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PROGRESS REPORT

PRELIMINARY METALLURIGICAL TESTS

JOHNSON CAMP VENTURE #2

JAN. 18, 1973

PROGRESS REPORT

Preliminary Metallurgical Tests

Johnson Camp Venture #2

For:

QUINTANA MINERALS CORPORATION
1892 West Grant Road
Tucson, Arizona 85705

Submitted by:

MOUNTAIN STATES RESEARCH & DEVELOPMENT
Post Office Box 17960
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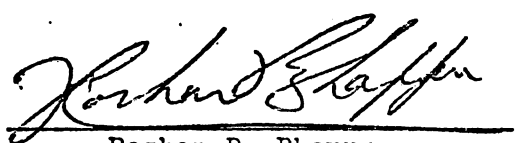
Project No. 2019

January 18, 1973

By 

Roshan B. Bhappu
Vice President and
General Manager

PROGRESS REPORT

Preliminary Metallurgical Tests

Johnson Camp Venture #2

INTRODUCTION

At the request of Mr. William E. Saegart, President, Quintana Minerals Corporation, the Mountain States Research and Development initiated a metallurgical test program on selected samples representing copper ore from Johnson Camp Venture #2 deposit.

The scope of this initial metallurgical test program was as follows:

1. Conduct initial leaching tests on coarse reject samples from Hole T-2. These samples to be grouped into four lots corresponding to various categories of copper and gangue mineralogy. The drill hole groups of these four lots are as follows:

Group No.	Footage Interval	$CuSiO_3/CuO$	Gangue Type
I	582' - 811'	2/1	Marble-moderate to strong lime
II	884.5' - 990.5'	2/3	Diopside-weak to moderate lime
III	990.5' - 1051'	2/3	Garnet-moderate lime

Group No.	Footage Interval	$CuSiO_3/CuO$	Gangue Type
IV	1051' - 1270'	1/1	Garnet and Shale-weak to moderate lime

The leaching test program would include investigation of both acid and ammonia solvents to determine copper extraction, rate of leaching, and reagent consumption.

2. MSRD to conduct a study consisting of check assaying in order to resolve the discrepancies in analyses reported by Southwestern Assayers and Chemists due to inconsistent atomic absorption calibration.

3. Set up chemical analysis control based on umpire assaying at three different laboratories in order to determine the accuracy of the assayers' calibration upon receipt of the assay reports.

4. A detailed mineralogical examination on selected core samples by Laszlo Dudas.

MINERALOGICAL EXAMINATION

A detailed mineralogical examination was conducted on selected core samples by Laszlo Dudas. Mr. Dudas' report is attached to this report as Appendix I.

ANALYTICAL STUDIES

The analytical studies as outlined under items (2) and (3) of the Scope were conducted by Mr. Frank Tindall, our Analytical Consultant, and the necessary

data and analyses were submitted to the Sponsor as soon as they were available.

EXPERIMENTAL WORK AND RESULTS

Standard laboratory agitation tests were run on four groups of representative samples. The major variables investigated were as follows:

(1) Sulfuric acid concentration at pH=1.0 and 1.5.

(2) Sulfuric acid plus auxilliary reagents such as H_2O_2 (oxidizer), phosphate (P_2O_5), and fluoride (F).

(3) Ammonia with carbonate (Cu_2), sulfate and oxidizer ($CuSO_4$) - both hot and cold.

The results of these tests are reported in attached Leach Test Data sheets and are summarized in Table I.

DISCUSSION OF RESULTS

1. The detailed mineralogical examination of the ore samples (as reported by Mr. Dudas) reveal that the predominent oxide copper minerals are chrysocolla and copper bearing layered silicates. About 60% of the total copper is contained in the hydrated copper silicate (chrysocolla) which is readily solubilized by acid. The rest of the copper content is tied up in layered silicates and this metal content will be difficult to extract. With excess of acid, extraction in the range of 70 to 75% may be achieved. However, extractions above these levels would be difficult to attain with acid alone and would require drastic treatment such as preheating to a critical temperature.

Since marble is the predominent gangue mineral in the first three groups of samples, the acid consumption would be quite high. For this reason, ammonia leaching would be the preferred solvent for the ore samples under investigation. However, because of the nature of copper mineralization as outlined above, pretreatment of the ore samples (such as heating) may be necessary for successful leaching with ammonia.

2. The results of these initial studies indicate that the composite sample representing Group I is the highest acid consumer with an acid consumption of about 555 (pH=1.5) to 761 (pH=1.0) pounds H_2SO_4 per ton of ore leached. The copper extraction in these cases amounted to about 65 to 86% respectively.

It is obvious that a practical acid leaching process based on such high acid consumption, even with a relatively low-cost acid, would not be economically or technically feasible (due to the production of a large amount of detrimental gypsum, $CaSO_4$).

Since Group I type ore represents a substantial resource of the total reserve in the deposit, it is essential that an effective treatment process for this refractory ore be determined first in order to sustain successful exploitation of the deposit under investigation.

3. The copper extraction by acid leaching of Group II ore amounts to about 61 and 72% with acid consumptions of approximately 154 (pH=1.0) and 114 (pH=1.5) pounds H_2SO_4 per ton respectively. These figures represent acid consumptions of 24.3 and 27.2 pounds H_2SO_4 per pound of copper leached. Such consumptions may or may not sustain an economically feasible leaching operation depending on the price of acid prevailing at the time of operation.

4. In the case of Group III sample, the acid consumptions are about 134 (pH=1.5) and 185 (pH=1.0) pounds H_2SO_4 per ton of ore leached, with copper extraction of about 65 and 74% respectively. This represents acid consumptions of 21.5 and 25.3 pounds H_2SO_4 per pound of copper extracted.

It is evident that this Group III ore resembles the Group II ore from economic as well as operational viewpoints, and for all practical purposes these two groups may be combined into a composite sample for further leaching studies.

5. Finally, Group IV ore sample indicates acid consumptions of about 77 (pH=1.5) and 112 (pH=1.0) pounds H_2SO_4 per ton of ore leached with corresponding copper extractions of approximately 70 and 73% respectively. At these rates, the acid consumptions per pound of copper leached amount to 9.8 and 14.4 pounds H_2SO_4 respectively.

The above acid consumption figures are relatively lower than those of the other three groups and does represent economically favorable conditions for a practical operation.

6. Auxilliary reagents such as H_2O_2 (oxidant), phosphates or fluorides added to the acid leach systems did not appear to exert a beneficial effect on extraction nor do they help in reducing acid consumption.

7. In general, ammonia leaching, either hot or cold, with a reagent combination including ammonium carbonate, hydroxide, sulfates, etc., does not appear to be effective for extracting copper from all four groups of samples. The copper extractions in these tests varied from a low of 9.2% to a high of 16.5%.

The above results clearly reveal that the treatment of these high acid consuming ore samples using ammoniacal solutions will not be successful. This is in accord with the results of other investigations concerning ammonia leaching of oxide-copper ores. However, there is a good possibility that such ores can be treated effectively by pre-heating the ores to a critical temperature followed by ammonia leaching. In this case the optimum temperature range is about 700 to 900°F.

RECOMMENDATIONS

1. Since Group I type ore represents a major portion of the ore deposit and reserves, further research efforts should be concentrated only on this sample for determing the optimum treatment procedure. There is a good chance that such a procedure may be applicable to samples representing the other three groups.

2. Investigate the effectiveness of preheating of Group I samples prior to ammonia leaching. The temperature range to be investigated should include from 300°F to 900°F. Acid and ammonia leaching of totally calcined samples should also be examined.

3. A combination of carbonate (marble or calcite) flotation and acid leaching of flotation tailings should be evaluated with the aim of lowering the acid consumption.

4. If anyone of the above tests are successful on Group I samples, the optimum procedure should be evaluated for the treatment of samples of the other three groups.

Table I:
AGITATION LEACH TESTS

200 gram samples

50% solids

Sample/ Test No.	Reagents and Concentration	Extraction, % T. Cu	Extraction, % Ox. Cu	Acid Consumption lbs/ton	Acid Consumption lbs/lb Cu	Preg Soln gpl	Leach Residue T. Cu	Leach Residue Ox Cu	Calc. Heads
DF - 1 Group I	H_2SO_4, pH 1.0	81.10	86.37	761.2	64.02	2.75	.14	.08	.73
DF - 2 Group I	H_2SO_4, pH 1.5	54.88	64.91	555.0	77.19	2.01	.29	.20	.655
DF - 3 Group II	H_2SO_4, pH 1.0	72.32	81.00	154.4	27.21	2.02	.11	.05	.39
DF - 4 Group II	H_2SO_4, pH 1.5	61.18	65.76	114.3	24.32	1.38	.15	.09	.38
DF - 5 Group III	H_2SO_4, pH 1.0	73.96	86.59	185.4	25.28	2.09	.13	.05	.49
DF - 6 Group III	H_2SO_4, pH 1.5	64.77	64.73	134.6	21.53	1.81	.17	.13	.48
DF - 7 Group IV	H_2SO_4, pH 1.0	73.73	72.50	112.1	14.41	2.64	.14	.10	.53
DF - 8 Group IV	H_2SO_4, pH 1.5	70.17	75.41	77.2	9.82	2.47	.17	.09	.56
DF - 9 Group II	NH_3 - 50 gpl CO_2 - 15 gpl SO_4 - 15 gpl	9.80	0.00	--	--	0.28	.28	.26	.31
DF - 10 Group II	NH_3 - 50 gpl CO_2 - 25 gpl	4.89	0.00	--	--	0.17	.40	.29	.41
DF - 11 Group II	Ammonium Thio-Sulfate-50 gpl	10.48	0.00	--	--	0.28	.36	.26	.396
DF - 12 Group II	H_2SO_4, pH 1.0 P_2O_2, 5 gpl	80.49	77.10	105.0	21.40	1.72	.06	.06	.30
DF - 13 Group II	H_2SO_4, pH 1.0 HF - 5 gpl	64.68	68.92	154.4	27.75	2.16	.15	.08	.43
DF - 14 Group II	H_2SO_4, pH 1.0 P_2O_2 - 5 gpl HF - 5 gpl	61.36	68.88	154.4	30.04	2.05	.16	.08	.42

Table I:
AGITATION LEACH TESTS

Sample/ Test No.	Reagents and Concentration	Extraction, % T. Cu	Extraction, % Ox. Cu	Acid Consumption lbs/ton	Acid Consumption lbs/lb Cu	Preg Soln gpl	Leach Residue T. Cu	Leach Residue Ox. Cu	Calc. Heads
DF - 15 Group II	Ammonium Bi-flouride, 5gpl Ammonium Carbonate, 15 gpl NH_4OH, 50 gpl	16.46	17.30	--	--	0.46	.25	.22	.30
DF - 16 Group II Condition 15 minutes	H_2SO_4, pH 1.0 P_2O_2, 5 gpl	66.84	77.28	173.0	29.00	1.84	.15	.06	.44
DF - 17 Group III Condition 15 minutes	H_2SO_4, pH 1.0 P_2O_2, 5 gpl	75.55	83.97	164.2	20.66	2.33	.13	.06	.52
DF - 18 Group IV Condition 15 minutes	H_2SO_4, pH 1.0 P_2O_2, 5 gpl	75.15	86.38	93.5	11.25	2.87	.14	.05	.55
DF - 19 Group II	H_2SO_4, pH 1.0 H_2O_2, 50%, 5gpl	72.51	81.01	182.7	31.91	1.49	.11	.05	.39
DF - 20 Group III	H_2SO_4, pH 1.0 H_2O_2, 50%, 5gpl	70.95	81.26	181.8	25.04	1.58	.15	.07	.51
DF - 21 Group IV	H_2SO_4, pH 1.0 H_2O_2, 50%, 5gpl	72.94	83.68	112.9	14.25	2.06	.15	.06	.54
DF - 22 Group I	NH_4CO_2, 25gpl NH_4OH, 50 gpl $CuSO_4$, 1 gpl Cold	9.20	--	--	--	0.402	.56	--	.61
DF - 23 Group I	NH_4CO_2, 25 gpl NH_4OH, 50 gpl Hot	10.95	--	--	--	0.76	.66	--	.73

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group I Test No. DF-1

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent

Concentration 96 Concentration

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	9:00		8.5				
	9:45		1.0	19.2 ml			
	10:15		1.0	2.55			
	10:35		1.0	1.85			
	11:10		1.0	1.90			
	11:40		1.0	1.85			
	12:15		1.0	1.80			
	12:55		1.0	2.05			
	1:15		1.0	1.55	Total Acid Added - 44.00		
	1:45		1.0	1.70			
	5:00		1.0	9.55			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1744	Preg Soln	160 ml	2.75 gpl	0.440	30.0
1736	Wash Soln	640 ml	1.17 gpl	0.749	51.1
1656	Leach Tail	197.8 g	0.14 %	0.277	18.9
				1.466	100.0

Head (calculated) 0.733% T Cu

Head (assayed) 0.66% T Cu

Remarks

Leach Tailing Assay = 0.08% Ox Cu

Accountability = 111%

Extraction 81.1 (%) Reagent Consumption 761.2 lbs acid/ton of ore

64.02 lbs acid/lb. Cu leached

*Actual acid consumption minus free acid.

Performed By

Type Agitation Leach Date November 28, 1972

Project No. 2019 Sample No. Group I Test No. DF-2

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.5 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	9:45		9.7				
	9:55		1.5	18.6 ml			
	10:15		1.5	0.9			
	10:35		1.5	1.15			
	11:10		1.5	1.45			
	11:40		1.5	1.15			
	12:15		1.5	0.70			
	12:55		1.5	0.55	Total Acid Added - 31.65 ml		
	1:15		1.5	0.95			
	1:45		1.5	0.75			
			1.5	5.45			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1745	Preg Soln	155 ml	2.01 gpl	0.311	23.74
737	Wash Soln	560 ml	0.73 gpl	0.408	31.14
1657	Leach Tail	203.6 g	0.29 %	0.59	45.12
				1.31	100.00

Head (calculated) 0.655% T Cu

Head (assayed) 0.66% T Cu

Remarks

Accountability = 99%

Extraction 54.88 (%) Reagent Consumption 555.0 lbs acid/ton of ore
77.19 lbs acid/ lb Cu leached

*Actual Acid consumption minus free acid.

Performed By

Type Agitation Leach Date November 28, 1972

Project No. 2019 Sample No. Group II Test No. DF-3

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature

Reagent H_2SO_4 Auxil. Reagent

Concentration 96.0 % Concentration

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:00		8.4				
	10:05		1.0	6.10 ml			
	10:40		1.0	1.30			
	11:00		1.0	0.20			
	11:40		1.0	0.25			
	12:15		1.0	0.45			
	1:15		1.0	0.35	Total Acid Added - 9.65		
	2.45		1.0	1.00			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1746	Preg Soln	146 ml	2.02 gpl	0.294	37.50
1738	Wash Soln	650 ml	0.42 gpl	0.273	34.82
1658	Leach Tail	197.6 g	0.11 %	0.217	27.68
				0.784	100.00

Head (calculated) 0.392% T Cu

Head (assayed) 0.37% T Cu

Remarks

Leached Tailing Assay - 0.05%Ox Cu

Extraction 72.32 (%) Reagent Consumption 154.4 lbs acid/ton of ore

27.21 lbs acid/lb. Cu leached

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group II Test No. DF-4

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.5 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:10		8.4				
	10:15		1.5	4.5 ml			
	10:45		1.5	0.45			
	11:00		1.5	0.30			
	11:40		1.5	0.45			
	12:15		1.5	0.30			
	12:55		1.5	0.25	Total Acid Added - 6.7 ml		
	1.15		1.5				
	1:45		1.5	0.25			
	2:45		1.5	0.20			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1747	Preg Soln	161 ml	1.38 gpl	0.224	29.02
39	Wash Soln	615 ml	0.40 gpl	0.246	32.16
1659	Leach Tail	198.1 g	0.15 %	0.297	28.82
				0.765	100.00

Head (calculated) 0.382% T Cu

Head (assayed) 0.37% T Cu

Remarks

Leached Tail Ox Cu - 0.09%

Extraction 61.18 (%) Reagent Consumption 114.3 lbs acid/ton of ore
24.32 lbs acid/lb. of Cu leac

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group III Test No. DF-5

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:15		7.9				
	10:25		1.0	7.25 ml			
	10:45		1.0	1.30			
	11:00		1.0	0.65			
	11:40		1.0	0.30			
	12:15		1.0	0.35			
	12:55		1.0	0.15			
	1:15		1.0	Total Acid Added - 11.40 ml			
	1:45		1.0	1.10			
	3:15		1.0	0.30			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1748	Preg Soln	190 ml	2.09 gpl	0.397	40.06
740	Wash Soln	510 ml	0.66 gpl	0.336	33.90
1660	Leach Tail	198.6 g	0.13 %	0.258	26.04
				0.991	100.00

Head (calculated) 0.49% T Cu

Head (assayed) 0.45% T Cu; OxCu 0.37%

Remarks

Leached tails OxCu - 0.05%

Extraction 73.96 (%) Reagent Consumption 185.4 lbs acid/ton of ore

25.28 lbs acid/ lb. of Cu leach

*Actual Acid consumption minus free acid.

Performed By

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group III Test No. DF-6

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature

Reagent H_2SO_4 Auxil. Reagent

Concentration 96 % Concentration

pH 1.5 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:30		7.9				
	10:30		1.5	6.4 ml			
	10:45		1.5	0.50			
	11:00		1.5	0.10			
	11:40		1.5	0.25			
	12:15		1.5	0.20			
	12:55		1.5	0.15			
	1:15		1.5				Total Acid Added - 7.85 ml
	1:45		1.5	0.20			
	3:15		1.5				

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
49	Preg Soln	185 ml	1.81 gpl	0.335	34.72
1741	Wash Soln	500 ml	0.58 gpl	0.290	30.05
60	Leach Tail	200.6 g	0.17 %	0.340	35.23
				0.965	100.00

Head (calculated) 0.48% T Cu

Head (assayed) 0.45% T Cu; Ox Cu 0.37%

Remarks

Leached tails Ox Cu = 0.13%

Extraction 64.77 (%) Reagent Consumption 134.6 lbs acid/ton of ore

21.53 lbs acid/ lb. of Cu leach

*Actual Acid consumption minus free acid

Performed By

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group IV Test No. DF-7

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:00		7.8				
	11:00		1.0	4.05 ml			
	11:15		1.0	1.15			
	11:40		1.0	0.65			
	12:15		1.0	0.35			
	12:55		1.0	0.10			
	1:15		1.0				
	1:45		1.0	0.30			
	3:55		1.0	0.65			
		Total Acid Added -		7.25 ml			

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay ($_{Cu}$)	Content	Distribution (%)
1750	Preg Soln	185 ml	2.64 gpl	0.488	46.25
1.42	Wash Soln	590 ml	0.50 gpl	0.290	27.48
1662	Leach Tail	197.8 g	0.14 %	0.277	26.27
				1.055	100.00

Head (calculated) 0.53% T Cu

Head (assayed) 0.51% T Cu

Remarks

Leached Tails Assay OxCu - 0.10%

Extraction 73.73 (%) Reagent Consumption 112.10 lbs acid/ton of ore

14.41 lbs acid/lb. of Cu leach

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date November 28, 1972

Project No. 2019 Sample No. Group IV Test No. DF-8

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.5 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:00		7.9				
	11:10		1.5	3.80 ml			
	11:40		1.5	0.30			
	12:15		1.5	0.15			
	12:55		1.5	0.10			
	1:15		1.5				
	1:45		1.5	0.10			
	3:55		1.5	0.15			

Total Acid Added - 4.60 ml

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
1751	Preg Soln	190 ml	2.47 gpl	0.469	41.83
1743	Wash Soln	610 ml	0.52 gpl	0.317	28.28
1663	Leach Tail	197.0 g	0.17 %	0.335	29.89
				1.121	100.00

Head (calculated) 0.56% T Cu

Head (assayed) 0.56% T Cu

Remarks

Leached Tails OxCu - 0.09%

Extraction 70.11 (%) Reagent Consumption 77.2 lbs acid/ton of ore

9.82 lbs acid/lb. Cu leache

*Actual acid consumption minus free acid.

Performed By

Type Bottle Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF-9

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent NH_3 - 50 gpl, CO_2 - 15 gpl, SO_4 - 15 gpl Auxil. Reagent Ammonium Carbonate - 35.5 g, Ammonium Persulphate - 6 g, NH_4OH - 148.8 ml

Concentration Concentration

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln.	132 ml	0.28 gpl	0.0370	6.0
Wash Soln.	590 ml	0.04 gpl	0.0236	3.8
Leach Tlg.	198.1 g	0.28 %	0.5547	90.2
Total			0.6153	100.00

Head (calculated) T Cu = 0.307%

Head (assayed) T Cu = 0.37%

Remarks

Leach Tailing Assay = 0.26% Ox Cu

Extraction 9.8 (%) Reagent Consumption

Performed By

Type Bottle Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 10

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent NH_3 - 50 gpl Auxil. Reagent

CO_2 - 25 gpl

Concentration Concentration

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	135 ml	0.17 gpl	0.0229	2.77
Wash Soln	670 ml	0.02 gpl	0.0134	1.62
Leach Tlg.	197.8 g	0.40 %	0.7912	95.61
Totals			0.8275	100.00

Head (calculated) T Cu = 0.414%

Head (assayed) T Cu = 0.37%

Remarks

Leach Tailing Assay = 0.29% Ox Cu

Extraction 4.39 (%) Reagent Consumption

Performed By

Type Bottle Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 11

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent Ammonium Thiosulphate Auxil. Reagent

Concentration 50 gpl Concentration

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	190 ml	0.28 gpl	0.0532	6.70
Wash Soln	750 ml	0.04 gpl	0.0300	3.78
Leach Tlg.	197.4 g	0.36 %	0.7106	89.52
Totals			0.7938	100.00

Head (calculated) T Cu = 0.396%

Head (assayed) T Cu = 0.37%

Remarks

Leach Tailing Assay = 0.26% Ox Cu

Extraction 10.48 (%) Reagent Consumption

Performed By

Type Agitation Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 12

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 to pH 1 Auxil. Reagent P_2O_2

Concentration 96% Concentration 5 gpl

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Total H_2SO_4 added = 6.85 ml

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	180 ml	1.72 gpl	0.3096	50.90
Wash Soln	600 ml	0.30 gpl	0.1800	29.59
Leach Tlg.	197.9 g	0.06 %	0.1187	19.51
Totals			0.6083	100.00

Head (calculated) T Cu = 0.304%

Head (assayed) T Cu = 0.37%

Remarks

Condition 5 minutes.

Leach Tailing Assay = 0.06% Ox Cu

Extraction 80.49 (%) *Reagent Consumption 105 lbs Acid/ton of ore

21.4 lbs Acid/lb Cu extracted

*Actual acid consumption minus free acid

Performed By

Type Agitation Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 13

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent Flouride, 5 gpl

Concentration 96% Concentration H F Acid, 10 ml

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Total H_2SO_4 added = 9.65 ml

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	138 ml	2.16 gpl	0.2980	34.64
Wash Soln	680 ml	0.38 gpl	0.2584	30.04
Leach Tailing	202.6 g	0.15 %	0.3039	35.32
Totals			0.8603	100.00

Head (calculated) T Cu = 0.43%

Head (assayed) T Cu = 0.37%

Remarks

Condition 15 minutes. Added acid over period of 4 hours.

Leach Tailing Assay = Ox Cu 0.08%

Extraction 64.68 (%) *Reagent Consumption 154.4 lbs Acid/ton of or 27.75 lbs Acid/lb Cu extracted

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 14

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent P_2O_2, 5 gpl F , 5 gpl

Concentration 96% Concentration

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Total H_2SO_4 added = 9.65 ml

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	137 ml	2.05 gpl	0.2808	33.53
Wash Soln	630 ml	0.37 gpl	0.2331	27.83
Leach Tlg.	202.3 g	0.16 %	0.3236	38.64
Totals			0.8375	100.00

Head (calculated) T Cu = 0.418 %

Head (assayed) T Cu = 0.37 %

Remarks

Conditioned 15 minutes.

Leach Tailing Assay = 0.08% Ox Cu

Extraction 61.36 (%) *Reagent Consumption 154.4 lbs Acid/ton of ore

30.04 lbs Acid/lb Cu extracted

*Actual acid consumption minus free acid.

Performed By

Type Bottle Date 12/6/72

Project No. 2019 Sample No. Group II Test No. DF - 15

Weight Solids 200 g Size or Grind Minus 35-Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent Auxil. Reagent Ammonium Biflouride, 5 gpl
Ammonium Carbonate, 15 gpl
NH_4OH, 50 gpl

Concentration Concentration

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	146 ml	0.46 gpl	0.0672	11.38
Wash Soln	600 ml	0.05 gpl	0.0300	5.08
Leach Tlg.	197.4 g	0.25 %	0.4935	83.54
Totals			0.5907	100.00

Head (calculated) T Cu = 0.295%

Head (assayed) T Cu = 0.37%

Remarks

Leach Tailing Assay = 0.22% Ox Cu

Extraction 16.46 (%) Reagent Consumption

Performed By

Type Agitation Date December 14, 1972

Project No. 2019 Sample No. Group II Test No. DF-16

Weight Solids 200 Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent P_2O_2

Concentration 96% Concentration 5.0 Gpl 7.46 Grams/Liter

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:05		1.0	7.2 ml			
	11:30			1.35 ml			
	12:30						
	1:15			1.45 ml			
	2:40			0.70 ml			

Total Acid Consumption 10.7 ml

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2039	Preg Soln	175 ml	1.84 gpl	0.322	36.17
2045	Wash Soln	650 ml	0.42 gpl	0.273	30.67
2023	Leach Tail	196.9 g	0.15 %	0.295	33.16
				0.890	100.00

Head (calculated) 0.44% T Cu

Head (assayed) 0.37% T Cu

Remarks

Conditioned pulp 15 minutes before acid addition

Leached tailing assay - 0.06% Ox Cu

Extraction 66.84 (%) Reagent Consumption 173 lbs acid/ton of ore

29 lbs acid/lb Cu leached

*Actual acid consumption minus free acid.

Performed By

Type Agitation Leach Date December 14, 1972

Project No. 2019 Sample No. Group III Test No. DF-17

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent P_2O_2

Concentration 96% Concentration 5 gpl (7.46 Grams/Liter)

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:05		1.0	6.5 Ml			
	11:30			1.2 ml			
	12:30						
	1:15			1.85 ml			
	2:40			0.65 ml			

Total Acid Consumption 10.2 ml

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2039	Preg Soln	195 ml	2.33 gpl	0.4543	43.20
2045	Wash Soln	630 ml	0.54 gpl	0.3402	32.35
2023	Leached Tail	197.7 g	0.13 %	0.2570	24.45
				1.0515	100.00

Head (calculated) 0.52% T Cu

Head (assayed) 0.45% T Cu

Remarks

Conditioned pulp 15 minutes before acid addition

Leach Tailing assay - 0.06%Ox Cu

Extraction 75.55 (%) Reagent Consumption 164.2 lbs acid/ton of ore

20.66 lbs acid/ lb Cu leache

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date December 14, 1972

Project No. 2019 Sample No. Group IV Test No. DF-18

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent P_2O_2

Concentration 96% Concentration 5 gpl (7.46 Grams/Liter)

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:40		1.0	3.5 ml			
	11:05			0.9 ml			
	11:30			1.0 ml			
	11:15						
	2:40			0.8 ml			

Total Acid Consumption 6.2 ml

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2041	Preg Soln	190 ml	2.87 gpl	0.5453	49.29
047	Wash Soln	650 ml	0.44 gpl	0.2860	25.86
2025	Leach Tail	196.3 g	0.14 %	0.2748	24.85
				1.1061	100.00

Head (calculated) 0.55% T Cu

Head (assayed) 0.51% T Cu

Remarks

Conditioned Pulp for 15 minutes

Leached Tailing Assay - 0.05%Ox Cu

Extraction 75.15 (%) Reagent Consumption 93.5 lbs acid/ton of ore

11.25 lbs acid/ lb Cu leache

*Actual acid consumption minus free acid.

Performed By

Type Agitation Leach Date December 14, 1972

Project No. 2019 Sample No. Group II Test No. DF-10

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent H_2O_2

Concentration 96% Concentration 50% 5 gpl

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Acid Addition 11.25 ml

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	215ml	1.49 gpl	0.3204	40.58
Wash Soln	600ml	0.42 gpl	0.2520	31.93
Leach tail	197.3 g	0.11 %	0.2170	27.49
			0.7894	100.00

Head (calculated) 0.39% T Cu

Head (assayed) 0.37% T Cu

Remarks

H_2O_2 added slowly over a period of 4 hours. 100 ml total

Leached tailing assay - 0.05%Ox Cu

Extraction 72.51 (%) Reagent Consumption 182.7 lbs acid/ton of ore

31.91 lbs acid/ lb Cu leache

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date December 14, 1972

Project No. 2019 Sample No. Group III Test No. DF- 20

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent H_2O_2

Concentration 96% Concentration 50% 5 gpl

pH 1.0 Time

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Acid consumption 11.20 ml

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	275 ml	1.58 gpl	0.4345	42.46
Wash Soln	620 ml	0.47 gpl	0.2914	28.49
Leach Tail	198.1 g	0.15 %	0.2972	29.05
			1.0231	100.00

Head (calculated) 0.51% T Cu

Head (assayed) 0.45% T Cu

Remarks

Added H_2O_2 over a period of 4 hours. 100 ml total.

Leached Tailing Assay - 0.07%Ox Cu

Extraction 70.95 (%) Reagent Consumption 181.8 lbs acid/ton of ore

25.04 lbs acid/lb Cu leache

*Actual acid consumption minus free acid.

Performed By

Project No. 2019 Sample No. Group IV Test No. DF-21

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent H_2O_2

Concentration 96% Concentration 5 gpl

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Acid Consumption 7.3 ml

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2044	Preg Soln	275 ml	2.06 gpl	0.5665	52.17
2050	Wash Soln	550 ml	0.41 gpl	0.2255	20.77
2028	Leach Tail	195.8 g	0.15 %	0.2937	27.06
				1.0857	100.00

Head (calculated) 0.54% T Cu

Head (assayed) 0.51% T Cu

Remarks

H_2O_2 added over a period of 4 hours. 100 ml total.

Leached Tailing Assay - 0.06%x Cu

Extraction 72.94 (%) Reagent Consumption 112.9 lbs acid/ton of ore

14.25 lbs acid/ lb. Cu leache

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date December 14, 1972

Project No. 2019 Sample No. Group I Test No. DF- 22

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent NH_3 as NH_4OH Auxil. Reagent CO_2 as NH_4CO_2 15 gpl

Concentration 50 gpl Concentration 1 gpl $CuSO_4$ or .398 g Cu++/Liter

pH 1.0 Time 4 Hour Leach

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg Soln	156 ml	0.40 gpl	0.0627	5.14
Wash Soln	620 ml	0.08 gpl	0.0496	4.06
Leach Tail	197.8 g	0.56 %	1.1077	90.80
			1.2200	100.00

Head (calculated) 0.61% T Cu

Head (assayed) 0.66% T Cu

Remarks

Agitation Cold

Leached Tailing Assay -

Extraction 9.20 (%) Reagent Consumption

*Actual acid consumption minus free acid.

Performed By

Type Agitation Date December 14, 1972

Project No. 2019 Sample No. Group I Test No. DF-23

Weight Solids 200 grams Size or Grind -35 Mesh

Leach Conditions

Solids 50 (%) Temperature 90°C

Reagent NH_4OH Auxil. Reagent NH_4CO_2

Concentration 50 gpl Concentration 25 gpl

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

Assay No.	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2208	Preg Soln	131 ml	0.76 gpl	0.0995	6.79
209	Wash Soln	610 ml	0.10 gpl	0.0610	4.16
2195	Leach Tail	197.7 g	0.66 %	1.3048	89.05
				1.4653	100.00

Head (calculated) 0.73% T Cu

Head (assayed) 0.66% T Cu

Remarks

Agitation Hot

Extraction 10.95 (%) Reagent Consumption

Performed By

APPENDIX I

Mineralogical Report

Tucson, Arizona
January 6, 1973

To: Dr. Roshan B. Bhappu
Vice President
Research and Development
Mountain States Mineral Enterprises
Vail, Arizona.85641.

From: Laszlo Dudas
Consulting Mineralogist
4737 E Adams St.
Tucson, Az. 85712.

Subject: Mineralogical Examination of Diamond Drill Core Samples from Johnson Camp, Dragoon Mountains, Arizona. Project No.: 2019.

Purpose: To determine the mineral composition of the sample in particular respect to copper mineralization.

Samples: On December 1, 1972, Fourteen diamond drill core samples from hole No. T-2 were chosen, with the help of Mr. Jay Quick, for mineralogical study, as follows:

1. 602.5'-603.5'
2. 713' - 717.5'
3. 723'
4. 893'
5. 896'
6. 897.5'-899.5'
7. 919.5'-924'
8. 1065'
9. 1070'
10. 1128'-1129'
11. 1132.5'-1135'
12. 1199.5'
13. 1257'-1262'
14. 1262-1263'

For metallurgical testing, a quarter split of each of the diamond drill cores (hole T-2, from 582' to 1270') was crushed to -½". These were divided into four groups according to their overall mineral composition. Samples for mineralogical study were taken from the composites of these four groups, to ascertain the percentile distribution of the mineral components.

Thin and polished sections were made from the samples and observed under transmitted and reflected light polarizing microscopes, respectively. (This includes 12 thin sections and four polished sections. Polished sections were made of 1070', 1132.5'-1135', and two of 1262'-1263'. Six thin sections and three polished sections were made from the composites; group II. is missing among the sections, because it was used up in testing.)

Results follow:

Mineralogy:

Transparent Minerals:

Calcite and Dolomite are the predominant minerals in the cores of group one; they are forth and sixth in groups three and four respectively. By inference from t sections of the "as is" samples, calcite-dolomite probably takes fourth place in group two. They occur as densely packed, anhedral grains - as marble - or as veins and veinlets. The calcite and dolomite grains intergrow with each other, and are irregularly distributed in the marble. They intergrow mostly with serpentine (perhaps derived from forsterite) and chlorite. These two minerals, without exception, are impregnated by hydrous iron oxides. Calcite occasionally also intergrows with garnet, diopside and amphiboles (tremolite-actinolite, hornblende). The grain size of the calcite-dolomite varies between 30 and 3000 microns in the "as is" cores, and between 3 and 600 microns in the crushed (-¼") samples.

Quartz predominates in groups three and four, while taking the sixth place in group one, and possibly in group two (inferred from "as is" cores). It occurs in large to medium-sized anhedral grains or as extremely fine-grained chalcedony, as vein fillings. The quartz intergrows with feldspar, garnet-epidote, and amphiboles. The chalcedony occasionally grades into chrysocolla. The grain size of the quartz varies between 5 and 2000 microns in the "as is" cores, and between 2 and 650 microns in the crushed (-¼") samples.

Chlorite and Serpentine are second in frequency in groups one and three, third in group two (by inference), and fourth in group four. They occur in fine to medium flaked aggregates, or as an interwoven mesh. They are intricately intergrown with diopside, amphiboles, calcite and dolomite. Most of the chlorite and serpentine aggregates are impregnated by hydrous iron, and, to a lesser degree, by hydrous copper solutions. For this reason, they show yellow, brown and green colorations of variable intensity. Some chlorites have a dark green color of their own, which has nothing to do with copper impregnation. The grain size of the individual flakes of the chlorite and serpentine is small - 1 to 20 microns - but the size of the aggregates may reach 5000 microns in the "as is" cores. In the crushed (-¼") cores, the grain size is small, varying between 1 and 300 microns.

Diopside is presumably second in group two, fourth in group one, sixth and ninth in groups three and four, respectively. The diopside appears in these samples mainly in two forms: 1. as coarse, subhedral prisms; 2. as small to medium sized, anhedral grains. It is intricately intergrown with chlorite, serpentine, muscovite, amphiboles, and calcite-dolomite. Occasionally, it is impregnated by hydrous iron solutions; hence, a portion of the diopside shows yellow-brown color.

The grain size is from 50 to 4000 (in elongation) microns in "as is" cores, while it ranges from 10 to 550 microns in the crushed (-¼") cores.

Garnet predominates in group three, takes the second place in group four, and is sixth in group one and also, probably, in group two. It occurs in coarse, euhedral to anhedral crystals and grains. Occasionally, some skeletal crystals are present. The garnet is mostly present as andradite - it is dark colored (reddish-brown) - but there are some light or even colorless grains which belong to the grossularite variety. Both varieties are intimately associated with epidote. Furthermore, it is intergrown with chlorite, serpentine, muscovite, quartz, amphibole and calcite. The grain size of the garnet is from 80 to 4000 microns in "as is" cores, and 50 to 600 microns in the crushed (-¼") cores.

Epidote ranks third in group three, and fifth and perhaps sixth in groups four and two, respectively. It is only a trace component in group one. The epidote occurs in coarse, anhedral grains of variable color (some are light to colorless, while others display a strong, pistachio green color). It is intimately intergrown with garnet, chlorite, muscovite, quartz, amphiboles, and calcite. Occasionally, it shows the brown color of hydrous iron oxide impregnation. The grain size is from 30 to 3000 microns in "as is" cores, and 20 to 450 microns in the crushed (-¼") cores.

Amphibole (tremolite, actinolite, hornblende) is sixth in frequency in groups one and three, and sev enth in group four. Perhaps, it is a dominating component in group two, as indicated by the observation of the "as is" cores (see Table II). The amphibole occurs in three varieties in these samples, two of which are needle types: tremolite is colorless, while actinolite is a light green variety; the third is hornblende, with dark green color and strong pleochroism. The latter is sometimes altered to chlorite. Some of the tremolite shows moderate to strong hydrous iron oxide impregnation. The amphibole is intricately intergrown with diopside, chlorite, serpentine, muscovite, sericite, quartz, calcite, garnet and epidote. The grain size of the individual needles and prisms is relatively small - 20 to 550 microns-but the felted aggregates may reach 6000, or even 15,000 microns. The section of core 897.5'-899,5' is almost totally composed of felted actinolite with small amounts of other components. The size of the aggregates in the crushed (-¼") cores ranges from 40 to 300 microns.

Muscovite (sericite) occupies the third place in frequency in group four, but it is only a trace mineral in group one, and was not noted in group three. It is possible that it is the fourth or fifth ranking component in group two. Composition-wise, muscovite and sericite are identical, but they differ in grain size. Coarse-

flaked, white, potassium mica is muscovite, while the fine-flaked variety is sericite. Both of them are present in the observed samples, mostly intricately intergrown with each other and with tremolite, actinolite, diopside and quartz. The flake size varies between 5 to 200 microns in "as is" cores, and between 1 to 100 microns in the crushed (-¼") cores.

Feldspar is a major component in group four, where it takes the third place in frequency. It is mosty represented by orthoclase, and some microcline. A large portion of it shows a brown, semi-opaque coating, which is a sign of progressing kaolinitization. The feldspar is intricately intergrown with quartz and occasionally with other silicates, particularly sericite.

Forsterite is a trace component which is present in the marbles. It is mostly altered to serpentine, and chlorite, which show strong hydrous iron oxide impregnation. Occasionally, some unaltered remnants are visible in the serpentine.

Sphene is a minor to trace component in these samples. It occurs mostly in small, anhedral grains. A few large, subhedral grains are also present. The sphene is mostly associated with amphiboles and calcite.

Fluorite is only a trace component. It occurs in medium-sized, resorbed grains, in association with calcite, diopside, amphiboles, quartz and feldspar.

Chrysocolla is a minor to trace component. The general distribution of this mineral in the crushed sample is just below one volumetric percent. In the "as is" cores, the chrysocolla shows very erratic distribution varying between traces and four volumetric percent. The highest amount can be found in group one cores. Groups two and four show only traces to one volumetric percent.

Opaque Minerals.

Magnetite and Martite are the predominant opaque components in the samples. They are third in group one, and fifth and seventh in groups three and four. Perhaps they occupy the fourth place in group two, according to the counts on the "as is" core sections. Most of the magnetite is partially oxidized to hematite, which is called martite. The martite forms needle-like networks which invade the magnetite along cleavage planes and cracks, replacing and altering it to hematite. A large portion of the magnetite is disseminated in the rock. It also forms large, anhedral to subhedral crystal aggregates or veins. The magnetite is associated with calcite-dolomite, and most of the silicates. It occasionally carries small inclusions (3 to 20 microns) of chalcopyrite. A portion of the magnetite is oxidized to goethite, or is almost decomposed to hydrous iron oxides (limonite). This provides a portion of the iron which impregnates the layered silicates (the other portion of the iron is derived from the Fe-content of the silicates). The grain

size of the magnetite varies between 10 to 300 microns. The size of the aggregates may reach 6000+ microns.

Hematite is present in trace amounts, exclusively as a complete decomposition product of magnetite.

Goethite-Hydrous Iron Oxides are ubiquitous in the observed samples, although there are some sections in which they are missing completely. Goethite and hydrous iron oxides rank perhaps fifth in group two, sixth in group one, and seventh and eighth in groups three and four, respectively. They form individual grains, occasionally intricately intergrown with carbonates and silicates. The hydrous iron oxides usually grade into iron impregnations, due to repeated leaching (through vadose water) and precipitation. By this, they mix with copper solutions, and thus may contain Cu values as high as 15 wt %. The prevalence of the brown color subdues the blue-green of the copper; thus it is not visible. The individual grain size is moderate, 5 to 200 microns, but the aggregate size may reach 3000 microns.

Rutile is a minor to trace component in the samples. It occurs only in a few of the cores. The rutile is present as small disseminated grains, mostly in the amphiboles.

Chalcopyrite, Covellite, Chalcocite and Native Copper are present only as trace components. They occur as small inclusions in magnetite and in the silicates. In many instances, they are present in the vicinity of chrysocolla or copper-iron impregnated chlorites (layered silicates). The grain size of the above minerals is between 2 to 30 microns.

Pyrite is present infrequently. It occurs in small (5 to 20 microns) grains.

Discussion:

1. Transparent (gangue) minerals (silicates and carbonates) constitute the major portion (79 to 86 vol %) of the observed samples. The remaining portion is composed of opaques: magnetite and hydrous iron oxides (13 to 21 vol %).

2. Chrysocolla is the only copper mineral present in the observed samples in appreciable amounts. It is most frequent in group one, in which it averages just below one volumetric percent. In the other groups, its amount is a half of one volumetric percent or lower. The distribution of the chrysocolla is erratic. It occurs mostly in association with crack-filling chalcedony, or with layered silicates (chlorites and serpentines) in the vicinity of fractures where supergene solutions were circulating. In both cases, the chrysocolla grades into its host minerals.

3. Copper sulfides are present, but only in tiny grains, in minute quantities.

4. The major copper carriers are the layered silicates (chlorite and serpentine) and the hydrous iron oxides. These do not have copper in their lattices. The copper is present in them only by physical means, by impregnation. The circulating surface water dissolves iron, copper, manganese, etc. from their sulfides, oxides, etc. and carries them into solution. Then the physical-chemical conditions permit, the surface water precipitates hydrous oxides, sulfates, hydrous silicates etc. of these elements. The layered silicates and the porous hydrous iron oxides are the most susceptible minerals for this action, because the circulating water over long periods of time, penetrates between the layers and into pores, depositing copper, iron, and manganese compounds. The true identity of the impregnating iron, copper, and manganese compounds can be determined neither by optical means nor by x-ray diffraction, due to their amorphous (non-crystalline) habits and highly variable characters. The copper impregnation shows a typical green, or greenish-blue color, but when iron is present the color changes to yellow-green, and if iron is prevalent, the copper color disappears, though the copper is still present in the host mineral. Several tests showed that this copper can be reduced or sulfidized in place in the host minerals, so that chlorite, serpentine and limonite showed metallic copper or covellite particles; but this did not aid copper recovery. The copper content of the host minerals varies between a few tenths of a percent up to 15 wt.%.

5. The presence of native copper indicates that in the upper portion of the deposit, not only oxidizing, but occasionally reducing conditions also existed locally.

6. The recovery of the copper mineral (chrysocolla) from this ore by acid-leaching should be easy. The recovery of the copper content of the ore by a simple acid-leach process faces difficulties in two areas, namely: 1. most of the copper present in the ore is in foreign host minerals as impregnation; 2. carbonates (calcite and dolomite) are abundant in the ore, particularly in group one. These two factors may defeat an acid leach attempt, because the layered-silicate host minerals may serve as fine, molecular filters, so they may frustrate the removal of copper. The circulating surface water acted upon these silicates over long periods of time, but when the removal of copper should be accomplished in a few hours, these silicates are not conditioned enough to release their impregnating compounds. The presence of carbonates in large quantity, will neutralize the leach solution, preventing it from removing even the fairly accessible chrysocolla.

7. No copper oxide minerals - tenorite-melaconite, delafossite, or cuprite - were detected. An attempt was made by x-ray diffraction powder methods, to determine the velvety black botryoidal mineral on some of the pieces. The result

showed only the presence of amorphous hydrous iron oxides and no copper oxides.

Conclusion:

1. The bulk of the copper is present as impregnation in layered silicates and hydrous iron oxides.

2. Chrysocolla is the major discrete copper mineral in the samples. The sulfides and native copper are present only in trace amounts.

3. Copper recovery is difficult, due to the layered silicates which act as micro-filters and the high amounts of carbonates which neutralize the acid leach solutions. Carbonates, during acid leaching, also produce gypsum in excess amounts which ultimately clog the way of the circulating solutions.

Recommendation:

1. It seems that a -48 mesh grind and preroasting would condition the ore for a more penetrating or vigorous leach by an ammonia-type solution.

Tables of volumetric percent distribution of the mineral components and photo-micrographs to illustrate the discussed problems are presented.

Laszlo Dudas

Table I.

Estimated Volumetric Percent Distributions of Mineral Components in the Diamond Drill Cores from Hole T-2, Johnson Camp, Dragoon Mountain, Arizona.

	Groups														
	I.			II.				III.	IV.						
Name of Minerals	602.5'-603.5'	713'-717.5'	723'	893'	896'	897.5'-899.5'	919.5'		1065	1070	1128'-1129'	1132.5'-1135'	1199.5'	1257'-1262	1262'-1263'
Calcite-Dolomite	49	2	25	2	15		6		3		12	Tr			
Quartz	4	3							8		18	26	19	30	
Diopside	12	30	14	25	20		26		1						
Garnet	1	Tr	Tr						27		34			24	
Epidote						5	3		19		19		2	18	
Chlorite-Serpentine		7	5	6	5	10	8		3		9	12	20		
" impregn. by Cu Solution	1	10	11	2	3		2		3				1		
" impregn. by Fe Solution	8	15	18	12	14		5		10			3	5		
Tremolite-Actinolite etc.	5	11	5	35	18	62	40		12		3	Tr		5	
Feldspar												24	20	3	
Muscovite-Sericite		5	4			10	3		3			24	16	5	
Forsterite	2		2												
Sphene						3	4		3			Tr			
Fluorite	Tr											Tr			
Chrysocolla	Tr	2	4		Tr		1		Tr		1		Tr		
Magnetite	6	3	2	6	6	3	1		1	Tr	1	2	8	7	7
" altered to Martite	9	4	2	4	9	4	1		2	2	1	7	6	4	8
Hematite															
Goethite, Hydrous iron oxides	3	8	8	8	10	3			5	4	2	2	3	4	4
Rutile										Tr					
Native Copper										Tr					Tr
Chalcopyrite												Tr			Tr
Covellite															Tr
Pyrite															
Total	100	100	100	100	100	100	100		100	100	100	100	100	100	100

No samples were taken because of similar gross mineralogy to group four. Missing

Transparent gangue 94 vol. %

Transparent gangue 81 vol. %

Table II.

Volumetric Percent Distribution of Mineral Components in the Crushed (-¼") Composite Samples of Diamond Drill Cores from Hole No.T-2, Johnson Camp, Dragoon Mountains, Arizona.

Name of Minerals	Groups I.	II.	III.	IV.
Calcite-Dolomite	34		10	8
Quartz	4		21	20
Diopside	11		5	1
Garnet	4		21	13
Epidote	Tr		13	9
Chlorite-Serpentine	12		11	7
" Impregn. by Cu solution	2		1	1
" Impregn. by Fe solution	7		3	3
Tremolite-Actinolite etc.	4		5	5
Feldspar				12
Muscovite-Sericite	Tr			12
Forsterite	1			
Sphene				
Chrysocolla	Tr			
Fluorite	Tr			
Magnetite	8		3	2
" altered to Martite	9		4	3
Hematite	Tr		Tr	Tr
Goethite-Hydrous Fe Oxides	4		3	4
Chalcopyrite	Tr			Tr
Pyrite	Tr		Tr	Tr
Total	100		100	100

Material of this group was used up in testing

Transmitted Light
Parallel Nicols
Magn.: 30x

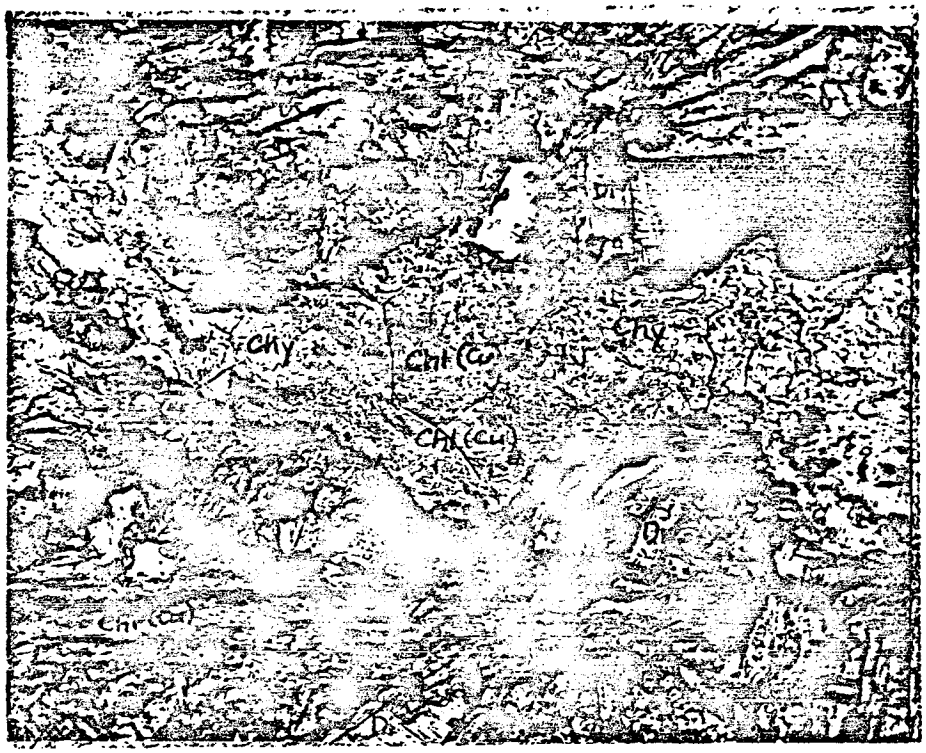
Scale

Mesh		Microns
100		147
48		295
24		590

Figure 1.
Group I. 602.5'

A calcite vein bordered by diopside cuts across dolomite marble. At the contact of the marble with diopside a few grains of fluorite are present, indicating a more diversified mineralization. The dark grains in the marble are iron impregnated serpentine, an alteration product of forsterite.

Code: Ca= calcite; Di= diopside; Do= dolomite; Fl= fluorite; Sep(Fe)= serpentine (forsterite) impregnated by Fe solutions.

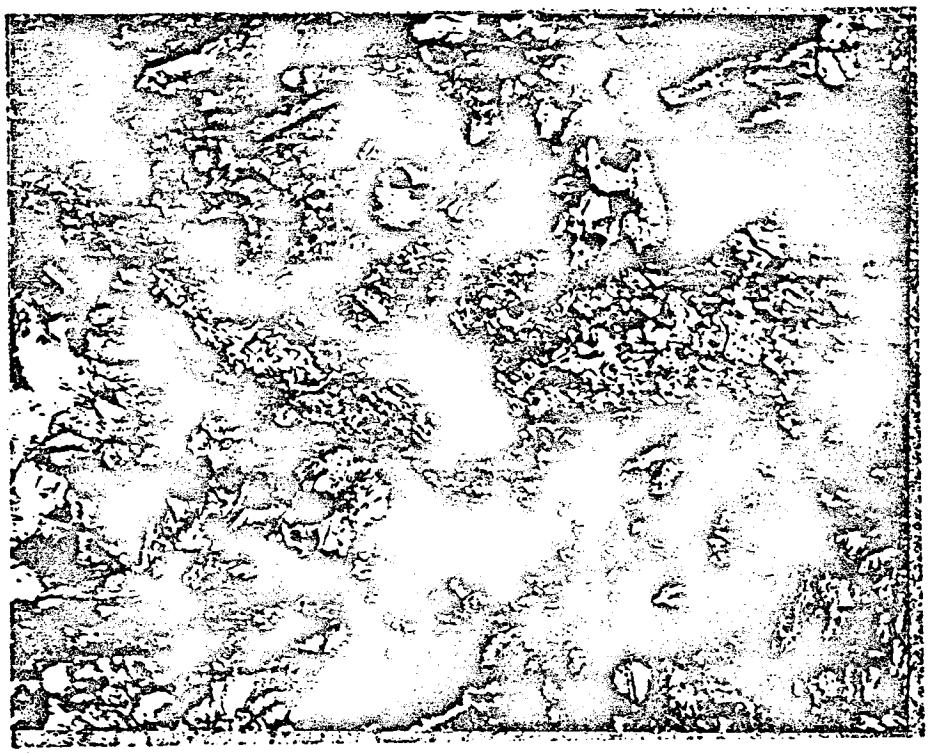


Transmitted Light
Parallel Nicols

Magn.: 100x

Scale:

Mesh	Microns
400	38
200	74
100	147
48	295

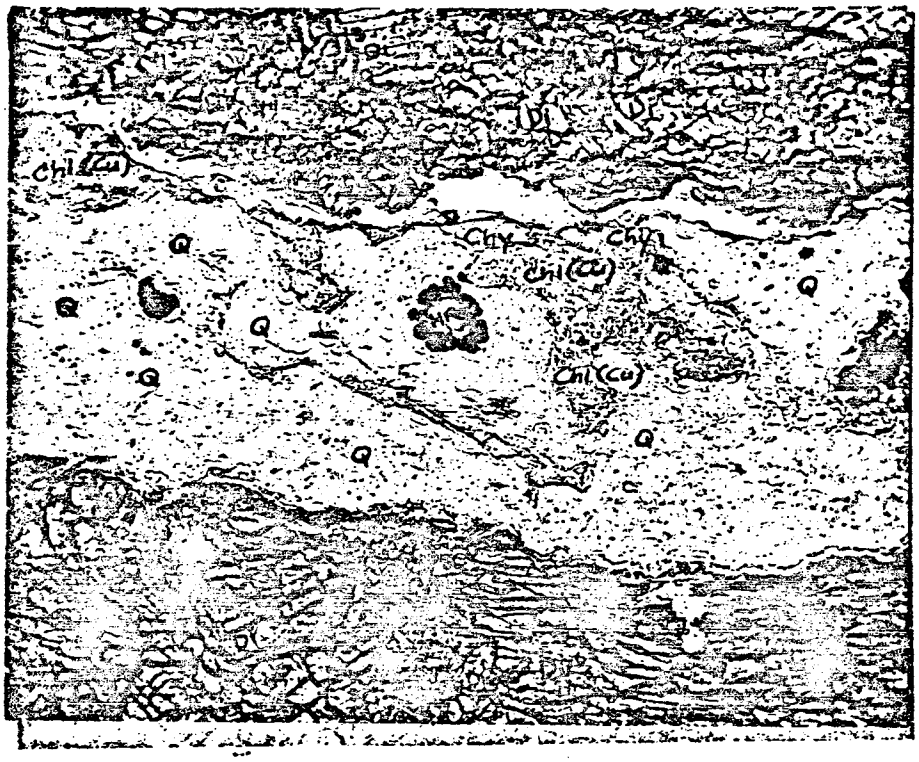


Transmitted Light
Crossed Nicols

Figures 2 and 3.
Group I, 713 - 717.5'

A vein and an auxiliary veinlet of chrysocolla and chlorite impregnated by copper solution cut through coarse, crystalline diopside. Under crossed Nicols, the mosaic of the chrysocolla grains is visible.

Code: Chl(Cu)= chlorite impregnated by Cu solution; Chy= chrysocolla; Di= diopside.

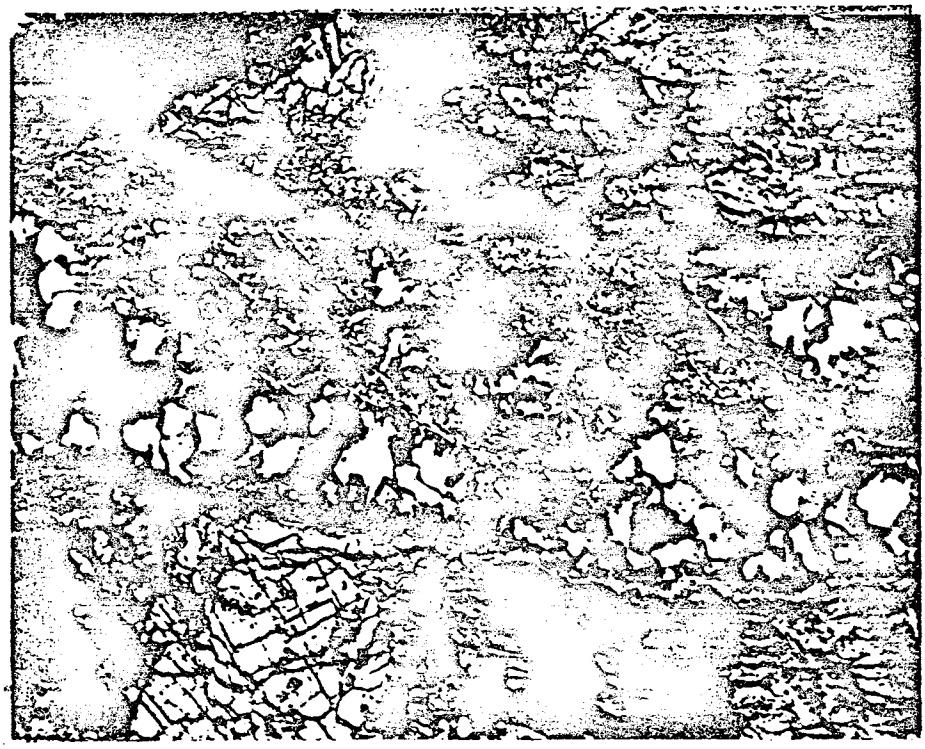


Transmitted Light
Parallel Nicols

Magn.: 100x

Scale

Mesh		Microns
400		38
200		74
100		147
48		295

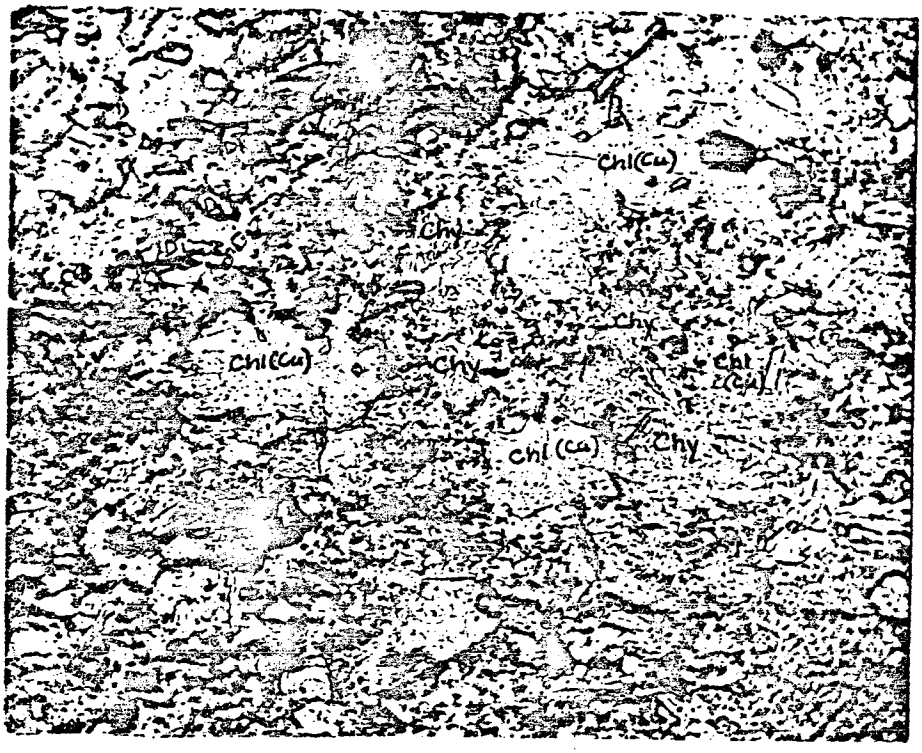


Transmitted Light
Crossed Nicols

Figures 4 and 5.
Group I, 713-717.5'

A chalcedony vein cuts coarse, prismatic diopside (note typical cleavages) incorporating copper impregnated chlorite and chrysocolla. A chlorite globule is impregnated by hydrous iron solution (center). The fine grain size of the chlorite is shown under crossed Nicols.

Code: Chl(Cu)= chlorite impregnated by Cu solution; Cly= chrysocolla; Di=diopside; HFe= hydrous Fe oxide; Q= quartz-chalcedony.

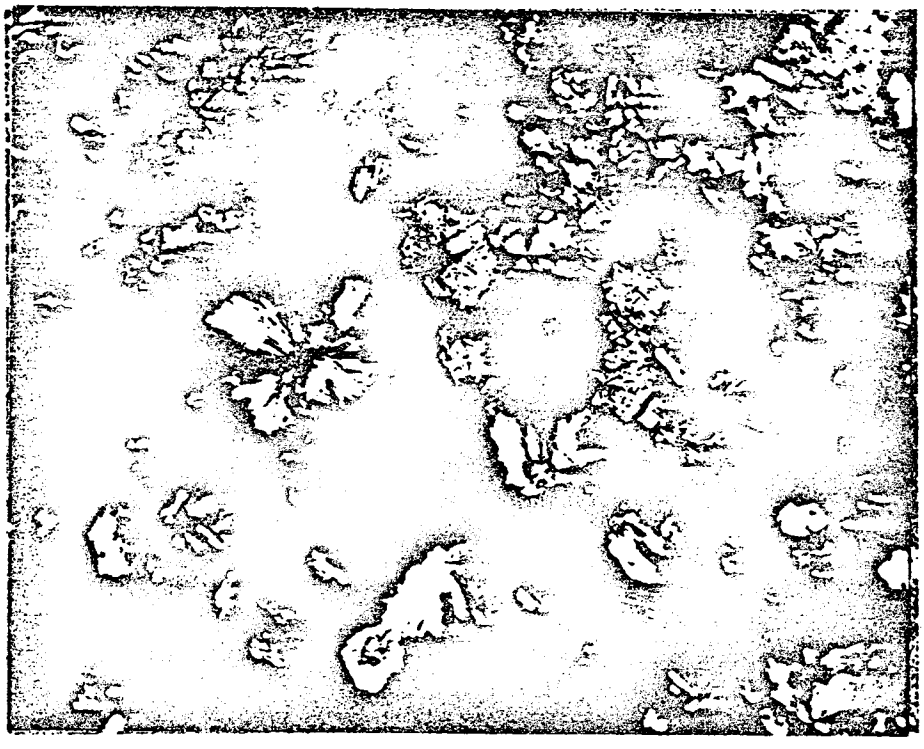


Transmitted Light
Parallel Nicols

Magn.: 100x

Scale

Mesh		Microns
400		38
200		74
100		147
48		295



Transmitted Light
Crossed Nicols

Figures 6 and 7.
Group I, 722'

An area near a large fracture in marble is filled with iron and copper impregnated chlorite. The green color of copper impregnation grades into the yellow and brown of Fe impregnation. Some chrysocolla grains are present, as shown in the fine mosaic structure visible under crossed Nicols. Remnant prisms of diopside are also present.

Code: Chl(Cu)= chlorite impregnated by Cu solution; Chy= chrysocolla; Di= diopside.

Transmitted Light
Parallel Nicols

Magn.: 100x

Scale

Mesh	Microns
400	38
200	74
100	147
48	295

Transmitted Light
Crossed Nicols

Figures 8 and 9.
Group II, 356'

A large patch of chrysocolla appears interstitially in the midst of a dense coarse, prismatic diopside aggregate. This indicates that the distribution of chrysocolla and the Cu impregnated chlorite-serpentine is very erratic. They are not confined only to fracture areas, but invade interstitial spaces in dense silicates, also.

Code: Chy= chrysocolla; Di= diopside.

Transmitted Light
Parallel Nicols

Magn.: 100x

Scale

Mesh	Microns
400	38
200	74
100	147
48	295

Figure 10.
Group IV. 1065'

Garnet occurs mostly in dense, anhedral, grain aggregates, but often forms skeletal crystals with lots of interstitial space. These interstices are filled by chlorite impregnated by iron, or copper, and by dark green hornblende. The occurrence of the Cu impregnated chlorite and chrysocolla is not as frequent in Groups three and four as it is in Groups one and two.

Transmitted Light
Parallel Nicols

Magn.: 100x

Figure 11.
Group IV. 1132.5'-1133'

In Group four, hornfels appears as one of the major rock types. The picture shows a quartz-feldspar-chlorite intergrowth. Note the presence of a large, resorbed fluorite grain, and the disseminated epidote and sphene. The feldspars are moderately altered to kaolinite, indicated by the dark color.

Code: Chl= chlorite (no impregnation); Ep= epidote; Fel= feldspar; Fl= fluorite; Gr= garnet; Ho= hornblende; Q= quartz; Spe= sphene.

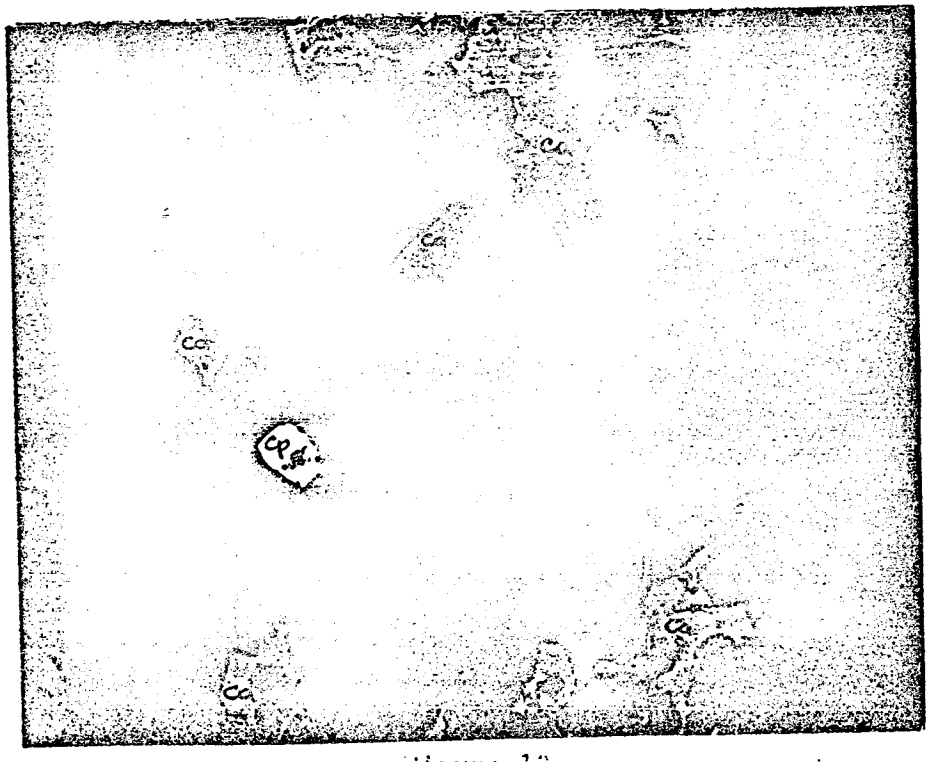
Reflected Light
Parallel Nicols

Magn.: 200x

Mesh	Micron
400	38
200	74
100	147
48	295

Figure 12.
Group IV. 1132.5'-1135'

Magnetite is the dominant opaque mineral in all the observed samples. A large grain-aggregate is shown here, which is heavily martitized. Martitization is an oxidation process by which the magnetite turns to hematite. The replacement progresses along cracks and cleavage planes. One chalcopyrite inclusion is shown in the magnetite. By further oxidation, the magnetite-martite will decompose to hydrous iron oxide, as shown on the left.



Reflected Light
Parallel Nicols

Magn.: 200x

Figure 13.
Group IV. 1265'-1268'

Chalcopyrite also occurs disseminated in the transparent gangue (garnet).

Code: Ca= calcite; Cp= chalcopyrite; Gr= garnet; HFe= hydrous iron oxide; Mag= magnetite; Mar= martite.

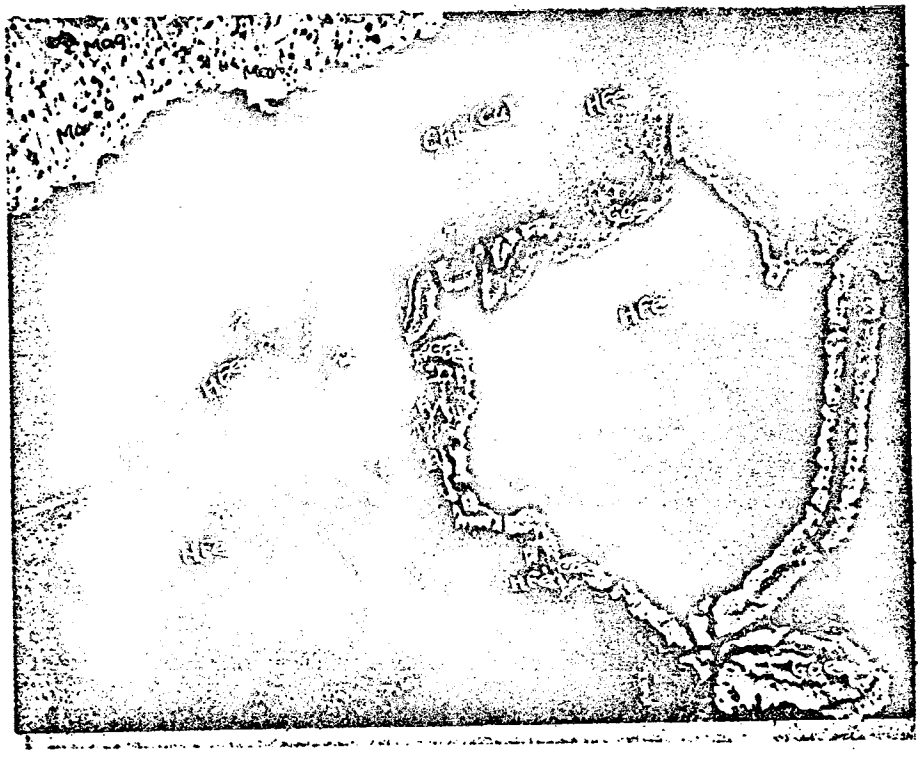
Reflected Light
Parallel Nicols
Magn.: 200x

Scale

400 — 38
200 — 74
100 — 147
48 — 295

Figure 14.
Group IV. 1262'-1263'

By the action of the circulating surface water, some of the original chalcopyrite is altered to covellite. The surrounding chlorite, which fills the interstices of a garnet aggregate, is impregnated by copper solution, also as a result of the alteration of the chalcopyrite.

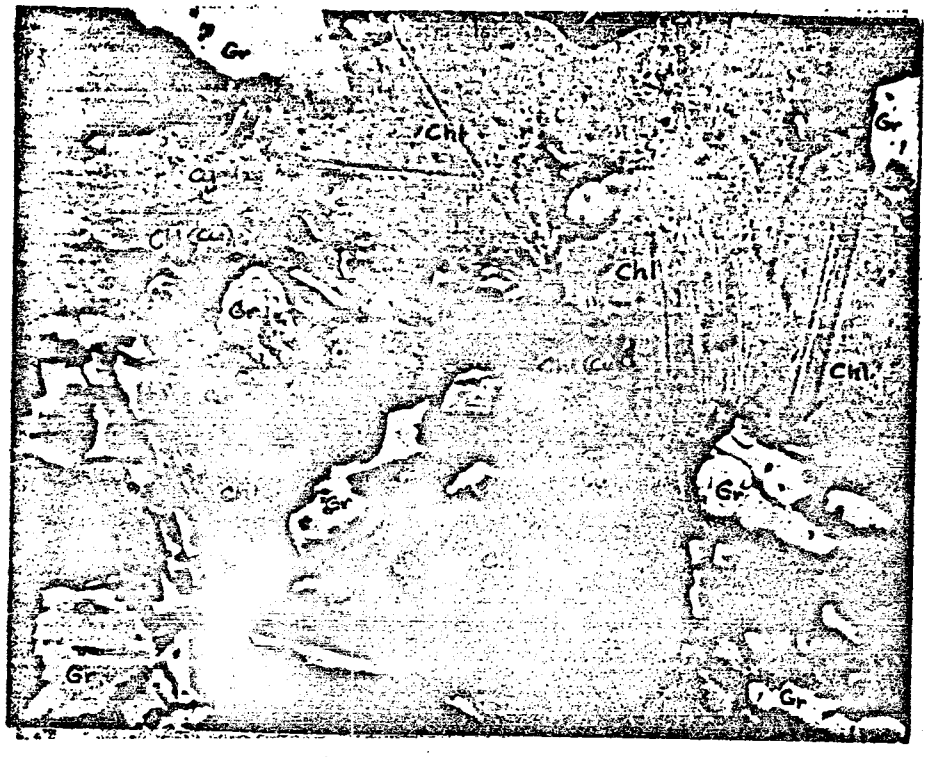


Reflected Light
Parallel Nicols
Magn.: 200x

Figure 15.
Group IV. 1132.5'-1135'

This picture shows the alteration and decomposition products of the magnetite-martite. Here, three different hydrous iron oxides are present, of which only the goethite is crystalline, the other two being amorphous. Between the magnetite and the hydrous iron oxide grains, some copper impregnated chlorites are present. Thus, perhaps these hydrous iron oxides also contain copper which is not visible.

Code: Chl(Cu)= chlorite impregnated by Cu solution; Cov= covellite; HFe= hydrous iron oxide; Goe= goethite; Gr= garnet; Mag= magnetite; Mar= martite.



Reflected Light
Parallel Nicols

Magn.: 200x

Scale

400 — 38
200 — 74
100 — 147
48 — 295

Reflected Light
PARallel Nicols

Figures 16 and 17.
Group IV. 1570'

Native copper grains occur in chlorite, and filling interstices in garnet. This indicates that reducing conditions existed locally, besides the prevailing oxidizing environment.

Code: Chl= chlorite; Chl(Cu)= chlorite impregnated by copper solution; Cov= covellite, Gr= garnet.

JOHNSON

Republic Mine

Burro Pit

OK Mine

Keystone Mine

S-2
Bedrock 360'
360' 500 0.02% Cu
T.D. 360' (Rotary)

DC-9
Bedrock 528'
630' 102 @ 0.87% Cu (Ox)
T.D. 1900'

S-3
Bedrock 338'
No ore intervals
T.D. 1223'

S-1
Bedrock 400'
420' 266 @ 0.66 % Cu (Ox)
T.D. 1211'

S-4
Bedrock Approx 280'
319' 111 @ 0.56 % Cu (Ox)
470' 79 @ 0.56 % Cu (Ox)
T.D. 960'

Willcox

Interstate 10

N

27 | 26
34 | 35

26 | 25
35 | 36

25 | 30
36 | 31

30 | 29
31 | 32

9
7
5
3
2
1

34 | 35
3 | 2
10 | 11

35 | 36
2 | 1
11 | 12

36 | 31
1 | 6
12 | 7

31 | 32
6 | 5
7 | 8

T. 15 S.

T. 16 S.

LEGAL TENDER
(Min. Pat.)

T-1
Bedrock 540'
852' 216 @ 0.78% Cu (Ox)
1300' 107 @ 0.68% Cu (Sul)
T.D. 1501'

T-10
Bedrock Approx 300'
T.D. 300' (Auger)

T-7
Bedrock 365'
843' 80 @ 0.38% Cu (Ox)
1072' 775 @ 0.37% Cu (Ox)
1822' 52 @ 0.57% Cu (Sul)
T.D. 2450'

Bedrock 480'
582' 222 @ 0.76% Cu (Ox)
884.5' 360 @ 0.48 % Cu (Ox)
T.D. 1464'

T-2

T-3
Bedrock 420'
No ore intervals
1657' 3' @ 0.36% Cu (Ox)
T.D. 1660'

T-9
Bedrock 190'
1578' 92 @ 0.58% Cu (Sul)
1578' 96 @ 0.34% Cu (Sul)
1845' 166 @ 0.43 % Cu (Sul)
T.D. 2163'

T-8
Bedrock 366'
1007.5' 121.5 @ 0.45% Cu (Ox)
1199' 118' @ 0.61% Cu (Ox & Sul)
T.D. 1828'

T-4
Bedrock 376'
385' 246 @ 0.32% Cu (Ox)
1300' 299 @ 0.30 % Cu (Sul)
T.D. 1600'

T-5
Bedrock 360'
992' 96 @ 0.44% Cu (Ox)
T.D. 1524'

T-11
Bedrock 797'
1060' 69 @ 0.33% Cu (Ox)
T.D. 2275'

T-6
Bedrock 215'
No ore intervals
T.D. 421'

T-12
Bedrock 564'
1703' 38 @ 0.33 % Cu (Sul)
T.D. 2065'

STANDARD 8
STANDARD 9
(Min. Pats.)

Interstate 10

Benson – Tucson

R. 22 E.

R. 23 E.

DRILL HOLE LOCATION
HOLE NO
T-2
DEPTH TO BEDROCK
Bedrock 480'
582' 222' @ 0.76% Cu
INTERVAL
FROM

EXPLANATION

- TEX CLAIMS (CJR OPTION)
- PROSPECT CLAIMS (CJR OPTION)
- STATE PROSPECTING PERMIT (CJR OPTION)
- PROPOSED DRILL HOLE
- ROTARY HOLE COMPLETED
- DIAMOND DRILLING IN PROGRESS
- DIAMOND DRILLING COMPLETED

REVISED:
September 6, 1972
October 5, 1972
November 14, 1972
December 18, 1972
February 8, 1973
April 30, 1973

QUINTANA MINERALS CORPORATION

PROPERTY AND DRILLING

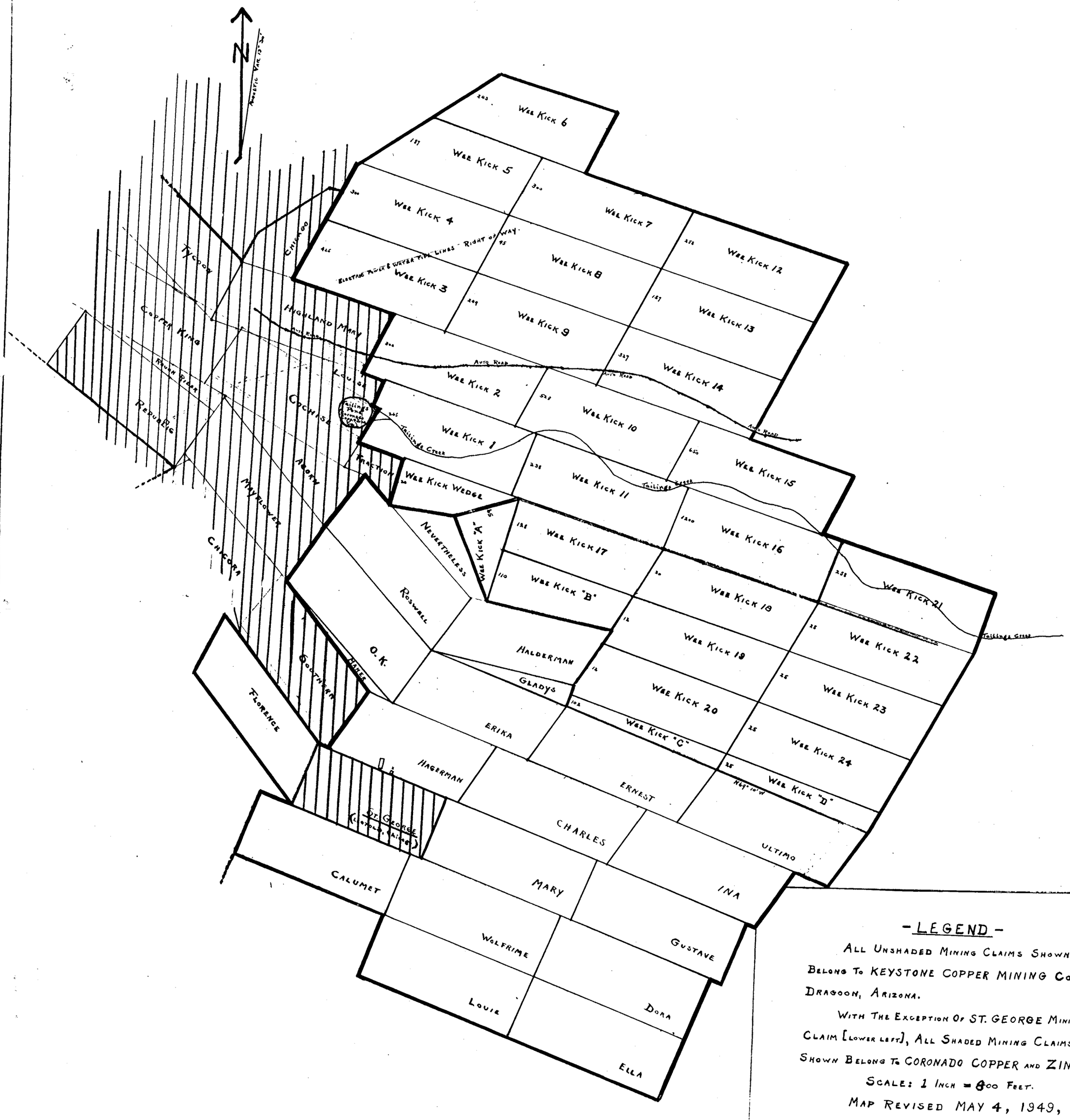
JOHNSON CAMP PROJECT

COCHISE COUNTY, ARIZONA

1" = 500'

500 0 500 1000 1500 FEET

JULY, 1972



- LEGEND -

All Unshaded Mining Claims Shown, Belong To KEYSTONE COPPER MINING Co., Dragoon, Arizona.

With The Exception Of ST. GEORGE Mining Claim [lower left], All Shaded Mining Claims Shown Belong To CORONADO COPPER AND ZINC Co.

SCALE: 1 Inch = 800 Feet.

MAP REVISED MAY 4, 1949,

By: NORMAN M. REHG.

SECTION 31

36 | 31

31 | 32

1 | 6

6 | 5

Legal Tender (Min. Pat)

T-1
Bdr. 540'
652': 216' @ 0.79% Cu (Ox)
1320': 110' @ 0.66% Cu (Sul)
T.D. 1501'

T-2
Bdr. 180'
582': 222' @ 0.76% Cu (Ox)
884': 238' @ 0.52% Cu (Ox)
T.D. 1464'

T-3
Bdr. 420'
No ore intervals
1657': 3' @ 0.36% Cu (Ox)
T.D. 1660'

T-4
Bdr. 376'
385': 248' @ 0.32% Cu (Ox)
1301': 299' @ 0.30% Cu (Sul)
T.D. 1600'

T-5
Bdr. 360'
982': 96' @ 0.44% Cu (Ox)
T.D. 1524'

T-6
Bdr. 215'
No ore intervals
T.D. 421'

T-7
Bdr. 565'
849': 85' @ 0.36% Cu (Ox)
1072': 77.5' @ 0.37% Cu (Ox)
1822': 52' @ 0.57% Cu (Sul)
T.D. 2450'

T-8
Bdr. 365'
1007.5': 121.6' @ 0.45% Cu (Ox)
1199': 118' @ 0.61% Cu (Ox & Sul)
T.D. 1528'

T-9
Bdr. 190'
1378': 98' @ 0.58% Cu (Sul)
1578': 96' @ 0.34% Cu (Sul)
1845': 166' @ 0.43% Cu (Sul)
T.D. 2183'

T-10
Bdr. approx. 300'
T.D. 310' (Rotary)

T-11
Bdr. 197'
1060': 69' @ 0.33% Cu (Ox)
T.D. 2275'

T-12
Bdr. 524'
1703': 38' @ 0.33% Cu (Sul)
T.D. 2065'

JD-1
Bdr. 188'
Log 87 ft (0.22-2.24% Cu)
T.D. 938' (La)

JD-2

JD-3
Bdr. 406'
80% Py
T.D. 748' (La)

JD-4
Bdr. 187'
Granite por.
T.D. 187'

JD-5
Bdr. 247'
Granite
T.D. 249'

JD-6
Bdr. 296'
Schist
T.D. 373'

JD-7
Bdr. 340'
371-492 & 0.008% Cu
(Rot. cuttings)
(Core 466-492
0.12% Cu
T.D. 512' (Ls)

JD-8
Bdr. 364'
Ls (lim/py or cpy)
T.D. 373'

1000 1050 1100 1150 1200 1250 1300 1350 1375 950

SECTION 1

SECTION 5

SECTION 7

T. 16 S.

R. 22 E.

R. 23 E.

Standard No. 8 (Min. Pat.)

Standard No. 9 (Min. Pat.)

N

EXPLANATION

- TEX location hole
- JD-4 United Geophysical drill hole
- T-6 Quintana drill hole
- 1100 Contours of total magnetic intensity in gammas (from Momag data of Minerals Exploration Company)
- * 250' equilateral triangle grid

REVISED:
October 19, 1972
November 28, 1972
May 2, 1973

QUINTANA MINERALS CORPORATION

TEX CLAIM GROUP

SHOWING MAGNETIC CONTOURS AND DRILL HOLES

1" = 200'

200 0 200 400 600 Feet

SEPTEMBER, 1972

PROGRESS REPORT

PRELIMINARY METALLURIGICAL TESTS

JOHNSON CAMP VENTURE # 2

MARCH 30, 1973

PROGRESS REPORT NO. 2

Preliminary Metallurgical Tests

Johnson Camp Venture #2

For:

Quintana Minerals Corporation
1892 West Grant Road
Tucson, Arizona 85705

Submitted by:

MOUNTAIN STATES RESEARCH AND DEVELOPMENT
Post Office Box 17960
Tucson, Arizona 85710

Project No. 2019

March 30, 1973

By _____
Roshan B. Bhappu
Vice President and
General Manager

INTRODUCTION

In the previous tests on selected samples (Groups I, II, III and IV) from Johnson Camp Venture #2 deposit, it was shown that about 70% to 80% of the total copper content of these samples can be extracted readily by acid leaching with varying acid consumption depending on the lime content of the sample (refer to MSRD report dated January 18, 1973). Thus:

Sample No.	Grade Cu (%)	Extraction Copper (%) (@ pH = 1)	Acid Consumption lbs/ton ore	Acid Consumption lbs/lb Cu
Group I	0.73	81.1	761	64.0
Group II	0.39	72.3	154	27.2
Group III	0.49	74.0	185	25.3
Group IV	0.53	73.7	112	14.4

The above results indicate that the major problem in the treatment of the above ore samples is obvicusly the high acid consumptions, especially for Group I ore, which presumably represents a major portion of the overall deposit. The acid consumptions incurred by Group II, III and IV ore samples on the other hand, though high, may be acceptable in a commercial plant in view of the availability of low cost acid in the future.

On the basis of the above data, it was recommended that further research efforts be concentrated only on Group I sample for determining the best treatment procedure.

The additional testing planned for this second phase of the metallurgical testing would include:

1. Effect of preheating on ammonia leaching systems with temperatures ranging from 300°F to 900°F and as high as 1050°C.

2. A combination of carbonate (marble or calcite) flotation followed by acid leaching with the hope of reducing the acid consumption.

3. Acid leaching on coarser ore fractions under vat-leaching conditions with the idea of forming synthetic gypsum ($CaSO_4$) coatings on coarser calcitic particles and thus preventing their further decomposition by acid. Such a technique may lead to an economic extraction of copper with acceptable acid consumption.

This Progress Report No. 2 covers the additional work carried out on the project since the presentation of the first report on January 18, 1973.

CONCLUSIONS

1. The heat treating of the Group I ore sample prior to ammonia leaching was not very effective. The maximum recovery of contained copper achieved amounted to only 24.3% on ore sample heated to 900°F.

2. The treatment scheme in which the Group I ore was subjected to flotation of calcitic gangue followed by acid leaching of the flotation tails also did not result in high enough over-all copper extraction. At best, about 67% copper extraction was achieved with an over-all acid consumption of 300 lbs. H_2SO_4/ton of original feed or 34 lbs. H_2SO_4/ lb. copper leached. In order to be economically viable, the extraction for the procedure under investigation should be in the range of 85 - 90%.

3. The proposed flowsheet involving acid leaching of coarser ore fractions (- 1/4" + 28-mesh) under vat leaching conditions and calcite flotation - acid leaching on finer ore (- 28-mesh) did not indicate economically acceptable over-all copper extraction although the acid consumption was reduced by about half of the original requirement. The reduction in over-all acid consumption was expected and attributed to the formation of synthetic gypsum on the surfaces of coarse calcite particles. Under the best test conditions, the extraction of copper using this flowsheet amounted to 65.3% with an acid consumption of 312 lbs. H_2SO_4/ton of ore treated. Obviously, this extraction is not high enough to sustain a practical vat leaching - flotation - agitation leaching operation.

4. On the basis of the results of the metallurgical studies on the treatment of Group I ore sample from the Johnson Camp Venture #2 deposit, it is quite apparent that none of the conventional or newly developed processes can be successfully applied for treating this ore. Since Group I ore constitutes the major portion of the over-all deposit it is doubtful if the deposit under investigation could be economically exploited under the current technological know-how. No doubt, that a successful treatment procedure will be found as a result of a great deal of additional research, but any such concentrated effort will involve considerable funds and time.

EXPERIMENTAL WORK AND RESULTS

1. EFFECT OF HEAT TREATING ON AMMONIA LEACHING

A series of ammonia $[(NH_4)_2CO_3 = 50$ gpl + $CO_2 = 25$ gpl$]$ agitation and bottle leach tests were conducted on Group I ore samples which were heat treated prior to leaching. The primary aim of the heat-treatment was to open up the pores of the layered silicate gangue present in the ore in order to facilitate the penetration of the leach solution and to prevent reabsorption of leached copper.

The results of these tests are given in the attached Leach Test Data sheets (DF-24, 25, 26, 27, 28, 29, 31 and 32) and are summarized as follows:

TABLE I

Test Number	Heat-Treatment Temperature	Heat-Treatment Time (hr.)	Leaching Time(hr)	Extraction Cu (%)	Remarks
DF-24(B)	Ambient	--	72	11.2	
DF-25(A)	1000°C	2	4	42.5	H_2SO_4 leach, Wt. Loss = 15.2%
DF-26(B)	1050°C	2	96	5.8	Wt. Loss = 20%
DF-27(A)	650°F	2	4	18.0	
DF-28(A)	300°F	2	4	8.2	
DF-29(A)	900°F	2	4	24.3	
DF-30(A)	Ambient	2	4	8.0	Unreported Data
DF-31(A)	850°F	2	4	20.4	
DF-32(A)	900°F	2	4	85.3	H_2SO_4 Leach (1151 lbs./ton)

(A) = Agitation leach and (B) = Bottle leach

The above results indicate that the heat treating of the Group I ore samples prior to ammonia leaching is not very effective. With heat treatment at suitable temperature (900°F) the extraction of copper is only 24.3% compared to 8.2% extraction at 300°F and 8.0% at ambient temperature. For this reason, the Johnson Camp Venture #2 ores cannot be treated economically by heat treatment-ammonia leaching process.

It should be noted that about 85% of the total copper can be recovered by heat treatment-H_2SO_4 leaching with a very high acid consumption amounting to 1151 lbs. acid/ton of ore. However, the same results are achievable without preheating of the ore.

LEACH TEST DATA

Type: Bottle Leach Date: 1/5/73

Project No.: 2019 Sample No.: Group I Test No.: DF-24

Weight Solids: 200 grams Size or Grind: - 35 mesh

Leach Conditions

Solids: 50 (%) Temperature: Ambient

Reagent: NH_4OH Auxil. Reagent: $(NH_4)_2CO_3$ (CO_2)

Concentration: NH_3 = 50 gpl Concentration: CO_2 = 25 gpl

pH: Time: 72-Hour leach

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2486	Preg. Soln.	154.0 ml	0.62 gpl	0.0955	7.91
2487	Wash Soln.	570.0 ml	0.07 gpl	0.0399	3.31
2481	Leach Residue	198.4 g	0.54 %	1.0714	88.78
	Total			1.2068	100.00

Head (calculated) 0.603 % Cu

Head (assayed) 0.66 % Cu

Remarks

Bottle leaches were run capped.

Accountability = 91.3%

Extraction: 11.22 (%) Reagent Consumption:

Performed By:

LEACH TEST DATA

Type: Agitation Leach Date: 1/10/73

Project No.: 2019 Sample No.: Group I Test No.: DF-25

Weight Solids: 100 grams (84.8) Size or Grind: - 35 mesh

Leach Conditions

Solids: 50 (%) Temperature: Ambient

Reagent: H_2SO_4 Auxil. Reagent:

Concentration: 96% Concentration:

pH: 1.0 Time: 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	10:40	12.2	12.2 initial				
	11:15			20.0 ml			
	11:30			6.35 ml			
	2:40			Total Acid Consumption = 26.35 ml			

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg. Soln.	110.0 ml	0.74 gpl	0.0814	15.50
Wash Soln.	590.0 ml	0.24 gpl	0.1420	27.00
Leach Residue	94.5 g	0.32 %	0.3020	57.50
Totals			0.5254	100.00

Head (calculated) 0.62% Cu

Head (assayed) 0.66% Cu

Remarks

Calcined for 2 hours, 100 gram sample. Temperature 950 - 1050°C. Cooled and then acid leached. Loss of weight 15.2 grams during calcination.

Accountability = 94%

Extraction: 42.50 (%) Reagent Consumption: 1095 lbs acid/ton ore
211 lbs acid/lb Cu leached

Performed By:

LEACH TEST DATA

Type Bottle Leach Date 1/11/73

Project No. 2019 Sample No. Group I Test No. DF-26

Weight Solids 115 grams (94.5) Size or Grind - 35 mesh

Leach Conditions

Solids 31 (%) Temperature Ambient

Reagent NH_4OH Auxil. Reagent $(NH_4)_2CO_3$

Concentration NH_3 = 50 gpl Concentration CO_2 = 25 gpl

$(NH_4)_2$ = 2 gpl Ammonium Persulphate

pH Time 96-Hour Leach

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2630	Preg. Soln.	198.0 ml	0.14 gpl	0.0277	4.69
2631	Wash Soln.	700.0 ml	< 0.01 gpl	0.0070	1.18
2654	Leach Residue	104.9 g	0.53 %	0.5560	94.13
	Totals			0.5907	100.00

Head (calculated) 0.59 % Cu

Head (assayed) 0.66 % Cu

Remarks

Calcined 115 gram sample at 1050°C for 2 hours. Weight loss 20.5 grams.

Accountability = 85%

Extraction 5.8 (%) Reagent Consumption

Performed By

Type Agitation Leach Date 1/15/73

Project No. 2019 Sample No. Group I Test No. DF-27

Weight Solids 20 grams Size or Grind - 35 mesh

Leach Conditions

Solids 33 (%) Temperature Ambient

Reagent NH_4OH Auxil. Reagent $(NH_4)_2CO_3$

Concentration NH_3 = 50 gpl Concentration CO_2 = 25 gpl
Ammonium Persulphate = 2.0 gpl

pH Time 4 hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2652	Preg. Soln.	48.0 ml	0.31 gpl	0.015	12.8
2653	Wash Soln.	210.0 ml	0.03 gpl	0.006	5.2
2655	Leach Residue	19.2 g	0.50 %	0.096	82.0
	Totals			0.117	100.0

Head (calculated) 0.59 % Cu

Head (assayed) 0.66 % Cu

Remarks

Heated sample in muffle furnace for 2 hours, temperature 600°F - 650°F and then leached with acid.

Accountability = 89.5%

Extraction 18.0 (%) Reagent Consumption

Performed By

LEACH TEST DATA

Type Agitation Leach Date 1/16/73

Project No. 2019 Sample No. Group I Test No. DF-28

Weight Solids 20 grams Size or Grind - 35 mesh

Leach Conditions

Solids 33 (%) Temperature Ambient

Reagent NH_4OH Auxil. Reagent $(NH_4)_2CO_3$

Concentration NH_3 = 50 gpl Concentration CO_2 = 25 gpl
Ammonium Persulphate = 2 gpl

pH Time 4 hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2659	Preg. Soln.	42.0 ml	0.15 gpl	0.0063	5.72
2660	Wash Soln.	270.0 ml	0.01 gpl	0.0027	2.45
2667	Leach Residue	19.4 g	0.52 %	0.1010	91.83
	Totals			0.1100	100.00

Head (calculated) 0.58 % Cu

Head (assayed) 0.66 % Cu

Remarks

Heat sample at 300°F for 2 hours and leach.

Accountability = 88%

Extraction 8.17 (%) Reagent Consumption

Performed By

Type Agitation Leach Date 1/16/73

Project No. 2019 Sample No. Group I Test No. DF-29

Weight Solids 20 grams Size or Grind - 35 mesh

Leach Conditions

Solids 33 (%) Temperature Ambient

Reagent NH_4OH Auxil. Reagent $(NH_4)_2CO_3$

Concentration NH_3 = 50 gpl Concentration CO_2 = 25 gpl
Ammonium Persulphate = 2 gpl

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2665	Preg. Soln.	33.0 ml	0.75 gpl	0.0248	20.06
2666	Wash Soln.	260.0 ml	0.02 gpl	0.0052	4.20
2668	Leach Residue		0.48 %	0.0936	75.74
	Totals			0.1236	100.00

Head (calculated) 0.61% Cu

Head (assayed) 0.66% Cu

Remarks

Heat sample at 900°F for 2 hours. Cool and leach.

Accountability = 92.5%

Extraction 24.26 (%) Reagent Consumption

Performed By

Type Agitation Leach Date 1/17/73

Project No. 2019 Sample No. Group I Test No. DF-31

Weight Solids 100 grams Size or Grind - 35 Mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent NH_4OH Auxil. Reagent $(NH_4)_2CO_3$

Concentration NH_3 = 70 gpl Concentration CO_2 = 50 gpl
$(NH_4)_2S_2)_4$ = 2 gpl

pH Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2682	Preg. Soln.	64.0 ml	1.45 gpl	0.0928	12.75
2683	Wash Soln.	280.0 ml	0.20 gpl	0.0560	7.69
2689	Leach Residue	98.1 g	0.59 %	0.5790	79.56
	Totals			0.7278	100.00

Head (calculated) 0.73 % Cu

Head (assayed) 0.66 % Cu

Remarks

Heat 100 grams sample to 850°F and hold for 2 hours. Leach 4 hours.

Accountability = 110%

Extraction 20.44 (%) Reagent Consumption

Performed By

LEACH TEST DATA

Type Agitation Leach Date 1/17/73

Project No. 2019 Sample No. Group I Test No. DF - 32

Weight Solids 100 grams Size or Grind - 35 mesh

Leach Conditions

Solids 50 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	12:00		8.5				
	12:08		1.0	12.10 ml			
	12:15		1.0	4.95			
	12:55		1.0	4.15			
	2:00		1.0	8.95			
	4:00		1.0	2.45			

Total Acid Added = 32.60 ml

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2718	Preg. Soln.	55.8 ml	2.92 gpl	0.1629	24.57
2719	Wash Soln.	320.0 ml	1.26 gpl	0.4030	60.78
2725	Leach Residue	98.4 g	0.097 %	0.0970	14.75
	Totals			0.663	100.00

Head (calculated) 0.663 % Cu

Head (assayed) 0.66 % Cu

Remarks

Heat sample at 900°F for 2 hours. Leach 4 hours.

Accountability = 100.5%

Extraction 85.35 (%) Reagent Consumption 1151 lbs acid/ton of ore

102.3 lbs acid/lb Cu leached

Performed By

2. CARBONATE FLOTATION - ACID LEACHING SYSTEM

Realizing that the excessive consumption of acid in the case of Group I type ore is due to the high content of lime (marble) in the ore, efforts were made to float off the major portion of the carbonate rock using conventional calcite flotation system. The tailings from this flotation step will then be subjected to acid leaching with the advantage of lower over-all acid consumption.

A series of flotation-leach tests were performed using fatty acid type anionic collectors (oleic acid and Pamak 25) to selectively float calcite away from the copper bearing minerals followed by acid leaching of the flotation tails.

One additional flotation test (DF-38) was run to determine the effectiveness of amine-flotation of chrysocolla ($CuSiO_3 \cdot 2\ H_2O$) and other copper-bearing minerals while depressing the calcite and other gangue minerals.

The results of these tests DF-33 to DF-37 are reported in the attached Data Sheets and are summarized in Table II.

The results of the above tests reveal that the technique encompassing calcite flotation, followed by acid leaching of flotation tails does not result in high enough over-all copper extractions. At the best (Test DF-34), about 67% extraction of the total copper was achieved with an over-all acid consumption of 300 lbs. H_2SO_4/ton original feed or 34 lbs. H_2SO_4/lb. Cu leached. Attempts to lower the acid consumption (Test DF-37) resulted in lowering the over-all copper extraction to 58.5% with an acid consumption

amounting to 73 lbs. H_2SO_4/ton original feed or 9.5 lbs. H_2SO_4/ lb. Cu leached. In this latter case, the acid consumption is very favorable, but this is attained at the expense of copper recovery. For this process to be economically viable, the recoveries should be in the range of 85 - 90%.

Results of Test DF-38 indicated that the selective flotation of copper-bearing minerals by amine type collector was not successful. Most of the chrysocolla remained in the tails along with the major portion of calcitic gangue.

TABLE II: SUMMARY OF RESULTS

Preliminary Lime Flotation Tests with Acid Leaching of Flotation Tails

Test No.	% Wt. Ore to Leach	Calc. Head % Cu	Tailing, % Cu			Cu Recovery, %				Acid Consumption		
			Lime Conc.	Leach Residue	Comb. Tail	Float %	Leach %	Overall %	lb/ton	# / # Cu	#/ton Orig. Ore	#/ton Leached
DF-1	100.00	.73	--	.14	.14	--	81.1	81.1	11.9	65	777	777
DF-33	72.51	.714	.246	.14	.168	90.5	83.6	76.5	10.9	54	584	805
DF-34	57.76	.665	.294	.18	.221	81.3	82.2	66.8	8.9	34	300	503
DF-35	16.80	.666	.42	.39	.394	49.0	83.0	40.6	5.4	6.2	33.6	200
DF-36	23.00	.675	.290	.27	.300	66.0	83.0	54.8	7.2	10	72	300
DF-37	24.00	.65	.260	.32	.277	70.3	83.2	58.5	7.7	9.5	73	310
DF-38	64.00	Not Effective - Flotation was not selective enough.										

(A) Flotation Conditions

1) DF-33 & 34: Oleic acid, C. Reagent 710, pH = 8.5
2) DF-35 : Pamak 25 (2.0 lb/ton), Na_2SiO_3 (0.3 lb/ton), pH = 9.6.
3) DF-36 : Pamak 25 (1.5 lb/ton), Na_2SiO_3 (0.3 lb/ton), pH = 8.5.
4) DF-37 : Pamak 25 (1.5 lb/ton), Na_2SiO_3 (0.3 lb/ton), NH_4OH = pH 9.6.
5) DF-38 : Amine D-Acetate (1.0 lb/ton), Kerosene (1.5 lb/ton), Na_2CO_3 (0.3 lb/to
In this test efforts were made to selectively float chrysocolla and other copper minerals while depressing the calcitic gangue.

(B) Note: Cleaning of lime rougher concentrate in Test DF-34, 35, 36 and 37 would probably improve copper rejection to tailing, and therefore, overall copper recovery, at expense of somewhat higher acid consumption.

MOUNTAIN STATES RESEARCH & DEVELOPMENT

Project No. 2019

FLOTATION TEST LOG SHEET | Date 1/18/73 | Table No. | Test No. DF-33

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.	Na_2CO_3	NaSil	NaCN	710	Oleic Acid		Machine RPM
Grind	5	67			1.2	1.4	.20				
Condition #1	1	22	9.0					.12			1600
Rougher #1	6							.12 added @ 2 min .12 added @ 4 min			
Condition #2	1							.12	.11 added @ 2 min		
Rougher #2	6							.12 .12	.11 .11 added @ 4 min		

Remarks

Only small amount limestone floating with 710 alone
Good lime float with combination of Oleic + 710

METALLURGICAL RESULTS

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu						Cu		
Heads Assay		0.66								
Lime Ro. Conc.	27.49	0.246						9.9		
Lime Ro. Tail	72.51	0.85						90.1		
Calc. Head	100.00	0.684						100.0		

Ratio of Concentration

Remarks Lime Ro. Tail to acid consumption test - See DF 33

Performed By_____

Type Agitation Leach Date 1/19/73

Project No. 2019 Sample No. Group I Test No. DF-33 (McA-1)

Weight Solids 100 grams Size or Grind

Leach Conditions

Solids 33 (%) Temperature Ambient

Reagent H_2SO_4 Auxil. Reagent

Concentration 96% Concentration

pH 1.0 Time 4 Hours

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:55		.75	22.8 ml			
	4:00		1.0				

Metallurgical Balance

	Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
2787	Preg. Soln.	150.0 ml	2.65 gpl	0.398	44.66
2788	Wash Soln.	660.0 ml	0.54 gpl	0.356	39.96
2828	Leach Residue	92.7 g	0.148 %	0.137	15.38
	Total			0.891	100.00

Head (calculated) 0.89%

Head (assayed) 0.85%

Remarks

Leached rougher tails after flotation.

% Accountability = 94.1%

Extraction 84.62 (%) Reagent Consumption 805 lbs/ton of Ro. Tail
53.3 lbs acid/lb Cu leached in Ro. tail.

Performed By

FLOTATION TEST LOG SHEET | Date | Table No. | Test No. DF 33

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton					
	Time Mins	Solids (%)	pH	Temp.						

Remarks

SUMMARY SHEET

METALLURGICAL RESULTS Flotation + Leaching of Ro. Tail

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads Assay		0.66								
Lime Ro. Conc.	27.49	0.246			.068			9.5		
Lime Ro. Tail										
1) Preg. + Wash	587.33	0.093			.546			76.5		
2) Leach Residue	67.49	0.14			.100			14.0		
3) Loss in Weight	5.02									
Calculated Head	100.00	0.714						100.0		

Ratio of Concentration

Remarks

Cu Recovery, lbs/ton original ore	10.9
Acid Consumption, lbs/lb. Cu	53.5
lbs/ton original ore	584
lbs/ton lime Ro-Tlg	805

Performed By _____

FLOTATION TEST LOG SHEET | Date 1/21/73 | Table No. | Test No. McA-2 (DF-34)

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.	Na_2CO_3	NaSil	NaCN	710	Oleic Acid		Machine RPM
Grind	5	67			1.2	1.8	0.20				
Condition	1	22	9.0					0.20	0.22		1800
Rougher	6							0.20	0.22	staged	@ 2 min
								0.20	0.22	staged	@ 4 min

Remarks

Appeared to be good selective lime float.

Cleaning may help Cu rejection from the calcite concentrate and, thus, would report in the leach feed.

METALLURGICAL RESULTS

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads Assay		0.66								
Lime Ro. Conc.	42.24	0.294			0.124			17.8		
Lime Ro. Tail	57.76	0.99			0.573			82.2		
Calc. Head	100.00	0.69			0.697			100.0		

Ratio of Concentration

Remarks

Performed By

LEACH TEST DATA

Project No. 2019 Sample Flotation Tails Test No. DF-34

Weight Solids 100 grams Size of Grind -35 mesh

Leach Conditions

Solids 33 (%) Temperature Ambient pH 1.0 Time 4 hours

Reagent H_2SO_4 Auxil. Reagent

Concentration Concentration

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay ( )	Extraction (%)	Reagent Consump. ( )
	11:00		0.85	15.2 ml			
	3:00		1.00				

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Preg. Soln.	160 ml	2.69 gpl	0.430	46.38
Wash Soln.	390 ml	0.84 gpl	0.328	35.38
Leach Residue	93.7 g	0.18%	0.169	18.24
Total			0.927	100.00

Head (Calculated) 0.927% Cu

Head (Assayed) 0.915% Cu

Remarks: Leached rougher tails after lime flotation.

Extraction 81.76 (%) Reagent Consumption 503 lbs. Acid/ton Ro. Tailing
300 lbs. Acid/ton ore
29 lbs/lb Cu leached

Operator

PROJECT NO. 2019 | Date 1/20/73 | Table No. | Test No. DF-34

FLOTATION TEST LOG SHEET

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.							

Remarks

SUMMARY SHEET

METALLURGICAL RESULTS Flotation & Leaching of Ro. Tail

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads		0.66								
Lime Ro. Conc.	42.24	0.294			0.124			18.8		
Lime Ro. Tail	57.76	0.927			0.535					
1. Preg. & Wash	550.00ml	0.138			0.437			66.8		
2. Leach Residue	54.20	0.180			0.098			14.4		
3. Loss in Weight	4.56									
Calculated Head	100.00				0.659					

Ratio of Concentration

Remarks

Cu Recovery, lbs/ton original ore	10.1
Acid Consumption, lbs/lb Cu	29.0
lbs/ton original ore	300.0
lbs/ton Ro. Tailings	503.0

PROJECT NO. 2019 | Date 1/23/73 | Table No. | Test No. DF-35

FLOTATION TEST LOG SHEET

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.	Na_2SiO_3		Pamak 25				
Grind	5	67									
Condition	1	22	9.6		0.3		1.5				
Rougher	6	22									

Remarks

Lime flotation with Pamak 25.
Rougher tailings leached with H_2SO_4 at pH 1.0.

SUMMARY

METALLURGICAL RESULTS - Flotation + Leaching

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads Assay		0.66								
Lime Ro. Conc.	83.2	0.42			0.344			51.4		
Lime Ro. Tail	16.8	1.92			0.322			--		
1) Preg + Wash	595 ml	0.045			0.268			40.6		
2) Leach Residue	16.7 g	0.32			0.054			8.0		
Calculated Head	100.00	0.67			0.666			100.00		

Ratio of Concentration

Remarks

Cu Recovery, lbs./ton original ore = 5.4
Acid Consumption, lbs/lb. Cu leached = 6.2
lbs/ton original ore = 33.6
lbs/ton lime Ro. Tail = 200

PROJECT NO. 2019 | Date 2/2/73 | Table No. | Test No. DF-36

FLOTATION TEST LOG SHEET

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.	Na_2SiO_3		Pamak 25				
Grind	5	67									
Condition	1	22	8.5		0.3		1.5				
Rougher	6	22									

Remarks

Lime flotation with Pamak 25.

Rougher tailings leached with H_2SO_4 at pH 1.0.

SUMMARY

METALLURGICAL RESULTS - Flotation + Leaching

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads Assay		0.66								
Lime Ro. Conc.	77.0	0.29			0.225			33.0		
Lime Ro. Tail	23.0	1.89			0.450			--		
1) Preg + Wash	590 ml	0.055			0.374			54.8		
2) Leach Residue	22.8 g	0.273			0.076			12.2		
Calculated Head	100.0	0.68			0.675			100.0		

Ratio of Concentration

Remarks

Cu Recovery, lbs./ton original ore = 7.2

Acid Consumption, lbs./lb. Cu leached = 10.0
lbs./ton original ore = 72
lbs./ton Lime Ro. Tail = 300

PROJECT NO. 2019 | Date 2/6/73 | Table No. | Test No. DF-37

FLOTATION TEST LOG SHEET

CONDITIONS AND REAGENTS

Point of Addition	Conditions				Reagents-Pounds Per Ton						
	Time Mins	Solids (%)	pH	Temp.	Na_2SiO_3		Pamak 25	NH_4OH			
Grind	5	67									
Condition	1	22	9.6		0.3		1.5	--			
Rougher	6	22									

Remarks

Lime flotation with Pamak 25 and pH adjusted with ammonia.
Rougher tailings leached with H_2SO_4 at pH 1.0.

SUMMARY

METALLURGICAL RESULTS Flotation + Leaching

Product	Weight (%)	Assays (%)			Contents			Distribution (%)		
		Cu			Cu			Cu		
Heads Assay		0.66								
Lime Ro. Conc.	76.0	0.26			0.200			30.8		
Lime Ro. Tail	24.0	1.90			0.452			--		
1) Preg. + Wash	585 ml	0.065			0.380			58.5		
2) Leach Residue	23.9 g	0.32			0.077			10.7		
Calculated Head	100.00	0.65			0.652			100.0		

Ratio of Concentration

Remarks

Cu Recovery, lbs./ton original ore = 7.7

Acid Consumption, lbs./lb. Cu leached = 9.5
lbs./ton Original Ore = 72
lbs./ton Lime Ro. Tail = 310

3. MODIFIED VAT LEACHING TO REDUCE ACID CONSUMPTION

One of the ways under which excessive acid consumption may be reduced is by inducing synthetic gypsum coatings on coarse calcitic gangue particles, which retard their dissolution. Such coating can be induced by allowing strong acid solution to come in contact with coarse ore particles as would be possible under vat leaching conditions. In this case, the ore would have to be reduced in size sufficiently to expose most of the oxide copper mineralization. Also, for the technique to work effectively it would be necessary to remove the fine fraction (say minus 28 or 35 mesh) of the ore which would be the higher acid consuming material. This fine fraction may then be subjected to flotation (without grinding) to remove most of the calcite followed by acid agitation leaching of the flotation tails.

The proposed flowsheet for the above treatment procedure is shown in Figure 1.

As can be seen, the ore is crushed through 1/4-inch and screened on 28-mesh to obtain a coarse fraction (-1/4-inch + 28-mesh) which goes to vat leaching and a fine fraction (- 28-mesh) which is subjected to flotation for calcite removal. The tailings from this step are then leached in a conventional agitation-leach process or the Leach-Precipitation-Flotation process.

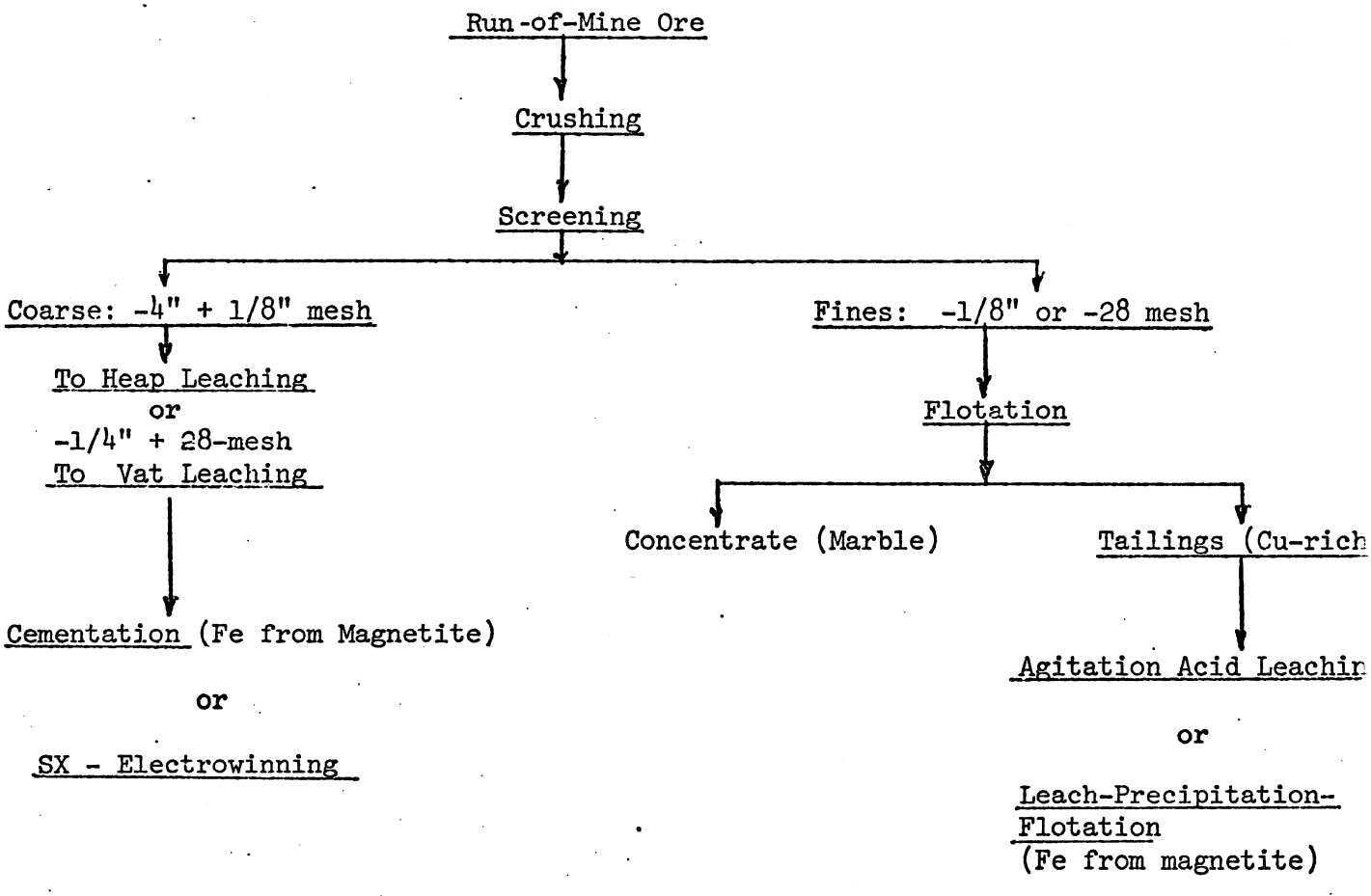
The selection of 28 mesh as the dividing size for coarse and fine fraction is based on the following reasonings:

1. Minus 28-mesh ore fraction can be subjected to flotation without additional grinding.
2. This size fraction can be readily handled in the agitation leach step to follow after flotation.
3. The size-assay analyses as shown in Tables III, IV, V and VI for

Figure 1

PROPOSED FLOWSHEET

GROUP I ORE



PREMISE

1) The proposed flowsheet is based on effective acid leaching of coarse ore under which a protactive layer of gypsum is formed on coarser particles. Such a coating prevents further dissolution of marble and thus, excessive acid consumption.

2) This phenomenon is not favorable for treatment of fines which are subjected to flotation to remove most of the marble followed by leaching of flotation tailings.

Group I, II, III and IV samples respectively clearly indicate that the plus 28-mesh is a convient split because a major (about 67%) portion of the copper content is distributed above this size.

It should be noted that the proposed process would be a more expensive system as compared to heap leaching and for this reason, the process must realize an over-all recovery of 85 - 90% in order for it to be economically attractive.

A series of vat-leach simulation tests were run on minus 1/4"+ 28-mesh fraction with varying concentrations of acid in effort to optimize the gypsum coating on coarse calcite particles. It was found that a 50 gpl H_2SO_4 solution provided the best coating conditions. However, the acid consumption was still quite high.

The experimental data and results of a typical leach test are shown in the following Leach Test Data sheets and these results are graphically presented in Figure 2.

As can be seen, an over-all extraction of about 65% copper was attained over a 10-day leaching period with an acid consumption of 429 lbs. H_2SO_4/ton of ore (-1/4" + 28-mesh fraction) treated or about 50 lbs. H_2SO_4/ lb. Cu leached. These results represent a reduction of over-all acid consumption from about 800 lbs. H_2SO_4/ton ore to about 430, a reduction of nearly half of the original acid consumption level. However, the extraction of copper amounting to 65% is not high enough for this type of leaching system (at Inspiration Vat leaching recovers about 90 to 94% copper). Also, the rate of leaching is slower than is achieved in commercial vat leaching operation (90% extraction

in about 6 days leaching time).

The over-all metallurgical results of the proposed process can be evaluated from the following metallurgical balance.

Size Fraction	Treatment Step	Weight (%)	Extraction (%)	Over-all Extraction (%)	Acid Consumption lbs H_2SO_4/ton ore
- 1/4" + 28-Mesh	Vat Leaching	68	65	44.1	294
- 28-Mesh	Flotation + Leaching	32	66*	21.2	18
Total		100		65.3	312

* Flotation + leach step recovery is estimated at 65% under optimum conditions.

The above table shows that the over-all recovery of copper using the proposed process would amount to 65.3% with an acid consumption of 312 lbs. H_2SO_4/ton of ore treated. For the Group I ore assaying 0.66% Cu, this amounts to a recovery of 8.6 lbs. copper with an acid consumption of 36.3 lbs. H_2SO_4 per lb. copper recovered.

It is understandable that the above metallurgical results may not be economically favorable for a commercial operation (set target of about 85 - 90% over-all recovery). However, it does provide a possible process for treating high lime containing ore (Group I), especially, if it could be blended with lower lime containing ores (Group II, III and IV).

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SIEVE TESTS
MADE WITH

Tyler Standard Screen Scale Sieves

TABLE III

Name Quintana Minerals Date 11/29/72 Test Number

Address Made by

Material Group I Composite Moisture

SCREEN SCALE RATIO √2 or 1.414				(A) Assay Number			(B) Weight			(C) Assay % Cu			(D) % Copper Distribution			(E)			(F)		
				Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes		
Openings		Tyler Mesh	U. S. No.	Weight Between Sieves		Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)
Inches	Milli-meters																				
1.050	26.67																				
.742	18.85																				
.525	13.33																				
.371	9.423																				
.263	6.680	3			1580		137.3	13.25			0.42			8.05							
.185	4.699	4	4		1581		180.3	17.40			0.55			13.85	21.90						
.131	3.327	6	6		1582		145.0	13.99			0.53			10.74	32.64						
.093	2.362	8	8		1583		114.2	11.02			0.65			10.36	43.00						
.065	1.651	10	12		1584		79.2	7.64			0.73			8.08	51.08						
.046	1.168	14	16		--		---	---			--			---	---						
.0328	.833	20	20		--		---	---			--			---	---						
.0232	.589	28	30		1585		150.5	14.52			0.81			17.02	68.10						
.0164	.417	35	40		1586		30.2	2.91			0.95			3.99	72.09						
.0116	.295	48	50		1587		27.0	2.60			1.03			3.89	75.98						
.0082	.208	65	70		1588		21.0	2.03			1.05			3.08	79.06						
.0058	.147	100	100		1589		24.0	2.32			0.97			3.26	82.32						
.0041	.104	150	140		--		---	---			--			---	---						
.0029	.074	200	200		1590		41.5	4.00			0.85			4.92	87.24						
.0000	.000	Pan	Pan		1591		86.0	8.32			1.06			12.76	100.00						
			Totals				1036.2	100.00						100.00							

REMARKS Head 1579 T Cu 0.66% Calculated 0.69% Cu
Ox Cu 0.58%

By

Note: Sieves omitted may be indicated with a dash (—) in weight column.

THE W. S. TYLER COMPANY, CLEVELAND, OHIO 44114, U. S. A. 2-65

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SIEVE TESTS
MADE WITH

Tyler Standard Screen Scale Sieves

TABLE IV

Name Quintana Minerals Date 11/29/72 Test Number

Address Made by

Material Group II Composite Moisture

SCREEN SCALE RATIO √2 or 1.414				(A) Assay Number			(B) Weight			(C) Assay % Cu			(D) % Copper Distribution			(E)			(F)		
				Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes		
Openings		Tyler Mesh	U. S. No.	Weight Between Sieves		Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)
Inches	Millimeters																				
1.050	26.67																				
.742	18.85																				
.525	13.33																				
.371	9.423																				
.263	6.680	3			1605		38.0	3.92			0.27			3.03							
.185	4.699	4	4		1606		141.0	14.55			0.26			10.79	13.82						
.131	3.327	6	6		1607		129.0	13.31			0.27			10.25	24.07						
.093	2.362	8	8		1608		113.0	11.66			0.29			9.65	33.72						
.065	1.651	10	12		1609		79.0	8.20			0.29			6.80	40.52						
.046	1.168	14	16		--		---	---			--			---	---						
.0328	.833	20	20		--		---	---			--			---	---						
.0232	.589	28	30		1610		158.0	16.30			0.34			15.81	56.33						
.0164	.417	35	40		1611		33.5	3.46			0.37			3.66	59.99						
.0116	.295	48	50		1612		29.5	3.04			0.44			3.83	63.82						
.0082	.208	65	70		1613		22.0	2.27			0.47			3.06	66.88						
.0058	.147	100	100		1614		23.3	2.40			0.50			3.43	70.31						
.0041	.104	150	140		--		---	---			--			---	---						
.0029	.074	200	200		1615		40.0	4.13			0.49			5.77	76.08						
.0000	.000	Pan	Pan		1616		162.5	16.76			0.50			23.92	100.00						
			Totals				969.3							100.00							

REMARKS Head 1617

T Cu 0.37% Calculated 0.35% Cu
Ox Cu 0.26%

By

Note: Sieves omitted may be indicated with a dash (—) in weight column.

THE W. S. TYLER COMPANY, CLEVELAND, OHIO 44114, U. S. A. 2-65

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SIEVE TESTS
MADE WITH

Tyler Standard Screen Scale Sieves

TABLE V

Name Quintana Minerals Date 11/29/72 Test Number

Address Made by

Material Group III Composite Moisture

SCREEN SCALE RATIO √2 or 1.414				(A) Assay Number			(B) Weight			(C) Assay % Cu			(D) Copper Distribution			(E)			(F)		
				Time: Minutes			Time: Minutes			Time: Minutes			Time: Minutes			Time: Minutes			Time: Minutes		
Openings		Tyler Mesh	U.S. No.	Weight Between Sieves		Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)
nches	Milli-meters																				
.050	26.67																				
.742	18.85																				
.525	13.33																				
.371	9.423																				
.263	6.680	3			1618		71.3	6.62			0.36			5.42							
.185	4.699	4	4		1619		154.5	14.35			0.37			12.10	17.52						
.131	3.327	6	6		1620		137.2	12.74			0.43			12.49	30.01						
.093	2.362	8	8		1621		115.7	10.74			0.43			10.53	40.54						
.065	1.651	10	12		1622		84.1	7.81			0.37			6.59	47.13						
.046	1.168	14	16		----		---	---			---			---	---						
.0328	.833	20	20		----		---	---			---			---	---						
.0232	.589	28	30		1623		191.3	17.76			0.42			17.00	64.13						
.0164	.417	35	40		1624		46.9	4.36			0.39			3.87	68.00						
.0116	.295	48	50		1625		43.8	4.07			0.44			4.08	72.08						
.0082	.208	65	70		1626		34.0	3.16			0.45			3.24	75.32						
.0058	.147	100	100		1627		36.0	3.34			0.47			3.58	78.90						
.0041	.104	150	140		----		---	---			---			---	---						
.0029	.074	200	200		1628		58.4	5.42			0.50			6.18	85.08						
.0000	.000	Pan	Pan		1629		103.7	9.63			0.68			14.92	100.00						
			Totals				1076.9	100.00						100.00							

REMARKS Head 1630

T Cu 0.45% Calculated 0.44% Cu
Ox Cu 0.37%

By

Note: Sieves omitted may be indicated with a dash (—) in weight column.

THE W. S. TYLER COMPANY, CLEVELAND, OHIO 44114, U. S. A. 2-68

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SIEVE TESTS
MADE WITH

Tyler Standard Screen Scale Sieves

TABLE VI

Name Quintana Minerals Date 11/29/72 Test Number

Address Made by

Material Group IV Composite Moisture

SCREEN SCALE RATIO √2 or 1.414				(A) Assay Number			(B) Weight			(C) Assay % Cu			(D) Copper Distribution			(E)			(F)		
				Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes			Time:____ Minutes		
Openings		Tyler Mesh	U. S. No.	Weight Between Sieves		Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)	Weight Between Sieves	% Between Sieves	Cum. % (Total % on each Sieve)
nches	Milli-meters																				
.050	26.67																				
.742	18.85																				
.525	13.33																				
.371	9.423																				
.263	6.680	3			1631		131.4	12.99			0.22			5.84							
.185	4.699	4	4		1632		159.6	15.77			0.33			10.62	16.46						
.131	3.327	6	6		1633		115.4	11.40			0.37			8.62	25.08						
.093	2.362	8	8		1634		99.0	9.78			0.39			7.78	32.86						
.065	1.651	10	12		1635		74.7	7.38			0.46			6.92	39.78						
.046	1.168	14	16		--		---	---			--			---	---						
.0328	.833	20	20		---		---	---			--			---	---						
.0232	.589	28	30		1636		166.3	16.43			0.52			17.44	57.22						
.0164	.417	35	40		1637		39.4	3.89			0.59			4.70	61.92						
.0116	.295	48	50		1638		37.0	3.65			0.60			4.47	66.39						
.0082	.208	65	70		1639		28.1	2.78			0.67			3.80	70.19						
.0058	.147	100	100		1640		20.6	2.04			0.70			2.92	73.11						
.0041	.104	150	140		1641		23.8	2.35			0.71			3.41	76.52						
.0029	.074	200	200		1642		26.4	2.62			0.81			4.33	80.85						
.0000	.000	Pan	Pan		1643		90.2	8.92			1.05			19.15	100.00						
			Totals				1011.9	100.00						100.00							

REMARKS

By

Note: Sieves omitted may be indicated with a dash (—) in weight column.

THE W. S. TYLER COMPANY, CLEVELAND, OHIO 44114, U. S. A. 2-68

2/20 - 3/2/73

LEACH TEST DATA

Project No. 2019 Sample Group I Test No. DFHM-41

Type Vat Leach Weight Solids 4000 grams Size of Grind -1/4 + 28 Mesh

Leach Conditions

Solids (%) Temperature Ambient pH Time 10 days

Reagent H_2SO_4 Auxil. Reagent

Concentration (see remarks) Concentration

Sample Date/#	Time ( )	Volume (ml)	pH	Reagent Add/Use	Assay (Cu)	Extraction (%)	Reagent Consump. ( )
2/20 1	4:00pm	2275	1.5		1.68 gpl	14.73	
2/21 2	7:30am	3550	1.5		0.72 gpl	24.59	
2/21 3	10:00am	610	1.5		0.48 gpl	25.71	
2/21 4	4:30pm	1700	1.5		0.43 gpl	28.52	
2/22 5	7:15am	2475	1.4		0.49 gpl	33.17	
2/22 6	10:00am	800	1.0		0.36 gpl	34.29	
2/22 7	4:15pm	1400	1.0		0.35 gpl	36.18	
2/22 8	7:15am	1600	1.0		0.35 gpl	38.44	
2/23 9	10:00am	1400	1.0		0.20 gpl	39.52	
2/23 10	3:00pm	2000	1.0		0.24 gpl	41.36	
2/26 11	7:30am	1700	1.0		0.18 gpl	53.16	
2/27 12	7:30am	8500	1.0		0.11 gpl	56.77	
2/28 13	7:30am	7850	1.0		0.117gpl	60.31	
3/1 14	7:30am	7920	1.0		0.103gpl	63.47	
3/2 15	7:30am	8500	1.0		0.076gpl	65.98	

Metallurgical Balance

Product	Weight/Volume	Assay (Cu)	Content	Distribution (%)
Leach Residue	3843 grams	0.23 %	8.84	34.08
Total Solution Content			17.10	65.92
Total			25.94	100.00

Head (Calculated) 0.65% Cu

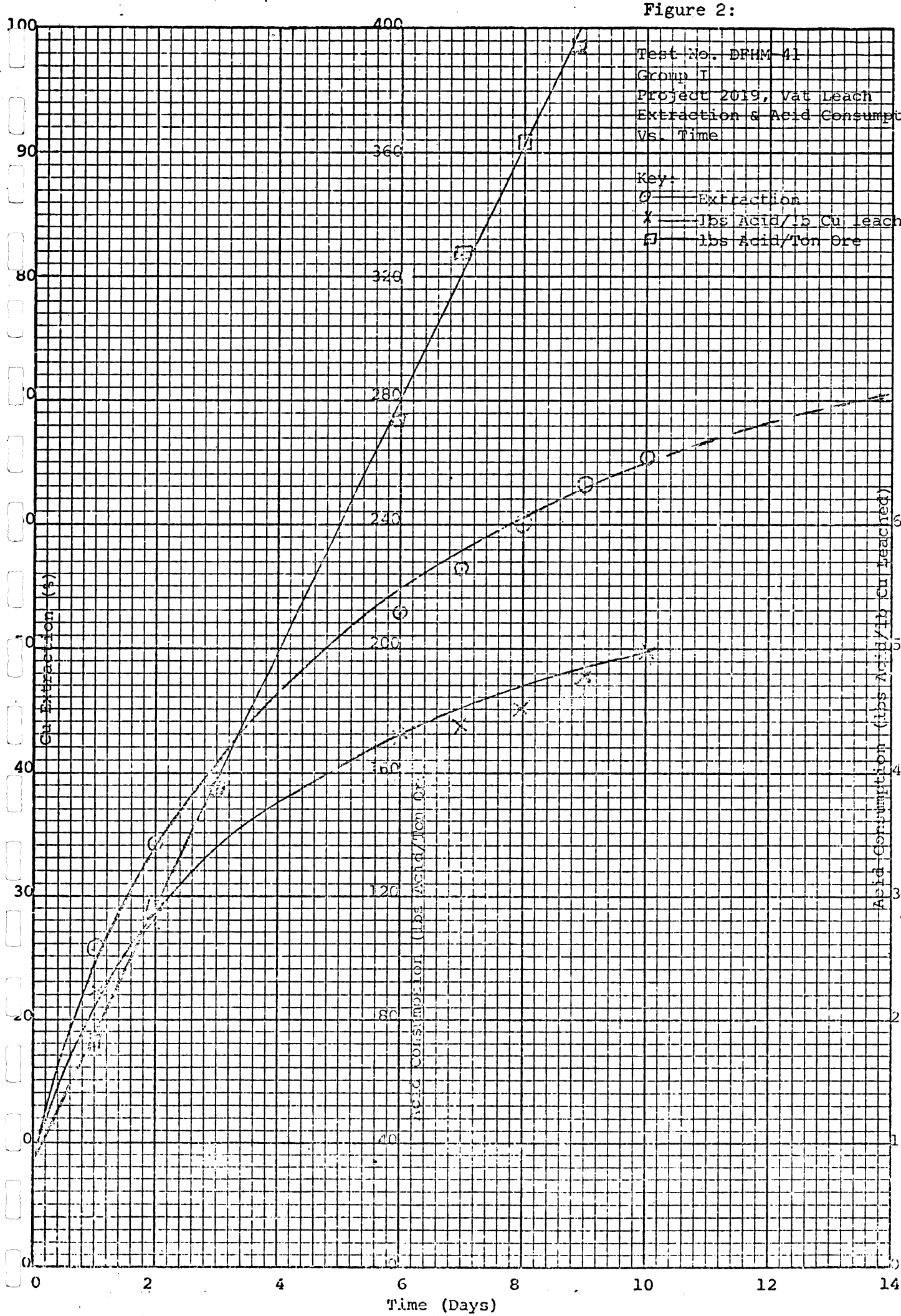
Head (Assayed) 0.66% Cu

Remarks: To a liter of 50 gpl acid was added 4000 grams sample charge, and 30 gpl of acid was perculated through between 5 & 6 cc/min.

Extraction 65.92 (%) Reagent Consumption 429 lbs. Acid/Ton Ore

Operator 50 lbs. Acid/lb. Cu Leached

Figure 2:


Test No. DFHM-41
Group I
Project 2019, Vat Leach
Extraction & Acid Consumption
Vs. Time
Key:
Extraction
lbs Acid/lb Cu leached
lbs Acid/Ton Ore
Cu Extraction (%)
Acid Consumption (lbs Acid/Ton Ore)
Acid Consumption (lbs Acid/lb Cu Leached)
Time (Days)
0
2
4
6
8
10
12
14
100
90
80
70
60
50
40
30
20
10
0
400
360
320
280
240
200
160
120
80
40
0
60
50
40
30
20
10
0

Mineral Survey No. 4197 ~ 2 Sheets

ARIZONA Land District.

PLAT

OF THE CLAIM OF

KEYSTONE COPPER MINING CO.

KNOWN AS THE

ERIKA, ERNEST, ULTIMO, HAGERMAN, CHARLES, INA, MARY, GUSTAVE, CALUMET, WOLFRIME, DORA, LOUIE, & ELLA Lodes

IN COCHISE *MINING DISTRICT,*
COCHISE *COUNTY,* ARIZONA

Containing an Area of _____ *Acres.*

Scale of 300 *Feet to the inch.*

Variation 14° 45' E.

SURVEYED May 28 - July 10, 1937 *BY*

T. N. Stevens, *Mineral Surveyor;*

The Original Field Notes of the Survey of the Mining Claim from which this plat has been made under my direction, have been examined and approved, and are on file in this Office; and I hereby certify that they furnish such an accurate description of said Mining Claim as will, if incorporated into a patent, serve fully to identify the premises, and that such reference is made therein to natural objects or permanent monuments as will perpetuate and fix the locus *thereof.*

I further certify that Five Hundred Dollars worth of labor has been expended or improvements made upon, or for the benefit of, each location embraced in said mining claim by claimant

_____ *or its grantors and that said improvements consist of* 36 Shafts, 1 Cut, & 2 Trenches, with 823 ft. of drifts & crosscuts, Total value $53,310.00

that the location of said improvements is correctly shown upon this plat, and that no portion of or interest in said labor or improvements has been included in the estimate of expenditures upon any other claim.

And I further certify that this is a correct plat of said Mining Claim made in conformity with said original field notes of the survey thereof, and the same is hereby approved.

Public Survey Office
Phoenix, Arizona,
November 8, 1937

Sherman C. Fiske
Office Cadastral Engineer for
ARIZONA

Mineral Survey No. 4197 ~ Sheet 2

Land District.

PLAT

OF THE CLAIM OF

KNOWN AS THE

IN MINING DISTRICT,
COUNTY,
Containing an Area of Acres.
Scale of 300 Feet to the inch.
Variation
19 BY
,
Mineral Surveyor;

The Original Field Notes of the Survey of the Mining Claim from which this plat has been made under my direction, have been examined and approved, and are on file in this Office; and I hereby certify that they furnish such an accurate description of said Mining Claim as will, if incorporated into a patent, serve fully to identify the premises, and that such reference is made therein to natural objects or permanent monuments as will perpetuate and fix the locus *thereof.*

I further certify that Five Hundred Dollars worth of labor has been expended or improvements made upon, or for the benefit of, each location embraced in said mining claim by claimant

or grantors and

that said improvements consist of

that the location of said improvements is correctly shown upon this plat, and that no portion of or interest in said labor or improvements has been included in the estimate of expenditures upon any other claim.

And I further certify that this is a correct plat of said Mining Claim made in conformity with said original field notes of the survey thereof, and the same is hereby approved.

Public Survey Office

, 19

Office Cadastral Engineer

Amount deposited $139.
Cost of O.W.& S. 139.

SEC. 31

T. 15 S.,
R. 23 E.

TOWNSHIP Nº 16 SOUTH, RANGE Nº 23 EAST, GILA AND SALT RIVER MERIDIAN, ARIZONA.

SUPPLEMENTAL PLAT OF SECTION 6.

2192

Scale: 10 Chains (660 feet) to 1 inch.

NOTE

The area of the conflict of Lot 2 with Legal Tender lode claim, Mineral Survey 3229, is 2.12 acres.

Public Survey Office,
Phoenix, Arizona, April 22, 1938.

This supplemental plat of Section 6, Township No. 16 South, Range No. 23 East, of the Gila and Salt River Meridian, Arizona, based upon the plat of said township approved January 23, 1897, plat of Township No. 15 South, Range No. 23 East, approved March 23, 1910, supplemental plat of Section 31, Township No. 15 South, Range No. 23 East, approved March 9, 1922, and official records of Mineral Surveys Nos. 3204 and 3229, Arizona, showing lottings of fractional areas created by the segregation of the Standard No. 8 and Standard No. 9 patented lode claims of Mineral Survey No. 3204 from Lot 6 and SW¼SE¼ of said Section 6, and by the segregation of the Legal Tender patented lode claim of Mineral Survey No. 3229 from Lot 1 of said Section 6; to accommodate homestead entry Phoenix 073063 in accordance with instructions contained in the Commissioner's letter "M" dated April 6, 1937, and to accommodate claims of all patented mining claims now located within said Section 6, is hereby approved.

Sherman E. Burke
Office Cadastral Engineer.

United States
Department of the Interior
General Land Office,
Washington, D.C., May 20, 1938.

The amended lottings represented by this supplemental plat having been correctly made in accordance with regulations of this office, the plat is hereby accepted.

Assistant Commissioner.

I-10 Deposit
"The Thing" Cochise Co., Ariz.