁾
Clnr Nonfloat	5.2	41.7	15.8		1.8	25.1	
Mag. Separation (Primary Flotn Nonfloat)							
Nonmag.	30.1	12.2	1.35		8.0	12.4	
Secondary Flotn (Mag. Concentrate)							
Clnr Float	2.7	61.7	21.5		3.6	17.8	
Clnr Nonfloat	1.5	64.3	9.80		2.1	4.4	
Ro Nonfloat	59.2	64.4	1.65		83.3	29.9	

1/ Calculated value.



REV'S	DATE	COLORADO SCHOOL OF MINES RESEARCH FOUNDATION INCORPORATED GOLDEN, COLORADO	
		TITLE:	Flowsheet - Flotation Test No. 2
		SPONSOR:	Walker - Martel Mining Company
		SCALE:	None
		DRAWN BY:	FTM
		CHECKED BY:	
		APPROVED BY:	
		DATE:	4-5-66
		PROJECT NO:	260211
		SHEET NO:	OF
		DRW. NO:	

EXHIBIT 4

Flotation Test No. 3

Purpose: To separate the magnetic material from the nonmagnetic prior to flotation of both fractions for copper recovery and sulfur removal.

Refer to flowsheet.

Sample: 1250 grams of Composite Sample No. 1 (3 grinds were used to produce the material).

Test Conditions:

	Time min	Solids %	pH		Reagents, Pounds per Ton of Flotation Feed			
			Start	Finish	Z.-6	CuSO ₄	Pine Oil	NaCN
Grind	20	50						
Mag. Separation								
Mag. Fraction								
Cond	2	30	3.9			0.2		
Cond	1	30						
Flotn	3	30		5.8	0.05		0.07	
Cond	1	30	4.1					
Flotn	3	30		5.3	0.05		0.035	
Nonmag. Fraction								
Cond	2	20	9.2					
Flotn	5	20			0.015		0.05	
Clnr Cond	2		10.2	8.3				
Clnr Flotn	3							
2nd Clnr Cond	2		10.4	10.2				
2nd Clnr Flotn								0.05

Test Notes: Golden tap water was used. Magnetic fraction was demagnetized prior to flotation. H₂SO₄ and lime were used for pH control.

Results: See following page.

EXHIBIT 4

Flotation Test No. 3 (continued)

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)		46.8	3.08	0.059			
Feed (calculated)	100.0	46.9	2.02		100.0	100.0	
Mag. Separation							
Primary Flotn (Mag. Fraction)							
Ro Nonfloat	56.7	65.3	0.20		78.9	5.6	
Ro and Scav Float	8.8	59.7	12.0		11.2	52.6	
Secondary Flotn (Nonmag. Fraction)							
Clnr Float	0.5	30.4	21.8	12.6	0.3	5.1	100.0 ⁽¹⁾
2nd Clnr Nonfloat	0.4	28.2	21.2	2.46	0.2	4.2	15.0 ⁽¹⁾
1st Clnr Nonfloat	4.0	24.1	7.64		2.1	15.2	
Ro Nonfloat	29.6	11.5	1.18		7.3	17.3	

^{1/} Calculated values.

Observations: Calculated sulfur content of feed is low; however, calculated sulfur content of magnetic product is 1.78, a reasonable amount.

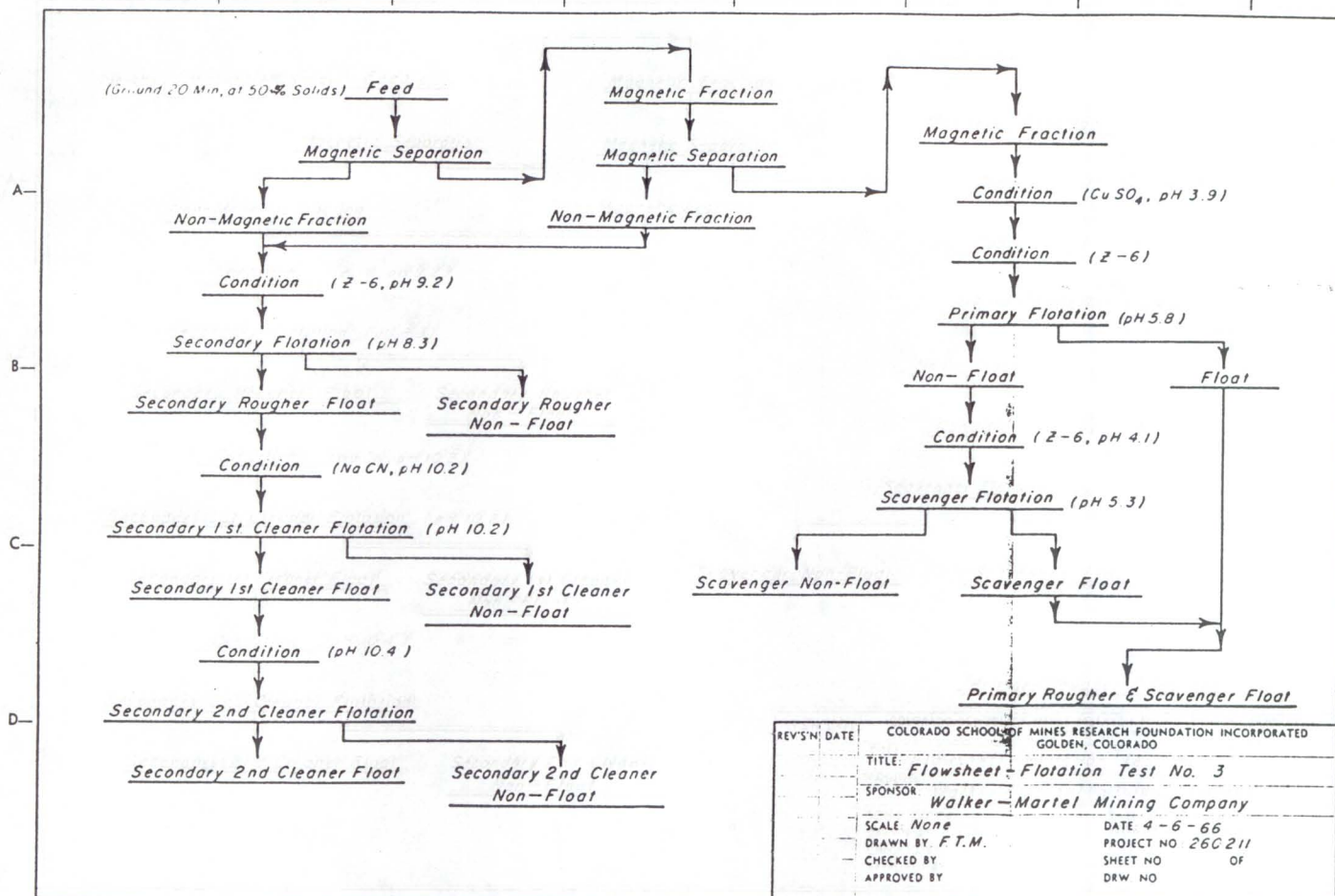


EXHIBIT 4

Flotation Test No. 4

Purpose: Similar to Flotation Test No. 2 with attempt to improve the grade of magnetic concentrate by regrinding prior to flotation for sulfur removal. Refer to flowsheet.

Sample: 500 grams of Composite Sample No. 1.

Test Conditions:

	Time min	Solids %	pH		Reagents, Pounds per Ton of Ore		
			Start	Finish	Z-6	CuSO ₄	Pine Oil
Grind	20	50					
Cond	2	20	8.1		0.02		0.07
Flotn	5	20		7.8			
Cond	1	20			0.01		
Flotn	3	20		7.8			
Mag. Separation on Nonfloat Product							
Mag. Conc							
Grind	30	50					
Cond	2	15	5.8			0.1	
Cond	2	15			0.01		0.035
Flotn	5	15		6.3			
Cond	2	15			0.01		
Flotn		15		6.8			

Test Notes: Golden tap water was used. H₂SO₄ used for pH control.

Results: See following page.

EXHIBIT 4

Flotation Test No. 4 (continued)

Results:

Product	Weight %	Chemical Analysis			Per Cent Distribution		
		Fe %	S %	Cu %	Fe	S	Cu
Feed (assayed)		46.8	3.08	0.059			
Feed (calculated)	100.0	47.3	3.64		100.0	100.0	
Primary Flotn Ro and Scav Float	9.8	35.4	15.2	0.58	7.3	40.9	96.3 ⁽¹⁾
Mag. Separation (Scav Nonfloat) Nonmag. Fraction	27.2	11.2	0.89		6.5	6.6	
Secondary Flotn (Mag. Fraction) Ro and Scav Float	7.0	58.3	3.85		8.6	7.4	
Nonfloat	56.0	65.6	2.93		77.6	45.1	

^{1/} Calculated value.

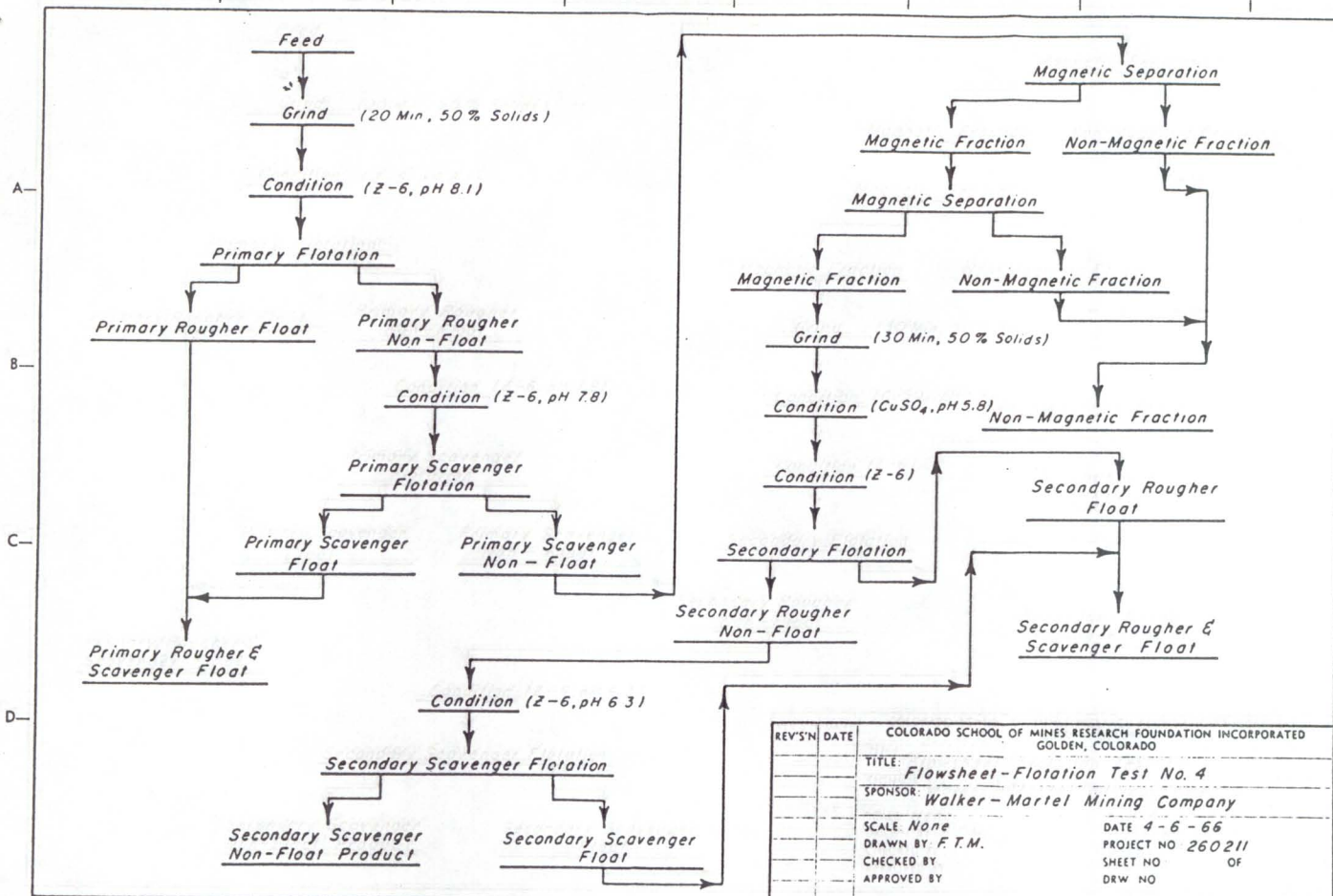


EXHIBIT 5

ROASTING TEST

Roasting Test No. 1

Purpose: To reduce the sulfur content of a magnetic concentrate by roasting.

Sample: Three charges of 500 grams of Composite Sample No. 1, -20M.

Procedure: The charge was ground in the laboratory ball mill for 30 min and then a magnetic separation was made using the Sala. The magnetic concentrate was given a magnetic cleaning step with the nonmagnetic fractions discarded.

The magnetic fraction was dried and a sample split for head analysis. The remaining concentrate was blended with 0.5% -100M bentonite and pelletized using a laboratory disk pelletizer. The pellets were sized between 0.525" and 0.371" and then dried for 3-1/2 hr at 250°F. Four charges of pellets were placed into the muffle furnace at 1500°F. The temperature was raised to 2250°F in 2 hr and the first dish was taken out. The remaining dishes were removed at 15 min intervals. Air was introduced to the last dish for 15 min and the temperature dropped to 2200°F.

Results:

<u>Product</u>	<u>Increase Weight %</u>	<u>Chemical Analysis S %</u>	<u>Per Cent Removal S</u>
Feed		3.50	
Dish No. 1 (0 min at 2250°F)	1.3	0.006	99.8
Dish No. 2 (15 min at 2250°F)	0.6	0.006	99.8
Dish No. 3 (30 min at 2250°F)	0.1	0.004	99.9
Dish No. 4 (45 min at 2250°-2200°F)	0.5	0.004	99.9

Observations: The fired pellets were black, but when pulverized, the ground material was purple or hematite was evident.

EXHIBIT 6

SCREEN TESTS

Screen Test No. 1

Sample: 130 grams of the feed to Davis Tube Test No. 1.

Procedure: The sample was wet screened on a 325M screen and the +325M fraction was dried and dry screened using the Ro-Tap.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Passing
Feed	100.0		
-20 +28	0.2	0.2	100.0
-28 +35	0.8	1.0	99.8
-35 +48	4.1	5.1	99.0
-48 +65	8.5	13.6	94.9
-65 +100	14.9	28.5	86.4
-100 +150	16.0	44.5	71.5
-150 +200	11.8	56.3	55.5
-200 +270	7.6	63.9	43.7
-270 +325	7.2	71.1	36.1
-325	28.9	100.0	28.9

EXHIBIT 6

Screen Test No. 2

Sample: 130 grams of the feed to Davis Tube Test No. 2.

Procedure: Same as Screen Test No. 1.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Cumulative Passing
Feed	100.0		
-35 +48	Tr		
-48 +65	0.1	0.1	100.0
-65 +100	0.9	1.0	99.9
-100 +150	5.8	6.8	99.0
-150 +200	11.9	18.7	93.2
-200 +270	11.7	30.4	81.3
-270 +325	13.0	43.4	69.6
-325	56.6	100.0	56.6

EXHIBIT 6

Screen Test No. 3

Sample: 130 grams of the feed to Davis Tube Test No. 3.

Procedure: Same as Screen Test No. 1.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Passing
Feed	100.0		
-65 +100	Tr		
-100 +150	0.2	0.2	100.0
-150 +200	1.6	1.8	99.8
-200 +270	3.9	5.7	98.2
-270 +325	8.7	14.4	94.3
-325	85.6	100.0	85.6

EXHIBIT 6

Screen Test No. 4

Sample: 500 grams -20M Composite Sample No. 1.

Procedure: The sample was charged to the ball mill with 500 cc of H₂O. The mill was turned at 74 rpm for 20 min. The charge was then wet screened on a 325M and the +325M fraction was then dry screened using the Ro-Tap for 25 min.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Cumulative Passing
Feed	100.0		
+100	0.6	0.6	100.0
-100 +150	4.3	4.9	99.4
-150 +200	12.4	17.3	95.1
-200 +270	12.2	29.5	82.7
-270 +325	14.8	44.3	70.5
-325	55.7	100.0	55.7

EXHIBIT 6

Screen Test No. 5

Sample: 500 grams of -20M Composite Sample No. 1.

Procedure: Same as Screen Test No. 4 except the grind was for 30 min.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Passing
Feed	100.0		
+100	0.1	0.1	100.0
-100 +150	1.1	1.2	99.9
-150 +200	5.4	6.6	98.8
-200 +270	8.4	15.0	93.4
-270 +325	12.9	27.9	85.0
-325	72.1	100.0	72.1

EXHIBIT 6

Screen Test No. 6

Sample: 500 grams of -20M Composite Sample No. 1.

Procedure: Same as Screen Test No. 4 except the grind was for 40 min.

Results:

Screen Product (Tyler) Mesh	Weight %		
	Direct	Cumulative Retained	Passing
Feed	100.0		
+100	<0.1	<0.1	100.0
-100 +150	0.4	0.5	99.9
-150 +200	2.8	3.3	99.5
-200 +270	5.8	9.1	96.7
-270 +325	12.2	21.3	90.9
-325	78.7	100.0	78.7

EXHIBIT 6

Screen Test No. 7

Sample: 83.2 grams of magnetic concentrate produced as pellet feed for Roasting Test No. 1:

Procedure: The sample was wet screened on a 325M screen and the plus material was dry screened using a Ro-Tap for 25 min.

Results:

Screen Product (Tyler) Mesh	Weight %	
	Direct	Cumulative Retained
Feed	100.0	
+150	0.1	0.1
-150 +200	6.5	6.6
-200 +270	8.5	15.1
-270 +325	12.8	27.9
-325	72.1	100.0

EXHIBIT 6

Screen Test No. 8

Sample: 245.2 grams of magnetic concentrate from Flotation Test No. 4 (regrind product).

Procedure: The sample was wet screened on a 325M screen and the plus material dry screened using a Ro-Tap for 25 min.

Results:

<u>Screen Product (Tyler) Mesh</u>	<u>Weight %</u>	
	<u>Direct</u>	<u>Cumulative Retained</u>
Feed	100.0	
+150	0.0	0.0
-150 +200	0.1	0.1
-200 +270	0.1	0.2
-270 +325	0.5	0.7
-325	99.3	100.0

EXHIBIT 7

BALL MILL GRINDABILITY TEST

Ball Mill Grindability Test No. 1

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number.

Sample: Composite Sample No. 1 (-1/2" sample reduced to 6M).

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test

Conditions: Mesh of grind: 200

Weight of undersize product for 250% circulating load: 546.8 gm

Weight % of undersize material in ball mill feed: 13.7

Results:

Stage No.	New Feed gm	Undersize		Revolutions	Undersize in Product gm	Undersize Produced	
		In Feed gm	To Be Ground gm			Total gm	Per Mill Revolution gm
1	1,913.7	262.2	284.6	200	459.7	197.5	0.988
2	459.7	63.0	483.8	490	552.3	489.3	0.999
3	552.3	75.6	471.2	472	575.9	500.3	1.060
4	575.9	78.9	467.9	441	541.7	462.8	1.049
5	541.7	74.2	472.6	451	566.2	492.0	1.091
6	566.2	77.6	469.2	430	542.5	464.9	1.081
7	542.5	74.3	472.5	437	546.7	472.4	1.081

Average last three = 1.084

EXHIBIT 7

Ball Mill Grindability Test No. 1 (continued)

Ball Mill Work Index Computations

$$Wi = \frac{44.5}{P_1^{0.23} \times Gbp^{0.82} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right)}$$

Wherein: P_1 = 100% Passing Size of Product = 74.2 microns
 Gbp = Grams per Revolution = 1.084
 P = 80% Passing Size of Product = 50 microns
 F = 80% Passing Size of Feed = 1,940 microns

$$Wi = 13.03$$

EXHIBIT 7

Ball Mill Grindability Test No. 1 (continued)

Screen Analyses of Test Feed.

Procedure: Sample was screened dry on a Ro-Tap for 30 min on screens listed below.

Results:

Screen Product (Tyler) Mesh	Weight %	
	Direct	Cumulative Passing
-6 +8	13.5	100.0
-8 +14	30.0	86.5
-14 +28	17.4	56.5
-28 +48	9.9	39.1
-48 +100	8.2	29.2
-100 +200	7.3	21.0
-200	13.7	13.7

