Chemical Mining—A Study of Leaching Agents

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Introduction

Chemical mining may be defined as the in-place extraction of metal values from ores located within the confines of a mine (that is, broken or fractured ore, stope fill, caved material, or ores in permeable zones) or ores located in dumps, prepared ore heaps, or tailing ponds. The term *solution mining* has, in general, been used to mean the in-place dissolution of water-soluble mineral salts with leach solutions.

Chemical mining is presently in its embryonic stage. Basic research needs to be done to understand the influence of physical and chemical factors in chemical mining, and applied research needs to be done on both a small and a large scale to develop practical techniques of field practice. The areas of research include the following:

(1) Leaching chemistry, the influence of physical, chemical, and biological factors in in-place leaching

(2) Hydrodynamics of chemical mining systems

(3) The development of new, as well as old, leaching agents for use under chemical mining leach conditions

(4) The development of new means of contacting fine and coarse materials (with solutions) under static leach conditions

(5) The development of new processes for generating and regenerating chemical mining leach solutions

(6) The development of metal recovery processes

Several of these topics have been investigated by the authors of this paper. Work has been done on generation and regeneration of acid-ferric sulfate leaching solutions (Johnson, 1965; Johnson and Bhappu, 1968a); physical, chemical, and biological factors involved in sulfide and oxide copper dump leaching (Bhappu et al.; Johnson and Bhappu, 1968b); chemical mining techniques (Johnson and Bhappu, 1967); theoretical and practical aspects of chemical mining (Johnson and Bhappu, 1967); leaching of low-grade molybdenum ores (Bhappu, 1968; Bhappu, Reynolds, and Stahmann, 1963); and metal recovery processes (Reynolds, Long, and Bhappu, 1966).

This paper describes a laboratory investigation wherein a variety of common and commercially available chemicals was tested as leaching agents on many different types of ores that can be exploited by chemical mining methods. Ores of copper, nickel, lead, zinc, silver, beryllium, uranium, vanadium, and molybdenum were contacted for long periods of time with solutions containing NaCl, NaCl plus Fe- and acids, thiosulfates, acids, acids plus Fe^{+++}, and carbonates and bicarbonates. These tests were carried out at room temperature and at atmospheric pressure.

As a result of this study, improved chemical mining processes have been devised for treating calcareous and noncalcareous, sulfide and nonsulfide ores of lead and zinc; copper ores with high pyrite content; lateritic nickel ores; vanadiferous shales; and uranium ores. Patent potentials of several of these processes are currently being studied by the New Mexico Institute of Mining and Technology.

PROCEDURES

Both static and column leach tests were used in this study. Generally, static leach tests were used as a means of screening an array of ore samples with a particular leaching agent. Column tests were used to obtain information about their heap or in-place leaching characteristics.

For the static leach tests, a weighed amount of ore was placed in a 600 ml Erlenmeyer flask with a measured volume of solution and allowed to sit for a period of time (up to several months) at room temperature. A cotton plug was placed loosely in the throat of the flask to retard evaporation. At specific intervals, the flask was shaken and a 5 to 10 ml sample of the solution was taken for analysis.

For the column leach tests, a vertically mounted plastic tube was filled with ore that was treated by upward or downward percolation of the leach solution. In downward percolation tests, solution was added at the top of the ore column and allowed to drain from the bottom. In upward percolation tests, solution flowed by gravity through a plastic line into the bottom of the column, up through the bed, and out of the system via a port in the side of the column. In both systems, the effluent solutions were measured and analyzed daily. The ore beds in some of the upward percolation column tests were removed from the columns, mixed, and replaced daily.

ORES

Table 1 is a compilation of the ores and minerals used, their physical and chemical characteristics, and their sources.

EXPERIMENTAL DATA

Experimental data are presented in tabular form for both static and column leach tests. Tables 2 and 3 show results of tests wherein sodium chloride was the major reagent used, Tables 4 and 5 list thiosulfate tests, Tables 6 and 7 show acid and acid-ferric sulfate tests, and Tables 8 and 9 are tabulations of sodium carbonate-bicarbonate tests.

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Discussion of Results

Several different leaching agents were tested in this study on many types of ores. The results of these tests have been arranged into four main groups according to the principal leaching agent(s) used: (1) sodium chloride, (2) thiosulfates, (3) sulfuric acid, and (4) carbonates-bicarbonates.

SODIUM CHLORIDE TESTS

Under the long-term leach conditions common to chemical mining processes, solutions of sodium chloride or sodium chloride plus acids or ferric iron were found effective in leaching lead and zinc from calcareous and noncalcareous oxide and sulfide ores, copper from semioxidized pyritic copper tailings, and nickel from magnesium-rich nickel laterites.

In tests S1 through S9, both sulfide and semioxidized calcareous lead-zinc ores were contacted with saturated salt solutions for long periods of time. Solutions containing up to 2.37 gpl Pb and 13.4 gpl Zn were obtained in tests S1 and S7. Column test C2 showed a lead and zinc buildup in the saturated salt solution to 0.275 and 0.038 gpl lead and zinc, respectively.

Sodium chloride solution was also very effective in leaching semioxidized pyritic tailings containing 0.31 per cent copper. The recovery of copper amounted to 87 per cent in 43 days of leaching with a saturated NaCl solution (test S37) and 55.5 per cent when leached with 53 pounds of sulfuric acid (test S67) per ton of ore for 13 days. Apparently, the sodium chloride solution dissolved some of the semioxidized pyrite contained in the ore and produced a ferric chloride-sodium chloride solvent which, in turn, leached the semioxidized sulfide copper minerals. In test S40, a saturated solution of sodium chloride dissolved 0.22 gpl of iron from an equal weight of minus 48 mesh, acid-washed pyrite in 9 days. Possibly, sodium chloride-ferric chloride solution for subsequent use as a leaching agent could be produced by passing an oxygenated sodium chloride solution over pyrite. The pyrite could be in the ore or in a separate heap or bed.

When small amounts of acid or ferric iron reagents, such as H_2SO_4 HCl, $Fe_2(SO_4)_3$, $FeCl_3$, were used in combination with a sodium chloride solution, a more effective dissolution was achieved on calcareous lead and zinc ores and magnesium-rich nickel laterite ores than when plain acid or saturated sodium chloride solutions were used. In tests S3 and S4, addition of small amounts of HCl or H_2SO_4, increased the dissolution of Pb and Zn from an oxidized tailing over the amounts dissolved with saturated sodium chloride solutions (tests Si and S2). The dissolution of lead from a calcareous lead-sulfide ore in test S9 increased from a value of 0.095 in 70 days with saturated NaCl solution to 0.82 in 5 days of contact with saturated NaCl solution containing a small amount of ferric sulfate.

Solutions of sodium chloride plus acid or ferric chloride were more effective in leaching a magnesium-rich nickel laterite than were the same solutions without NaCl (tests S15 and S79). Test C3 gives an indication of the solubility of a magnesium-rich nickel laterite in NaC1 and NaCl-plus-acid solution under upward percolation leach conditions.

THIOSULFATE TESTS

Solutions of sodium or ammonium thiosulfate were found to be selective leaching agents for calcareous and noncalcareous oxide copper ores. To a much lesser degree, a vandiferous shale also responded to thiosulfate 'leaching.

The data presented in Tables 4 and 5, as well as data obtained from other studies, indicated that thiosulfate leaching of copper has the following characteristics:

1. Thiosulfate ion is a very selective solvent for copper contained in chrysocolla, malachite, and azurite ores in the pH range of 5 to 10. Sulfide minerals are not leached to an appreciable extent by thiosulfate solutions.

2. The rate of dissolution is very fast in an agitated system and relatively slow in a static system. Near maximum dissolution of chrysocolla required 1 hour under agitated conditions and several days under static leach conditons.

3. Slightly increased reaction rates were observed at temperatures above room temperature and below 80℃. The reagent decomposes at temperatures above 80° C.

4. The leaching reaction appears to be influenced by light. When tests were conducted in clear flasks, the reaction rates were about twice what they were when the same tests were conducted in brown bottles.

5. Complete dissolution of a mineral particle may be difficult to obtain. Maximum dissolution in both agitated and column tests was generally in the 50 to 80 per cent range. Thiosulfate solutions seem to be somewhat limited in their ability to penetrate rock or mineral particles.

6. There is a solubility limit for copper in thiosulfate solutions. This value is roughly 1.0 gpl copper per 1 per cent $(NH_4)_2S_2O_3$ in solution.

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7. Solutions of copper thiosulfate decompose and form a black copper-sulfide precipitate under certain conditions; for example,

  (a) when the copper-loading capacity of the solution is exceeded,
  (b) when solutions are exposed to light,
  (c) when solutions are heated, and
  (d) when the pH drops below about 5.

Sulfite ion seems to buffer the pregnant copper-thiosulfate solutions and retard their decomposition for a few days. In a commercial leaching application, solution strengths, temperatures, and contact times would have to be controlled to prevent precipitation of copper sulfides within the ore mass.

8. Addition of soluble sulfides, such as $(NH_4)_2S$, CaS or Na_2S, causes copper to be precipitated from pregnant, copper-thiosulfate solutions and the thiosulfate leaching agent to be regenerated. The copper sulfide product assays about 80 per cent copper.

Tests C4, C5, and C6 are indicative of thiosulfate column leaching results. In test C5, 56 per cent of the copper was extracted from a chrysocolla ore assaying 0.6 per cent copper in about 70 days of leaching with medium strength (3 per cent) thiosulfate solution. Sixty per cent recovery of the oxide copper was obtained on a calcareous sulfide-oxide ore assaying 0.815 per cent total copper and 0.508 per cent oxide copper in 30 days of leaching with 3 per cent $(NH_4)_2O_3$ solution.

The cost of thiosulfate leaching appears to be reasonable. When a 15 per cent loss of the leaching reagent $[(NH_4)S_2O_3]$ is assumed per cycle, the cost of both the leaching reagent and the precipitating reagent $[(NH_4)_2S]$ has been estimated at 9 to 12 cents per pound of copper recovered.

SULFURIC ACID AND SULFURIC ACID—FERRIC SULFATE TESTS

Several sulfuric acid and sulfuric acid-ferric sulfate leach tests were run on copper and uranium ores to establish comparative test data and to examine long-duration sulfuric acid and sulfuric acid-ferric sulfate leaching. These data are shown in Tables 6 and 7.

Sulfuric acid and sulfuric acid-ferric sulfate leach tests (C7 through C11) show the effect of solution strength and particle size on the effectiveness of these reagents in leaching copper-oxide ores. The reagent consumption increased and the recovery decreased with increasing particle size in tests C9, C10, and C11, and high recovery and low reagent consumption were obtained when the leach solution was relatively low in its acid content (tests C7, C8, and C9) or when ferric sulfate was used in addition to sulfuric acid (test C12). Maximum economic recoveries were achieved on both coarse and fine ores in about 6 days of leaching. After this period, the reagent consumptions per pound of additional copper recovered became very high.

CARBONATE-BICARBONATE LEACH TESTS

Sodium carbonate-bicarbonate solutions were tested as leaching agents on several types of ores under long-duration, ambient temperature leaching conditions. These solvents were only effective in leaching secondary uranium ores and to a small extent, MoS_2 concentrates.

Tests C15, C16, and C17 show that carbonate-bicarbonate leach solutions are quite effective in leaching relatively coarse uranium ores in several days at ambient temperatures. Recoveries in excess of 90 per cent were obtained in column leach tests C15 and C17.

Low-temperature, long-duration, carbonate-bicarbonate leaching of uranium ores appears to have an advantage over high-temperature, short-duration, carbonate-bicarbonate leaching in that at these low-temperature conditions, sulfide and silicate minerals are not appreciably dissolved. These minerals consume large amounts of the carbonate and bicarbonate reagents under conventional, high-temperature leaching conditions.

Conclusions

Ambient temperature, long-duration leach tests were conducted on a wide variety of ores with four different types of leaching agents. These leaching agents consisted of solutions of (1) sodium chloride or sodium chloride plus acids, ferric sulfate, or ferric chlorides, (2) thiosulfates, (3) sulfuric acid or sulfuric acid-ferric sulfate, and (4) sodium carbonate-sodium bicarbonate.

Saturated sodium chloride solution was found to be an effective leaching agent for calcareous, semioxidized lead and zinc ores and pyritic, semioxidized copper ores. Small additions of acid or ferric iron greatly increased the lead and zinc dissolution from galena-sphalerite ores and nickel from magnesium-rich, nickel laterites.

Solutions containing thiosulfate compounds such as $Na_2S_2O_2$ and $(NH_4)_2\%O_3$ were found to be excellent solvents for oxide copper minerals under basic leach conditions. These reagents are very selective for copper, can be regenerated in the copper precipitation step, can be used on calcareous ores, and appear to be very low in cost. One problem with their use, however, is that the pregnant leach solutions decompose and precipitate a black, copper-sulfide product under a variety of conditions.

Sulfuric acid and sulfuric acid-ferric sulfate solutions were used to leach copper and uranium ores. This provided comparative data for other leach tests and illustrated the use of these solvents in chemical mining applications.

Bicarbonate-carbonate leach liquors were excellent solvents for leaching both low- and high-grade uranium ores under long-duration leach conditions. Much better utilization of reagents was achieved under these leach conditions than with conventional, carbonate-bicarbonate leaching of uranium ores.

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TABLE 1. ORES USED IN THIS STUDY

ORE NO.	SOURCE	PRINCIPAL MINERALS ORE	GANGUE	ASSAYS (%)	
1	Milford, Utah	Cu oxide	Ls,SiO_2	2.5	Cu
2	Brighton, Utah	Bornite, magnetite	Magnetite, SiO_2	1.8	Cu
3	Stanley, Idaho	ZnS	SiO_2	10.	Zn
4	New Mexico	Beryl	SiO_2	7.09	BeO
5	Paris, Idaho	Vanadium	Shale	1.99	V_2O_5
6	Cuba	Nickel laterite	Fe_2O_3	1.0	Ni
7	Mackay, Idaho	Chrysocolla	Fe_2O_3	2.2	Cu
8	Park City, Utah	Oxide, PbS,ZnS	Ls,SiO_2	1.8	Pb,1.7 Zn
9	Lakeshore mine, Ariz.	Chrysocolla	Ls,SiO_2	3.	Cu
10	Mountain City, Nev.	Oxide $CuFeS_2$	SiO_2	0.31	Cu
11	O.K. mine, Milford, Utah	Malachite	Monzonite	2.2	Cu
12	Guatemala	Nickel laterite	MgO,Fe_2O_3	2.2	Ni
13	Minnesota	Fe_2O_3	---	55.	Fe
14	Chino mine, N. Mex.	Chalcocite	SiO_2	2.2	Cu
15	Questa, N. Mex.	MoS_2 conc.	---	55.	Mo
16	Kelly, N. Mex.	PbS,ZnS	Ls,SiO_2	3.0	Pb,4.0 Zn
17	Questa, N. Mex.	Mo oxides	SiO_2	0.3	Mo
18	Grants, N. Mex.	Secondary U_3O_8	Ls,SiO_2	0.2	U_3O_8
19	Kelly, N. Mex.	Oxide PbS,ZnS tailings	Ls,SiO_2	1.0	Pb,1.2 Zn
20	Kelly, N. Mex.	Oxide PbS,ZnS tailings	Ls,SiO_2	1.0	Pb,2.0 Zn
21	Santa Rosa, N. Mex.	Oxide Cu chalcocite	Ls,SiO_2	2.5	Cu
22	Grants, N. Mex.	Secondary U_3O_8	Ls,SiO_2	1.96	U_3O_8
23	El Cerrito mine, Ariz.	Chalcopyrite conc.	---	30.	Cu
24	Guatemala	Nickel laterite	Fe_2O_3,MgO	0.6	Ni
25	Mexico	Malachite	Fe_2O_3	14.	Cu
26	Mexico	Ag, galena	Ls,SiO_2	15	oz Ag/ton
27	Midvale, Utah	Pyrite conc.	---	45.	Fe
28	---	Chrysocolla	---	30.	Cu
29	Pima mine, Ariz.	MoS_2 conc.	---	55.	Mo
30	Tyrone, N. Mex.	Oxide, semioxide Cu mins. in tailings	SiO_2	0.6	Cu
31	Grants, N. Mex.	Secondary U_3O_8	Ls,SiO_2	0.047	U_3O_8
32	Iron King mine, Ariz.	Pyrite tailings	SiO_2	20.	Fe
33	Water Canyon, N. Mex.	PbS,ZnS	Ls	6.57	Pb,4.0 Zn
34	Guatemala	Nickel laterite	Fe_2O_3, MgO	1.0	Ni
35	Tyrone, N. Mex.	Chrysocolla	Granodiorite	0.60	Cu
36	Fierro, N. Mex.	Chrysocolla	Ls,SiO_2	0.8	Cu
37	Garfield Co., Utah	Secondary U_3O_8	SiO_2	0.08	U_3O_8
38	Shirley Basin, Wyo.	Secondary U_3O_8	SiO_2	0.6	U_3O_8
39	Morocco	Chalcocite	SiO_2	2.0	Cu

TABLE 2. SODIUM CHLORIDE STATIC LEACH TESTS

TEST	ORE NO.*	ORE WT (gms)	REAGENT USED†	SOL. VOL. (ml)	ELEMENT (g/l in sol. @ — days)	
S1	8	300	none	300	Pb	1.63 @ 1, 1.78 @ 3, 2.37 @ 46
					Zn	0.14 @ 1, 0.22 @ 3, 0.25 @ 46
S2	8	300	none	900	Pb	0.68 @ 8, 0.86 @ 24, 0.92 @ 41
					Zn	0.045 @ 8, 0.07 @ 24, 0.09 @ 41
S3	8	300	0.2 ml HCl	300	Pb	2.0 @ 19
					Zn	0.27 @ 19
S4	8	300	0.1 ml H_2SO_4	300	Pb	2.42 @ 4
					Zn	0.43 @ 4
S5	20	300	none	300	Pb	0.28 @ 2, 0.52 @ 21
					Zn	0.20 @ 2, 0.25 @ 21
S6	20	300	2 gms $FeCl_3 \cdot 6H_2O$	300	Pb	0.72 @ 8
					Zn	0.76 @ 8
S7	19	300	none	300	Pb	0.015 @ 7, 0.15 @ 50
					Zn	3.6 @ 7, 13.4 @ 50
S8	16	300	none	300	Pb	0.023 @ 1, 0.095 @ 70
S9	16	300	1 gm $Fe_2(SO_4)_3 \cdot nH_2O$	300	Pb	0.82 @ 5, 0.09 @ 30
S10	18	300	none	300	U_3O_8	0.002 @ 6
S11	18	300	1 gm $FeCl_3 \cdot 6H_2O$	300	U_3O_8	0.068 @ 6
S12	6	200	none	200	Ni	0.0001 @ 7
S13	6	200	0.1 ml H_2SO_4	200	Ni	0.005 @ 6
S14	2	300	none	300	Ni	0.001 @ 1, 0.001 @ 15
S15	12	300	5 gms $FeCl_3 \cdot 6H_2O$ + 0.5 ml H_2SO_4	300	Ni	0.63 @ 16, 0.85 @ 21, 0.67 @ 60, 1.35 @ 150
S16	24	200	1 ml H_2SO_4	200	Ni	0.10 @ 1, 0.10 @ 15
S17	24	200	0.05 ml H_2SO_4 + 1 gm $FeCl_3 \cdot 6H_2O$	400	Ni	0.02 @ 1, 0.076 @ 9
S18	12	300	5% Na_2CO_3 + 1% $NaHCO_3$	300	Ni	0.003 @ 25
S19	17	200	none	200	Mo	0.006 @ 22
S20	17	200	1 gm Na_2CO_3	200	Mo	0.016 @ 130
S21	15	300	2 gms $FeCl_3 \cdot 6H_2O$	300	Mo	0.013 @ 150
S22	15	100	1% $NaHCO_3$ + 5% Na_2CO_3	300	Mo	0.020 @ 30, 0.057 @ 90
S23	9	200	none	200	V_2O_5	nil @ 60
S24	3	200	none	200	Zn	0.070 @ 7, 0.195 @ 90
S25	3	200	0.1 ml HCl	200	Zn	0.160 @ 50
S26	3	200	3 gm $FeCl_3 \cdot 6H_2O$	200	Zn	0.69 @ 5
					Cu	0.66 @ 5
S27	1	200	none	200	Cu	0.008 @ 7, 0.020 @ 14
S28	1	200	1 ml 5% NaOCl	200	Cu	0.007 @ 5
S29	2	300	none	200	Cu	0.001 @ 13
S30	2	300	2 gms $FeCl_3 \cdot 6H_2O$	200	Cu	0.16 @ 9
S31	7	300	none	300	Cu	0.005 @ 17
S32	7	300	0.1 ml HCl	300	Cu	0.0005 @ 9
S33	9	300	none	300	Cu	0.0005 @ 8
S34	9	300	5 gms $FeCl_3 \cdot 6H_2O$ + 0.5 ml H_2SO_4	300	Cu	6.3 @ 21
S35	9	300	none	300	Cu	.040 @ 16
S36	14	300	5 gms $FeCl_3 \cdot 6H_2O$ + 0.5 ml H_2SO_4	300	Cu	3.15 @ 16
S37	10	200	none	200	Cu	0.095 @ 1, 1.48 @ 7, 2.71 @ 43; 81% rec.
S38	30	100	15 g/l H_2SO_4	300	Cu	1.74 @ 2, 3.05 @ 20; 88.1% rec. @ 2.7 lbs H_2SO_4/lb Cu
S39	30	100	0.5 g/l H_2SO_4 + 4 gms $Fe_2(SO_4)_3 \cdot nH_2O$	300	Cu	1.56 @ 2, 2.70 @ 20; 95% rec. @ 0.3 lbs H_2SO_4/lb CU
S40	27	300	none	300	Fe	0.109 @ 1, 0.219 @ 9
S41	27	300	1 ml H_2SO_4	300	Fe	2.33 @ 6
S42	4	200	none	200	BeO	0.001 @ 13
S43	4	200	3 gms $FeCl_3$ + 0.5 ml H_2SO_4	200	BeO	0.020 @ 15

* See Table 1.

† In saturated NaCl solutions.

TABLE 3. SODIUM CHLORIDE COLUMN LEACH TESTS

TEST	ORE NO.*	ORE SIZE (mesh/in.)	ORE WT (gms)	COL. DIAM. (in.)	REAGENT USED†	SOL. FLOW RATE (ml/day)	TIME (days)	SOLUTION CONTENT	REC. (%)	ACID[Δ] CONS.
C1‡	30	− 100	6000	3.5	5 g/l H_2SO_4	2880	10	0.2 - 2 g/l Cu	69.4	3.22
C2	33	− 3	4000	2.5	none	500	0-9	0.275 g/l Pb 0.038 g/l Zn		
					none	250	9-13	0.244 g/l Pb 0.024 g/l Zn		
					1 ml H_2SO_4/l	1000	14-22	0.380 g/l Pb 0.102 g/l Zn		
					2 ml H_2SO_4/l	1000	22-27	0.550 g/l Pb 0.075 g/l Zn		
					4 ml H_2SO_4/l	500	28-35	0.680 g/l Pb 0.100 g/l Zn		
					4 ml H_2SO_4/l	250	35-40	1.110 g/l Pb 0.180 g/l Zn		
C3	34	− 1 in.	4000	2.5	none	250	0-18	0.020 g/l Ni		
					1 ml H_2SO_4/l	250	18-30	0.310 g/l Ni		

* See Table 1.

† In saturated NaCl solution

‡ Average values.

‡ Tests 1 and 3 were upward percolation, test 2 downward.

Δ Acid consumption at pounds of sulfuric acid per pound of copper.

TABLE 4. THIOSULFATE STATIC LEACH TESTS

TEST	ORE NO.*	ORE WT (gms)	REAGENT USED†	SOL. VOL. (ml)	ELEMENT (g/l in soln. @ – days)
S44	31	300	0.9 $(NH_4)_2S_2O_3$	300	U_3O_8 0.014 @ 21
S45	22	100	0.9 $(NH_4)_2S_2O_3$	300	U_3O_8 0.01 @ 21
S46	22	100	1.9 $Na_2S_2O_3 \cdot 5H_2O$	300	U_3O_8 0.05 @ 1, 0.36 @ 30
S47	5	50	1.5 $(NH_4)_2S_2O_3$	300	V_2O_5 0.27 @ 39
S48	6	200	1.8 $(NH_4)_2S_2O_3$	300	Ni 0.0005 @ 17, 0.003 @ 30
S49	3	100	1.5 $Na_2S_2O_3 \cdot 5H_2O$	300	Zn 0.001 @ 1, 0.0001 @ 14
S50	3	200	1.8 $(NH_4)_2S_2O_3$	300	Zn 0.11 @ 29
S51	3	300	1.8 $(NH_4)_2S_2O_3$	300	Zn 0.010 @ 28
S52	4	50	1.5 $Na_2S_2O_3 \cdot 5H_2O$	150	BeO 0.0001 @ 14
S53	12	100	1.5 $Na_2S_2O_3 \cdot 5H_2O$	300	Ni 0.001 @ 14
S54	13	300	1.8 $(NH_4)_2S_2O_3$	300	Fe 0.001 @ 39
S55	29	100	1.8 $(NH_4)_2S_2O_3$	300	Mo 0.033 @ 29
S56	26	300	3. $(NH_4)_2S_2O_3$	300	Ag 0.015 @ 33
S57	21	200	3. $Na_2S_2O_3 \cdot 5H_2O$	300	Cu 0.027 @ 4, 1.58 @ 60, 0.003 @ 135‡
S58	23	100	3. $Na_2S_2O_3 \cdot 5H_2O$	200	Cu 0.014 @ 1, 0.033 @ 45
S59	28	50	3. $Na_2S_2O_3 \cdot 5H_2O$	200	Cu 1.15 @ 2, 1.96 @ 32‡
S60	28	50	3. $Na_2S_2O_3 \cdot 5H_2O$	200	Cu 1.96 @ 2, 4.67 @ 32‡
S61	26	300	1.5 $Na_2S_2O_3 \cdot 5H_2O$	300	Cu 0.144 @ 1, 0.83 @ 34
S62	26	300	5. $Na_2S_2O_3 \cdot 5H_2O$	300	Cu 1.63 @ 6, 4.7 @ 30
S63	26	300	1.8 $(NH_4)_2S_2O_3$	300	Cu 0.57 @ 1, 1.89 @ 15‡
S64	26	300	60 $(NH_4)_2S_2O_3$	300	Cu 21.8 @ 1, 0.59 @ 20‡
S65	20	300	10 $(NH_4)_2S_2O_3$	300	Cu 1.12 @ 1, 6.97 @ 20‡
S66	1	50	1.5 $Na_2S_2O_3 \cdot 5H_2O$	300	Cu 0.10 @ 1, 1.13 @ 58

* See Table 1.
† Volume per cent of $(NH_4)_2S_2O_3$ solutions and weight per cent of $Na_2S_2O_3 \cdot 5H_2O$ solutions.
‡ Precipitate of black Cu_2S product.

Table 5. THIOSULFATE COLUMN LEACH TESTS (downward percolation)

TEST	ORE NO.*	ORE SIZE (mesh/in.)	ORE WT (gms)	COL. DIAM. (in.)	REAGENT USED (vol. %)	SOL. FLOW RATE (ml/day)	TIME (days)	SOLUTION CONTENT	REC. (%)	ACID CONS.
C4	9	– 8	7500	3.5	0.9 - 6 $(NH_4)_2S_2O_3$	150-1000	117†	2.0 g/l Cu	25	
C5	35	– 3	1000	1.5	6 $(NH_4)_2S_2O_3$	10-150	70†	8.0 g/l Cu	56	
C6	36	– 2 in.	12000	3.5	3 $(NH_4)_2S_2O_3$	1000	30	7.0 g/l Cu	60‡	

* See Table 1.
† Days of contact during which solution was added to column; about 90 per cent of this recovery was within 30 days.
‡ Recovery of oxide copper.

TABLE 6. SULFURIC ACID—FERRIC IRON STATIC LEACH TESTS

TEST	ORE NO.*	ORE WT (gms)	REAGENT USED†	SOL. VOL. (ml.)		ELEMENT (g/l in sol. @ — days)
S67	10	100	1.43 ml H_2SO_4	100	Cu	1.60 @ 8, 1.79 @ 13; 55.5% rec.
S68	10	100	2.87 ml H_2SO_4	100	Cu	1.53 @ 8, 1.72 @ 13
S69	10	100	2.5 gms $Fe_2(SO_4)_3 \cdot n\,H_2O$	100	Cu	1.53 @ 8, 1.65 @ 13
S70	10	100	none	100	Cu	0.87 @ 7, 0.97 @ 15
S71	30	100	1 ml H_2SO_4	200	Cu	2.39 @ 4; 64% rec. 3.5 lbs H_2SO_4/lb Cu
S72	30	100	3.15 ml H_2SO_4	200	Cu	2.91 @ 4
S73	30	100	15 g/l H_2SO_4	300	Cu	1.95 @ 20; 79% rec. @ 3.8 lbs H_2SO_4/lb Cu
S74	30	100	3 g/l H_2SO_4 + 4 gms $Fe_2(SO_4)_3 \cdot nH_2O$	400	Cu	0.95 @ 2, 1.41 @ 19; 78% rec. @ 1 lbs H_2SO_4/lb Cu
S75	30	100	15 g/l H_2SO_4 + 4 gms $Fe_2(SO_4)_3 \cdot nH_2O$	300	Cu	1.52 @ 2, 2.83 @ 19; 89% rec.
S76	30	100	none	400	Cu	0.28 @ 1, 0.348 @ 18; 19% rec.
S77	11	150	1.33 ml H_2SO_4	150	Cu	7.76 @ 9
S78	3	300	5 gms $Fe_2(SO_4)_3 \cdot n\,H_2O$	300	Zn	0.66 @ 2, 1.20 @ 21
S79	12	300	5 gms $FeCl_3 \cdot 6H_2O$ + 0.5 ml H_2SO_4	300	Ni	0.143 @ 2, 0.318 @ 10, 0.360 @ 16

* See Table 1.

TABLE 7. SULFURIC ACID—FERRIC SULFATE COLUMN LEACH TESTS
(upward percolation)

TEST	ORE NO.*	ORE SIZE (mesh/in.)	ORE WT (gms)	COL. DIAM. (in.)	REAGENT USED	SOL. FLOW RATE (ml/day)	TIME (days)	SOLUTION CONTENT	REC. (%)	ACID△ CONS.
C7	35	−3	6000	3.5	5 g/l H_2SO_4	10,000	6	0.12-0.9 g/l Cu	67.5	4.15
							9		72.7	4.50
C8	35	−3	6000	3.5	30 g/l H_2SO_4	5000	6	0.14-4.4 g/l Cu	67.2	5.32
							9		73.4	7.5
C9	35	−3	6000	3.5	15 g/l H_2SO_4	5760	6	0.80 - 5.2 g/l Cu	69.4	3.85
							9		74.5	4.25
C10	35	−1 in.	6000	3.5	15 g/l H_2SO_4	5760	6	0.23-1.93 g/l Cu	67.2	5.36
C11	35	−2 in.	6000	3.5	15 g/l H_2SO_4	5760	6	0.06-2.2 g/l Cu	53.6	5.23
C12	35	−3	6000	3.5	10 g/l H_2SO_4 + 5 g/l Fe^{+++}	5760	6	0.2-2.2 g/l Cu	79.6	3.18†
C13	30	−100	6000	3.5	5 g/l H_2SO_4 + 3 g/l Fe^{+++}	2880	9	0.2-3.2 g/l Cu	71.3	1.98†
C14	31	−3	6000	3.5	5 g/l H_2SO_4	5000	15	0.04-0.11 g/l U_3O_8	35.8	70‡

* See Table 1.
† Equivalent acid consumption.
△ Acid consumption at pounds of sulfuric acid per pound of copper.
‡ Pounds H_2SO_4 per ton of ore.

TABLE 8. CARBONATE–BICARBONATE STATIC LEACH TESTS

TEST	ORE NO.*	ORE WT (gms)	REAGENT USED (wt. %)	SOL. VOL. (ml.)	ELEMENT (g/l in sol. @ – days)	
S80	31	300	$1\ NaHCO_3 + 2\ Na_2CO_3$	300	U_3O_8	0.20 @ 2, 0.47 @ 21; 85% rec.
S81	31	300	$2\ NaHCO_3 + 4\ Na_2CO_3$	300	U_3O_8	0.28 @ 2, 0.46 @ 21; 83% rec.
S82	31	300	$1\ NaHCO_3 + 6\ Na_2CO_3$	300	U_3O_8	0.25 @ 2, 0.48 @ 21; 91% rec.
S83	22	100	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	U_3O_8	2.63 @ 29, 3.71 @ 60
S84	8	100	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Pb	0.09 @ 2, 0.09 @ 30
					Zn	0.005 @ 2, 0.008 @ 30
S85	3	300	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Zn	0.004 @ 64
S86	6	200	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Ni	0.001 @ 29
S87	12	100	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Ni	0.001 @ 65
S88	4	50	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	BeO	0.005 @ 90
S89	15	100	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Mo	0.025 @ 9, 0.070 @ 30, 0.10 @ 90
S90	32	300	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Fe	0.021 @ 30
S91	7	300	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Cu	0.054 @ 1, 0.05 @ 20
S92	28	50	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Cu	0.05 @ 30
S93	6	200	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Cu	0.04 @ 4, 0.14 @ 30
S94	13	300	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	Fe	0.02 @ 30
S95	5	50	$1\ NaHCO_3 + 5\ Na_2CO_3$	300	V_2O_5	0.075 @ 2

* See Table 1.

TABLE 9. CARBONATE–BICARBONATE COLUMN LEACH TESTS (upward percolation)

TEST	ORE NO.*	ORE SIZE (mesh/in.)	ORE WT (gms)	COL. DIAM. (in.)	REAGENT USED† (wt %)	SOL. FLOW RATE (ml/day)	TIME (days)	SOLUTION CONTENT	REC. (%)	ACID△ CONS.
C15	31	−3	6000	3.5	$1\ NaHCO_3 + 5\ Na_2CO_3$	1000	6	0.2-1.35 g/l U_3O_8†	89.3	
							9		92	
C16	37	−3	6000	3.5	$1\ NaHCO_3 + 5\ Na_2CO_3$	1000	12	0.10-0.70 g/l U_3O_8†	58.6‡	
C17	38	−10	1000	2.0	$1\ NaHCO_3 + 5\ Na_2CO_3 + 1\ g/l\ KMnO_4$	500	5	4.5 g/l U_3O_8+	67	
							12	0.51 g/l U_3O_8	91	

* See Table 1.
† These solutions were very low in $SiO_2^=$ and $SO_4^=$ content.
‡ This ore required an oxidant for high U_3O_8 recovery.