GL03956

PRELIMINARY FEASIBILITY REPORT

ON

IN SITU MINING METHODS

ASARCO'S SANTA CRUZ PROJECT

PINAL COUNTY, ARIZONA

For

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By

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# SECTION 1

## INTRODUCTION

Mountain States Research and Development was authorized by Asarco, Incorporated to evaluate in situ methods of copper recovery for application to their Santa Cruz Project, Pinal County, Arizona.

As a basis for the study, Asarco provided geologic, mineralogic and preliminary leach test data. From this data, MSRD formulated study parameters, developed the concept and order of magnitude capital cost estimates for two mining methods namely, Void-hole Fragmentation Technique and Block Cave In Situ Leach process which lend themselves to in situ leach operations.

This report contains descriptions, capital cost estimates and operating cost estimates for each mining system and the process plant.

A summary of the report is found in Section 3.

# SECTION 2

## CRITERIA AND DESIGN BASIS

This study was based upon information provided by ASARCO on the size and nature of the ore body to be developed. A series of assumptions were then made to establish design criteria against which the study was developed as described in the following sections of this report. Figure 2.1 shows the area under consideration and this study is based on mining Block B.

### ASARCO Data

1. The ore body consists of Block B only.

2. Block B has the following statistics:

   Surface dimensions - 2,500' x 2,500'

   Top of ore body - 1,300' depth

   Ore zone thickness - 800'

   Copper content - 0.42% copper

3. Mineralization is predominently oxide copper. Approximately 35 percent of the oxide copper is green vitreous minerals comprised of brochantite, atacamite and dioptase. Atacamite is a chloride mineral which will release chlorine to the leach solutions.

4. Acceptable copper recoveries can be attained with sulfuric acid leach.

### Assumptions

1. The ore zone formation must be fractured or broken to allow a reasonable solution flow rate for leaching in place.

2. The ore body is too deep to allow conventional mining techniques and treatment of the ore in a surface plant.

3. Leaching in place, or in situ leaching, will recover 50 percent of the total copper. Acid consumption averages 4.0 lbs. $H_2SO_4$ per lb. copper recovered.

4. The ore body will be mined at a rate that will provide a 25 year life for the project.

5. Copper will be recovered from solutions by a solvent extraction plant and electrodeposition to produce marketable cathodes.

6. The project is close enough to urban areas such that no housing will be provided.

7. Leaching solution strengths of 2.9 grams per liter copper can be attained initially.

8. The capital cost estimates will not include:

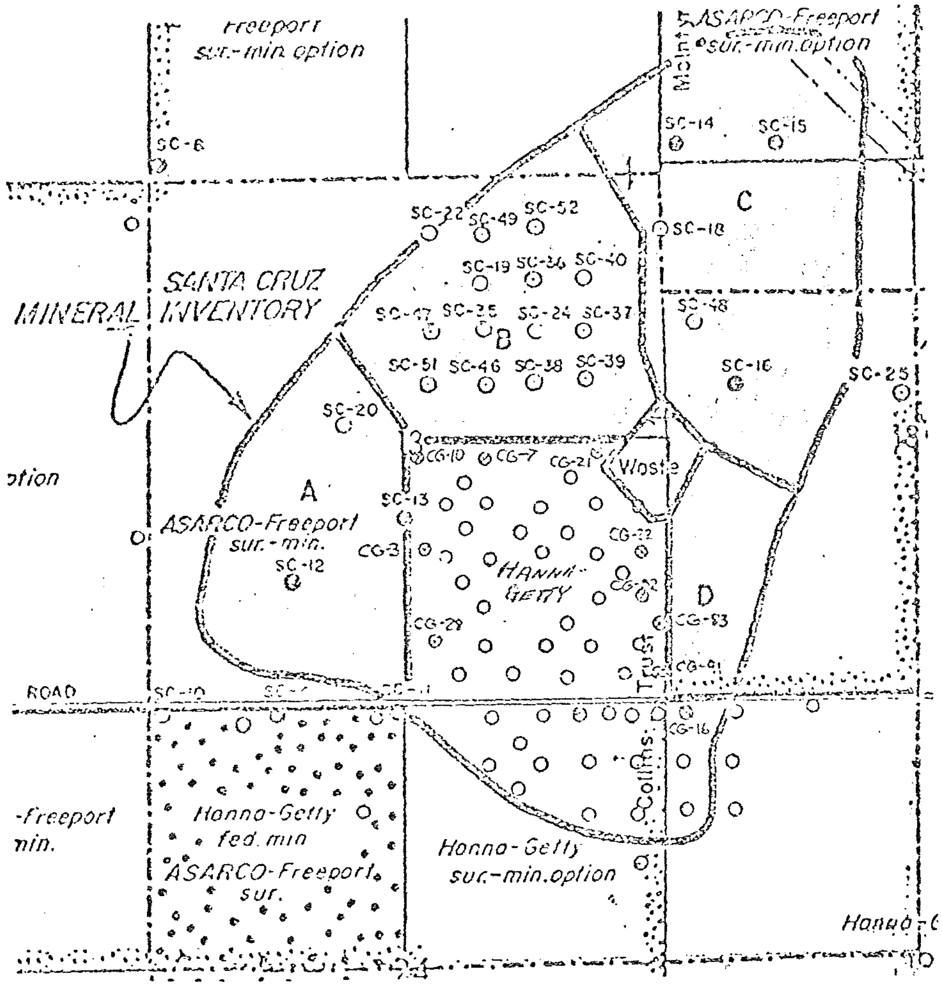
   Property purchase cost

   Access roads to the project

   Power generation or transmission

   Water development beyond the plant area.

PRELIMINARY
ROUGH DRAFT




OUTLINE OF AREA OF COPPER INVENTORY
IN AND ADJACENT TO SEC. 13 (T6S, R4E)

Pinal County, Ariz.

CG-22
⊗ Hanna-Getty holes used to estimate
ASARCO-Freeport Inventory

outline of Hanna-Getty/ASARCO-Freeport
area of exchange

H.G.K - F.T.G. December 12, 1979

1" = 2000'

FIGURE 2.1

## SECTION 3

## SUMMARY AND RECOMMENDATIONS

### 3.1 GENERAL

Two mining methods were examined as an approach to fracturing the ore body such that solutions could be circulated at a reasonable rate to recover an average of 174,904 pounds of copper as cathode each day. These methods are:

1. Void Hole Fragmentation In Situ Leach System

2. Block Cave In Situ Leach System

They are described respectively in Sections 4 and 5.

As leach solutions from in situ and heap leaching operations are generally too low in copper for direct deposition in an electrolytic cell, a solvent extraction (S-X) plant is included in the study. Pregnant solutions from the mine containing up to 2.9 grams per liter of copper initially will be treated in the S-X plant to produce a decopperized acid solution for return to the leach area and a strong copper solution (25-30 gpl copper) for feed to the electrolytic tankhouse. It will be necessary to expand the S-X plant after the fourth year to accommodate larger volumes with lower copper content. The process plant is covered in Section 6.

The scope of the study did not include such infrastructure as access roads, property acquisition, power supply and water supply. Additionally, environmental permit programs were not addressed.

### 3.2 CAPITAL COST

Order of Magnitude capital costs were developed for each mining system studied. As each system will produce the same solution flowrates and copper production rate, the same process plant will serve each system. It is also assumed that warehouse and office facilities are equivalent for all systems. A summary comparison of capital costs is presented in Table 3.1.

TABLE 3.1

SUMMARY CAPITAL COST COMPARISON
($000)

| | Void Hole System Case 1 | Void Hole System Case 2 | Block Cave System |
|---|---|---|---|
| Exploration | -- | -- | $ 3,205 |
| Mine Development - Preproduction | $ 75,620 | $ 71,498 | 22,422 |
| Surface Facilities | 1,750 | 1,750 | 3,300 |
| Process Plant - Preproduction | 12,646 | 12,646 | 12,646 |
| Total Direct Costs | $ 90,016 | $ 85,894 | $41,573 |
| Contingency | 22,066 | 21,036 | 7,500 |
| | $112,082 | $106,930 | $49,073 |
| S-X Plant Expansion - Year 5* | 3,370 | 3,370 | 3,370 |

*Note: Costs are as of 12/31/79.

### 3.3 OPERATING COSTS

Operating costs are based upon cost per pound of copper produced. The annual dollar costs may be calculated using a production rate of 63,840,000 pounds copper per year.

A summary comparison of operating costs is presented in Table 3.2.

TABLE 3.2

SUMMARY OPERATING COST COMPARISON
(Cents per pound copper)

| | Void Hole System Case 1 | Void Hole System Case 2 | Block Cave System |
|---|---|---|---|
| Mining and Well Install. | 56.3 | 52.2 | 12.8 |
| Underground Leaching | 7.5 | 7.5 | 10.5 |
| Surface Leaching | -- | -- | 2.0 |
| Process Plant | 11.3 | 11.3 | 11.3 |
| Administration & Indirects | 13.0 | 13.0 | 13.0 |
| Total Operating | 88.1 | 84.0 | 49.6 |
| Depreciation (Straight Line over 14 years) | 12.5 | 12.0 | 5.5 |
| | 100.6 | 96.0 | 55.1 |

Of the three systems, the block cave system is the only one which appears economically attractive.

### 3.4 RECOMMENDATIONS

The block cave mining system with in situ leaching followed by solvent extraction and electrowinning appears economically attractive on the basis of a 50¢ per pound copper cost. However, the capital costs do not reflect such influencing factors as interest and escalation and will require a study in greater detail.

It is recommended that an exploration program be implemented to include a shaft and a trial block to determine block caving characteristics. This program will provide bulk samples which can be column leach treated to establish the design criteria for the solution circuits as well as resolve the question of chlorine removal or recovery.

# SECTION 4

## VOID HOLE FRAGMENTATION IN SITU LEACH SYSTEM

### 4.1 DESCRIPTION

Discussion

The Void Hole Fragmentation technique is a conceptual approach (patent pending) to the fracturing of a large mineralized body such that solutions may be freely circulated throughout the body for recovery of the valuable minerals in solution.

The technique is based upon the principle that when rock is broken, voids are created throughout the mass and therefore the total volume expands. Openings, or voids, must be made into which the expanded mass can move. This patent-pending system is essentially an explosives column vertically (or angled) through the ore zone under its hundreds of feet of barren overburden. Each explosive column is surrounded by satellite large diameter drill holes that are left void. When the explosives are detonated the subsequent expansion can take place by moving into the void holes provided.

The percentage of voids (or expansion of volume) can be designed into the project by the number of void-holes drilled.

This technique appeared to be a viable system to apply to the Santa Cruz project and was developed as a potential alternate to the Block Cave system covered in Section 5 of this study.

Due to the potential high cost of drilling through the 1,300 of overburden, two cases were developed.

Case 1. Ground Level Drilling

In Case 1 all drilling will be from the surface as shown in Figure 4.1 with a pattern of void and explosive holes. The void holes will be drilled at 12-1/4-inch diameter to the top of the ore zone and from that point drilled and reamed to 26-inch diameter through the ore zone. The largest under-reamer now available is 26-inch diameter and the cost estimate is based upon this factor. However, special 30-inch bits are feasible on a special order basis and the extra cost involved may be more than offset by a 33 percent reduction in the number of void holes required.

The explosion holes will be drilled from the surface to the bottom of the ore zone at 12-1/4-inch diameter.

Leach blocks will be drilled and blasted to provide a fragmented zone measuring 250 feet by 250 feet by 800 feet deep. After blasting, a series of injection and production wells will be installed to form a conventional well field. Each leach block will have the following statistics for purposes of this study:

| | |
|---|---|
| Surface Area | 62,500 square feet |
| Volume | 50,000,000 cubic feet |
| Tonnage @ 2.56 S.G. | 3,993,600 tons |
| Total Copper @ 0.42% Cu | 33,546,240 pounds |
| Recoverable Copper @ 50% | 16,773,120 pounds |
| Voids, after Fragmentation | 4 % |
| Void Holes Required | 660 |
| Explosive Holes Required | 100 |
| Production Wells Required - 6" Dia. | 13 |
| Injection Holes Required - 4" Dia. | 15 |

The production-injection well pattern is a basic 5-spot with 50-foot spacing between the injection and production wells. Eight blocks will be developed during a two-year preproduction period to provide for initial operations at 5,000 GPM to 8 blocks, or a rate of 625 GPM per block. At 13 production wells per block, relatively small pumps of 50 GPM rated capacity will provide the needed capacity.

Case 2. Sub level Well Field

This concept was developed as an alternative to the extremely heavy drill schedule required to penetrate the 1,300 foot thick overburden. In this case, development shafts will be sunk to a depth about 80 feet over the top of the orebody and from that level drive rooms out over the orebody. From these rooms the vertical explosives and void holes can be drilled and shot. The 80-foot sill over the ore will maintain the room so that after the blast re-entry will be possible and leachants can be applied and pumped from the same room.

If the cost of underground preparation is less than drilling from the surface, these advantages will still obtain:

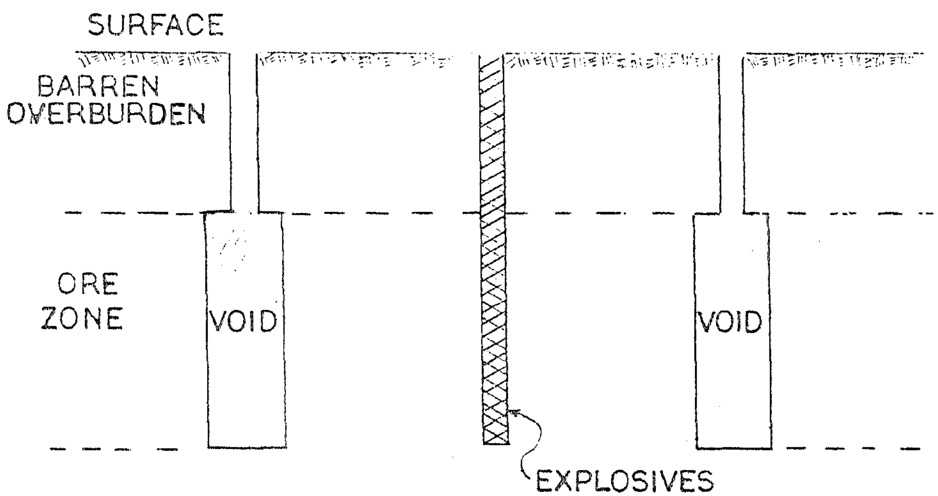
1. The full thickness of the ore can be blasted at once, which is safer and cheaper than the many drill, blast, muck cycles required to take it out in lifts.

2. Miner exposure is minimized and the environment can be controlled.

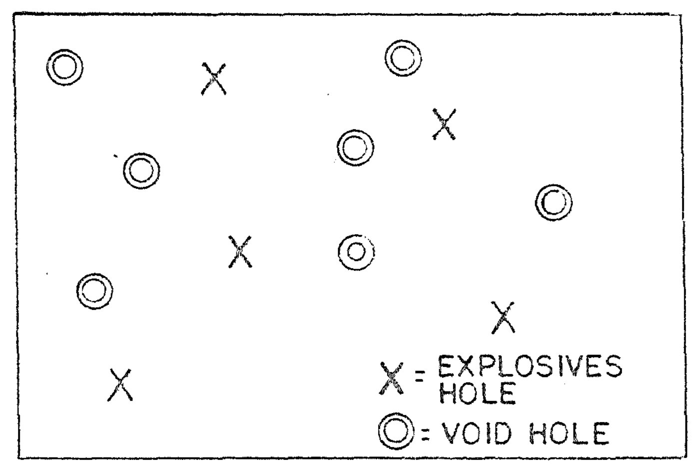
3. Underground development work will be minimal.

This concept is shown in Figure 4.2.

The block statistics are the same as previously given under Case 1.




CROSS-SECTION




PLAN AT TOP OF ORE ZONE

FIGURE 4.1 VOID HOLE FRAGMENTATION

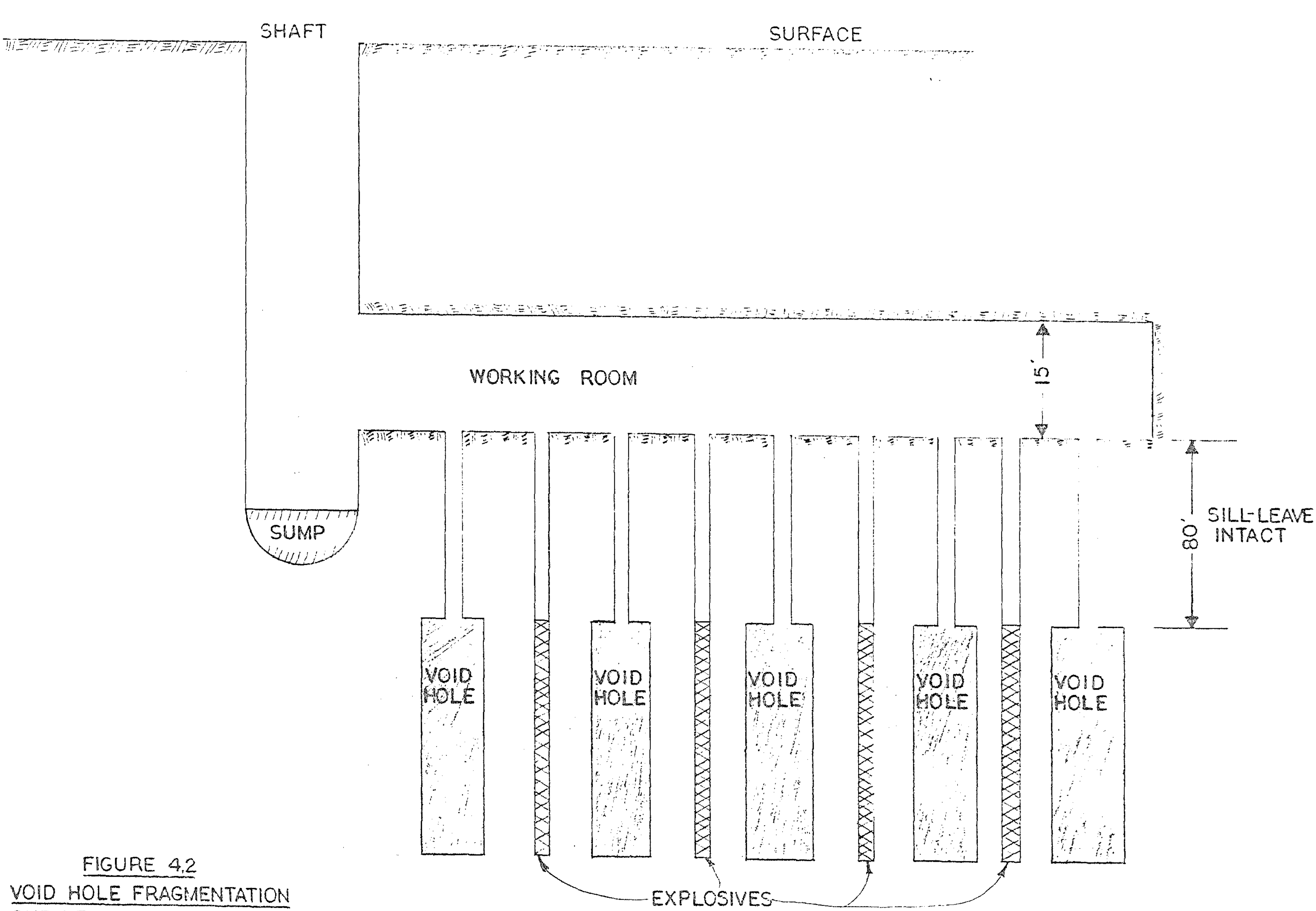



FIGURE 4.2
VOID HOLE FRAGMENTATION
SUB. LEVEL WORKING ROOM

4.2 CAPITAL COSTS

Estimated capital costs were developed for Cases 1 and 2 to provide a comparison between the two concepts. They are based upon the use of unit costs derived from similar operations and from qualified contractors. As both cases are conceptual in nature, the estimates are considered to be order of magnitude only. All costs are in 1980 dollars.

CASE 1. GROUND LEVEL DRILLING.

1. DRILLING AND BLASTING (CONTRACT)

| | Cost ($000) | |
|---|---|---|
| Drilling | | |
| 660 (12-1/4") Holes x 1350 ft. each @ $2.78/ft. | $2,477 | |
| 100 (12-1/4") Holes x 2150 ft. each @ $2.78/ft. | 598 | |
| Total Drilling | $3,075 | |
| Reaming Void Holes | | |
| 660 (26") Holes x 800 ft. @ $3.57/ft. | $1,885 | |
| Explosives - 3.3#/cu. yd. | | |
| 6,111,112# @ $50/100# | $3,056 | |
| Detonation & Loading - $12.50/100# | 764 | |
| Total Explosives | $3,820 | |
| Total Cost - Drill & Blast | $8,780 | |
| Total Cost 8 Blocks - Preproduction | | $70,240 |

2. PRODUCTION AND INJECTION WELLS (8 BLOCKS)

| | | |
|---|---|---|
| Injection Wells (Ream, Case, Cement) | | |
| 15 x 8 x $10,000 | $1,200 | |
| Production Wells (Ream, Case, Cement) | | |
| 13 x 8 @ $12,000 | $1,248 | |
| Pumps, Installed 13 x 8 @ $23,000 | $2,392 | |
| Surface Piping, Storage, Instrumentation | | |
| 27,000 ft. @ $20.00 | 540 | |
| Total Cost Production and Injection Wells | $5,380 | $ 5,380 |
| Total Preproduction Direct Costs | | $75,620 |
| Contingency (25%) | | 18,905 |
| Total Estimated Cost | | $94,525 |

CASE 2. SUB LEVEL WELL FIELD.

| | | Cost ($000) | |
|---|---|---|---|
| 1. | SHAFT - 1,300 ft. @ $1,200/ft. | | $ 1,560 |
| 2. | WORKING ROOM - 250 x 250 x 15' WITH PILLARS | $ 2,082 | |
| | Total - 8 Rooms | $16,656 | $16,656 |
| 3. | DRILLING AND BLASTING (CONTRACT) | | |
| | Drilling | | |
| | 660 (12-1/4") Holes x 80 @ $2.78/ft. | $ 147 | |
| | 100 (12-1/4") Holes x 880 @ $2.78/ft. | 245 | |
| | Total Drilling | $ 392 | |
| | Reaming | | |
| | 660 (26") Holes x 800 @ $3.51/ft. | $ 1,885 | |
| | Explosives - 3.3#/cu. yd. | | |
| | 6,111,112# @ $50/100# | $ 3,056 | |
| | Detonation & Loading - $12.50/100# | 764 | |
| | Total Explosives | $ 3,820 | |
| | Total Cost - Drill & Blast | $ 6,097 | |
| | Total Cost 8 Blocks - Preproduction | | $48,776 |
| 4. | PRODUCTION AND INJECTION WELLS (8 BLOCKS) | | |
| | Injection Wells (Ream, Case, Cement) | | |
| | 15 x 8 x $8,000 | $ 960 | |
| | Production Wells (Ream, Case, Cement) | | |
| | 13 x 8 x $9,600 | 998 | |
| | Pumps, Installed, 13 x 8 @ $21,000 | 2,184 | |
| | Booster Pumps/Pipe to Storage | 100 | |
| | Distribution Piping, 22,000 ft. @ $12.00 | 264 | |
| | Total Cost Production & Injection Wells | $ 4,506 | $ 4,506 |
| | Total Preproduction Direct Costs | | $71,498 |
| | Contingency (25%) | | 17,875 |
| | Total Estimated Costs | | $89,373 |

4.3 OPERATING COSTS

Operating costs for the void hole system are considered to be the cost of developing new blocks each year plus the cost of solution circulation. The operating cost estimates were calculated and presented herein on the cost per pound of copper produced. Separate costs are given for Case 1 and Case 2.

Mining

| | |
|---|---|
| Block Size - 250' x 250' x 800' | |
| Block Area, Square Feet | 62,500 |
| Total Tons @ 2.56 S.G. | 3,993,600 |
| Copper Recovery - % | 50 |
| Copper Recoverable/Ton @ 0.42% Cu | 4.2 |
| Copper Recoverable - Total Pounds | 16,773,120 |
| Copper Recoverable/Square Foot | 268 |

CASE 1

| | |
|---|---|
| Cost per Block | $9,452,500 |
| Cost per Square Foot | $ 151 |
| Cost/lb Copper - $151/268 | 56.3¢ |

CASE 2

| | |
|---|---|
| Cost per Block | $8,742,000 |
| Cost per Square Foot | $ 140 |
| Cost/lb Copper - $140/268 | 52.2¢ |

| | | Cost (¢/lb. Copper) |
|---|---|---|
| **CASE 1** | | |
| 1. Mining and Well Installation | | 56.3¢ |
| 2. Leaching | | |
| Acid - 2.5 lb. @ 1.0¢ | 2.5¢ | |
| Pumping | 5.0¢ | |
| Total Leaching | 7.5¢ | 7.5¢ |
| Total Direct Operating Costs | | 63.8¢ |
| Administration & Indirects | | 13.0¢ |
| Total (Excluding Process Plant) | | 76.8¢ |
| **CASE 2** | | |
| 1. Mining and Well Installation | | 52.2¢ |
| 2. Leaching | | |
| Acid - 2.5 lb. @ 1.0¢ | 2.5¢ | |
| Pumping | 5.0¢ | |
| Total Leaching | 7.5¢ | 7.5¢ |
| Total Direct Operating Costs | | 59.7¢ |
| Administration & Indirects | | 13.0¢ |
| Total (Excluding Process Plant) | | 72.7¢ |

It should be noted that the above capital and operating costs are based on assuming that 4 percent void space would be sufficient to provide the required permeability in the fractured ore. Previous studies have indicated that about 8 percent void space is necessary to obtain the desirable percolation rates. Accordingly, the cost may be substantially higher if the higher void space is required.

# SECTION 5

## BLOCK CAVE IN SITU LEACH SYSTEM

### 5.1 DESCRIPTION

The Block Cave In Situ Leach System described herein was conceptually developed for this study as a potentially viable approach to solution mining of the deep-lying Block B oxide ore zone. At first inspection, the 1300' thick overburden represented a sizeable drilling cost to install a typical drilled well field and so an alternate solution was sought. In developing this system the following criteria were considered:

1. The method would provide maximum permeability for solution flow.

2. Operations would be restricted to the ore zone.

3. Maximum solution control could be maintained using presently developed technology.

4. Known mining technology could be employed.

The Block Cave In Situ Leach System meets all of these conditions.

In concept, the system is based upon sinking shafts to the bottom level of the ore zone (approximate depth 2,300 feet) and driving a series of haulage and crosscut drifts as in a conventional block caving system. The ore body will then be undercut with approximately 10 percent of the ore body hoisted to the surface, stockpiled and heap leached. Concurrently, a series of solution drifts will be driven at the 1,300 foot level to provide access to the top of the ore body for solution injection.

As mining progresses, blocks 150' x 150' by 800' deep will be induced to cave such that the entire block is fractured. Solutions will then be circulated through the block until the copper extraction rate decreases to an uneconomic level.

Mining will continue on the two levels throughout the 25 year life of the project at such a rate as to provide new blocks for leaching as earlier blocks are exhausted.

Figures 5.1 and 5.2 show respectively plan and cross section views of the mine development.

<u>Shafts</u>

Access to the ore body will be provided by a main shaft containing both hoisting and service facilities with a ventilation shaft provided at the opposite side of the ore body. The main haulage drift at the 2,300 foot level will connect the two shafts. The shaft will be capable of hoisting 7,000 tons per day on a 5-day work week.

Mining

Based upon results obtained at Miami Copper, San Manuel Copper and the Lawrence Radiation Lab curve developed from nuclear underground explosions, it is expected that 10 percent of the ore column must be removed to break the column. This, however, will have to be confirmed or revised by underground testing to assure block cave arch failure to the top of the ore body.

The mining system will incorporate lateral transfer by slushers on the control level with rail haulage on the main level. The slusher drifts will be on 30 foot centers with draw raises on 17 foot centers with a 15 foot pillar between panels. These pillars will be crushed and broken for leaching.

A main drift will connect the main and ventilation shafts on the 1,300 foot level to provide for ventilation and the main solution distribution headers. Lateral crosscuts will be driven over each block to provide access for distribution of leaching solutions to the blocks.

Each 150 x 150 x 800 leach block has the following statistics:

| | |
|---|---|
| Area for solution injection | 22,500 square feet |
| Solution flow @ 0.0045 gpm/ft$^2$ | 100 GPM |
| Total volume | 18,000,000 cubic feet |
| Volume to be mined | 1,800,000 cubic feet |
| Tonnage to be mined @ 2.56 SG | 144,000 tons |
| Mining days at 7,000 TPD | 20.6 days |
| Blocks per year @ 250 days/yr | 12.1 |

As the entire Block B contains 6,250,000 square feet, a total of 40 leach blocks will be developed during preproduction and 238 blocks over a period of 19.7 years and leaching completed in 25 years.

Solution Operations

Block B contains an estimated 380,000,000 tons of ore averaging 0.42 percent copper. The block is approximately 2,500 x 2,500 feet with an average thickness of 800 feet. At 50 percent recovery, total recoverable copper is:

$380 \times 10^6 \times 0.42\% \times 50\% = 1,596 \times 10^6$ pounds copper.
At a projected 25 year mine life average production will be:
$1,596 \times 10^6/25 = 63.84 \times 10^6$ pounds copper per year.

At 365 days per year:
$63.84 \times 10^6/365 = 174,904$ pounds copper per day.

A strong solution of 2.9 grams per liter (gpl) flowing at 5,000 gallon per minute (GPM) will produce copper at this rate using the first solvent extraction module; however, as the leach blocks decrease in copper grade the total flow must be incresed to compensate for lower extraction rates and weaker solutions produced. The projected production schedule giving consideration to block development time and leaching times are given in Table I. At approximately year 5, the second phase S-X plant will be placed in service to provide capacity for the increased flows.

TABLE I

PRODUCTION SCHEDULE

| Production Year | Flow GPM | Assay G/L Cu | Copper/Year Lbs x $10^6$ |
|---|---|---|---|
| 1 | 4,000 | 2.6 | 45.6 |
| 2 | 5,000 | 2.5 | 54.7 |
| 3 | 6,000 | 2.4 | 63.2 |
| 4 | 7,000 | 2.3 | 70.6 |
| 5 | 8,000 | 2.1 | 73.7 |
| 6 | 9,000 | 1.8 | 71.1 |
| 7-25 | 10,000 | 1.46 | 1,217.1 |
| Total | | | 1,596.0 |

Table II shows the number of blocks in service at various total flow rates if a constant solution distribution rate of 0.0045 GPM per square foot of leach area is maintained.

TABLE II

TOTAL FLOW/BLOCK RELATIONSHIP

| Total Flow GPM | Blocks in Service |
|---|---|
| 4,000 | 40 |
| 5,000 | 50 |
| 6,000 | 60 |
| 7,000 | 70 |
| 8,000 | 80 |
| 9,000 | 90 |
| 10,000 | 100 |

As individual blocks will vary in copper content and total leaching time, the actual operating schedule will be developed to maintain a balance of new and almost depleted blocks such that overall copper production will be at a relatively constant level.

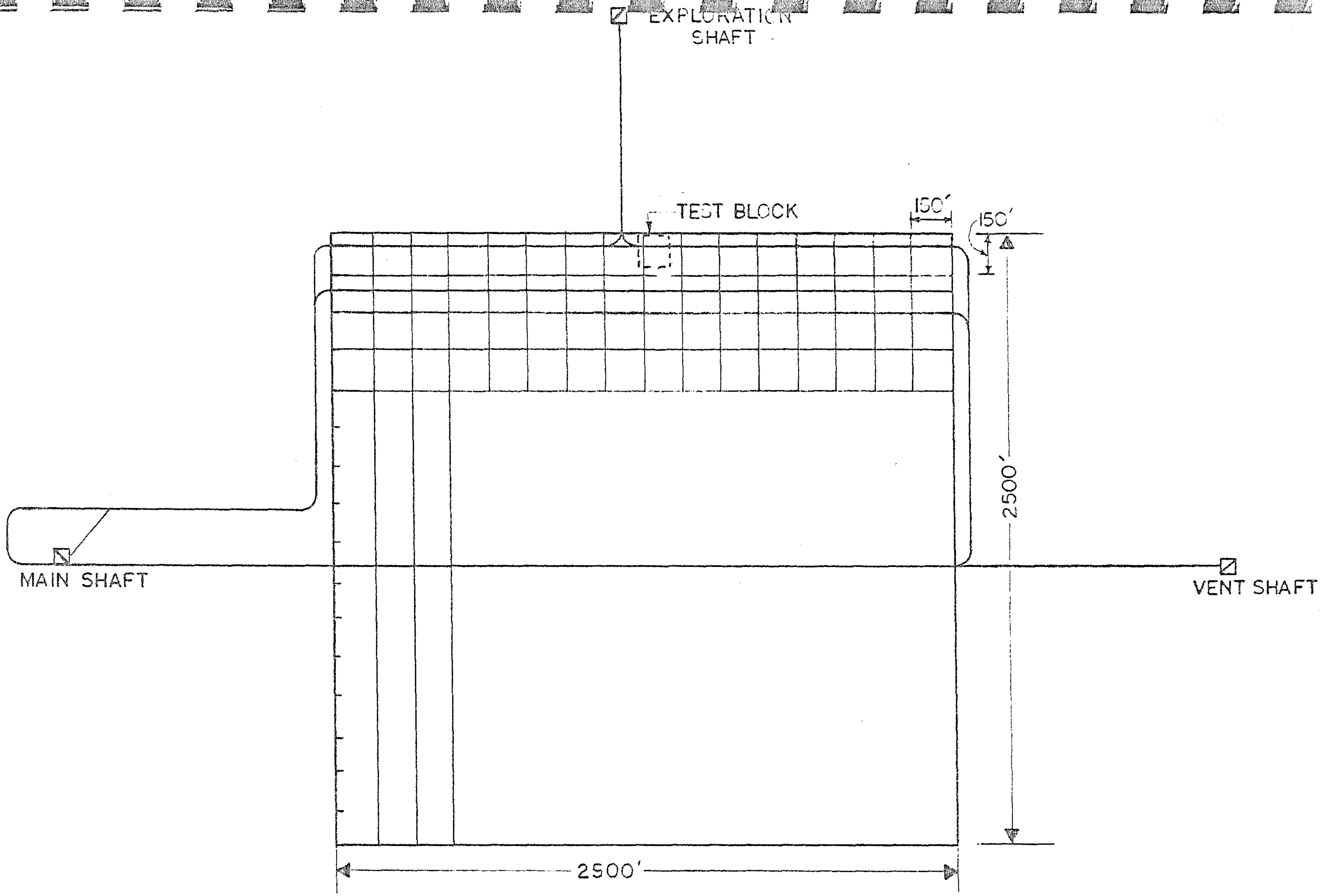



FIGURE 5.1

BLOCK CAVE LEACH IN PLACE

PLAN VIEW

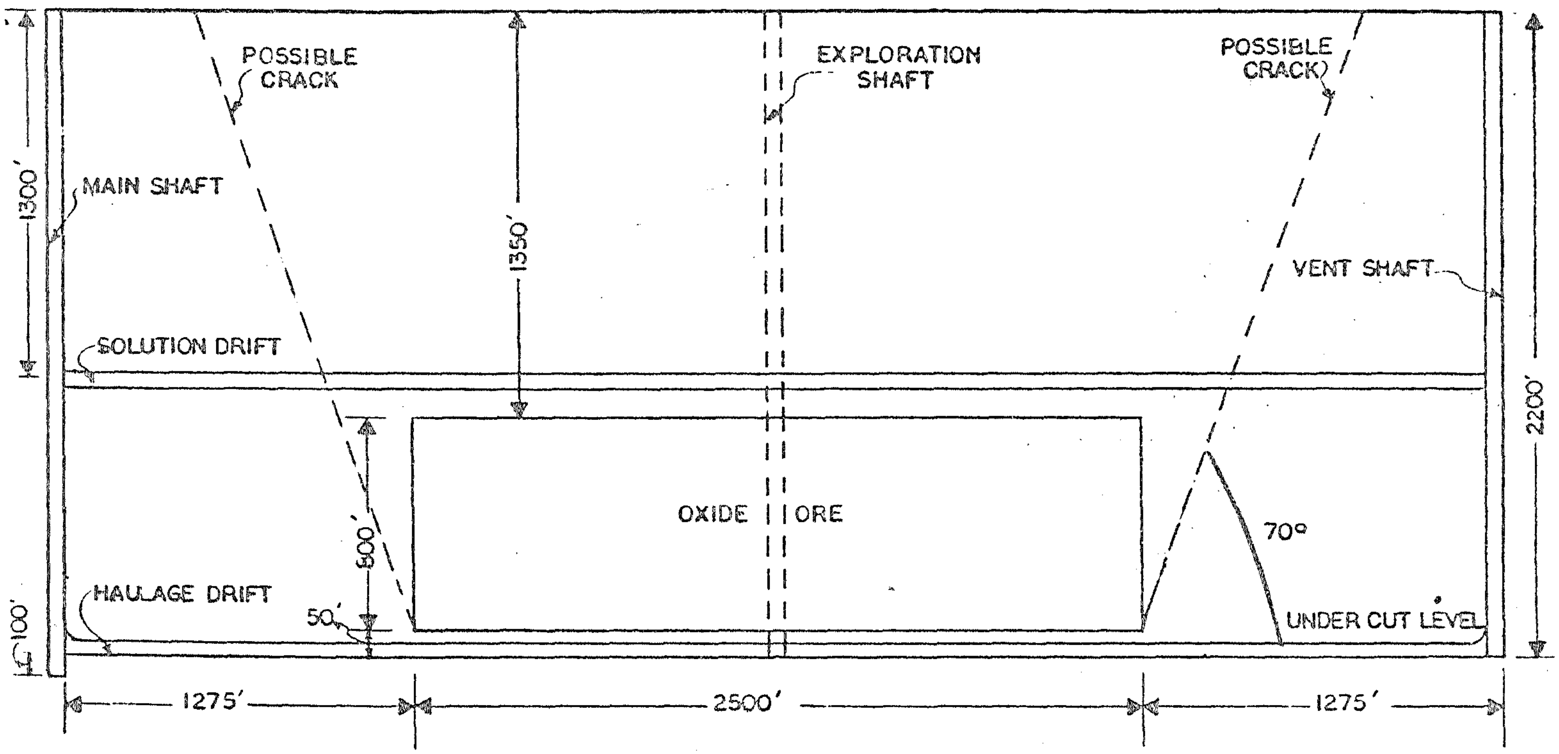



FIGURE 5.2
BLOCK CAVE LEACH IN PLACE
TYPICAL CROSS SECTION

NOTE:

SHAFTS OFFSET 1275' FROM BLOCK CAVE AREA TO ALLOW FOR 70° VERTICAL SHAFT SUPPORT COLUMN.

## 5.2 CAPITAL COSTS

Estimated capital costs were developed for the Block Cave System to include:

1. Exploration Costs

2. Mine Development (Preproduction)

3. Surface Facilities (EXCLUDING PROCESS PLANT)

The estimates are based upon unit cost factors and cost factors derived from similar type operations. All costs are in 1980 dollars.

| | | |
|---|---|---|
| 1. | Exploration Costs | $ 3,205 |
| 2. | Mine Development Costs (Preproduction) | 22,422 |
| 3. | Surface Facilities (Excluding Process Plant) | 3,300 |
| | Contingency (15%) | 4,339 |
| | Total | $33,266 |

| | | Cost ($000) | |
|---|---|---|---|
| 1. | EXPLORATION COSTS | | |
| 1.1 | Development | | Shaft |
| | Design | $ 40 | |
| | Development Setup | 400 | |
| | Shaft Sinking & Equipping - 2,200 ft. @ $1,000 | | 2,200 |
| | Total Development Shaft | $2,640 | $2,640 |
| 1.2 | Exploration - Underground | | |
| | Development Drifts - 1,200 ft. @ $300 | $ 360 | |
| | Drill Cut-Outs - 20 @ $2,000 | 40 | |
| | Drilling BX Core - 6,000 @ $15 | 90 | |
| | Exp. Leach Block Launders | 40 | |
| | Total Exploration - Underground | $ 530 | $ 530 |
| 1.3 | Surface Leach Pad and Dump | $ 35 | $ 35 |
| | Total Direct Exploration Costs | | $3,205 |

|  | Cost ($000) |  |
|---|---|---|
| 2. MINE DEVELOPMENT - PREPRODUCTION |  |  |
| 2.1 Hoisting Shaft |  |  |
| Design | $ 120 |  |
| Sinking & Equipping - 2,300 ft. @ $1,600 | 3,680 |  |
| Haulage Level Station | 200 |  |
| Solution Level Station | 120 |  |
| Pocket | 400 |  |
| Total Hoisting Shaft | $4,520 | $ 4,520 |
| 2.2 Underground Facilities |  |  |
| Pump Station | $ 800 |  |
| Electrical Sub Station | 200 |  |
| Ventilation Drifts - 3,000 ft. @ $300 | 900 |  |
| Haulage Drift - 12,000 ft. @ $300 | 3,600 |  |
| Supply Elevator Raises - 1,000 ft. @ $450 | 450 |  |
| Standby Power | 150 |  |
| Total Underground Facilities | $6,100 | $ 6,100 |
| 2.3 Developed Blocks (4 Total) | $2,102 | $ 2,102 |
| 2.4 Shaft Equipment |  |  |
| Hoist | $2,000 |  |
| Cage & Skips | 200 |  |
| Elevator | 600 |  |
| Hoist & Cage - Vent Shaft | 600 |  |
| Total Shaft Equipment | $3,400 | $ 3,400 |
| 2.5 Mine Equipment |  |  |
| Mining Equipment | $1,500 |  |
| Haulage Equipment | 3,000 |  |
| Fans | 400 |  |
| Pipe | 600 |  |
| Pumps | 800 |  |
| Total Mine Equipment | $6,300 | $ 6,300 |
| Total Direct Mine Development Costs |  | $22,422 |

| | | Cost ($000) | |
|---|---|---|---|
| 3. | Surface Facilities (Excluding Process Plant) | | |
| 3.1 | Hoist House, Change Room Office, Shops | $1,000 | $1,000 |
| 3.2 | Head Frame and Bins | 500 | 500 |
| 3.3 | Electrical Distribution, Transformer Stations | 800 | 800 |
| 3.4 | Water Development | 1,000 | $1,000 |
| | Total Surface Facilities | | $3,300 |

| | Cost ($000) |
|---|---|
| UNIT MINING COST (150 x 150 FOOT BLOCK) | |
| Haulage Drift - 150 ft. @ $300 | $ 45 |
| Chutes - 5 @ $2,200 | 11 |
| Transfer Raise - 100 ft. @ $70 | 7 |
| Slusher Drift - 750 ft. @ $240 | 180 |
| Fringe Drift - 150 ft. @ $240 | 36 |
| Draw Raises - 1,200 ft. @ $30 | 36 |
| Undercutting - 18,000 square feet @ $7 | 126 |
| Sub-Total | $ 441 |
| Solution Distribution Drifts - 150 ft. @ $450 | 67.5 |
| Drill Holes - 1,600 ft. @ $10 | 16 |
| Sub-Total | 83.5 |
| Total Cost per Block | $525.5 |

At 22,500 Square Feet

| | | |
|---|---|---|
| Mining: | $441,000/22,500 = | $19.60 per square foot |
| Solution: | $ 83,500/22,500 = | $ 3.71 per square foot |
| Total | | $23.31 |

5.3 OPERATING COSTS

Operating costs for mining and solution operations were estimated on the basis of cost per pound of copper produced. The estimates are in 1980 dollars and based on the following factors:

Mining Cost

The average head of ore to be broken is 800 feet with 10 percent of the column to be mined and leached in dumps on the surface. The 80 feet mined represents 80/12.5 or 6.4 tons per square foot of leach block. The cost per ton is estimated as:

| | | |
|---|---|---|
| Drawing | - | 70¢ |
| Haulage | - | 20¢ |
| Hoisting | - | 35¢ |
| Dumping | - | 20¢ |
| Total | | $1.45 |

$1.45 x 6.4 = $9.28 per square foot leach area.

$9.28 + $23.32 = $32.59 total mining cost per square foot of leach block developed.

Recoverable Copper

Recoverable copper per square foot of leach area:

380 x $10^6$ tons ore at 4.2 lbs/ton = 1,596 x $10^6$ pounds copper.

2500' x 2500' = 6.25 x $10^6$ square feet.

1,596/6.25 = 255 lbs. recoverable copper/square foot.

Acid Consumption

Leach test results indicate 4.0 pounds sulfuric acid required per pound copper recovered.

The S-X plant returns 1.5 pounds in the raffinate for a net consumption of 2.5 pounds.

| | | | Cost (¢/lb. Copper) |
|---|---|---|---|
| 1. | Mining | | |
| | $32.59/255 = | | 12.8¢ |
| 2. | Underground Leaching | | |
| | Acid - 2.5 lb. @ 1.0¢ | 2.5¢ | |
| | Solution Distribution | 3.0¢ | |
| | Underground Pumping | 5.0¢ | |
| | Total Leaching | 10.5¢ | 10.5¢ |
| 3. | Surface Leaching | | |
| | Distribution & Pumping | | 2.0¢ |
| | Total Direct Operating Costs | | 25.3¢ |
| | Administration and Indirects | | 13.0¢ |
| | Total (Excluding Process Plant) | | 38.3¢ |

## SECTION 6

## PROCESS PLANT

### 6.1 DESCRIPTION

This study is based upon a solvent extraction-electrowinning plant for recovery of copper from strong leach solutions and the production of marketable cathode copper. The strong leach solutions from the mine and dump heap leach operations will be treated in the solvent extraction (S-X) section for removal of copper by an organic solvent, and then returned to the mine and dump solution circuits as new lixiviant. The organic solvent will be treated with strong sulfuric acid solution to strip the copper from the organic and produce a high copper content aqueous solution suitable for electrolysis. This solution will then be sent to the tankhouse for electro-deposition of the copper as cathodes. The spent electrolyte from the tankhouse is returned to the S-X section for stripping of loaded organic.

#### Mining Solutions

A dilute sulfuric acid solution will be used in the mining operation. An initial acid charge will be required to acidulize the ground water in the ore zone and produce the first strong leach solutions. As operations progress, decopperized acid solutions (raffinate) from the S-X plant will be reinjected to the leach zone. Make up acid and water will be added to the mining solution circuit to maintain the desired acid and volume conditions.

The process plant will include storage tanks for receipt of strong leach solution ahead of the S-X section, raffinate storage tanks, acid storage tanks, fresh water storage, organic storage, kerosene storage and necessary pumps.

#### Solvent Extraction Section

The solvent extraction section consists of a series of mixer settler tanks which provide for the mixing of the aqueous organic phases followed by a quiescent settling zone for separation of the two phases. To provide for 95 percent extraction of the copper to the organic phase from the aqueous phase, a typical three stage circuit is described.

The strong leach solution is fed to the mixer of No. 1 mixer-settler where it is mixed with the organic overflow of No. 2 mixer-settler. The mixed phases are pumped by the mixer impeller to the settler portions of the cell where the two phases are allowed to settle and separate. The lighter organic overflows the top of the settler and is pumped to the stripper circuit. The aqueous phase discharges over a bottom weir and flows to No. 2 mixer where it is mixed with the organic overflow from No. 3 settler. After separating in the No. 2 settler the aqueous progresses to the No. 3 mixer where it is mixed with barren organic solvent from the stripper circuit. After separating in the No. 3 settler, the aqueous phase, or raffinate, is pumped to storage and subsequent recycling to the mining area. As can be seen, the barren organic enters the extraction circuit at No. 3 mixer-settler and progresses countercurrent to the aqueous phase.

The copper loaded organic from No. 1 mixer is fed to a series of mixer-settlers in the stripper circuit where the copper is stripped into a strong acid solution. As just described, the organic and aqueous phases progress countercurrent to provide maximum stripping efficiency.

The entire S-X Section is designed as an all weather installation and does not require any building or enclosures other than those provided on the tanks and mixer-settlers.

Electrolytic Tankhouse

The tankhouse comprises a single building with a full basement area under the cell floor to provide adequate headroom for gravity return of solutions from the cells to the appropriate pump sumps. The cells will be monolithic concrete tanks with preformed paraliners. A number of the cells will be dedicated to starting sheet production with stainless steel cathodes and lead-antimony anodes. Rolls, shears and looping machines are provided for preparation of the starting sheets on a daily basis. These starting sheets become the core cathode in the production cells. A rectifier station will provide the required electrolytic current.

A cathode storage and shipping area is provided adjacent to the main building as well as solution storage tanks.

Evaporation Ponds

Due to the presence of Atacamite ($Cu_2(OH)_3Cl$), the pregnant solutions will gradually increase in chlorine content to the point that adverse reactions will occur in the solvent extraction circuit. With an assumed tolerance of 20 grams per liter of chlorine by the LIX 64N solvent used in the S-X circuit, the pregnant solutions will be allowed to increase in chlorine to this level and then controlled by a bleed stream discard of raffinate (low copper) to evaporation ponds. An average discard rate of approximately 40 gallon per minute is required to maintain the solution in balance. Under this system copper losses will be minimal but will involve an acid loss.

Two evaporation ponds 660 feet x 660 feet x 5 feet deep will be provided to meet the evaporation requirements. Each pond will be membrane lined and provide a four foot evaporation depth with one foot of freeboard.

Sizing was based upon the following information and assumptions:

1. 31% of the copper in Block B occurs in zones containing Brochantite, Atacamite, and Dioptase. These are all designated as vitreous green oxides. (H.G. Kreis memorandum of July 28, 1978).

2. 65% of the vitreous green oxide minerals occur as the chlorine containing Atacamite.

3. Atacamite contains 59.5% Cu and 16.6% Cl.

4. Production solutions average 2.9 grams per liter copper.

5. Net evaporation rate is 60 inches per year.

6. Maximum LIX64N tolerance for chlorine in low copper-acid solutions is 20 grams per liter.

## 6.2 CAPITAL COSTS

Basis of the Estimate

The capital cost estimate for the process plant is a factored estimate determined through application of an exponent factor of 0.7 to the capacity ratio between this plant and a similar project for which reliable data is available.

It is based upon the following general conditions and assumptions:

1. The project site is located within reasonable commuting distance of an adequate supply of craft labor.

2. Project is to be on a "turn key" basis.

3. The estimate includes facilities within the battery limits for the solvent extraction plant, electrolytic plant, and evaporation ponds only.

4. The estimate is based on 1980 dollars.

5. The accuracy of this estimate is assumed to be within ± 35%.

Estimates

Estimates for the process plant are given on Sheets S-1 and S-2 following.

Phase I covers the process plant facilities to treat strong leach solutions at a rate of 5,000 gallon per minute with a copper content of 2.9 grams per liter and produce electrolytic cathode at a rate of 200,000 pounds per day.

Phase II covers an additional solvent exchange section for the treatment of an additional 5,000 gallon per minute of leach solution. This addition will be required to handle increased solution flow rates to compensate for a decrease in solution grade as a result of ore body depletion. There will be no expansion of the tankhouse due to the offsetting effects of increased solution flow with decreased copper content.

mountain states engineers

REV. DATE

PROJECT IN SITU COPPER LEACH PROCESS | PREPARED BY JDK | FIRST ISSUE

CLIENT ASARCO | APPROVED BY | SHEET S-1 of S-2

| A/C NO. | AREA PHASE I PROCESS | QUANTITY | UNIT | MANHOURS PER UNIT | MANHOURS TOTAL | COST UNITS L M E | TOT. or SUB | LABOR | MATERIAL | EQUIPMENT | SUB CONTRACTOR | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | FACILITY | | | | | | | | | | | |
| | STAGE I SOLVENT EXTRACTION (5000) GPM | | | | | | | | | | | 1,975,000 |
| | ELECTROWINNING PROCESS (200,000 LB/DAY) | | | | | | | | | | | 6,350,000 |
| | EVAPORATION PONDS - NO ROCK ASSUMED | | | | | | | | | | | 1,131,000 |
| | SUBTOTAL - DIRECT COSTS | | | | | | | | | | | 9,456,000 |
| | | | | | | | | | | | | |
| | SUPPLEMENTAL DIRECT COSTS | | | | | | | | | | | 1,417,000 |
| | FIELD INDIRECT COSTS | | | | | | | | | | | 424,000 |
| | | | | | | | | | | | | |
| | TOTAL FIELD COSTS | | | | | | | | | | | 11,297,000 |
| | | | | | | | | | | | | |
| | HOME OFFICE SUPPORT | | | | | | | | | | | 220,000 |
| | ENGINEERING | | | | | | | | | | | 1,129,000 |
| | | | | | | | | | | | | |
| | SUBTOTAL | | | | | | | | | | | 12,646,000 |
| | | | | | | | | | | | | |
| | CONTINGENCY (+25%) | | | | | | | | | | | 3,161,000 |
| | ESCALATION - NONE: ALL COSTS ARE THROUGH | | | | | | | | | | | |
| | 1/31/80 | | | | | | | | | | | |
| | TOTAL EXCLUDING FEE* | | | | | | | | | | | 15,807,000 |
| | IF ROCK ENCOUNTERED IN EXCAVATING EVAP.PONDS CAPITAL COST MIGHT INCREASE $260,000 | | | | | | | | | | | |

* THE ACCURACY OF THIS ESTIMATE IS ASSUMED TO BE WITHIN ± 35%

PROJECT NUMBER 2 1 4 0

FORM S-301

mountain states engineers

REV. DATE

PROJECT IN SITU COPPER LEACH PROCESS PREPARED BY RHK 12/26/79 FIRST ISSUE

CLIENT ASARCO APPROVED BY SHEET S-2 of S-2

| A/C NO. | AREA PHASE II PROCESS | QUANTITY | UNIT | MANHOURS PER UNIT | MANHOURS TOTAL | COST UNITS L M E | COST UNITS TOT. or SUB | COSTS LABOR | COSTS MATERIAL | COSTS EQUIPMENT | COSTS SUB CONTRACTOR | COSTS TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | FACILITY | | | | | | | | | | | |
| | STAGE II SOLVENT EXTRACTION | | | | | | | | | | | |
| | (5000 GPM) | | | | | | | | | | | 1,975,000 |
| | | | | | | | | | | | | |
| | SUPPLEMENTAL DIRECT COSTS | | | | | | | | | | | 300,000 |
| | FIELD INDIRECT COSTS | | | | | | | | | | | 120,000 |
| | | | | | | | | | | | | |
| | TOTAL FIELD | | | | | | | | | | | 2,395,000 |
| | | | | | | | | | | | | |
| | HOME OFFICE SUPPORT | | | | | | | | | | | 50,000 |
| | ENGINEERING | | | | | | | | | | | 250,000 |
| | | | | | | | | | | | | |
| | SUBTOTAL | | | | | | | | | | | 2,695,000 |
| | | | | | | | | | | | | |
| | CONTINGENCY (+25%) | | | | | | | | | | | 675,000 |
| | ESCALATION - NONE: ALL COSTS ARE THROUGH | | | | | | | | | | | |
| | 12/31/79 | | | | | | | | | | | |
| | | | | | | | | | | | | |
| | TOTAL EXCLUDING FEE* | | | | | | | | | | | 3,370,000 |

*THE ACCURACY OF THIS ESTIMATE IS ASSUMED TO BE WITHIN ± 35%

PROJECT NUMBER 2 1 4 0

FORM S-301

### 6.3 OPERATING COSTS

Direct operating costs for the process plant were developed using general experience factors from similar type plants and operations. Repair labor and supply costs were taken as a percentage of the plant capital cost. The estimates are presented in cost per pound copper recovered in 1980 dollars.

| | | Cost (¢/lb. Copper) |
|---|---|---|
| 1. Labor | | |
| Supervision | 0.427¢ | |
| Operating Labor | 1.548¢ | |
| Fringe Benefits - 45% | .889¢ | |
| Total Labor | 2.864¢ | 2.864¢ |
| 2. Repair Labor and Supplies | | 2.256¢ |
| 3. LIX and Kerosene | | 0.912¢ |
| 4. Sulfuric Acid | | 2.500¢ |
| 5. Power | | 2.800¢ |
| Total Direct Operating Costs | | 11.332¢ |

Note: Administration and indirect overhead costs are included in mine operating costs - Subsection 5.3.

RECENT ADVANCES IN MINING OF LOWER GRADE ORES

Part I

IN SITU MINING - THEORETICAL AND PRACTICAL ASPECTS

Part II

BLOCK CAVE - IN PLACE LEACHING

by

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## Introduction

The conventional approach to the mining and processing of low-grade ore deposits is "giantism". The economic advantages of large-scale and mass-production methods have been utilized for lowering overall production costs, thus compensating for lower grade of ore. The large, open-pit mines in the Southwestern United States, Chile, Peru and the USSR are classic examples of reducing unit cost by spreading the cost of highly mechanized automated facilities over the largest possible volume of output.

However, large capital investment and considerable risk are involved in these larger mineral development projects as the following figures for two new copper mining projects will show.

| Project Name | Reserves (million tons) | Grade (%) | Capital Required (million $) | Mill Capacity (tons/day) |
|---|---|---|---|---|
| Sierrita - Arizona, USA | 414 | 0.35 Cu<br>0.036 Mo | 163 | 80,000 |
| Bethlehem Copper Co., Highland Valley British Columbia, Canada | 1,000 | 0.48 | 300 | 75,000 |

Technological advances in mining have been the subject of several recent technical meetings and are well documented; the topics covered include larger trucks and conveyors used for transporting mined ores, larger milling equipment, studies of back slopes of open pits, construction of large tailing dams, recent advances in underground mining and ground supports, rapid excavation, mechanized mining, raise boring, shaft sinking and many other relevant items.

Rather than a representation of published information on the above techniques, the present paper discusses newer developments in mining technology which are readily applicable to lower grade and submarginal ore deposits. These include in situ mining technology and the block cave - in place leaching concept, two classical examples of which are the economically successful in place leaching operation at Miami, Arizona, and the more recent application of the process at the Old Reliable Copper deposit near Mammoth, Arizona. In the latter, 4 million pounds of explosives were employed to shatter a near-surface deposit of copper containing about 4 million tons of ore, to a size that will permit satisfactory in place leaching operations.

## PART I

## IN SITU MINING - THEORETICAL AND PRACTICAL ASPECTS[1]

In situ mining may be defined as the extraction of metals from ores located within the confines of a mine (broken or fractured ore, stope fill, caved material, ores in permeable zones) or in dumps, prepared ore heaps,

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[1]This contribution was prepared in cooperation with Dr. Paul Johnson former Metallurgist with N.M. Bureau of Mines and Mineral Resources, Socorro, N.M. The authors gratefully acknowledge the cooperation of Mr. Don H. Baker, Jr., Director of the Bureau for permitting us to present this contribution.

slag heaps, and tailing ponds on the surface. These materials represent an enormous, untapped, potential source of all types of metals. The field of in situ mining, now in its infancy, encompasses the preparation of ore for subsequent in place leaching, the flow of solutions and ionic species through rock masses and within rock pores, the leaching of minerals with inexpensive and regenerable lixiviants under prevailing conditions of the in place environment, the generation and regeneration of such solutions, and the recovery of metals or metal compounds from the metal-bearing liquors.

It is not inconceivable that eventually our ore reserves will consist largely of low-grade, refractory, and inaccessible new deposits and low-grade zones near previously worked deposits, caved and gob-filled stopes, waste dumps, tailing ponds, and slag heaps. In situ mining promises economic recovery from these types of deposits, but full appreciation of its potential needs a much better understanding of its chemical and physical aspects.

This kind of mining has previously largely been limited to the extraction of copper from low-grade materials. The potential is, however, much greater as practically all metals are susceptible to leaching in the in situ environment. Processes will soon be developed for the in situ extraction and recovery of metals such as copper, lead, zinc, nickel, manganese, uranium, silver, gold, molybdenum, and mercury.

Any process used in mining or mineral processing has certain advantages and disadvantages. A few for chemical mining are listed below:

Advantages

1. In situ mining can often be used to recover metals economically from materials that could not be so treated by more conventional mining, milling, and smelting processes.

2. An in situ mining plant usually requires less capital investment than a conventional mine and mill plant.

3. An in situ mining process usually increases a mine's ore sources and reserves. Low-grade or inaccessible ore zones, gob and caved fill, and dumps and tailings may become ores.

4. The leach liquors obtained through in situ mining usually lend themselves to a variety of metal recovery processes. The pure metal or metal compounds so obtained may be of greater value than the sulphide or oxide products normally obtained by conventional milling processes.

5. In situ mining may prove applicable for ores that are too refractory for conventional recovery processes.

6. In situ mining can often be used in conjunction with a conventional mining or milling process to boost metal recoveries and increase ore reserves.

### Disadvantages

1. Both physical and chemical restraints may limit the usefulness of a chemical mining process. The effectiveness of contacting ore with solutions and the recovery of leach solutions from the system without appreciable loss are two important physical factors. Dissolution or dissolution rates, metal precipitation, and solution regeneration are major chemical factors.

2. Testing an in situ mining process short of actual field operation sometimes proves difficult.

3. Groundwater contamination may result from some chemical mining operations.

4. Basic information on the physical and chemical factors involved is presently lacking.

## In Situ Mining Technology

The field of in situ mining may be considered under the headings of (1) mining economics and ore evaluation, (2) elements of the leaching phase, (3) preparation of ores, (4) practical aspects of in situ leaching, (5) reagent generation and regeneration, and (6) recovery of metals from leach liquors.

### Mining Economics and Ore Evaluation

In considering the economic exploitation of a deposit through in situ mining, one must determine the size of the deposit, tonnage of ore in place, and amount of metal contained therein. In past as well as present mining operations, the cut-off grade has been governed by the total operating cost, including mining, which usually constitutes a significant portion of the overall cost. In chemical mining considerations, however, the cost of mining would be minor and the cut-off grade can be lowered correspondingly. This would inevitably increase the tonnage as well as the metal content of the deposit, which in turn would influence the overall economics of the venture.

Unfortunately, information concerning the relationship between tonnage and grade is largely lacking in the literature. No doubt records of mining companies may contain such valuable information, and some attempts should be made to obtain pertinent data from these sources.

Lasky (1950), Musgrove (1965) and a few others have studied this relationship through a statistical analysis of known deposits and perusal of past records of some mining companies. These studies reveal that there is an exponential relationship between grade and tonnage of ore reserves. Especially for deposits in which there is a gradation from relatively rich to relatively lean material, there appears to be a consistent mathematical relation between tonnage and grade, according to the equation $G = K_1 - K_2 \log T$, (1) where T is the tonnage produced to a given time plus the estimated reserves, G is the weighted average grade of this tonnage, and $K_1$ and $K_2$ are constants to be determined for each deposit. Using equation (1), Lasky (1950) showed that for a typical porphyry copper deposit, the tonnage increases at a compound rate of 14.9 per cent for each 0.1 per cent decrease in grade.

Another important aspect of <u>in situ</u> mining on which hardly any data are available is determining a minimum reserve and grade for profitable exploitation. The only operational data available are from copper dump leaching and in place leaching practices in which the grade of material treated is above 0.16 per cent.

The important factor in chemical mining, as in dump leaching, is making sure that the major portion (+90%) of the specified volume of leach solution fed to the deposit or dump is recovered with a given minimum amount of metal content in solution over the life of the economic operation. This minimum metal content in the specified volume is such that the value of the recovered metal will provide for the cost of operation, amortization and profits.

Naturally, the metal content and its value in leach solution differs from metal to metal. From a hydrometallurgical recovery viewpoint alone, it is estimated that at the current prices of metals and operating conditions, the break-even contents for a minimum operation of 200,000 gallons a day are 250 ppm (0.25 g/l) copper, 50 ppm (0.05 g/l) molybdenum, and 10 ppm (0.01 g/l) uranium. If the mining and development cost, overhead, and profit amount to 200 per cent of the metallurgical treatment cost, then the metal contents in leach solution must be 750 ppm (0.75 g/l) copper, 150 ppm (0.15 g/l) molybdenum, and 30 ppm (0.03 g/l) uranium for an economic operation.

In general, it may be safe to assume that because of lower treatment and capital costs incurred in <u>in situ</u> mining, a sufficiently large deposit containing half the grade of deposits currently mined and milled could be treated economically. Thus, deposits containing 0.25 per cent copper, 0.12 per cent molybdenum, and 0.1 per cent uranium could profitably be mined with this technique. In actual practice, it may well be possible to treat even lower-grade deposits than these.

It may be emphasized, however, that utilizing <u>in situ</u> mining schemes would require a new approach on all phases of the mining operation, especially in exploration, reserve estimation and overall evaluation of leaching and metal recovery parameters.

## Elements of the Leaching Phase

Accessibility, physicochemical interaction and transport constitute the elements involved in the leaching phase of <u>in situ</u> mining. Limitations imposed on any of these factors restrict the leaching process.

Accessibility is essential because interaction between the desired constituents and the lixiviant cannot take place in the absence of contacts, which depends on exposure and penetrability. The factors to consider are locations of the metal values, their volume and shape distribution, exposure area, specific surface, particle size, porosity, capillary pressure, viscosity pressure, solubility of gases in the lixiviant and surface roughness.

Physicochemical interaction converts the desired constituents from a fixed to a mobile condition and is governed by the solubility of the solid in leach solutions and vapour pressure in gases. Knowledge of free energies of reactants and products helps to determine whether a reaction is possible. The kinetic factors involved include time, concentration, diffusivity, specific rate constants and wettability.

The first two elements by themselves do not ensure successful leaching without transport of products away from and reactants to the zone of interaction, through diffusion and convection. Diffusion is governed by concentration gradient and diffusivity, which in turn are influenced by particle size, micropore radius, temperature and molecular mass. On the other hand, convective flow concerns interparticle penetration and is restricted by pressure gradient, permeability, viscosity and surface roughness.

Broadly speaking, the factors governing leaching can be grouped as either physical or chemical. The majority of leaching studies in the past have emphasized chemical factors; it is, however, essential that we also consider the physical factors since they definitely influence the leaching process. We must develop new techniques for physical and chemical testing of ore samples and for establishing the limitations and optimum parameters for successful extraction of values from the broken ore.

## Preparation of Ores

To be processed by in situ mining techniques, ores may be:

(a) in place but requiring fragmentation prior to leaching;

(b) in place and permeable enough to permit flow of solutions through them; or

(c) previously mined or fragmented.

Waste dumps, tailings, filled stopes and caved ground fall into the last group.

Several means have been proposed for fragmenting an ore body prior to in situ leaching. In recent years, various authors[2] have proposed the use of nuclear explosives. Griswold (1967) suggested using hydrofracturing techniques to break ore for subsequent leaching; liquid explosives would be injected into ore bodies along planes of weakness and detonated at a slow rate.

Although conventional mining methods have seldom been used to prepare ore for in situ leaching, there is no reason why they could not be used. Present methods such as caving techniques and shrinkage stoping are well adapted to breaking ore for subsequent underground leaching.

---

[2] Johnson (1959), Smith and Young (1960), and Hansen and Lombard (1964).

In some instances, it may be advantageous to use a conventional mining method for selectively removing the higher-grade ores from an ore body prior to in situ leaching of the lower-grade materials. If, for example, in the mining of an ore body by a shrinkage method, a lower-grade zone adjacent to the higher-grade body were to be drilled as the stope progressed upward and the holes were loaded as the stope was drained, the low-grade material could be broken into the stope cavity and then leached.

Varying degrees of preparation may be required when the ore is already broken. When it is located underground as stope fill, very little ore preparation is required. Ores to be treated by in situ mining techniques on the surface, however, may or may not require some preparation prior to leaching. Waste dumps, slag heaps, and the like may require crushing and stacking on prepared pads, whereas fine materials like tailings may require rebedding, slime removal, or placement on an impervious pad.

Figure 1 illustrates how fine tailings might be leached by downward percolation techniques. In this system, an impermeable pad of plastic, asphalt, or concrete would prevent solution losses through ground seepage. Alternating layers of coarse rock and fine mill tailings would then be laid down over the pond area. Leaching, either concurrent with tailings deposition or following deposition, would be by downward percolation through the tailings beds of limited thickness. Percolation rates would be considerably higher through beds of limited thickness than unlimited ones. Solutions could be fed on the top bed or injected to selected beds through wells. After percolating through the tailings, the pregnant leach solutions would flow to a central recovery well. Possibly, gases could be injected to displace solutions and to react with the metal-bearing minerals.

## Practical Aspects of In Situ Leaching

In in situ mining, large volumes of ore are in contact with relatively large volumes of leaching solution over a period of time. The mechanics or pattern of solution flow varies according to the chemistry involved, the means available for solution containment and recovery, and the need to prevent groundwater contamination.

Two principal types of solution flow through a porous ore bed are

(a) downward percolation under the influence of gravity and

(b) flow within an immersed system.

In a downward percolation system, leach liquor is usually distributed over the top of a pile of ore and allowed to flow through the pile to a liquor collection system. This type of solution-ore contact has the advantage that only the floor of the ore bed need be impervious to the leach solutions. This type of flow allows some circulation of air within the ore bed, possibly an important factor in the oxidation of ore minerals. Use of downward percolation in an underground ore bed could prevent solution seepage into the groundwater strata. The chief disadvantage of this type is that incomplete solution-ore contact can result from localized impermeable zones and from channelling.

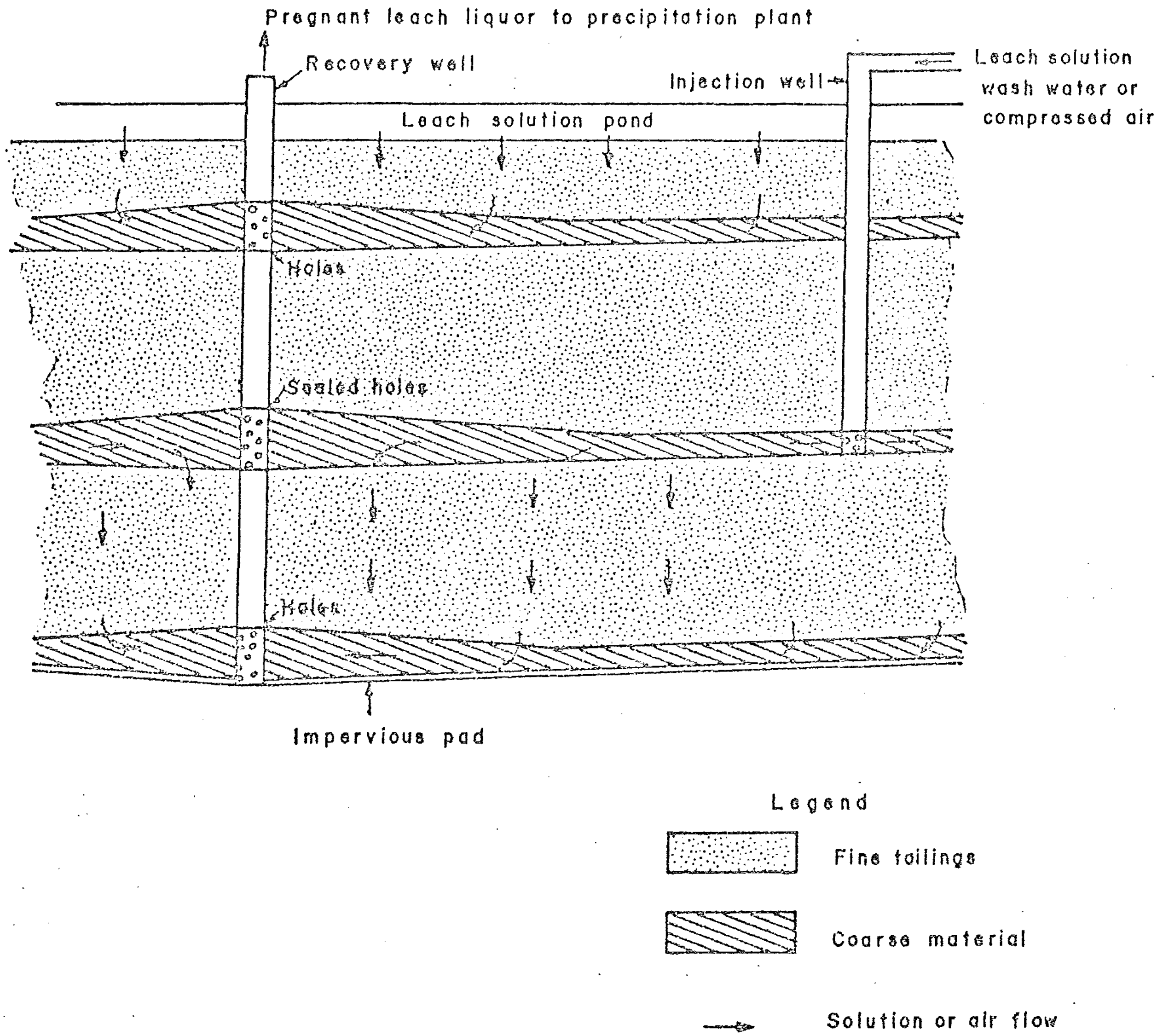


Figure I
Bedding of Fine Tailings for Subsequent In Situ Leaching.

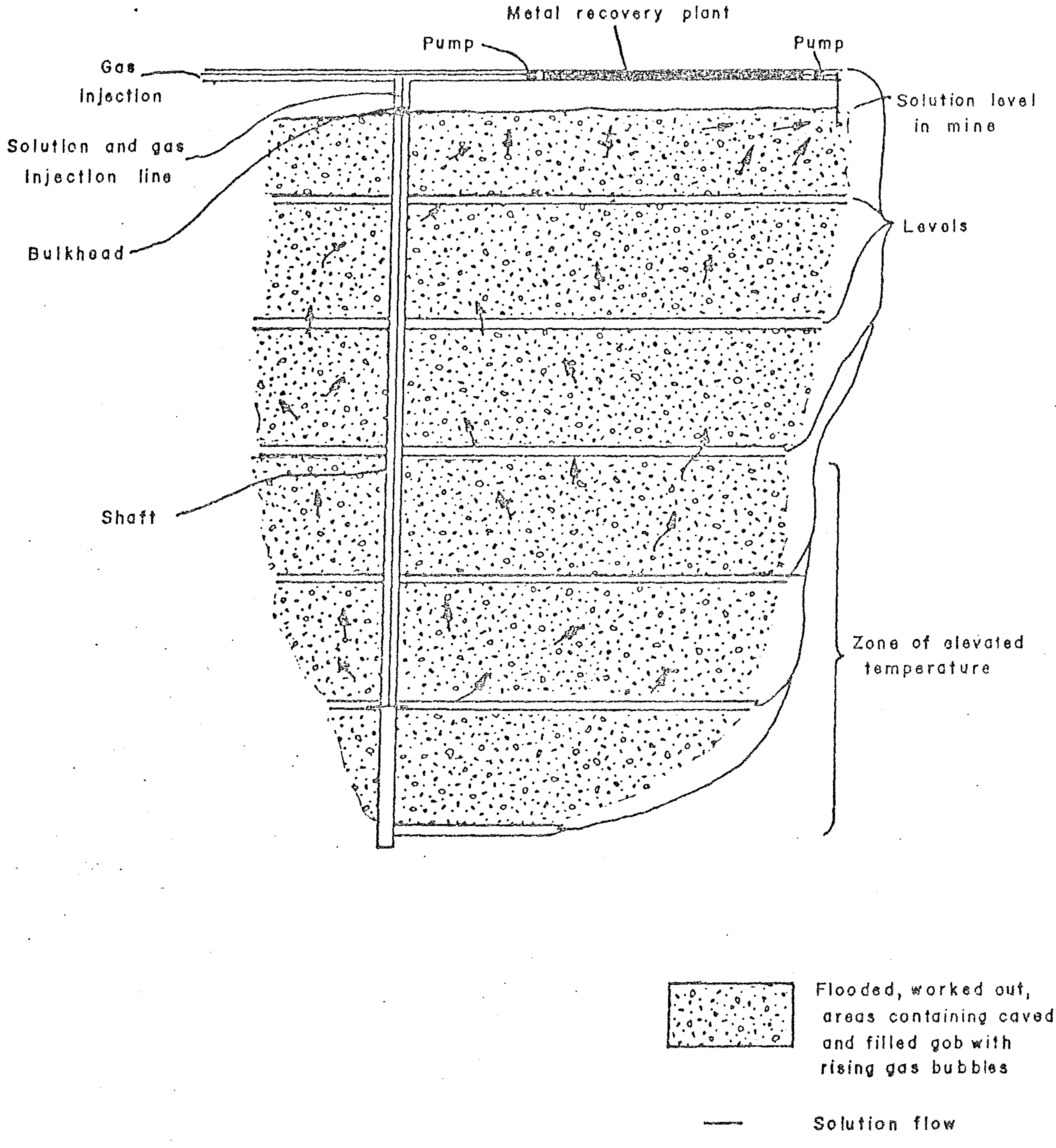


Figure 2
The Leaching of an Immersed Mine Under Autoclave Conditions

Downward percolation has been, and probably will continue to be, the principal method of leaching ore beds. The technique, which has been discussed by several authors[3], has been used in the leaching of waste dumps[4], crushed and uncrushed ores on prepared pads[5], mill tailings[6], and filled and caved workings[7]. Solution transfer may be either by convection or by diffusion. Convection may be caused by mechanical means or by differences in the density of the solution at different points within the system. This type of flow offers positive movement of solutions through an ore bed at a desired flow rate and complete contact of the solution with the bed. The main disadvantages of this type of flow are the necessity of having the ore in an impermeable container to prevent solution loss and contamination of groundwater, the restriction of natural oxidation by air circulation, the necessity of pumping solutions, and the large amount of solution involved in the leaching system.

Immersion techniques have been used in in situ mining. Copper oxide-sulphide ores have been leached in concrete tanks by upward percolation techniques at Inspiration - Robie (1928) - for many years. Utah Construction Company - Robie (1967) - recently conducted leaching tests on an unmined uranium ore body by using a series of injection, monitor, and recovery wells to force a leaching solvent through the permeable uranium ore body and to recover the pregnant leach solutions. Pirson (1959) proposed similar techniques for the in situ leaching of phosphate beds. Uranium is being recovered from mine waters at Grants, New Mexico. Copper has recently been recovered from water in the flooded Rio Tinto mine, Mountain City, Nevada.

Certain features inherent in an underground environment can assist leaching. One is the hydrostatic pressure imposed on an immersed deposit at depth and another is the natural increase in rock temperatures with depth. If gas(es) were introduced into an inflowing stream of solution entering a deposit under a hydrostatic pressure and/or the rock temperatures were above the normal temperatures at the surface, one could possibly use this system as a huge, low temperature-high pressure autoclave. Leach reaction rates can usually be increased manyfold when temperatures and gas pressures are increased.

Figure 2 illustrates how autoclave conditions might be imposed on an immersed mine. This system consists of an abandoned mine flooded with a leaching solution. A pipe introduced into the bottom of the mine via a

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[3] Seidel; Levine and Hassialis (1962); Sullivan and Bayard (1931).

[4] Irving (1921); Power (K.L.), Sawyer (1929); Argall (1963); Bogert (1961).

[5] Thompson (1948) and Nashbir (1964).

[6] Greenawalt (1912).

[7] Greenawalt (1912); Wormser (1923); Thomas (1938).

shaft would carry a gas (usually air) and/or spent leach liquor into the bottom of the system. These conditions would greatly enhance the rates of dissolution of most oxide and sulphide ore minerals. Solution circulation within the system would result from the air-lift and convection effects of rising gas bubbles, convection currents caused by the variation in temperature between the top and bottom of the system, and pumping of solution in closed circuit within the system. Metal would be recovered from the pregnant solution in its circuit from the top to the bottom of the mine.

A similar technique could leach a particular area underground; only the leaching zone would need immersion. If the area constituted a worked-out part of the mine, bulkheads placed in appropriate drifts or openings would seal it off. If it were a fragmented or permeable area underground reached by boreholes (Fig. 3), no seal would be required. Solution and/or gases would enter the leaching zone under pressure and either would be forced back to the surface by the internal pressure in the system or would require pumping. If the volume of the cavity leached were not too large, heating the influent leach solution could prove advantageous.

If the effluent stream of gas and solution discharges from the leached zone at a lower pressure than the influent stream, one could introduce a low-pressure gaseous stream at the top of the influent pipe and allow the downward-flowing liquid to compress the gases during their flow. The downward-flowing stream would act as a hydraulic compressor and as an autoclave. Possibly, a hydraulic compressor-autoclave of this type would serve to oxidize or chemically change the leaching solution prior to its use in either a percolation or immersion type of system.

Another idea that may warrant consideration is that the flow of a low-amperage current could be directed from an electrode on one side of a broken ore zone undergoing leaching to another electrode on the other side. Under given conditions of voltage and amperage, one might change the leaching solutions chemically, accelerate physicochemical reactions, and cause an increased rate of ion migration to a local area of solution recovery.

## Solution Generation and Regeneration

Inasmuch as the chemical reagents used in *in situ* mining greatly influence leaching and generally constitute a major cost item, reagent generation and regeneration is a very important part of any chemical mining process. Some reagents can be generated and regenerated by natural processes in the leaching cycle, whereas others require various chemical processes.

Practically, the only time a leach solution is generated and regenerated by natural processes is in the production of sulphuric acid and ferric sulphates from pyrites and spent ferrous sulphate leach liquors under air oxidation conditions. Although this process is used in leaching great quantities of copper from both sulphide and sulphide-oxide copper ores, very little is really known about the reaction mechanisms; reaction rates and factors influencing them, such as oxygen availability, temperature, bacterial activity, and degree or extent of solution generation or regeneration; location of chemical processes within the dump; or means of enhancing this natural process. Kennecott Copper Corp. and others have attempted in recent years to solve some of these problems.

Various industrial methods are used for producing and regenerating leaching solvents. Sulphuric acid is produced by the contact process, ferric sulphate - Thomas and Ingraham (1963) - can be made by the air oxidation of ferrous sulphate solutions in the presence of sulphuric acid, and all of the salts, acids, or bases used in leaching - NaOH, $Na_2CO_3$, $NaHCO_3$, $NaHClO_3$ and NaCN - are prepared commercially and can be regenerated from spent leach liquors. Unfortunately, the use and regeneration of these reagents are uneconomical in many instances.

Johnson (1965) described a sulphuric acid-ferric sulphate solution generation and regeneration process for chemical mining. A specially designed, air-agitated autoclave generates sulphuric acid and ferric sulphate leach solutions from pyrite and concurrently oxidizes and hydrolyzes a spent ferrous sulphate leach liquor to sulphuric acid and ferric sulphate.

The key to low-cost lixiviants for in situ mining purposes probably lies in the efficient use of either low-cost materials found in the ores to be leached, such as pyrite, or raw or low-cost prepared materials readily available, like pyrite, sodium chloride, trona, or liquid ammonia. Much research work is required to determine how these materials can be used efficiently.

The authors' current studies will determine the effectiveness of common leaching reagents, such as acids, acid-iron salts, and salts like NaCl, $Na_2CO_3$, $NaHCO_3$, in leaching ores of copper, lead, zinc, nickel, silver, gold, uranium, and molybdenum under chemical mining conditions. These tests include percolation leach conditions, static leach conditions, and high gas pressures--low temperature (below $100^\circ$ C) conditions. A large column-type autoclave may be used soon to leach coarse materials under each of these conditions.

## The Recovery of Metals from Leach Liquors

The last phase of any hydrometallurgical process, including in situ mining, is the recovery of metals from leach liquors. Conventional purification of a metal-containing solution followed by recovery of metals or compounds from the solution by either chemical or electrolytic precipitation is employed to obtain the marketable product. This recovery technique is adequately covered in the literature and its effectiveness is clearly demonstrated in several successful plant practices.

In connexion with in situ mining applications, however, the recovery phase poses certain technical problems that may influence the overall effectiveness of the process. One such difficulty concerns treating a large volume of very dilute metal-bearing solutions that may contain more than one valuable metal. Unlike copper, not all metals easily precipitate on scrap iron. Recirculation of the leach solution may be required to build up the metal content, with bleeding off of a small part of the concentrated leach stream for metal recovery.

Newer techniques of ion exchange, solvent extraction, and charcoal sorption may effectively concentrate dilute leach liquors. These procedures have proved very effective for processing large volumes of leach solutions containing more than one valuable metal. The Climax process - Johnson (1966) - for recovering oxide molybdenum values by charcoal sorption and the New Mexico Bureau of Mines procedure - Reynolds, Long and Bhappu (1966) - recently developed for recovery and selective separation of molybdenum, tungsten, and rhenium by sorption processes are typical of techniques that metal recovery systems will increasingly employ.

Since the crux of the <u>in situ</u> mining process is the particular lixiviant in leach solution, any recovery phase that regenerates the solvent or provides an essential component of the leach solution would be the preferred procedure. Also, since <u>in situ</u> mining depends on the continuous circulation of the leach solution at its peak volume, it is imperative that the retention time in the metal recovery step be as short as possible. This requirement necessitates a metal recovery procedure that is relatively quick and capable of handling large volumes effectively.

## Discussion

This paper outlines a novel approach for extracting the metal values contained in low-grade deposits, worked-out mines, dumps, and tailing piles. With the ever increasing demand for today's metals, the necessity of treating complex and low-grade ores, increasing operational costs, and the public awareness of environmental pollution factors, future metal production will inevitably employ chemical mining on an increasing scale. The scope of this mining method encompasses interdisciplinary science and technology, requiring application of the principles of basic sciences, economics, mining, metallurgy, hydrology, and allied disciplines. Although some technological information is available from several dump leaching and a few in place leaching operations for copper and uranium, not enough is known about this technique.

Some researchers have tried to develop newer metal recovery techniques, but a much more concentrated endeavour would take fullest advantage of the <u>in situ</u> mining process for extracting metal values from marginal, low-grade deposits, dumps, and tailings.

The dearth of information illustrates the need for broad and imaginative research. Inasmuch as in place leaching occurs on coarse materials on a large scale over a period of time and under conditions considerably different from most other leaching practices, study of the process from both basic research and practical application standpoints is of great importance.

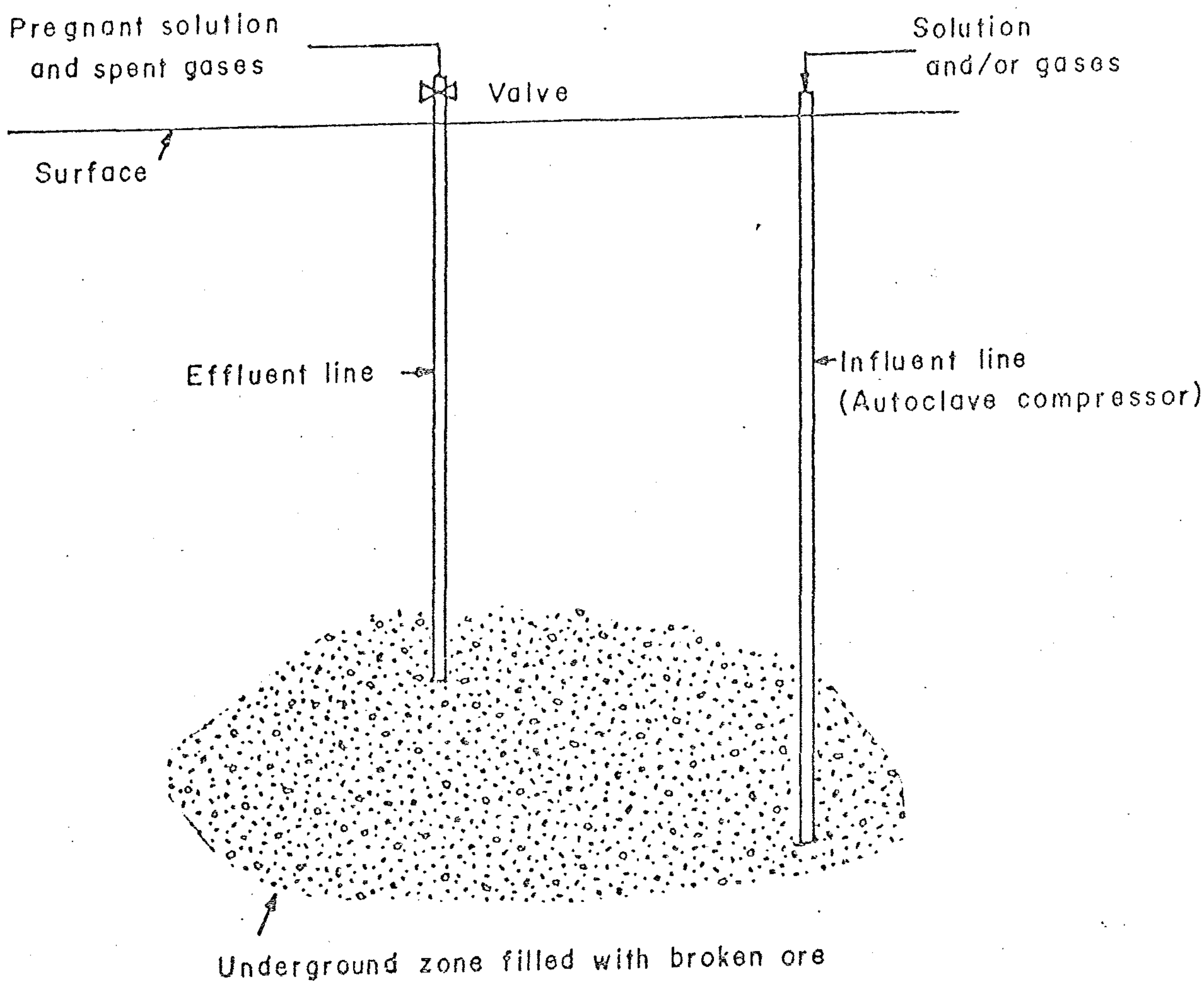


Figure 3
The Leaching of a Local Underground Area Under Autoclave Conditions

## PART II

## BLOCK CAVE - IN PLACE LEACHING

To introduce the concept of Block Cave-In Place Leaching, a mixed oxide-sulphide (chalcocite) copper deposit has been selected. It comprises 102,000,000 tons, assaying 0.50 per cent total copper (0.25 per cent oxide and 0.25 per cent sulphide). The ore body averages 425 feet in thickness with 2,000 feet of overburden. This hypothetical ore body can be used for the study of deposits of different types, shapes and contained minerals.

Porphyry ore bodies are usually highly altered and fractured with much of the mineralization occurring in the seams or fractures. This type of ore body caves readily and breaks into small pieces (5% + 10", 25% + 4" and 70% - 4") which would be ideal for leaching.

An ore body of this type must be proven by drilling. The normal block cave mine, before development, is usually drilled on 200 foot centres. However, due to the extreme depth of the hypothetical ore body, a surface drilling on 600 foot centres would probably justify shaft sinking and additional drilling from underground. All such holes should be saved for future use.

### Mining

From studies relating to block cave mining and underground nuclear explosion (chimneys), it is apparent that 10 per cent to 15 per cent of the ore column must be removed to break the column. This will have to be tested and monitored underground. It is necessary to get the typical <u>block cave arch failure to the top of the ore in order to have the entire ore column</u> broken for leaching. The exploratory drill holes can be used for monitoring the progress of the cave.

Using the drill hole data, a contour map of the bottom and top of the ore body can be drawn. With this information, a block cave-in place leach plan can be formulated. The most suitable and economical plan would be lateral transfer by scrapers on the control level with belt conveyor haulage. This system allows the undercut to follow the slope of the ore body. For this study, a 140 foot wide panel system, with 20 foot pillars between panels was selected. Slusher drifts on 30 foot centres and draw raises on 17 1/2 foot centres along the slusher drifts were selected. With this system a good draw control is obtained and additional pillars can easily be left. Blocks of 150' x 140' comprise a unit within the panel. Figures 1, 2 and 3 give details concerning the operation.

From a block cave layout a $12.54 per sq. ft. mining cost was calculated (Appendix 1).

### Leaching

Test leaching of all the exploratory drill holes, cores and leaching tests of underground work should be made. From observations of results from numerous leaching operations, a 70% recovery of 0.50% copper ore is considered conservative. This is the point where the grade of the pregnant solution drops below 1 g/l. This average recovery is applied to the entire ore body and gives 7.0 lbs. copper recovery per ton of ore. With 425 ft. of ore at 7 lbs. per ton this gives

$$\frac{425}{12.5} = 34 \text{ tons per sq. ft.} \tag{2}$$

$$34 \times 7 = 238 \text{ lbs. Cu/sq. ft.} \tag{3}$$

Operating Cost

(a) Mining: $\frac{12.54}{238} = 5.3$ cents per lb. Cu (4)

(b) Underground leaching:

| | | |
|---|---|---|
| Acid at $20/ton and net consumption of 2 lbs/lb Cu (3.5 lbs minus 1.5 lbs recovered from electrowinning) | - 2.0¢ | |
| Solution distribution | - 2.0¢ | |
| Underground pumping | - 3.0¢ | = 7.0¢ per lb Cu |

(c) Surface leaching = 1.0¢ per lb Cu

(d) SX[8] electrowinning = 8.0¢ per lb Cu

(e) General = 5.0¢ per lb Cu

Total 26.3¢ per lb Cu

[8] SX = solvent extraction.

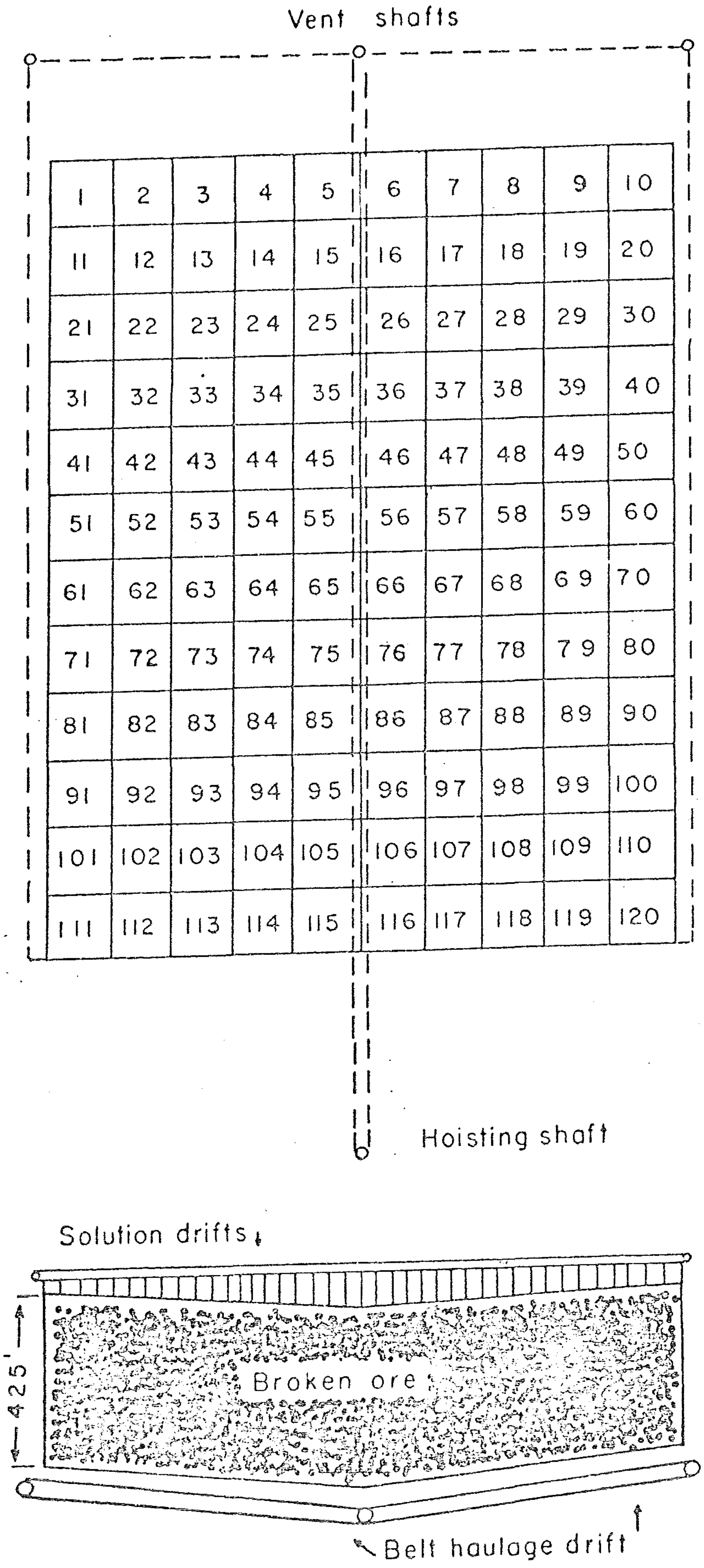


Figure 1 A Typical Block Cave System

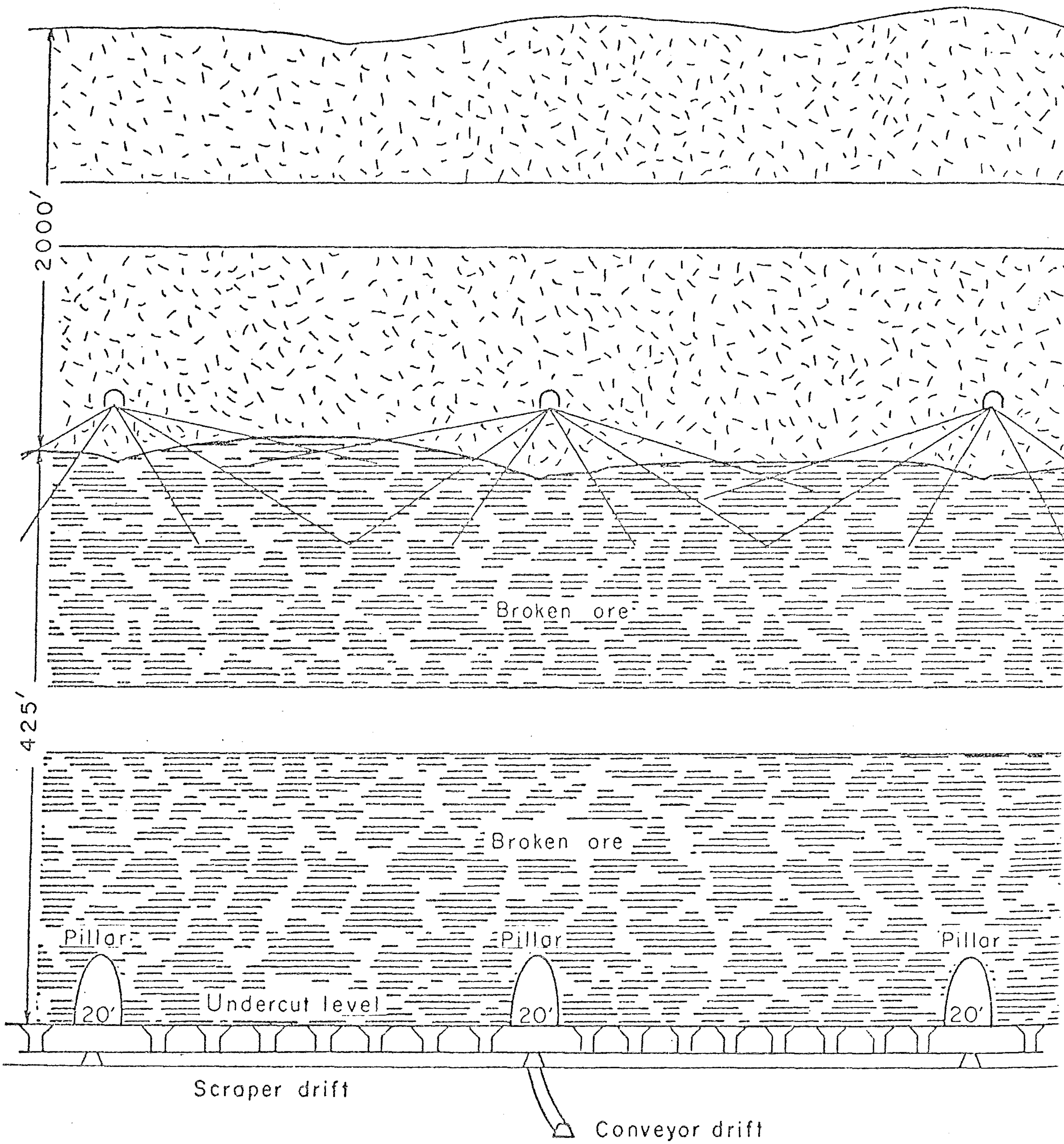


Figure 2 A Cross Section of the Block Cave System

With a sale price of 50¢ per lb. Cu this gives an indicated operating margin of 23.7¢ per lb. Cu.

## Capital Cost

The capital cost estimated to bring the mine into production is estimated at $30,000,000 in Appendix 2. The $1,154,000 cost listed therein for developing four blocks is not properly capital investment but the money will be needed prior to production. Block preparation should go into the deferred development account and is actually part of the mining cost.

## Plant

A 4,000 g.p.m. plant was selected for this study. With an average of 2 g/l copper this gives a production of 96,000 lbs. Cu/day or 35,000,000 lbs. Cu per year.

$$\frac{96{,}000 \text{ lbs. Cu/day}}{238 \text{ lbs. Cu/sq. ft.}} = 404 \text{ sq. ft. of development required/day} \quad (6)$$

$$\frac{714{,}000{,}000 \text{ lbs. Cu}}{35{,}000{,}000 \text{ lbs. Cu/yr}} = 20.4 \text{ years life} \quad (7)$$

This indicates a mining crew of 120 men per day on a five-day week basis, or 80 man shifts per leaching day (30 day month). This gives 5 sq. ft./man/day. With 425 ft. of ore to break and 10% to hoist gives 17 tons/man shift or 2,000 tons per day, using a five-day week.

A 4,000 tons per day hoisting system should be installed to take care of the initial development and any possible expansion.

## Operations

A completely developed block must be left between the leaching and development blocks to prevent any breakthrough of solution into the working areas. On the block lay out plan (fig. 1) blocks 2, 11, and 12 would have to be completely developed before leaching of block no. 1 started. Block 3 and 13 before leaching of no. 2, blocks 4 and 14 before leaching of no. 3, etc.

A system of water doors will be needed to store solution in case of power failures. Or a stand-by power system could be used.

Four hundred g.p.m. of new water will be needed to maintain the 4,000 g.p.m. flow.

## Ecology

This process should have environmental advantages because there would be no smelting of concentrates nor disposal of tailings, and a minimum disturbance of surface is envisioned. The solution will all be re-cycled with no seepage loss.

The heap leach dumps would be placed in terraces which would be re-planted.

## Basis for Conclusion and General Remarks

### Mining

One of the problems in block caving is maintaining the openings while drawing the large tonnages of ore. The mining method proposed in this report surmounts this problem, as only enough ore is drawn to break the column. No repair work will be required on the extraction levels. The flow of solution can be maintained by a system of pipes (French drain) in the caved areas.

Guarding against caving at the surface is a serious problem. The mining must be watched very carefully so as not to overdraw, but the draw must break the ore column in order to successfully leach the ore. Due to compaction there may be a small sway in the surface (which can be detected by surveying) with no damage to building or structures. There could, however, possibly be a small crack at the 70° angle circle outside the caved line, which would damage pipe lines, railroads, etc. A fund should therefore be set aside for possible damage. A pillar study should be made as mining progresses.

### Leaching

The following are characteristics of leaching:

1. The grade of the pregnant solution varies with the amount of copper in the ore. As leaching progresses, the assay of the pregnant solutions drops.

2. The rate of recovery varies with the height of the column to be leached and the size of the material.

3. The percent recovery varies with the grade of the ore.

With computer programming the proposed mining system is flexible enough to give a reasonably steady production.

Several block cave mines have successfully leached the copper left after mining (dilution and pillars). Notable examples are the Ray Mines (Kennecott), Inspiration and Miami Copper. Miami Copper block caved in order to break the ore to leach in place.

## Construction of Leaching Rate Working Curves

In order to optimize dump or heap leaching operations under a given set of operating conditions such as, number of dumps or heaps, their heights, time required for preparing each dump or heap, rate of solution

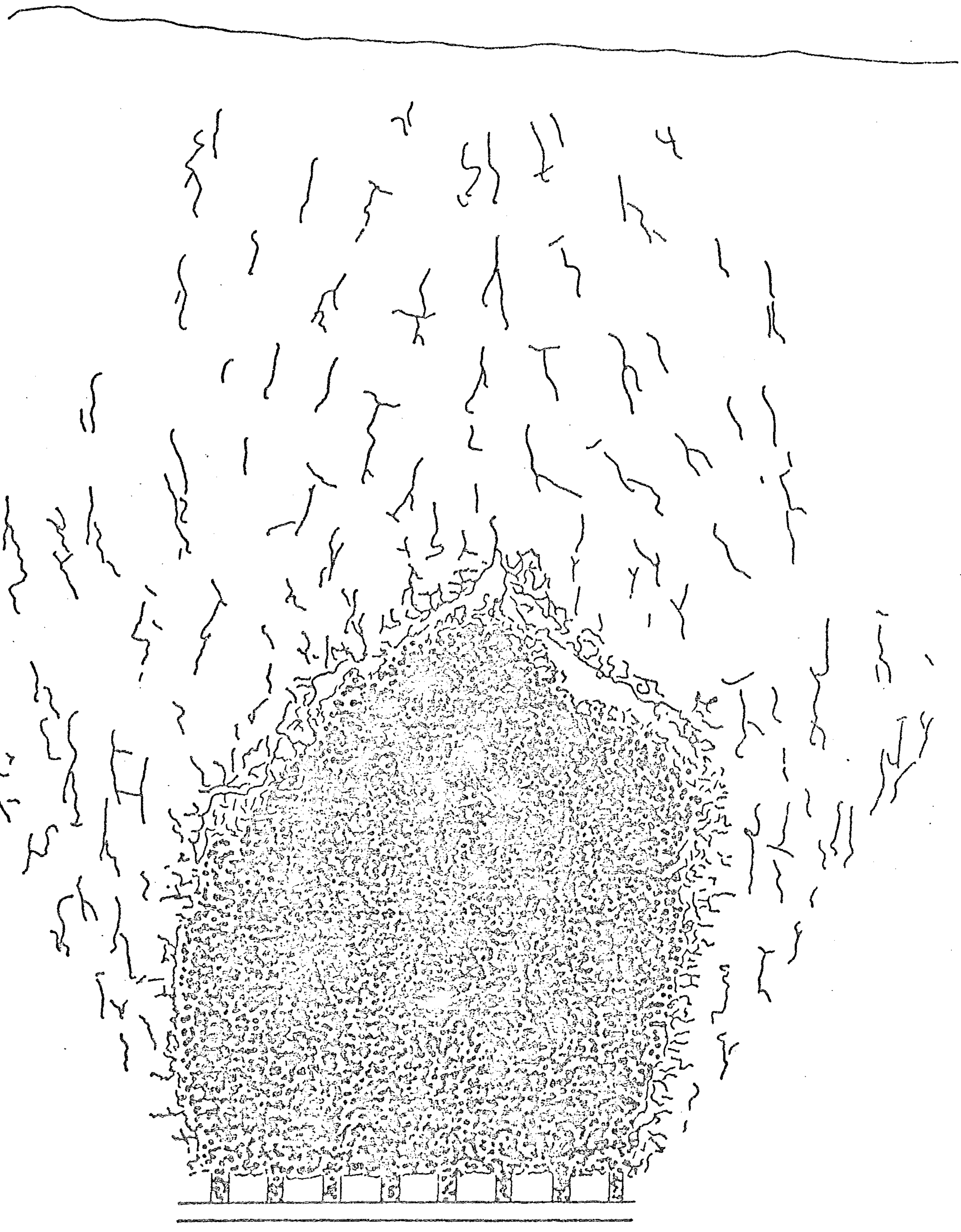

Figure 3    A Section Through Caving Block

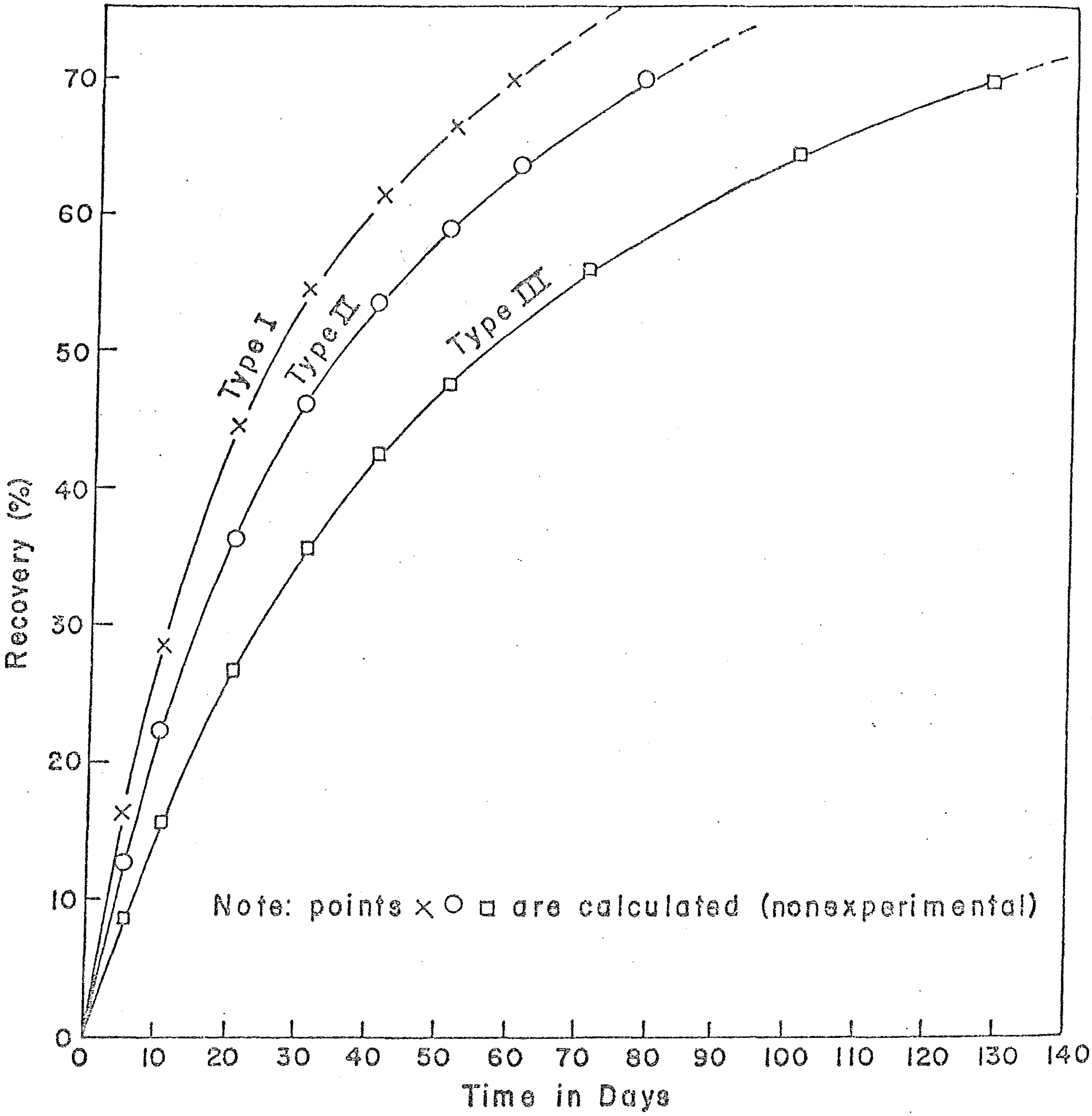


Figure 4 Recovery as a Function of Time, Heap Leaching Situation (Ore Size = 2½"; Heap Height = 50')

flow, and holding the metal concentration in leach solution at a given value (gpl), it is imperative to have a fair idea of the extent of metal recovery at a given time for a particular dump or heap. In order to achieve this goal, leaching rate working curves must be constructed as follows:

1. Recovery as a function of time for different types of ores (from the same deposit) encountered during the leaching operation under dump or heap leaching situations.

Such working curves for the different types of ores under consideration may be constructed by establishing the recovery at any one time for each type of ore by column leaching tests and by utilizing the following general mathematical expression (Pade's approximation):

$$\text{Recovery} = \frac{at}{(b + t)} \tag{8}$$

where,

a = maximum recovery possible,

t = time for obtaining specified recovery, and

b = a constant depending on the type of ore under consideration and depth of the dump or heap and numerically equal to the time required to obtain 50% of the maximum recovery.

It should be noted that because of the empirical nature of these working curves, they are more reliable for recovery values above 30%.

Experimental Data

In order to illustrate the construction of working curves, let us examine the experimental data (Table 1) obtained for a typical oxide copper ore containing malachite, azurite, and chrysocolla as the major oxide copper minerals along with lesser amounts of sulfide copper mineralization (the sulfide copper content being less than 25% of the total copper content).

TABLE 1

LEACHING RATES TO OBTAIN 70% RECOVERY

(day or months per given depth)

| Type Ore | Size: Depth: | Minus 2 1/2 inch 4-feet | Minus 2 1/2 inch 50-feet | Minus 12-inch 4-feet | Minus 12-inch 200-feet |
|---|---|---|---|---|---|
| Type I | (+90% oxide) | 4.6 days | 58 days | 13.0 days | 21 months (1.8 years) |
| Type II | (+75% oxide) | 6.2 days | 78 days | 17.5 days | 29 months (2.4 years) |
| Type III | (+50% oxide) | 10.2 days | 128 days | 28.6 days | 48 months (4.0 years) |

These results are obtained from actual laboratory tests using 4-foot columns of appropriate diameter. Let us assume that experimental data for minus 2 1/2 inch material at 50-feet depth represents a heap leaching situation while data for minus 12-inch material signifies a dump leaching operation. Also, let us assume that the leaching time is directly proportional to depth.

Working Curves

Since the times required for 70 per cent recovery for minus 2 1/2-inch material at 50-feet depth and minus 12-inch ore at 200-feet depth are known, we can now obtain the values for "b" as follows:

| Type Ore | Values of "b" 50-feet depth | 200-feet depth |
|---|---|---|
| Type I | 24.9 | 9.0 |
| Type II | 34.4 | 12.4 |
| Type III | 55.0 | 20.3 |

Using the above values of "b" and for given time "t" it is now possible to construct the curves for the three types of ores for the heap leaching situation (Fig. 4) and the dump leaching operation (Fig. 5).

Using the above leaching rate working curves, it is now possible to develop a tentative production schedule for predicting the length of time each dump or heap would be leached, the grade of the pregnant solution from each heap or dump (since the quantity of leach solution is fixed the concentration of copper in the leach solution at a known time is also fixed) and the daily copper production throughout the life of the mine. Such information is invaluable in predicting the net cash flow as well as the return on the investment (ROI), the criteria on which the modern mining business is based.

It is recognized that, in the construction of the leaching rate working curves and in the consequent development of the production schedule, the data and the techniques used are at best engineering approximations. Consequently, the results are not absolute, but are nevertheless, thought to be of value in predicting trends and possible problems in trying to achieve the optimum production rate as well as operation conditions.

Efforts are currently under way to correlate the usefulness of such working curves with actual plant operations. Moreover, additional laboratory experiments are being conducted to develop techniques for scale up from laboratory data to actual plant practice. Scale up rules are being developed which take into account variations in: (a) size distribution of the material being leached, (b) changes in the rate controlling step during leaching operation, and (c) fluctuations in solution flow rate through the material being leached. It is hoped that the results of these studies may enable predictions of leach solution grades and recoveries

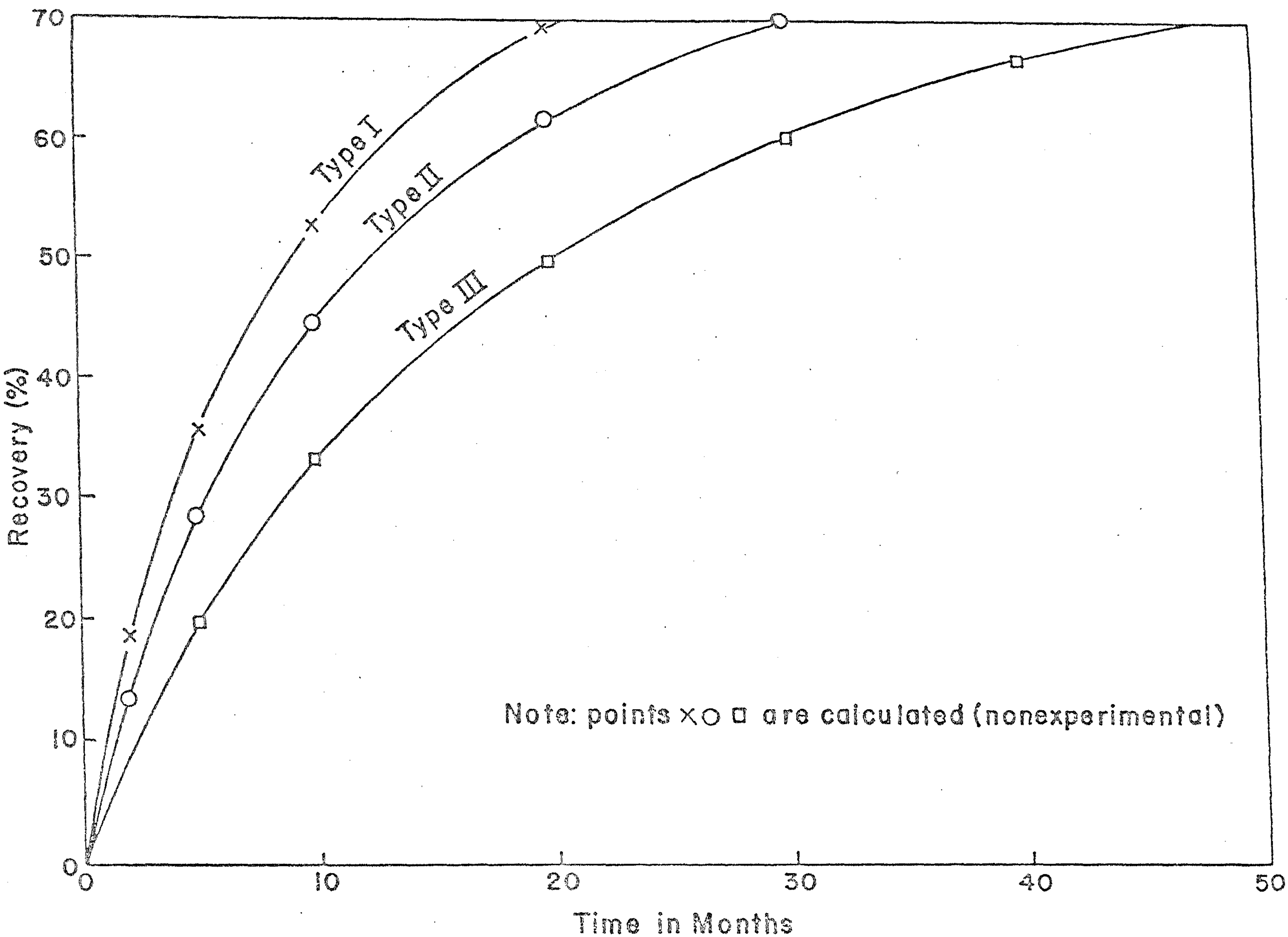


Figure 5 Recovery as a Function of Time, Dump Leaching Situation
(Ore Size = ~12"; Dump Height = 200')

in a large dump or heap to be made on the basis of three simple laboratory tests.

## Computer Programme for Estimating Production

A computer algorithm may be developed to predict copper production and acid consumption under different options of mine operation, which should be based on drill core data, metallurgical test data, and a mine plan. In order to ensure maximum usefulness from such a programme, the following factors should be considered in its development.

First, each stope should be assigned the mineralogy, metallurgy, and physical characteristics of the four or more surrounding drill holes weighted by the distance from each of the drill holes.

Second, detailed laboratory testing should be conducted to determine the rate at which acid is consumed by the rock (as distinct from the copper minerals) and the total amount of acid one ton of rock will consume. This should be done for all samples or until it is proven that both rate and total consumption are independent of location. This data should then be incorporated into a form compatible with the model of acid consumption.

Third; the decision to add 0, 1, or 2 new stopes to the system should be based on results of calculations for each of the three parameters (mineralogy, metallurgy and physical characteristics) rather than on the results of a past operational period. This would eliminate some over-shooting and undershooting taking place in a production schedule.

The computer programme has two major areas of usefulness. The first is to predict production levels from different operating options. The second is in conducting a sensitivity analysis on the variables of ore grade, ore tonnage, copper recovery, acid consumption and average solution grade. The sensitivity analysis would be valuable in suggesting areas in which the data should be firmed up.

Appendix 1

PRESENT DAY UNIT MINING COST IN UNITED STATES

150 Ft. x 140 Ft. Block
24,000 Sq. Ft. (includes pillar)

| | |
|---|---|
| 150 ft. Conveyor Drift at $135 | $ 22,250 |
| 5 Chutes at $1,000 | 5,000 |
| 100 ft. Conveyor Raise at $30 | 3,000 |
| 800 ft. Slusher Drift at $110 | 88,000 |
| 150 ft. Fringe Drift at $110 | 16,500 |
| 1,600 ft. Draw Raises at $10 | 16,000 |
| Undercutting 21,000 sq. ft. at $3/sq. ft. | 63,000 |
| Sub-Total<br>$8.40/sq. ft. | $213,750 |
| To Introduce Solution | |
| 150 ft. Distribution Drift at $200 | $ 30,000 |
| 1,600 ft. Drill Holes at $5 | 8,000 |
| Sub-Total<br>$1.58/sq. ft. | $ 38,000 |
| TOTAL | $251,750 |

$$\frac{\$251,750}{24,000} = \$10.50 \text{ per sq. ft.} \tag{9}$$

The average head to be broken is 425 ft. with 10% of the column to be mined, this gives 42.5 ft. to be mined 15 ft. of this tonnage will be mined in the undercutting. This gives 42.5 ft. or 3.4 tons per sq. ft. to draw to break the ore column.

| Per Ton | | |
|---|---|---|
| Drawing | = | 30¢ |
| Conveying | = | 5¢ |
| Hoisting | = | 15¢ |
| Dumping | = | 10¢ |
| TOTAL | | 60¢ |

$$3.4 \times 60¢ = \$2.04 \text{ per sq. ft.} \tag{10}$$

TOTAL $2.04 + $10.50 = $12.54/sq. ft.

Appendix 2

CAPITAL COST

| | |
|---|---|
| Hoisting shaft 16' dia 2425' at 800 | $ 1,940,000 |
| Design | 50,000 |
| Exploration and ventilation station | 50,000 |
| Haulage station | 90,000 |
| Pocket | 150,000 |
| Contingency at 15% | 342,000 |
| Sub-Total | $ 2,622,000 |
| Three 40" dia ventilation holes 2425 at 150 | $ 1,091,000 |
| Pump station | 300,000 |
| Electrical sub-station | 100,000 |
| Explorator entry and ventilation drift 2650 ft. at 200 | 530,000 |
| Ventilation drift 5,000 ft. at 150 | 750,000 |
| Haulage drift 2,600 ft. at 200 | 520,000 |
| Supply elevator 500 ft. at 200 | 100,000 |
| Transfer raises 2,000 ft. at 100 | 200,000 |
| Contingency at 15% | 539,000 |
| Sub-Total | $ 4,130,000 |
| Hoist house, change room, office and shops | $ 500,000 |
| Hoist | 900,000 |
| Head frame to bins | 200,000 |
| Cage and skips | 80,000 |
| Mine equipment | 500,000 |
| Electrical equipment, transformers, etc. | 400,000 |
| Pumps | 300,000 |
| Pipe | 250,000 |
| Belt conveyors 10,000 ft. at 100 | 1,000,000 |
| Elevator | 250,000 |
| Water development | 1,000,000 |
| 4 Blocks developed | 1,154,000 |
| Sub-Total | $ 6,534,000 |
| Surface plant, SX, etc. | $14,000,000 |
| Working Capital | 2,000,000 |
| TOTAL | $29,286,000 |

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