A SOCIOECONOMIC STUDY OF COPPER LEACHING AT SANTA RITA

by

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ABSTRACT

The recovery of copper by leaching of the waste dumps of the Chino Mines Division, Kennecott Copper Corporation, Santa Rita and Hurley, New Mexico, is described. Suggestions for improvements are made.

Assumptions on which the present operation is based are examined. Possible improvements in the assumptions are suggested; and areas of needed research are indicated.

A plan of operation based on shifting emphasis from milling to precipitation recovery of copper is described. The cost saving associated with such a modified plan of operation is estimated.

Probable socioeconomic trends requiring an eventual shift to copper recovery based wholly on precipitation methods are described. The probable long-range technical development of recovery methods is suggested.

ACKNOWLEDGEMENTS

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Special thanks go to the Kennecott Copper Corporation and Chino Mines Division for permission to conduct this study. The courtesy and cooperation of Mr. E.A. Slover, General Manager of Chino Mines Division, deserve special note. His generosity in allowing access to the operation made the study possible; his informative discussion added materially to the development of the study. Thanks go to Messrs. Frank Woodruff, T.J. Montgomery, C.H. Van Buskirk, William Baltosser, Rupert Spivey, Paul Lemke, and Bill Wallace, members of the staff of Chino Mines Division, for their aid and cooperation during the conduction of this study.

To my wife Caroleann goes my heartfelt appreciation for her words of encouragement and hours of typing.

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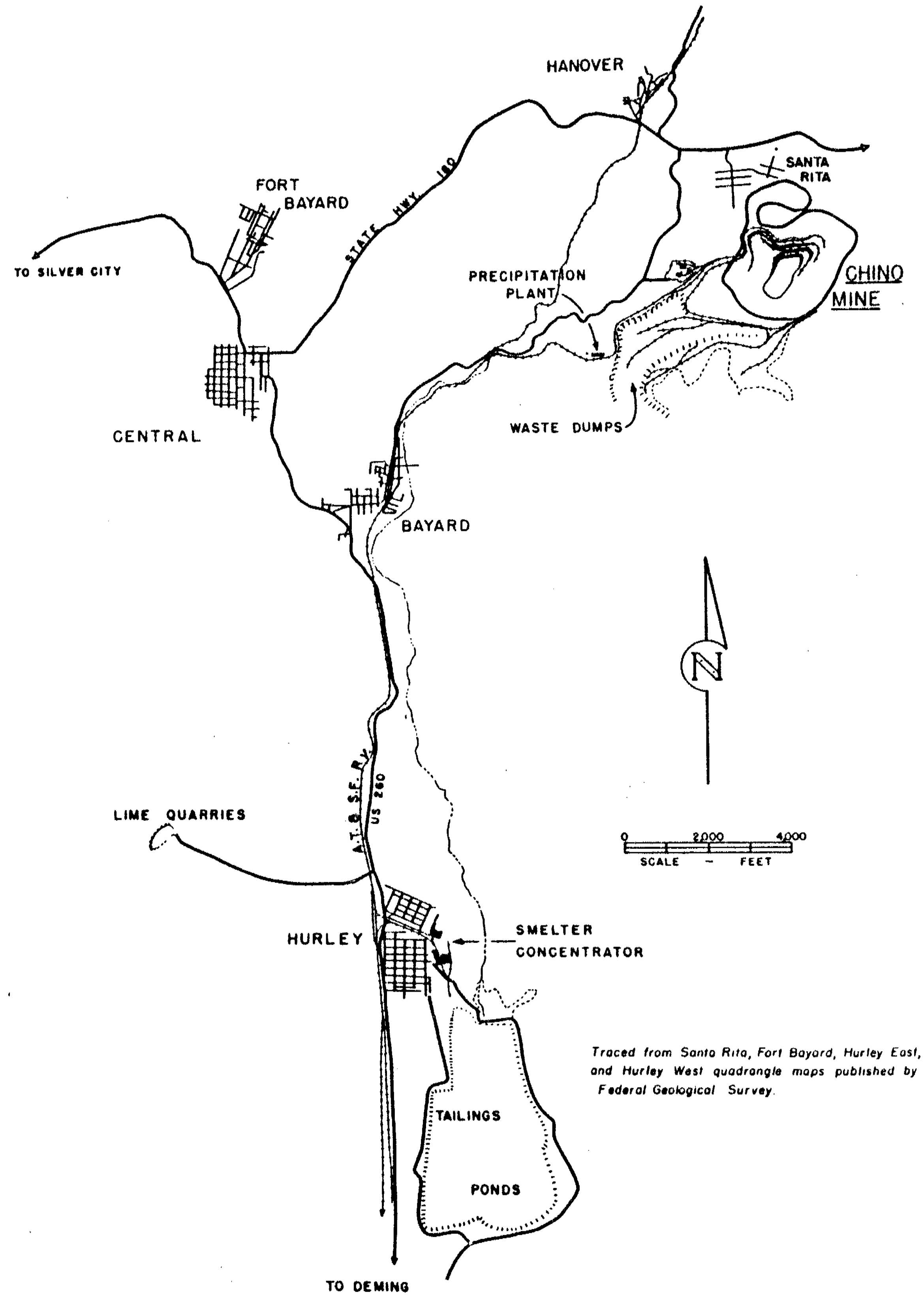
INTRODUCTION

This paper presents the results of a socioeconomic study of leaching of copper from dumps at the Chino Mines Division, Kennecott Copper Corporation, Santa Rita and Hurley, Grant County, New Mexico. The study was conducted in partial fulfillment of the requirements for the degree of Master of Science in Geology at the New Mexico Institute of Mining and Technology, Socorro, New Mexico.

"The Chino mine is at Santa Rita, in the Central Mining Area of Grant County, New Mexico; the concentrator and smelter are at Hurley, approximately 9 miles from the mine." (Hardwick, 1958, page 1) The accompanying map, from Hardwick (1958), shows the relative locations of the mine, dump, precipitation plant, mill, and smelter.

"The Chino Mines Division of Kennecott Copper Corporation is a completely integrated unit. Copper ore is mined, concentrated, and smelted. Fire-refined copper is produced and sold on the market. Molybdenum is recovered as a byproduct and sold" (Hardwick, 1958, page 1).



Location Map, Chino Mines Division, Kennecott Copper Corp.

Chapter I

HISTORY OF THE SANTA RITA MINES

Cabeza de Vaca, in the journal of his journey through the southwest, mentions copper articles in the possession of Indians (1536)(Rickard, 1932). Castaneda, the Recorder of the Coronado expedition, also mentions copper ornaments worn by the Indians (1541). Indian reports of the source of this copper indicate that it came either from Cananea, Sonora, Mexico; or from Santa Rita, New Mexico, some 90 miles to the north. Although this indicates a possible early knowledge of the mines, they were unknown to the Spanish and Mexicans until about 1800. The mines were first worked by Europeans in 1804, though Indians had recovered native copper from the area in the late 1700's. This European mining activity is preceded in New Mexico only by the Spanish-era workings of Chalchihuitl[a] and Mina del Tiro[b] in the Cerrillos district (circa 1680)(Jones, 1904). The mining at Santa Rita was the first major recovery of copper in what is now the United States; the Michigan copper deposits, though known as early as 1653, were not opened for major production until 1844 (Rickard, 1932).

An Apache Indian is credited with the discovery of the Santa Rita mines about the year 1798 (Jones, 1915). In 1800 an Apache showed the outcroppings of native copper to Lieutenant Colonel Josa Manuel Carrasco of the Spanish army in

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a- the Indian name for turquoise
b- also called Mina Del [illegible]

return for an act of kindness (Rickard, 1932). Colonel Carrasco did not attempt to work the property (Jones, 1904); but interested Don Manuel Francisco Elguea[a], a merchant of Chihuahua, in its business possibilities (Jones, 1904). Elguea, a subdelegate to the Spanish court, obtained a concession to work the deposit and a contract to supply copper for coinage to the government of New Spain (Spencer and Paige, 1935). In 1804 he purchased Carrasco's interest (Rickard, 1932) and began production[b] (Jones, 1904).

After Elguea's death in 1809, his widow retained ownership (Jones, 1904) but placed the mine under the management of Francisco Pablo de Lagera[c] (Rickard, 1932). The superintendent at that time was Juan Onis[d] (Jones, 1904).

Several versions of the history between 1809 and 1826 are available. Rickard (1932) states that Sylvester Pattie leased the mine from the Elguea family in 1825. A Frenchman named Courcier is reported to have succeeded Pattie and to have worked the mines for seven years. Rickard (1932) contradicts these statements by noting that Robert McKnight succeeded Courcier in 1826.

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a- also referred to as Deelguea (Spencer and Paige)
b- Lt. Zebulon Pike mentioned these mines in his report on the expedition of 1807, although he did not visit it. He is quoted as stating, "It is worked and produces 20,000 mule loads of copper annually" (Jones, 1904). One mule load of copper is about 300 pounds (Rickard, 1932); this amounts to about six million pounds of copper per year.
c- In 1825 the manager was Don Juan Ortiz (This Is Chino); this may be the Ortiz who later held the Ortiz Mine Grant near Santa Fe.
d- Also called Ori (Jones, 1904).

Jones (1904) states that "two Pattie brothers" leased the mine "for a number of years, paying $1000 per annum" (page 37); and that in 1827 the elder brother attempted to purchase the mine but was foiled by a swindle. Another version states that two trappers, Sylvester Pattie and his son James, took over operation of the mines in 1825. "Sylvester Pattie operated the mines with some success and was on the verge of buying the property when a trusted employee absconded with $30,000 of his working capital. Bankrupt, Pattie left in disgust" (This Is Chino).

Jones (1904) gives yet another version. He states that in 1822 Senora Elguea sold the mines. However, the purchaser was exiled in 1826; and the property passed into the hands of Robert McKnight.

In any event Robert McKnight gained ownership of the mines in 1826; and worked the mines until 1834[a] (Jones,1904), when Indian hostility forced their abandonment (Rickard, 1932). In 1840 Leonardo Siaqueiro took control and worked the mines until the late 1850's. (Rickard, 1932). In 1860 the property was held by Sweet and LeCoste (Spencer and Paige, 1935). Confederate troops under General Sibley invaded the area in 1862, destroyed the mine works, and confiscated a large amount of copper (Spencer and Paige, 1935). After the war Sweet and LaCoste reopened the mines and continued operation until 1871, when litigation was initiated by the

a - This date is also given as 1838 (This Is Chino)

Elguea heirs who claimed the property under the terms of the old Spanish grant (Rickard, 1932). In 1873 title was vested in the Elguea heirs by the Comissioner of Patents and the Secretary of the Interior. (Rickard, 1932). Matthew B. Hayes[a], a smelterman from Colorado, obtained title from the heirs; and located and patented the mines under American law (Spencer and Paige, 1935).

Hayes held the property until 1880, when it was purchased by J. Parker Whitney (Rickard, 1932). Under Whitney's ownership the property was developed and worked by lessees (Spencer and Paige, 1935). The Hearst estate leased the property during 1897-1899 to provide sulphide ores for the smelter at Silver City (Graton, 1910). The Santa Rita Mining Company, controlled by the Amalgamated Copper Company[b] (Rickard, 1932), purchased the property in 1899 (Graton, 1910). In that year the Santa Rita branch of the Silver City and Northern Railroad[c] was built into the area (Spencer and Paige, 1935).

An examination of the property was made by the Santa Rita Mining Company in 1906. The hoped-for bodies of high grade ore were not discovered (Spencer and Paige, 1935), and the property was sold to the Chino Copper Company in 1909 (Rickard, 1932). The Chino Copper Company, associated

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a- Also called Martin B. Hayes (<u>This Is Chino</u>)
b- now the Anaconda Company (Rickard, 1932)
c- now part of the Atchinson, Topeka, and Santa Fe Railway Company (Spencer and Paige, 1935)

with the Guggenheim interests, pursued a vigorous program of churn drilling, and in 1910 began shipping operations from the first open pit mining of the deposit. The company built a 5,000 t.p.d. mill at Hurley, nine miles south of the deposit by rail, which was completed in 1911 (Spencer and Paige, 1935).

In 1924 Chino Copper Company sold all its assets to Ray Consolidated Copper Company (Files, NMBMMR); and this company was in turn absorbed by the Nevada Consolidated Copper Company in 1926 (Anderson, 1957). By 1926 production had been expanded to 11,000 tons of ore per day (Files, NMBMMR). In 1933 the Kennecott Copper Corporation acquired the assets of the Nevada Consolidated Copper Company; and established it as a wholly-owned subsidiary, the Nevada Consolidated Copper Corporation. (Anderson, 1957). The Santa Rita-Hurley complex was established as the Chino Mines Division in that year.

The operations at Santa Rita were suspended from October, 1934 to January, 1937, due to adverse market conditions. During 1937 a molybdenum recovery circuit was added to the mill at Hurley. On May 3, 1939, a new smelter at Hurley was fired for the first time; and concentrator capacity was increased to 20,000 t.p.d. (Minerals Yearbook). The installation of a pit-side skip and other modernization of mining operations within the last three years insure Santa Rita's position as New Mexico's greatest single mineral resource operation.

The writer estimates, on the basis of incomplete production data, that total production through December 31, 1962 amounts to at least 4,800,000,000 pounds of copper, as well as considerable amounts of gold, silver and molybdenum.

Chapter II

MINING, MILLING, AND SMELTING OPERATIONS

"Chino's mine is an open pit which covers an area of roughly one square mile at the perimeter. From the uppermost section of uncovered ore the pit drops to a depth of nearly 1,000 feet" (This Is Chino).

The sides of the pit descend stepwise in a series of benches about 50 feet high. The upper surface of each bench forms an inward and downward spiral; more than 50 miles of standard gauge railroad track and roads are located on the spirals.

Electric- or diesel-powered churn drills put down 12-inch holes 55 to 60 feet deep along the edge of the bench. The holes are loaded with ammonium nitrate-fuel oil explosive, and the material is blasted from the bench. A simulated channel sample is taken across the pile of broken material; and it is designated as mill ore or dump material on the basis of the assays of the samples.

Electric-powered 5 cubic yard shovels load the broken material directly into railroad cars. Waste is loaded into 30-yard air-operated side-dump cars; electric locomotives pull the cars to the dump where the material is discharged. Ore is loaded into 40-yard gondola cars that are taken by electric locomotives to a gathering yard to be made up into trains. Ore and waste from the lower levels are loaded into 25-ton rear-dump trucks for delivery to feed bins for the pit-side skip. The material is hoisted in 40-ton loads to the upper rim of the pit for loading into waste or ore cars.

Future development plans call for a change-over to truck haulage throughout the mining operation.

The present ore mining capacity is 22,500 tons per day. About 40,000 to 45,000 tons of waste are sent to the dump each day.

After the trains of ore cars are made up in the gathering yard, they are sent to the mill at Hurley, via the Atchinson, Topeka, and Santa Fe Railway. Eight trains of about 58 cars each are sent to Hurley each day.

The flow sheet of the mill is shown on the following page (Hardwick, 1958).

A recovery of about 80 percent is reported for the mill. The final concentrate carries about 20 percent copper. The molybdenum concentrate, packaged for sale outside the organization, carries approximately 90 percent molybdenite (MoS_2).

Copper concentrate from the mill is mixed with cement copper discharged from a rotary dryer; and the mixture, blended with suitable reagents, is charged directly to a natural gas-fired reverberatory furnace. Matte[a] is tapped from one of the two reverberatories as needed for charge to one of three convertors. After 12 to 14 hours in the convertor, where iron and sulphur impurities are removed, the molten copper is sent to a holding furnace,

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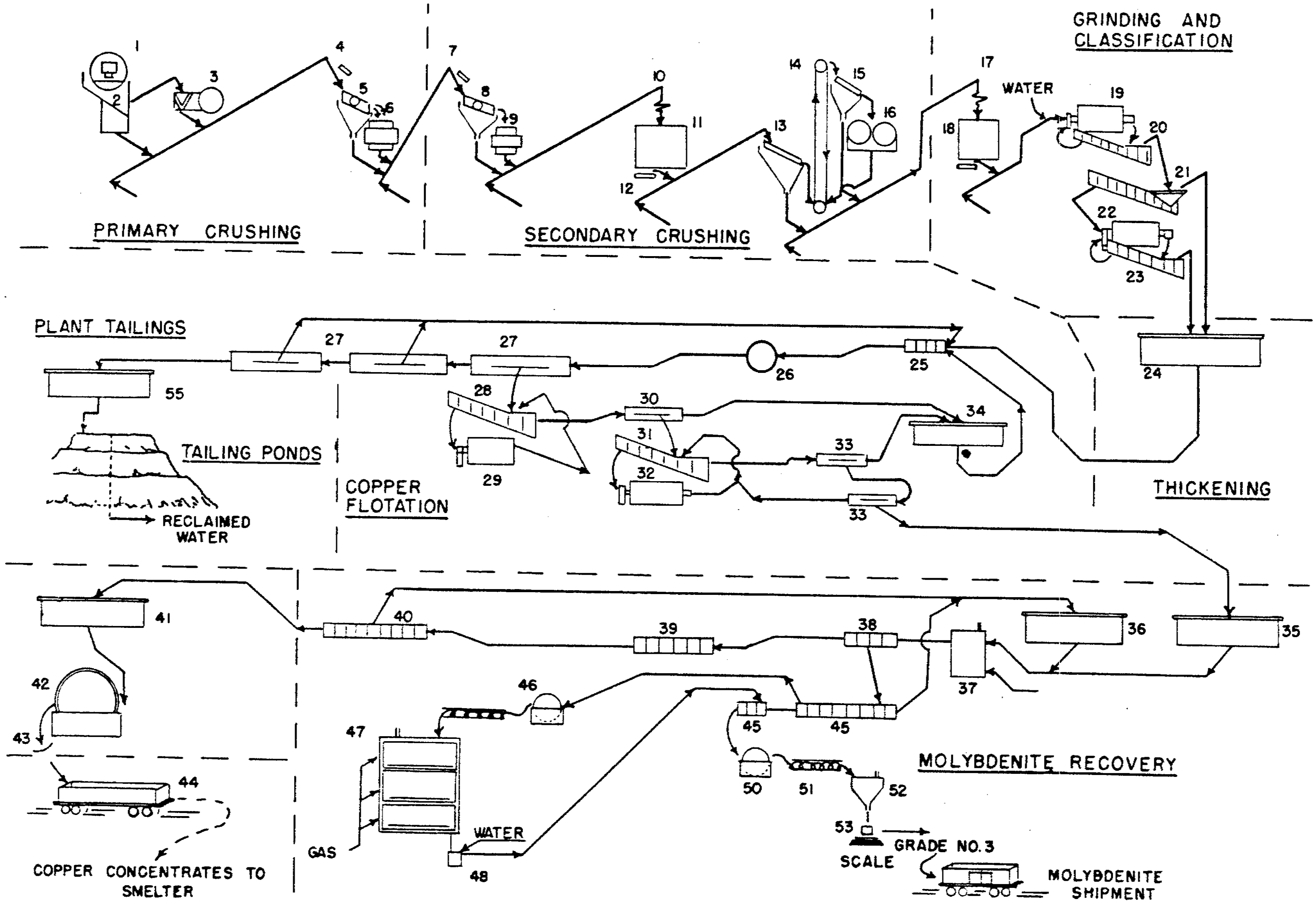
[a] - a molten mixture of iron, copper, and sulphur

or directly to the fire refining furnace. After refining, the copper is cast in finished shapes and shipped for marketing.



Mill Flowsheet of Concentrator, Hurley, N. Mex.

EQUIPMENT KEY FOR MILL FLOWSHEET

1. Mine car, rotary dumper
2. 8-inch grizzly in surge bin
3. Jaw crusher, 84 x 66 inches
4. Overhead magnet
5. 2 double-deck screens, 5 x 6 feet
6. 3 symons cone crushers, 7 feet
7. Overhead magnet
8. 2 Tyrock screens, 6 x 6 feet
9. 2 cone crushers, 5½ feet
10. 2 tripper conveyors
11. 18,500 ton ore bin
12. Belt feeders
13. 8 vibrating screens
14. 4 elevators, 80 feet
15. 16 vibrating screens
16. 4 crusher rolls, 72 x 20 inches
17. 2 tripper conveyors
18. 14 fine ore bins
19. 11 primary ball mills
20. 9 drag classifiers
21. 14 bowl classifiers
22. 22 secondary ball mills
23. Drag classifiers
24. 4 thickeners, 75 feet diameter; 1 with 230 feet diameter
25. 4-way distributor
26. 3 4-way distributors; 2 2-way distributors
27. 36 air cells in 12 rows of mechanical cells
28. 2 spiral drag classifiers
29. 2 ball mills, 8 x 6 feet
30. Cleaner air cells
31. 1 drag classifier, 9 x 30 feet
32. 1 ball mill, 7 x 12 feet
33. 4 recleaner cells
34. 2 thickeners, 48 feet diameter
35. 1 thickener, 26 feet diameter
36. 1 thickener, 26 feet diameter
37. Series of three steam tanks
38. 4-cell scavenger flotation unit
39. 6-cell scavenger flotation unit
40. 9-cell scavenger flotation unit
41. 1 thickener, 75 feet diameter
42. 4 drum filters, 14 x 14 feet
43. Conveyor belts
44. Bottom-dump railroad cars
45. 8-cell cleaner
46. 4-leaf disk filter, 4 feet diameter
47. 3-deck dryer, 14 feet diameter
48. Repulper, 2 feet diameter
49. 3-cell recleaner unit
50. 2-leaf disk filter, 4 feet diameter
51. Steam-jacketed spiral conveyor
52. Hopper
53. 700-pound barrels
54. Railroad boxcar
55. 2 thickeners, 250 feet diameter

Chapter III

HISTORY OF LEACHING AT SANTA RITA

During the early operation of the Santa Rita open pit mine bright grains of metallic copper were noticed where mine waters had come in contact with metallic iron present in the pit. It was also observed that waters discharged by a spring beneath the large waste dump showed a deep blue color. After determining that the blue color of the water was caused by copper in solution, and that metallic copper was produced by the reaction of the solutions with metallic iron, it was decided to attempt to recover the copper dissolved in the waters flowing from beneath the dump.

Accordingly, in 1924, simple lauders made of wood were erected on the sides of Santa Rita Creek below the dump. The water from the spring beneath the dump was led into the launders. Scrap iron resulting from the mining operation was placed in the launder. The aqueous copper solution reacted with the metallic iron, soon forming a coating of metallic copper on the surface of the scrap. When the pieces of scrap were completely covered with metallic copper they were moved about to jar the copper off and present a clean surface of iron to the action of the water. When sufficient copper precipitates had collected in the launders, the water was diverted from the launders and the cement copper was removed by hand shovel. After drying, the precipitates were packed in old cement sacks and stored until a carload was accumulated, when they were shipped to the El Paso smelter.

During this early period of cement copper production, an attempt was made to improve copper recovery by lengthening the launders. A larger portion of the copper in solution was recovered due to the additional iron surface available for reaction. Further efficiencies were introduced by the construction of a parallel set of launders. Thus while one launder was being cleaned of its copper precipitate content, the water was diverted to the parallel launder where copper continued to be recovered.

During the early period of production the operation depended on the natural flow of water from the dump. In 1936, the first artificial additions of water to the dump were made. Pit drainage and surface run-off water was pumped to the waste dump to increase production. This operation showed that mine drainage water could be used successfully to stabilize or increase the water supply. In 1939, it was decided to use the North Pit as a storage basin for run-off water from Santa Rita Creek. The availability of this and additional water from pit drainage and underground mining operations in the vicinity warranted construction of additional copper recovery facilities.

During 1939 a modern precipitation plant was built. The plant consisted of two units of six cells each, similar to one in operation at the Ray Mines Division, Kennecott Copper Corporation, Ray, Arizona. The plant had a capacity of 1,000 gallons per minute. After the plant was placed in operation it was realized that the quantity of new water available, averaging 475 gallons per minute, was insufficient to permit year-round maximum operation. It was therefore decided to pump tailing water from the precipitation plant back to the dump, thus allowing flow to be built up to maximum plant capacity.

The first tailing water was pumped to the dump in October 1939. Anticipated difficulties in the use of tailing water did not materialize, and experimental evidence of the desirability of tailing water as a leach solution was confirmed. The resulting effluent solution was found to contain large amounts of copper. It was decided therefore to utilize more tailing water in the leaching operation in order to build up a greater flow of water. The additional leach solution flow permitted the construction of a third six cell unit at the plant. This addition, completed in June 1941, increased the plant capacity to 1500 gallons per minute.

In 1947 mine water was obtained from the Ground Hog Mine of the American Smelting and Refining Company. The new plant flow made available by the aquisition permitted the addition of a fourth unit in January 1948. This new construction, coupled with modification on the existing units, brought total plant capacity to approximately 2,600 gallons per minute. Additional acquisitions of water permitted the construction of a fifth unit, which was placed in operation in May 1950. Plant capacity at that time was approximately 3,300 gallons per minute.

Since 1950 the acquisition of additional water for leaching and the modification of the precipitation operation has extended the plant capacity to about 7,500 gallons per minute. In 1955 a new pumping installation was built at the plant site, thus allowing the handling of greater quantities of water and the pumping of leach solutions to higher dump levels.

At this time the operation is being maintained at a capacity of 7,500 gallons per minute. Research is presently underway to develop improved techniques of recovery of copper from copper-bearing waters. Preliminary planning leading to the development of a new and improved operation has recently been initiated (1963).

The sources of the preceding discussion are papers by Messrs. W.H. Goodrich[a] and A.F. Morris[b], written while they were staff members of the Chino Mines Division of the Kennecott Copper Corporation; and conversations with Messrs. T.J. Montgomery and C.H. Van Buskirk, who currently are associated with the leaching operation at Santa Rita.

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a - formerly General Manager of Chino Mines Division;

b - presently General Manager, Ray Mines Division, Kennecott Copper Corporation, Ray, Arizona.

Chapter IV

PRESENT OPERATION

The waste dump[a], the immediate source of the copper recovered in the leaching operation, is derived from the open pit mining operation. The dump is composed of material stripped from the ore body to expose it for mining, and of low-grade or mineralogically-unfavorable portions of the ore body itself. Material that cannot be economically treated at the flotation mill at Hurley is discarded by being placed on the waste dump. Although not 'ore' in terms of the flotation recovery of copper, these 'waste' materials do contain more or less copper. The development of the leaching process has made it possible to recover these small but valuable amounts of metal.

Formation of the Dump

Material destined for the dump is removed from the pit via the new pit-side skip or via train haulage. Material removed from the pit by skip hoisting is loaded into 30-yard side-dump cars like those used in train haulage. Standard gauge electrified waste trains, composed

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a - The operation to be described here is commonly referred to in the literature as heap or pile leaching (Liddell, 1926). However, heap or pile leaching implies construction of the piles specifically for leaching purposes. The dump at Santa Rita was originally, and is still primarily, constructed for the disposal of waste material. Therefore, in keeping with current usage at Santa Rita, the denotations 'dump' and 'dump leaching' will be used throughout this paper.

of six to eight cars and one locomotive, dump the material along the edge of the bench currently being constructed. Rubber-tired loader-graders keep the material pushed over the edge and maintain a clear dump area for the trains. As the dump builds outward the track is moved to the edge by mechanized track movers.

The dump benches are built up in 50-foot layers. Of the material dumped over the edge, the larger masses tend to roll or carry farther down the side of the dump. Thus a rough stratification similar to graded bedding tends to be produced. This bedding may be observed on the sides of the dumps.

Application of Water to Dump

The purpose of the water application system is to make water available for the leaching and recovery of copper from the dump. At first water was distributed by spraying; later, by ponding. The plugging of spray hoses and pipes by precipitation of iron salts from recirculated tailing water created such problems that ponding was developed as an alternate method of application. Level ponds approximately 300 feet square were built on the upper surface of the dump by bulldozers. The water was allowed to stand in the ponds until it soaked into the dump. The application of water was accompanied by precipitation of iron salts; maintainance consisted of removal of precipitated salts and ripping to loosen the surface so as to permit water to enter the dump.

The present technique of application uses the 'difficulty' with surface plugging to good advantage. The precipitation of iron salts from the tailing water is used as a control technique. Progressive cementing of the surface by precipitates causes the point of water penetration to move across the surface; thus, leach water is eventually placed in contact with all of the dump.

Tailing water from the precipitation plant is pumped to the dump; and distributed across the surface in ten-inch Transite pipe. The water is then discharged into a system of distribution trenches. The distribution system[a] consists of a main trench with a series of secondary trenches branching away at convenient angles. The leach water initially sinks into the dump rather quickly; but as the iron salts precipitate and cement the surface the water is forced to flow farther before penetrating the dump. The point of penetration of the solution gradually moves downstream along each trench. A small dam diverts the water from the main trench into each of the subsidiary trenches. When a subsidiary trench is completely sealed by precipitates of iron, it is blocked off and the water is diverted into another trench.

The ditches or trenches are constructed by use of a rubber-tired loader-grader. The grader operator lays

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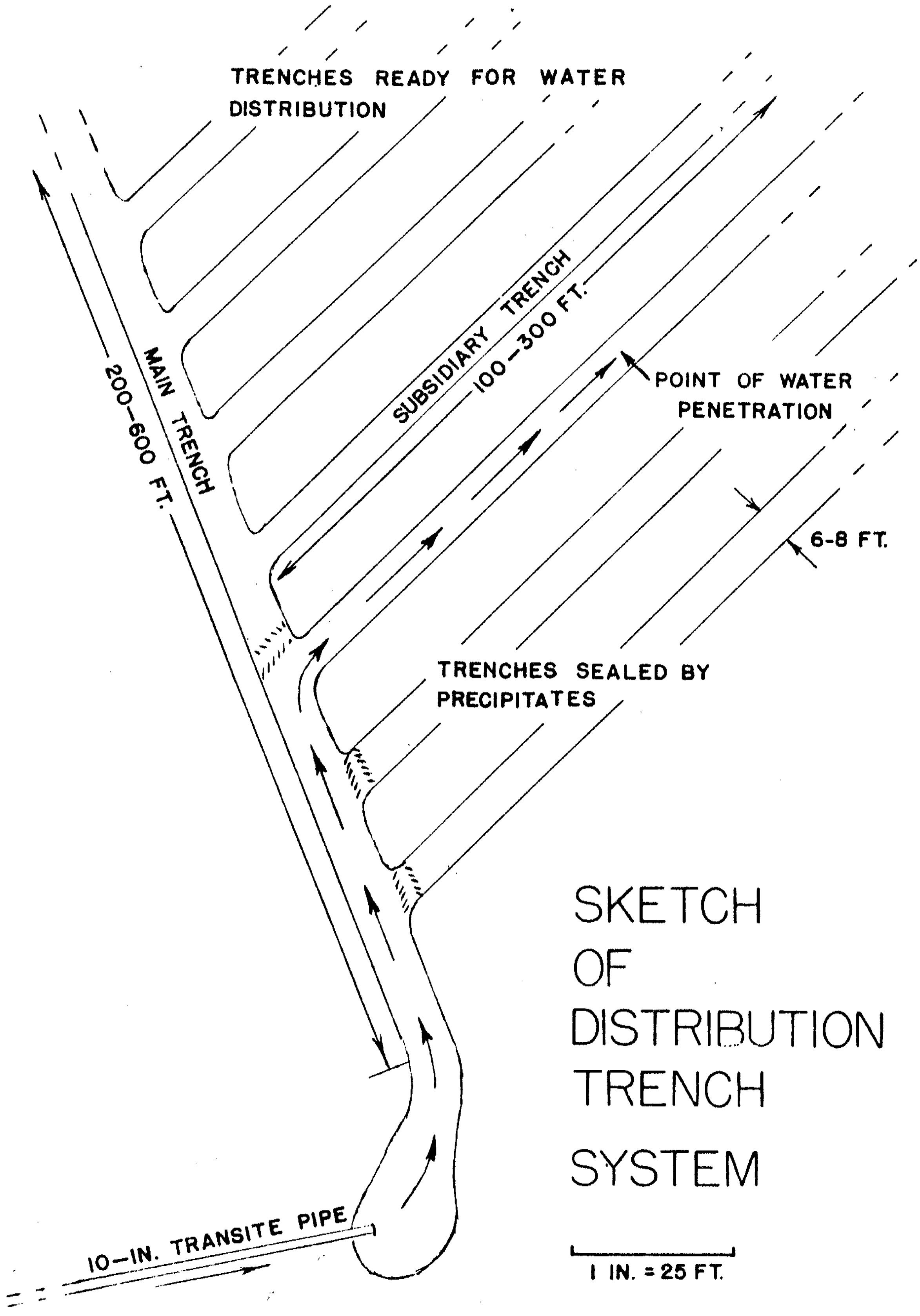
a - See accompanying sketch



SKETCH OF DISTRIBUTION TRENCH SYSTEM

out the pattern of trenches by eye, and then grades the system. The trenches are the width of the grader blade, with walls or sides six inches to one and one half feet high. After a trench system has been completely plugged by precipitated iron salts, the water is diverted to another system of trenches. The cemented dump surface is ripped to break the iron salt seal, the trenches re-graded, and the dump is prepared for a new cycle of water application.

This technique requires a minimal amount of attention and maintainance, and requires little preparation of the dump surface for leaching.

To effect economies in operation, certain procedures have been introduced in the re-laying of Transite pipe on the dump surface. When pipe is to be taken up, no attempt is made to salvage the Transite collars and rubber seals that join the 13-foot lengths of pipe. It has been found cheaper to break the old collars and use new ones when re-laying pipe, rather than take the time and effort necessary to recover successfully the old collars. Thus a substantial portion of the time of a five-man pipe crew is made available for other duties.

Collection of Effluent Solution

After passing through the dump and becoming enriched in dissolved copper, the leach water is re-captured by collection behind two dams along Santa Rita Creek. Water collected behind the higher dam, after metering and sampling, is gravity-fed directly to precipitation cells. That collected behind the lower dam is directed through pipes and launders to the pump house. A metering and sampling location is established on the launders just ahead of their discharge into the feed pump sump. Five 40-h.p. 2000-g.p.m. feed pumps draw copper-rich solution from this sump and force it through five 10-inch plastic-lined steel pipes across Santa Rita Creek to the precipitation cells.

Precipitation of Copper

The copper-bearing water, after being pumped across Santa Rita Creek, is directed into concrete precipitation cells. Here the solution is allowed to react with metallic iron, taking the iron into solution and precipitating metallic copper[x]. After the copper has been removed from the solution, the water is returned through a concrete launder across Santa Rita Creek to the pump house to be pumped onto the dump so that a new leaching cycle may begin.

A precipitation cell, shown in the accompanying diagram, is constructed of concrete. Each cell is 80 feet long, 5 feet wide, and 5 feet deep. The cell is doubled in the middle so that the water must make a 180° turn

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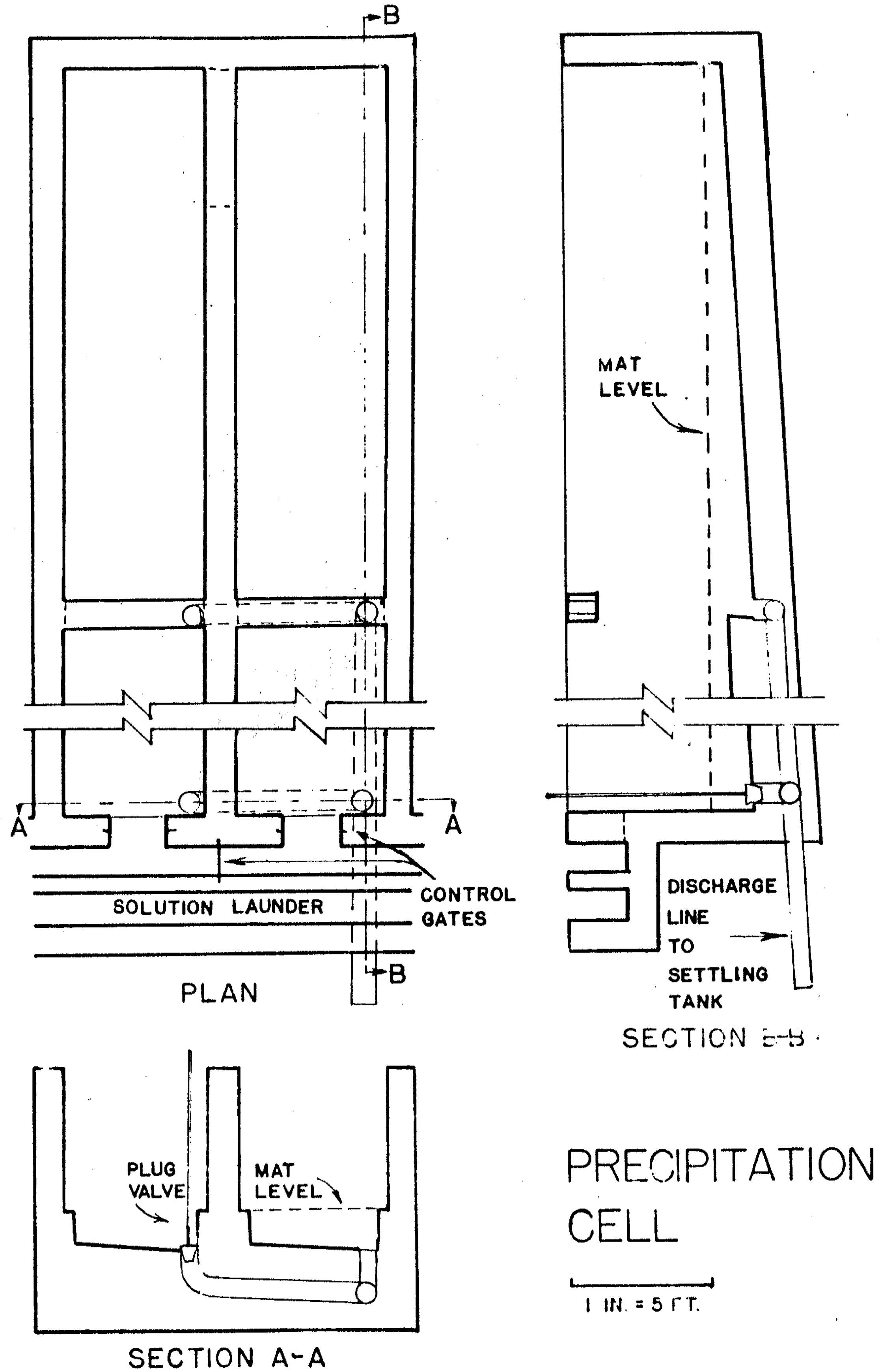
x - Commonly called 'cement copper'. The recovery of copper from solution by precipitation on metallic iron is often called 'cementation'.



PRECIPITATION CELL

midway through the cell. Protruding shoulders 12 to 18 inches above the floor of the cell support wooden mats or grids with square openings 5/8 of an inch on a side. The bottom of each half-cell is divided into two slopes, each leading to a simple plug valve. The valves from the upper half-cell permit the water to flow to the lower one-half; those from the lower one-half lead to settling ponds. The entrance and exit of each cell are fitted with gates, so that water may be diverted when necessary. Water is lead to and from the cells by concrete launders. The top of the concrete work is fitted with 2" x 12" timbers to protect the cells from breakage during charging with tin cans. The construction of the cells is designed so that the water flows down a 5 percent grade.

After a cell is readied for use, it is charged with burned and crushed tin cans by a crane fitted with an electromagnet. Copper-rich solution is then directed into the cell and the precipitation of copper is allowed to proceed. After most of the cans in a charge are consumed, the water is diverted from the cell and it is washed to recover the metallic copper.

The copper is often encrusted on the remaining tin cans in the cell, and must be washed from them with water under high pressure. After the water has been diverted from the cell, the plug valves are opened, and the copper precipitates are washed into drying cells. The wash water, supplied at 200 p.s.i. by two 75 h.p. pumps, carries the copper from the cans through the wooden mats and out through the wash lines served by the plug valves. After washing, the extraneous material present in the tin can charge, such as plastic and glass, is removed from the cell; and mats in poor condition are replaced. After recharging the cell with cans, water is allowed to enter and the process of precipitation is repeated.

The present operational plan directs the water through four cells in normal flow[a]; thus the water flows 320 feet in contact with the precipitating agent. The recovery has averaged about 87 percent for the last year. In periods of lower water flow the recovery has been as high as 99 percent, but flow in excess of rated capacity accounts for the present lower efficiency.

After the copper is washed from the cells to the settling tanks, the excess wash water is decanted and fed to the tailing water launder. A crane fitted with a stainless steel clamshell removes the copper to drying pads, where it is allowed to drain. When the water content has been reduced to about 25 percent the copper is loaded into gondola cars by the clamshell and is sent to the smelter at Hurley.

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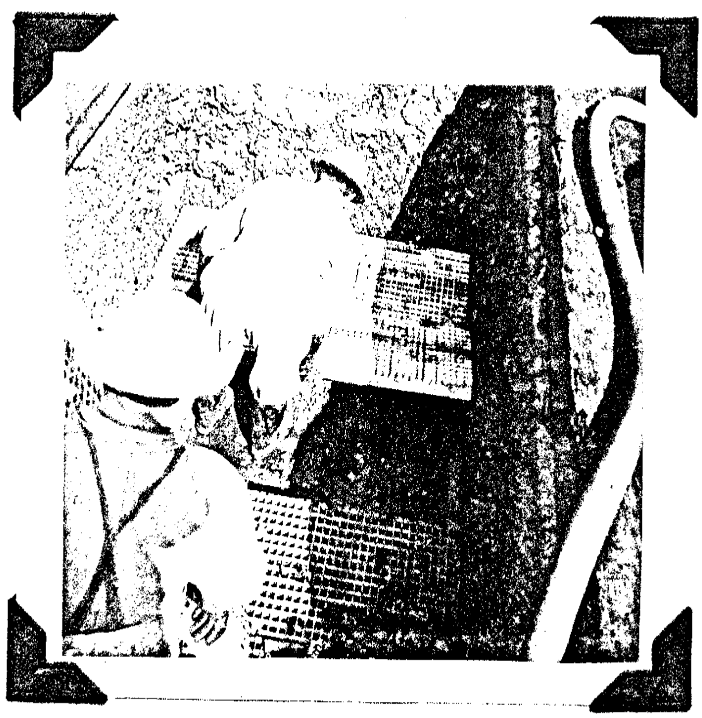
[a] - Six units of this type are in use. In addition, one six-cell unit is now in operation.

WASHING A CELL

REPLACING MATS IN A CELL

Note washed cans (lower center), and unwashed bed of cans and precipitates (upper center).



Recirculation of Tailing Water

Tailing water from the cells is directed through a concrete launder across Santa Rita Creek. After a tail sample has been taken, the water is directed into a large settling tank adjacent to the pump house. The overflow from this tank goes to the tailing sump. Ten high-lift pumps[a] rated at 1000 g.p.m. draw water from the sump and force it through 10-inch plastic-lined steel pressure pipe, or 10-inch Transite pressure pipe, to the top of the dump. The ten tailing return lines[b] discharge into Transite pipe which is used to distribute the water on the dump surface.

Make-Up Water

Losses due to evaporation and other reasons require the addition of water at various times. This make-up water presently is obtained from three sources.

Mine water pumped from American Smelting and Refining's Star shaft, adjacent to the precipitation plant, is added to tailing water for circulation to the dump as required. Pit drainage from the South Pit, pumped directly to the waste dump, amounts to a continuous addition

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a- Distributed as follows:
Six 250 h.p.,600 foot head, 1000 g.p.m. vertical motor
Two 200 h.p.,650 foot head, 1000 g.p.m. vertical motor
Two 200 h.p.,600 foot head, 1000 g.p.m. horizontal motor

b- Distributed as follows;
Six 10-inch plastic-lined steel lines
Four 10-inch Transite lines (to be replaced by three 14-inch plastic-lined steel lines)

to flow of 160 g.p.m. The old North Pit serves as a reservoir and flood control basin from which water may be drawn as needed by releasing it into Santa Rita Creek and collecting it at the dam below the precipitation plant. Projected flood control dams will provide additional water supply sources.

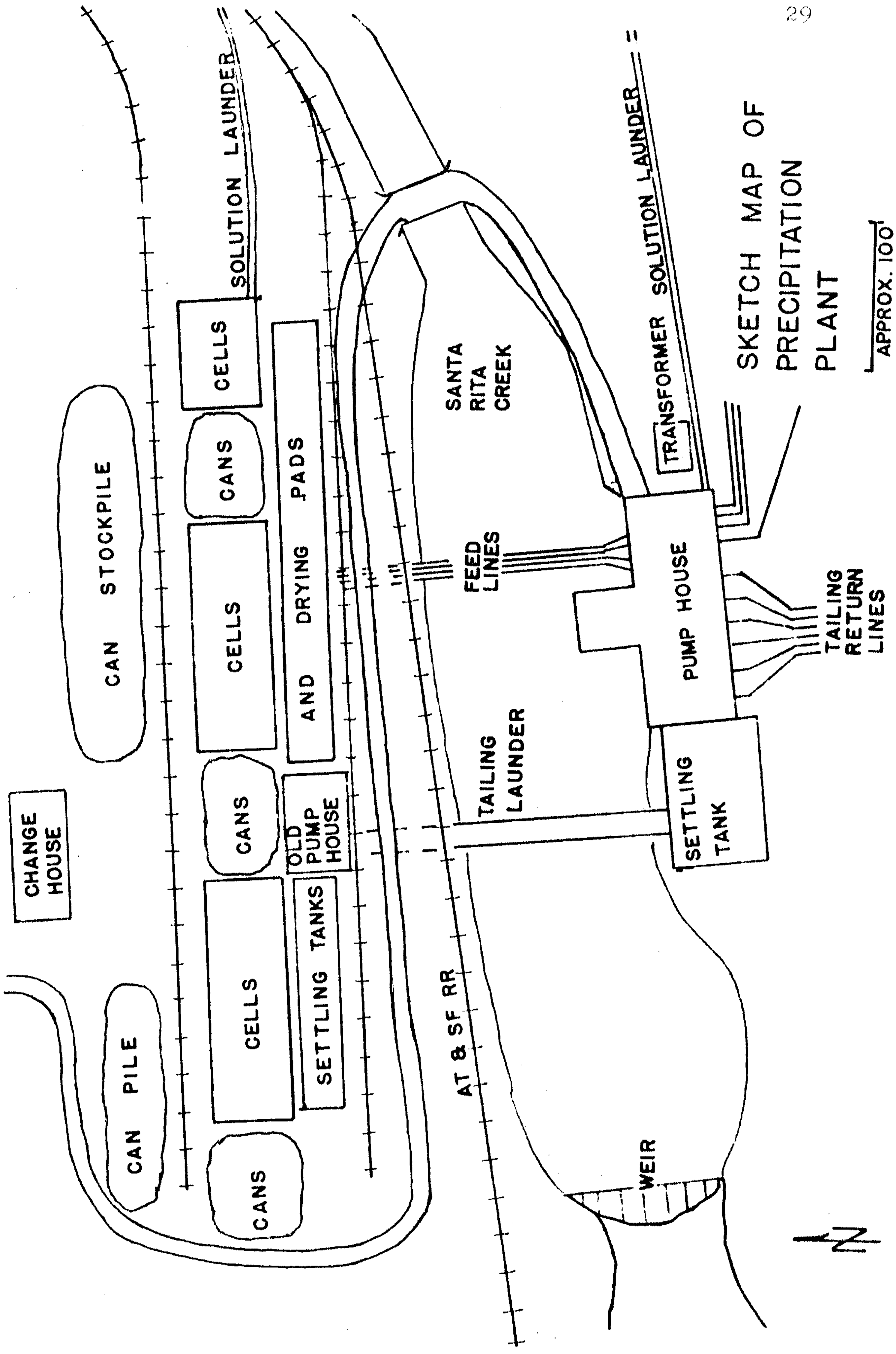
During wet weather periods, such as December 1962 - April 1963, water added to the dump by natural means is sufficient to meet make-up water demands. The sources noted above, except South Pit drainage, are then held in reserve.

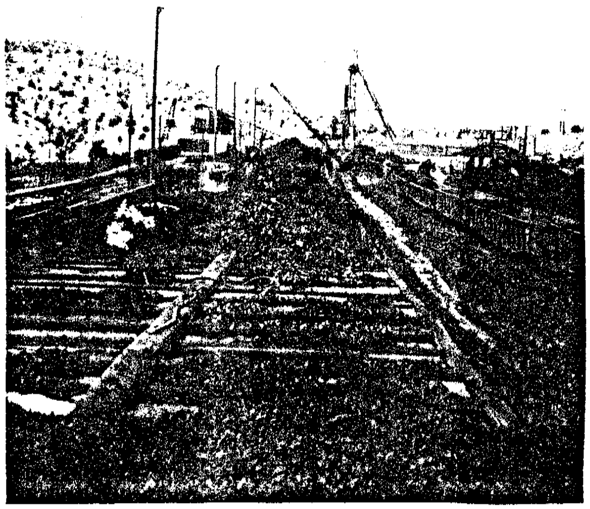
Plant Layout

A sketch map of the plant follows this page.

Santa Rita Creek and the Santa Rita branch of the Atchinson, Topeka, and Santa Fe Railroad pass through the precipitation plant, effectively dividing the pumping installation from the precipitation facilities. This makes the construction of pipe and launder bridges and tunnels necessary.

The precipitation facility is served by two railroad spurs, one on either side. One serves as a delivery spur. The carloads of tin cans for use in the process are spotted on this spur, where the cans are unloaded by the crane and electromagnet to the stockpiles or directly to the cells. The second spur, adjacent to the settling tanks


SKETCH MAP OF
PRECIPITATION
PLANT
APPROX. 100'
SOLUTION LAUNDER
SOLUTION LAUNDER
TRANSFORMER
TRANS
SANTA
RITA
CREEK
CELLS
CANS
CAN STOCKPILE
CELLS
DRYING AND PADS
CHANGE
HOUSE
CANS
OLD
PUMP
HOUSE
CELLS
SETTLING TANKS
CAN PILE
CANS
AT & SF RR
FEED
LINES
PUMP HOUSE
TAILING
RETURN
LINES
TAILING
LAUNDER
SETTLING
TANK
WEIR



VIEW OF PRECIPITATION CELLS

Looking west. Note can unloading spur (middle right). A.S.R. Star Shaft headframe in background.

and drying pads, is used for the loading of cars with copper precipitates by the clamshell crane.

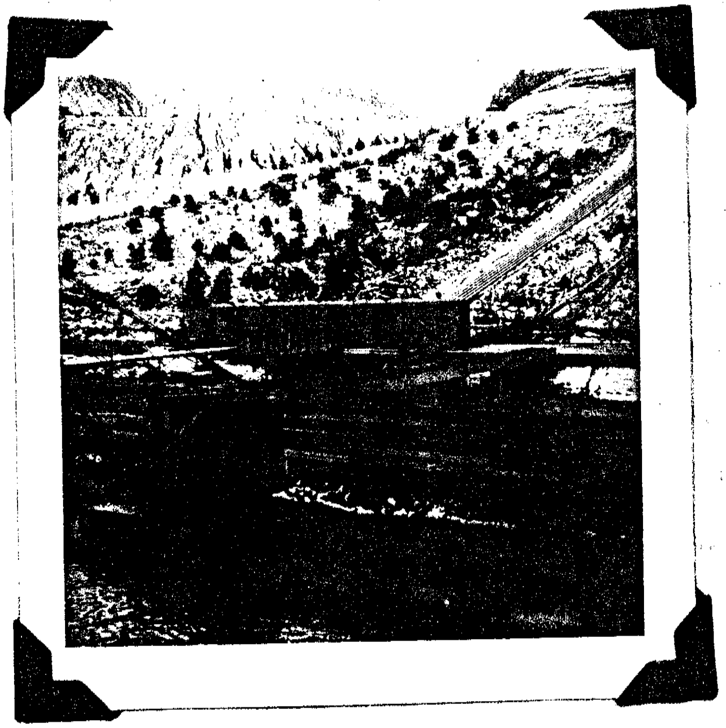
Below the pump house is a weir. This serves as a recovery dam, stopping the copper-rich water that has escaped the higher dams or collecting water sent down Santa Rita Creek from the North Pit. When sufficient water has backed up behind the dam, it is pumped to the tailing water sump by two 10 h.p. electric pumps. The weir also serves as an emergency reservoir in case of pump breakdown or flood.

Pump House

The pump house is a modern reinforced concrete and tile structure. The pumps are located on galleries around the interior, which is a sump fitted with inspection and cleaning ports to the intake sumps. The discharge lines pass through the tile walls above the reinforced concrete foundation, thus making the structure virtually water tight against flooding. The central control room overlooks the whole pump room. The plant offices are attached to the building.

Construction

Because of the corrosive nature of the solution handled in the plant, only inert or corrosion-resistant materials may be used in the construction of facilities. Materials used include stainless steel, plastic, concrete,

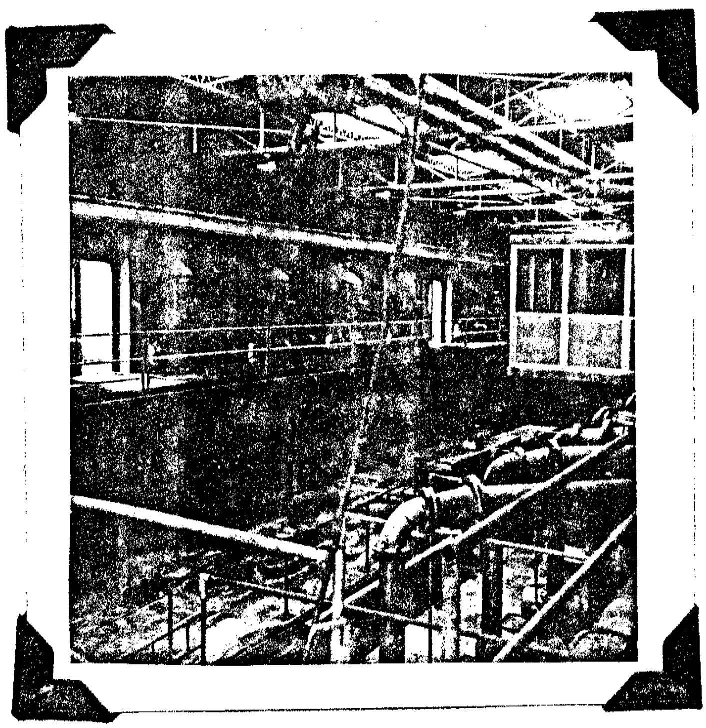


EXTERIOR VIEW OF PUMP HOUSE

Note tailing water launder and settling tank(middle right), tailing return lines to dump(upper right), and dump(upper left). Taken when ground snow covered.

INTERIOR VIEW OF PUMP HOUSE

Note feed pump discharge lines(lower right), control booth (middle right), and tailing water recirculation pumps(middle center and left).



lead sheeting, fiberglass, Transite, and non-corrosive rubber.

Power

Electrical power, used for pumping and operation of the cranes, is supplied by the company-owned power plant at Hurley. High lines carry the power to a transformer station at the precipitation plant, where it is converted to 3-phase 60-cycle 440 volts for use. About 33 percent of the total power consumption of Chino Mines Division is attributed to the precipitation plant.

Can Consumption and Copper Production

About 4000 tons of burned and crushed tin cans, shipped from as far away as Atlanta, Georgia, are consumed each month. The average 'can factor' for the last two years has been about 1.5 (i.e. 1.5 pounds of cans are consumed for each pound of copper produced). Total monthly production has been about 2,600 short tons of copper.

Grade of Product

Presented below are three monthly composite assays of precipitation plant product. The months represented are in mid-1962.

COMPOSITE MONTHLY ASSAYS
expressed as dry weight percent

month	Cu	Fe	CaO	Al_2O_3	S	unreported
A	84.31	2.1	0.1	1.8	0.4	10.0
B	85.86	2.2	0.2	2.0	0.8	7.6
C	83.45	2.1	0.1	2.3	---	9.3

Manpower

Twenty-three men are employed at the plant, under a supervisory staff of five. All are on rotating 5-day shift, except the general foreman and one foreman who are assigned permanent 5-day morning shift.

Additional maintainence personnel, such as electricians, carpenters, and mechanics, are drawn as needed from the mining department.

EMPLOYEES, BY JOB CLASSIFICATION

CLASSIFICATION	NUMBER
Washerman	12
Pumpman	6
Crane Operator	4
Laborer	1
Foreman	4
General Foreman	1

SHIFT DISTRIBUTION OF PERSONNEL

SHIFT	CLASSIFICATION	NUMBER
Morning	Washerman	4
	Pumpman	2
	Crane Operator	2
	Laborer	1
	Foreman	2
Afternoon	Washerman	4
	Pumpman	1
	Crane Operator	1
	Foreman	1
Evening	Pumpman	1
	Foreman	1

Chapter V

IMPROVEMENTS IN PRESENT OPERATION

The discussion to be presented here is restricted to improvements in the present operation that do not require major physical alterations in the plant.

The staff of the precipitation plant who has had a part in the evolution of the present operation should be complemented for their work. The operation as it is now being conducted has been developed to a high level of efficiency in terms of the present physical facilities and concepts of leaching. Few areas of fruitful improvement are left. However, suggestions that may add to the efficiency of the operation are noted here.

The principal consumption of manpower may be attributed to three occupations; maintainance, washing of cells, and laying of distribution piping on the dump surface. These three areas represent the possible subjects of improved manpower efficiency.

Maintainance is a necessary part of any operation and is not amenable to more than cursory control. A schedule of preventive maintainance, such as that now employed, may reduce the ultimate expense. However, certain necessities of maintainance arise that cannot be impressed in a schedule, and that must recieve immediate attention. Therefore, economies are not liable to occur in the operation beyond those already in force.

Washing of the cells consumes more man-hours than any other single occupation in the plant. Economies or improvements in this operation would be highly desirable. However, by the very nature of the physical facilities, the washing is constrained to a certain configuration. No extensive improvement in the washing is available in the context of the present facility.

Laying pipe on the surface of the pile requires the services of a five-man pipe crew. Therefore, this operation is a possible source of improvement.

The original use of Transite pipe was prompted by the corrosive nature of the solution being distributed. Hoses used in the early operation were quickly eaten away or blocked by deposits. The Transite pipe was not affected by the corrosion. However, it is brittle and easily broken by impact.

With the new developments in plastic and rubber, a suitable replacement for Transite pipe may now be available. The replacement should have the following characteristics:

(1) Chemical resistance to corrosion and scale formation;
(2) Physical resistance to impact and abrasion, and to extremes of weather conditions;
(3) Availability in pipe sizes required, with quick-connecting joints and positive seals.

If such a product is available at a reasonable investment it could well pay for its purchase in lowered maintainance and replacement costs, and reduced man-power requirements in relaying.

The Transite pipe on hand could be salvaged for use as semi-permanent distribution lines that would be moved only when a new dump bench is to be established.

The only raw material used in the operation is iron in the form of burned and shredded tin cans. Efficiencies in raw material consumption may be easily introduced.

In-flowing solutions are rich in ferric sulphate and sulfuric acid in addition to the desired copper sulphate. Both of these compounds are iron-consuming through reactions of the form:

(1) $Fe_2(SO_4)_3 + Fe = 3\ FeSO_4$; and

(2) $H_2SO_4 + Fe = FeSO_4 + 2\ H$.

These reactions compete with the precipitation reaction:

(3) $CuSO_4 + Fe = Cu + FeSO_4$.

It would be desirable, therefore, to purify the incoming solution and thereby reduce unproductive can consumption.

Several possible methods of purification are available, but only one could be introduced without major modification of the existing plant.

If the inflowing solution was passed over a bed of concentrate from the mill at Hurley, the following reactions would take place:[a]

(4) $Cu_2S + 2\ Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S$;

a- Note that these will not be the only reactions, but are representative of the type that will occur.

(5) $Cu_2O + H_2SO_4 = CuSO_4 + H_2O$.

The first reaction will reduce ferric sulphate to a form that will not consume iron. The latter reaction expends sulfuric acid, an iron-consuming compound. The sulfuric acid will, however, be incompletely consumed, since the amount of copper oxides in the mill concentrate is small.

Unless the purification is conducted in the absence of oxygen, ferric sulphate will be regenerated by the reaction:

(6) $4\ FeSO_4 + 2\ H_2SO_4 + O_2 = 2\ Fe_2(SO_4)_3 + 2\ H_2O$.

However, this reaction should not be restrictive if the bed of concentrate is sufficiently long. The reaction reduces the sulfuric acid content of the solution; and with sufficiently long bed or reaction area, the ferric sulphate will again be reduced by the sulphides of the concentrate. If the bed length is optimized, no sulfuric acid or ferric sulphate will exist in the solution going to the cells.

If the acid is completely consumed before the ferric sulphate, problems may develop through precipitation of ferric iron by hydrolysis reactions of the form:

(7) $Fe_2(SO_4)_3 + H_2O = Fe_2O_3 \cdot 2\ SO_3 \cdot 5\ H_2O + H_2SO_4$.

If the bed is moved forward in the reaction area these precipitates will be eventually removed by the action of the sulfuric acid in the in-flowing solution.

Note that the total consumption of iron is not altered. However, the recovery of copper per unit of iron consumed will be greatly increased, possibly approaching the theoretical limit of 1 pound of copper for each 0.88 pound of iron consumed. Further economy will result from the conversion of mill concentrate to a form that is more desirable for use in the smelting operation. The heat saving in smelting metallic copper rather than an equivalent amount of sulphide copper should be appreciable. This saving plus the saving incurred through the elimination of non-productive iron consumption constitute the added economy of the purification of the in-flowing solution. The economy is modified by the value attached to the concentrate used in the purification.

The treatment bed of concentrates may be formed in the collection launders now in existence. Thus no additional construction would be required.

Chapter VI

OPERATING ASSUMPTIONS

The benefits derived from development based on a set of assumptions are direct functions of the validity of the assumptions. Therefore, to maximize the benefits to be derived from an operation it is necessary to maximize the validity of the assumptions on which the operation is based.

Leaching at Santa Rita is based on a set of assumptions made in the early days of the operation. The assumptions, derived from observational data, have not been modified in any major particular during the twenty-five years of extensive operation. Before capitalizing additional effort based on the assumptions, it is necessary that they be reviewed and modified so that they conform more closely to the actual situation.

Range of Assumptions

Practically every aspect and phase of the operation lack definite data. Leaching is based on an assumed set of chemical and physical conditions in the pile; recirculation of tailing water, on an assumed chemical reaction; and cementation recovery, on an assumed economic situation.

It is the purpose here to examine the technical aspects of the leaching program; economic considerations will be treated elsewhere.

Mineralogical Assumptions

One of the principal assumptions employed in the leaching operation is that the major ore mineral in the dumps is chalcocite. The entire operational plan is established to recover effectively copper present in that form.

Santa Rita is described as one of the 'porphyry copper' deposits characterized by a 'blanket' of secondary chalcocite underlying a leached capping zone (Lindgren, 1933). Other deposits of the type are well known; in fact, nearly all the copper producers in the Pacific Southwest from Bingham Canyon, Utah, and Ajo, Arizona, to Santa Rita,, New Mexico, mine ores of this type.

Copper occurs in pyrite as well as in chalcocite at Bisbee, Arizona (Ransome, 1904). Chalcopyrite and copper-bearing pyrite are common in the ores of the Clifton-Morenci district, Arizona (Lindgren, 1904). At Ray and Miami, Arizona, appreciable amounts of copper occur in chalcopyrite; pyrite contains a small amount of copper (Ransome, 1919). The secondary ores at Bingham, Utah, carry covellite in addition to chalcocite (Lindgren, 1933). Spencer (1917, page 102), notes that the secondarily enriched ores at Ely, Nevada, contain a large amount of chalcopyrite. A composite assay of the ores produced to 1914 shows a chalcopyrite content of 1.8 weight percent and a chalcocite content of 1.9 weight percent. Thus about 36 percent of the total copper is contained in a mineral other that chalcocite. Pyrite, composing an average 4.8 weight percent of the ore, is reported to contain additional small amounts of copper.

A common feature of deposits of the secondary chalcocite type is the change of ore character with depth. Chalcocite content decreases; the proportion of primary or 'protore' minerals such as pyrite and chalcopyrite increases. The mineralogy of the copper, as well as the overall copper content, changes appreciably (Lindgren, 1933).

On the basis of these observations it is a reasonable conclusion that at Santa Rita an appreciable amount of copper in the dump occurs in a form other than chalcocite.

Wells (1923) reports chalcocite to be the most important copper mineral in the ore bodies at Santa Rita. However, chalcopyrite is widely distributed throughout the ore, as is pyrite bearing small amounts of copper. Additional copper minerals occur in the oxidised upper portion of the ore body; such as cuprite, malachite, azurite, and native copper (Wells, 1923).

Since the report just cited was written, mining has proceeded to greater depths. Therefore, less of the oxide copper minerals are to be expected in the ores. Further, the deep ores may be expected to carry a greater portion of the copper in chalcopyrite, while the relative importance of chalcocite must diminish somewhat.

At Santa Rita, chalcopyrite occurs to the greatest depths yet examined, together with cupriferous pyrite. However, the vertical range of chalcocite is extensive, and the expected diminution of chalcocite content may not yet be of major importance (Wells, 1923).

Microscopic examination of the cupriferous pyrite occuring at Santa Rita shows small inclusions or 'blebs' of chalcopyrite in the mineral. The appearance of the inclusions indicates physical mixing rather than exsolution texture, suggesting that little copper occurs within the body of the pyrite, and thus the copper content of the pyrite must be largely due to such inclusions of chalcopyrite.

Ores mined in the past from mineralisation in the Abo formation have contained an unusually large proportion of chalcopyrite. However, these ores were not extensive, and probably form only a small part of the dump material.

Present mining and milling operations are based on eight ore types. The four principle types are:

(1) free chalcocite ore;
(2) chalcocite-pyrite intergrowth ('ducktownite') ore;
(3) chalcopyrite ore; and
(4) chalcopyrite-magnetite intergrowth ore.

The remaining four types are variations of these major types, and occur in minor quantities. The proportions of the major ore types present in the material now being sent to the dump has been estimated to be, in the order given above, 40:25:15:15.

Future operations are to be extended into the Lee Hill area. Exploratory drilling shows that the ores to be mined from that area will contain a larger amount of chalcopyrite than has been contained in the ores mined to date. The waste material from the area will also be high in chalcopyrite.

'Protore' containing chalcopyrite is encountered on the margins of the irregular lower limit of the secondary enrichment zone. This material contains no secondary copper minerals, but is reported to contain 0.1-0.2 percent copper in chalcopyrite. Increasing amounts of this material will be sent to the dump as the mining operation is extended to greater depth.

To date, only ore enriched by secondary processes producing chalcocite have been mined. Therefore, the operating assumption mentioned earlier has been valid to date.

However, future mining plans call for the development and use of Lee Hill ores containing copper minerals other than chalcocite. Further, it is to be expected that in terms of long range operations an increasing portion of the copper mined will occur in chalcopyrite. Therefore, any change in the leaching operation should take account of the probable future change in the mineralogy of the dumps.

Physical Assumptions

Application of leach solution to the waste dump is based on the assumptions that:

(1) The leach solution has physical accessibility to the copper-bearing minerals; and
(2) The leach solution may be recovered after obtaining its load of dissolved copper.

The first assumption is usually expressed as two statements:

(1) The leach solution has access to each rock mass due to the high porosity of the piles; and
(2) The copper minerals occur on fracture planes within each rock mass, and are therefore available to the solution by its penetration of the fractures.

The second assumption is often stated: A factor in the successful development of leaching at Santa Rita is "the solid, impervious nature of the earth's surface under the dump" (Goodrich).

Drilling tests on the leaching piles at Utah Copper Division, Kennecott Copper Corporation, Bingham, Utah, indicate that water flow within the piles is essentially vertical. Examination of cuts through Chino's No. 8 dump, moved to allow mining of the Lee Hill area, indicated essentially vertical water flow. Water movement of this type indicates high porosity and permeability of the material relative to the amount of water applied (Tolman, 1932).

However, there is some indication that the porosity and permeability of the dump may be modified by the leaching process. Tests at the metallurgical research laboratory at Hurley indicate that appreciable amounts of hydrous ferric iron compounds are precipitated throughout the column of dump material being leached. The observed deposits were of such a magnitude that, if occuring in the dump, they could seriously modify the flow of leach solution.

The manner of construction of the dump may create relatively impermeable layers that restrict the flow of solution. During the construction of a bench or layer, the upper surface of the layer is subjected to weathering that tends to produce materials impermeable to water flow. Hydrous iron salts, precipitated during leaching and loosened by ripping but not removed, are left on the dump surface. Finally, the movement of heavy equipment on the dump surface tends to pack the impermeable materials; thereby creating a layer that, when covered by the construction of a new dump bench, is impervious to leach solution.

A 'breakout' of water on the side of the dump about two years ago tends to support this postulation of impermeable layers within the dump. The 'breakout', occuring near the precipitation plant, was located near the intersection of a bench top and the side of the dump. Leach solution apparently flowed down through the dump to the impervious surface, and then channeled outward above this surface to the edge of the dump where it emerged to the surface and flowed down the side of the dump. Efforts to stem the flow were not successful; only the cessation of water application stopped the 'breakout'. However, the problem apparently cured itself, since later application of leach solution has not resulted in a similar occurence.

The evidence cited here is not conclusive. However, there is a strong suggestion that leach solutions do not have free access to the rock masses at all times, especially after leaching has begun. Further investigation is definitely indicated.

Examination of rock from the dump supports the assumption that leach solutions have access to the interior of rock mass; and hence, to the copper minerals. Close spaced fracturing is a feature of all the material now being mined. Dissemination of ore minerals through the rock is reported to be minor; most of the copper minerals occur on the fracture planes.

A large piece of material was reduced to particles about $\frac{1}{4}$ to $\frac{1}{2}$ inch across by breaking with a hammer. Examination of the particles indicated two conclusions:

(1) Breakage occured primarily on fracture planes present in the rock; and
(2) Most of the copper minerals (95%) occur on the fracture planes.

The rock, lying on the surface of the dump in one of the leaching solution trenches, had been penetrated by leach solution to a depth of about two inches. The time required for this penetration is not known.

The assumption that copper minerals are available to the action of leach solutions appears to be soundly based. Only the kinetics of the process remain unknown. However, it has been shown that the penetration of fractures by leach solution is relatively fast (Sullivan, 1928).

It has been assumed to date that leach solution is recovered after it obtains its load of dissolved copper because of "the solid, impervious nature of the earth's surface under the dump" (Goodrich). However, not all of the leach solution applied to the dump is recovered. Losses of leach solution as high as 20 percent have been reported. To date these losses are attributed to evaporation.

Techniques for the exact computation of evaporative losses are extremely complex; further, a certain inherent error is introduced due to inability to quantitatively handle the complex and numerous variables present. However, certain estimates may be made to indicate the order of magnitude of the evaporative loss to be expected.

The mean annual temperature at Silver City, 15 miles to the west of Santa Rita, is 53.1°F. The average humidity is 46 percent (Long, 1946). The total water vapor capacity of air at 53.6°F. is 0.000666 pound of water per cubic foot of air (Hodgman, 1962). If it is assumed that one cubic foot of air immediately overlying one square foot of water surface becomes completely saturated in one minute, the loss of water by evaporation can be estimated to be 0.000045 gallon per square foot of water surface per minute at 53.6°F. and a relative humidity of 46 percent. Assume further that 20 acres of water surface are exposed, and that the air over the water is changed completely once each minute. The total water loss by evaporation can then be estimated to be 43.7 gallons per minute.

Another estimate may be made by comparing the losses to the evaporation loss from the open surface of the ocean. The yearly evaporative loss from the ocean surface has been estimated to be 106 centimeters[a] (Sverdrup, Johnson, and Fleming, 1942). This amounts to 0.000043 gallons per square foot of water surface per minute at average oceanic humidity and temperature. Again assuming 20 acres of exposed water surface, the loss of water by evaporation is estimated to be 41.76 gallons per minute.

Losses as high as 20 percent or, at the present rate of flow of 7500 gallons per minute, 1500 gallons per

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[a]- i.e. 106 cubic centimeters per square centimeter of surface).

minute have been reported. Assuming that the average loss is closer to 5 percent, the losses still amount to 375 gallons per minute. It is readily seen that even the lower figure is greater than the estimates noted above (43.7 and 41.76 gallons per minute) by a factor of 10.

The credibility of the estimates may be questioned on several points. However, the agreement between two estimates obtained through two separate lines of reasoning is such that their validity is strongly suggested. Further, the sum of all possible adjustments to the estimates is not likely to alter them by a factor of 10.

Therefore, additional sources of water loss must be sought.

Solutions entering the precipitation plant on April 23, 1963, carried 13.0 pounds of iron per 1000 gallons of solution, while on that date solution recirculated to the dump carried 30.7 pounds of iron per 1000 gallons. Thus, at a plant flow of 7500 gallons per minute, about 127.5 pounds of iron are deposited in the dump per minute. If the deposition is assumed to take place according to the hydrolysis reaction:

$$2\ Fe_2(SO_4)_3 + 18\ H_2O = 2\ Fe_2O_3 \cdot 5\ SO_3 \cdot 17\ H_2O + H_2SO_4,$$

a maximum amount of water is consumed as water of hydration (Morris, 1948). The stoichiometric relationship of iron and water in the reaction is 1.367 pounds of water consumed per 1 pound of iron deposited. Thus an estimated 20.9 gallons of water per minute may be lost.

However, this loss added to the estimated evaporation loss still does not compare with the experienced minimum or maximum losses. Water loss is known to range between 375 and 1500 gallons per minute. Only a maximum of 60 gallons per minute can be accounted for through evaporation and development of hydrated compounds. Therefore, 300 to 1400 gallons per minute are lost through the dump bottom. Assuming that the water carries an amount of dissolved copper comparable to that entering the precipitation plant (20.30 pounds per thousand gallons on April 23, 1963), 7200 to 36,400 pounds of copper are lost through the bottom of the dump each day. This may total 1314 to 6643 short tons of copper per year.

The preceding analysis indicates that water loss far in excess of that to be expected is occurring during the leaching process. The loss may be attributed to flow through the 'impervious' surface beneath the dump.

At Burro Mountain, leaching operations were conducted during World War II by Phelps Dodge Corporation. Water was applied to the area to be leached by spraying. Though the area was supposedly composed of 'solid' rock, enough water flow developed through the rock to support a sizable leaching operation. It is therefore reasonable to suggest that an analogous flow has developed through the surface beneath the dump at Santa Rita.

These considerations indicate that the assumption of an impervious surface beneath the dump is in error. Further, the copper losses are of such a magnitude that further investigation is definitely required.

Chemical Assumptions

The basic chemical assumption in the leaching process is that ferric sulphate in aqueous solution is a solvent for the cuprous sulfide chalcocite. The reaction is thought to take place in two steps:

(1) $Cu_2S + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + CuS$; and

(2) $CuS + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + S$.

Oxidation of pyrite according to the following reactions is assumed to produce the necessary ferric sulphate:

(3) $2\ FeS_2 + 2\ H_2O + 7\ O_2 = 2\ FeSO_4 + 2\ H_2SO_4$; and

(4) $4\ FeSO_4 + 2\ H_2SO_4 + O_2 = 2\ Fe_2(SO_4)_3 + 2\ H_2O$.

The equations may be ordered somewhat differently to permit examination. The initial reaction in the process, a combination of reactions (3) and (4), may be written:

(5) $4\ FeS_2 + 2\ H_2O + 15\ O_2 = 2\ Fe_2(SO_4)_3 + 2\ H_2SO_4$.

This expression indicates the oxidation of pyrite as a one-step reaction.

The next step in the process is the oxidation of chalcocite by the ferric sulphate produced in reaction (5).

The expression indicating the reaction, the sum of reactions (1) and (2), may be written:

$$(6)\ Cu_2S + 2\ Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S.$$

The final reaction, using the products of reactions (5) and (6), may be called the rejuvenation reaction. The reaction, thought to take place in both the dump and the recirculated tailing water, is identical to reaction (4):

$$(7)\ 4\ FeSO_4 + 2\ H_2SO_4 + O_2 = 2\ Fe_2(SO_4)_3 + 2\ H_2O.$$

The sum of reactions (5), (6), and (7) indicate the net chemical reaction assumed to take place in the dump. The sum is:

$$(8)\ Cu_2S + 4\ FeS_2 + 16\ O_2 = 2\ CuSO_4 + 2\ Fe_2(SO_4)_3 + S.$$

Note the products of this reaction; only cupric sulphate, ferric sulphate, and elemental sulfur should exist in the solution collected from the dump.

This is not strictly true, since ferric sulphate will not exist in a neutral environment. Precipitation of hydrolized basic ferric sulphate salts will produce sulphuric acid until the pH of the solution has reached a favorable level for the existance of the remaining ferric sulphate.

Therefore, the solution entering the precipitation plant from the dump should contain cupric sulphate, ferric sulphate, sulfuric acid, and elemental sulfur.

Assays of the solution entering the precipitation plant show considerable ferrous sulphate. According to reaction

(7), all ferrous sulphate in solution should be consumed by the acid produced from oxidation of pyrite and by excess oxygen. The reaction apparently has not progressed to completion. This lack of complete consumption may be attributed to the insufficient amount of one or the other of the reactants. However, both ferrous sulphate and sulfuric acid occur in measurable quantities in the effluent solutions withdrawn from the dump. Therefore, the lacking reactant must be oxygen.

If the analysis just completed is correct, verification may be found in a simple test. It has been stated that the ferrous sulphate content of the solutions is high; and that the anomolous ferrous sulphate content is due to the lack of oxygen to participate in reaction (7). However, the heads sample is taken after the solution has flowed from the dump and been in contact with air for some time, thus permitting the oxidation of ferrous iron and formation of ferric sulphate. Therefore, it is to be expected that solution withdrawn from the base of the dump and kept from contact with the air, thereby forestalling production of ferric sulphate through reactions of the form of expression (7), should exhibit an even higher ferrous sulphate-ferric sulphate ratio. If the ratio is in fact higher, the postulated oxygen deficiency is proved.

Such a test was made on April 23, 1963. A sample was withdrawn from the bottom of the dump by inserting a rubber tube into interstices in the dump and withdrawing solution by gravity flow. The sample was kept from the air by collection in a narrow-neck bottle and plugging the bottle immediately after collection. Assay of the solution was conducted at the mine assay laboratory. The techniques used were those employed in the daily assay of heads and tails samples from the precipitation plant. The results are presented below, with the heads sample taken on that date presented for comparison.

ANALYSIS OF SAMPLES, APRIL 23, 1963
reported in pounds per 1000 gallons

component	A	B
total copper(1)	24.7	21.3
sulfuric acid(2)	6.9	6.0
total iron(1)	11.7	13.0
ferrous iron(3)	7.9	8.2
ferric iron(4)	3.8	4.8
% total iron in ferric state(5)	32.5%	36.0%

A - sample taken from bottom of dump
B - heads sample taken on that date
(1)- by x-ray spectrograph
(2)- by titration with $NaCO_3$ with methyl orange indicator
(3)- by titration with $KMnO_4$
(4)- by subtraction
(5)- computed

The data indicates some variation of the percentage of total iron occuring in the ferric state. However, the difference is small, so that the significance of the data may be questioned.

Examination of the base of the dump in the area that the sample was taken showed that the dump was quite porous.

Oxygen from the atmosphere could easily have penetrated into the dump and promoted the oxidation of iron to the ferric state. However, it is thought that the data does clearly indicate the possibility of verification by such a technique. Future tests should include provisions for drawing the sample from the interior of the dump beyond the influence of atmospheric oxygen.

The insufficiency of oxygen is further suggested by an observed phenomenon. It has been found by experience that the recovery of copper from the dump is facilitated by the alternating application and withdrawal of leaching solution. During the withdrawal period the dump 'drys out' and air is drawn into the interstices. The dump is, in effect, oxygen-saturated. The renewed application of leaching solution to the dump cuts off the atmospheric source of oxygen. Only the oxygen dissolved in the leach solution is added to the dump, and the oxidation reactions necessary to leaching are inhibited. "It would seem, then, that the greater part of the oxidation must take place when the sulphides are merely moist rather than when they are flooded, because then the water could recieve oxygen from the air in contact with it at the same rate at which oxygen was being taken out of the solution by the reactions of oxidation"(Spencer, 1917, page 81).

The principal effect of oxygen deficiency is on rate of recovery rather than on ultimate recovery. Given enough time, oxygen will eventually reach all parts of the dump, and oxidation and leaching will be complete. However, the rate of recovery may be unfavorable for maintainance of an industrial operation.

Tests were conducted at Hurley during 1944-1946 to determine the rate of oxidation of copper-bearing materials. Two beds of concentrates were formed and exposed to the atmosphere. One bed was turned periodically by hand shovel, thus allowing atmospheric oxygen free access to the minerals. It was found that oxidation, and hence leaching, proceeded much more rapidly in the bed that received greater amounts of oxygen through periodic turning.

"It is of interest to inquire whether the oxygen dissolved in rain water could alone have effected the oxidation of the mass of material which has contributed to the secondary copper now contained in any given ore body. This query may be answered in the negative"(Spencer, 1917, page 80). Spencer estimates that for the oxygen dissolved in rainwater to oxidize the copper minerals from the capping over a secondary copper ore body a time comparable to the age of the earth would be required.

If the leaching dumps at Santa Rita are compared to the oxidized capping of a secondary copper deposit, it may be readily seen that the length of time required to effect leaching in the absence of oxygen other than that dissolved in the leach solution is very large.

Additional ramifications of oxygen deficiency are discussed in the literature concerning secondary enrichment of ore deposits.

Lindgren (1933, page 836) notes: "The secondary copper sulphides --- may be deposited at any place in the oxidized zone where there is a deficiency in oxygen and ferric sulphate." Further, "Pyrite of primarily mineralized material has acted as a reducing agent in precipitating copper from surface-derived solutions" (Spencer, 1917, page 102). "The decomposing power (of the surface-derived solutions) appears in general to be almost spent within a shell of material scarcely more than 3 feet thick (immediately overlying the zone of secondary deposition)" (Spencer, 1917, page 83).

The conclusions to be drawn from these statements are suggested in the preceding discussion. It is reasonable to postulate that the principal effect of leaching has been to relocate the copper in the dump; in effect, to create an artificial ore body. The copper recovered through collection of leach solutions may then be taken to represent only a portion of the total amount of copper moved.[a]

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[a]- Note that such an analysis indicates the loss of rather large amounts of copper during the formation of the secondary ore body now being mined at Santa Rita. Consideration of the arguments established here may lead to fruitful areas of prospecting for heretofore unknown ore deposits.

To summarize the chemical assumptions discussed thus far, the following excerpt is taken from Spencer (1917, page 83):

> "The foregoing discussion should make clear the following points: First, that waters from the surface which penetrate a body of porphyry ore (or porphyry dump material) will decompose strongly the metallic sulphides present so long as they contain ferric sulphate. Second, that where chalcocite, pyrite, and chalcopyrite are all present the chalcocite will be largely and perhaps fully decomposed before the other minerals are attacked. Third, that the decomposition of chalcocite, pyrite, and chalcopyrite effects the reduction of ferric salts contained in the solution. Fourth, that the decomposition of pyrite, chalcopyrite, and chalcocite each tends to produce sulfuric acid. Fifth, that the decomposition of chalcocite and of chalcopyrite furnish cupric sulphate to the solution. Briefly, then, when oxygen-bearing waters reach the upper part of the mass of sulphide-bearing rock the consumption of the dissolved oxygen begins at once, and before the waters can progress downward for any considerable distance all this free oxygen is used up in decomposing the sulphides."

Earlier, in the discussion of mineralogical assumptions, it was pointed out that in the future a larger amount of copper placed on the dumps will be contained in the mineral chalcopyrite. Certain comments may be made regarding the effect of the mineralogical change on the chemical assumptions.

Chalcopyrite, reduced to -100,+200 mesh and kept at room temperature, yields only 2 percent of its copper in 43 days when leached with a sulfuric acid-ferric sulphate solution (VanArsdale, 1953). The strength of the solution is unknown.

In the presence of excess oxygen, chalcopyrite is consumed by oxidation:

$$(9)\ CuFeS_2 + 8\ O = FeSO_4 + CuSO_4.$$

If sulfuric acid is present, the consumption of the chalcopyrite may proceed according to the reaction:

$$(10)\ 2\ CuFeS_2 + 16\ O + H_2SO_4 = Fe_2(SO_4)_3 + 2\ CuSO_4 + H_2O.$$

The rates of these reactions are not known; but it can be stated that they will be promoted by the presence of excess oxygen.

On the basis of these considerations it may be stated that the sum of the effects of changing mineralogy and oxygen deficiency will be to reduce the rate of recovery of copper from the dumps if present leaching practice is continued.

It has been assumed that the leaching operation is enhanced by "The added height of the dump through the years, which increases the leaching column and allows the solution to remain in contact with the rocks for long periods of time."(Goodrich).

Examination of the preceeding discussion of oxygen deficiency indicates that, rather than an enhancement, the added dump height is a hindrance to leaching. It appears that the oxygen is consumed very near the surface; and that reactions that reduce the copper content of the solution are taking place during the remainder of the time that the leach solution is in contact with the dump. Further, the added height of the dump makes penetration by atmospheric oxygen more difficult. Thus the leaching process may be impared by excess dump height.

In the future this factor will probably be an even greater drawback, since additional amounts of oxygen will be required to sustain leaching of material carrying a higher proportion of chalcopyrite.

Since 1939 it has been assumed that favorable results have been obtained in the leaching of the dumps by recirculation of tailing water from the precipitation cells. The ferric sulphate and sulfuric acid present in the solution are thought to promote the leaching process.

A typical analysis of tailing water recirculated to the dump is presented below. The assay techniques are the same as those noted earlier.

TAILING WATER ANALYSIS, APRIL 23, 1963
reported in pounds per 1000 gallons

total copper	4.3
total iron	30.7
ferrous iron	28.4
ferric iron	2.3
sulfuric acid	2.1

It may be seen that the amounts of ferric sulphate and sulfuric acid in the untreated tailing water are very small. It is questionable whether or not the benefit gained from recirculation of tailing water outweighs its disadvantages.

As tailing water is distributed across the surface of the dump, oxygen is absorbed by the solution. The result is the oxidation of ferrous iron, forming ferric sulphate. However, since the acid content of the solution is too small to support the existance of ferric sulphate in solution, hydrous ferric salts precipitate according to reactions of the form:

$$(11)\ Fe_2(SO_4)_3 + 6\ H_2O = Fe_2O_3 \cdot 2\ SO_3 \cdot 5\ H_2O + H_2SO_4.$$

These precipitates tend to collect on the surface and seal it against penetration by leach solution. Periodic removal of these precipitates is therefore required.

Such precipitates are not completely removed from the dump. Therefore, when a new layer or bench is added to the dump, the precipitates may tend to form a layer within the dump that is impervious to the flow of leach solution, thus cutting off a part of the dump from effective leaching.

Leaching tests, previously described, have shown the possibility that precipitation of ferric salts may continue throughout the leaching column. Therefore, serious modification of the physical access of leach solutions to the copper minerals may occur.

Other ill effects of recirculation of tailing water may be possible. "The solvent may contain several other acid-soluble elements in addition to copper. There may be iron (both ferrous and ferric), alumina, potassium, sodium and magnesium sulphates, arsenic, chlorine, nitric acid, molybdenum, etc. These may cause harm by building up to an extent requiring their removal by solution discard or otherwise. Other elements, such as arsenic, chlorine, and nitric acid, may be directly injurious." (VanArsdale, 1953).

One of the assumptions used in the leaching operation is that the material of the dump is non-acid consuming. Examination of the mineralogy of the material now being sent to the dump shows this is a good assumption to date.

However, plans for future mining call for the mining of material at Lee Hill that contains an appreciable amount of carbonates. Therefore, the approximation will probably not be valid after material from Lee Hill is placed on the dumps.

Before proceeding further, it may be useful to summarize the conclusions indicated by the analysis of chemical assumptions. They are:

(1) No alteration of basic chemical assumptions are necessary so long as chalcocite remains the primary copper-bearing mineral in the dump;
(2) When chalcopyrite becomes a major copper-bearing constituent of the dump, certain modifications in chemical assumptions will be required;
(3) Deficiency of oxygen in the leaching environment definitely inhibits recovery of copper from the dump, and the retardation of recovery can be expected to increase as chalcopyrite becomes a major constituent of the

dump;

(4) The effect of added dump height may be harmful rather than beneficial, as a consequence of item (3);

(5) Recirculation of untreated tailing water may harmful through precipitation in the dump of basic ferric salts, and build-up of other harmful elements; and

(6) To date the assumption that the dump is non-acid consuming has been a good approximation; but as the carbonate-rich material to be mined from Lee Hill comes to form a substantial portion of the dump, the assumption will become invalid.

Reserve Assumptions

Implicit in the initiation of planning for a new precipitation plant is the assumption that enough recoverable copper is available to support the leaching operation. This assumption must certainly be examined before proceeding farther.

The current leaching operation produces about 2600 short tons of copper per month. Mining operations currently send about 1,100,000 tons of material containing 0.3 weight percent copper (i.e. a total of 3300 short tons of metallic copper) to the dump each month. The present recovery estimates used at Santa Rita assume that only 60 percent of that copper is recoverable (i.e. 1980 short tons). Therefore, 620 short tons of copper per month are recovered from the material placed on the dump before the leaching operation reached its present magnitude.

An estimate of the total amount of copper placed on the dumps is made in the table on the next page. The number of tons of waste per day placed on the waste dump is taken to be 1.5 times the mill capacity during that period. The grade of the material placed on the dump is an estimate based on published or reported cut-off grades for mill feed during the period. The estimate is based on an average 280-day work year. It should be noted that the estimate varies less than 10 percent from values mentioned in conversation with staff members of the Chino Mines Division.

ESTIMATE OF COPPER PLACED IN DUMP

PERIOD	TONS PER DAY TO DUMP	GRADE % CU	TOTAL TONS COPPER
1911-1925	7,500	0.5	157,500
1926-1931	16,500	0.4	110,880
1932-1934(a)	8,250	0.4	27,720
1935-1936(b)	0	---	0
1937-1938(a)	8,250	0.4	18,480
1939-1960	30,000	0.3	554,400
1960-1962(c)	42,300	0.3	71,100
Total copper placed in dump			940,000 tons

(a) - operations curtailed
(b) - operations suspended
(c) - reported present rate of waste disposal

The total copper extracted from the pile is estimated below.

ESTIMATE OF COPPER EXTRACTED FROM THE DUMP

METHOD OF EXTRACTION			TOTAL TONS COPPER
natural loss due to rainfall	(1911-1939)	(a)	11,510
natural loss due to seepage	(1939-1962)	(b)	32,850
production of copper	(to 1956)	(c)	211,220
production of copper	(1956-1962)	(d)	240,000
Total copper extracted from dump			495,580

(a) - based on average yearly rainfall of 17 inches

(Long, 1946), 300 acres of dump area, and water capacity of 8 pounds of copper per 1000 gallons
(b) -based on loss of 300 gallons per minute and water capacity of 20 pounds of copper per 1000 gallons
(c) - from Parsons, 1957
(d) - based on estimated monthly production of 2500 tons

If the current estimate of 60 percent recovery is employed, it is seen that 564,000 tons of recoverable copper have been placed on the dump; and that 495,580 tons of that recoverable reserve have already been extracted from the dump. As noted earlier, 620 tons are required each month from this reserve to maintain the production at the present rate. It can be readily computed that, assuming the correctness of the estimates and continuation of the operation at the present level of production, the life of the operation on a 'status quo' basis is just over 9 years.

The principal value of the estimates just completed is to indicate the order of magnitude of the quantities being dealt with. The estimates indicate the probability that more detailed examination of reserves will show at least 70,000 tons of recoverable copper remaining in the dump.

Chapter VII

INDICATED AREAS OF INVESTIGATION

The preceding examination of the operating assumptions employed in the leaching operation indicate several areas of insufficient knowledge.

Mineralogy

An examination of core or sludge samples from exploratory drilling should indicate the mineralogy of the ores mined or to be mined. The correlation of the examination results with mining records or future mining plans should indicate the composition of the dump at any time past, present, or future.

Drilling Tests

Several possible benefits may be gained from drilling and drill hole examination of the dump.

Some question exists as to the freedom of movement of solution within the dump. Physical examination of the material removed during drilling should confirm or disprove the suggested modification of porosity and permeability by deposition of hydrous iron salts from the leach solution. Such an examination should also reveal the effect of the manner of construction of the dump on the flow of solution.

Should such physical examination fail to reveal the necessary information, modified oil well logging techniques should be effective in determining the porosity and permeability of the dump. These techniques might also reveal something of the nature of the physical-chemical changes accompanying leaching.

Access to the interior of the dump through drill holes should allow examination of the postulated oxygen deficiency mentioned earlier. The magnitude and result of the deficiency should be indicated by physical examination of the material produced by drilling. Postulated 'tertiary' deposition should be readily observed if it is of importance.

If the mineralogy of the dump at the location of drilling is known, the effect of leaching on the copper-bearing minerals may be qualitatively determined. Thus the relative rate of consumption of chalcocite and chalcopyrite may be estimated.

An added benefit of drilling the dump may be found in the possible verification of the reserve estimates made earlier.

Evaporation Tests

One of the more important suggestions made in the analysis of operational assumptions is that a rather large quantity of water, bearing important amounts of copper, may be lost by flow through the bottom of the dump.

The suggestion was based on the estimate that the water losses experienced could not be attributed to evaporation. Therefore, experimental verification of the water losses actually incurred through evaporation must be sought. Several techniques for the determination of evaporative loss are described in the literature of atmospheric science.

Hydrologic Studies

If the estimates made earlier are borne out by the evaporative loss studies just suggested, a hydrologic study should determine if in fact the abnormal water loss is attributable to seepage or flow through the bottom of the dump. The study may include drilling tests in the area of suspected loss.[a]

Any holes drilled in the hydrologic study may be used for the recovery of copper-bearing solutions encountered, if loss of the solution is in fact occuring.

Oxygen Deficiency Investigations

Refinement of the experimental procedure described earlier should confirm or disprove the postulated oxygen deficiency in the dump. Collection of a water sample from the bottom of drill holes may represent the most feasible method of verifying the deficiency.

Kinetic Studies

If the oxygen deficiency mentioned above is verified, the kinetic effect of the deficiency should be determined through laboratory experiment. The results of the laboratory tests should indicate the nature of any engineering

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[a] - Note that the study may produce as a side result indications or techniques for the location of other secondary ore deposits.

development and improvement of the leaching process required to improve the rate of recovery.

Examination of Recirculation of Tailing Water

It has been suggested that the recirculation of tailing water may be detrimental in that it induces modification of the porosity and permeability of the dump by deposition of hydrous iron salts. Tests conducted at the metallurgical research laboratory at Hurley indicate that the addition of acid to the tailing water will aid in the rectification of this condition. It is suggested that such tests, correlated with the results of porosity and permeability examinations permitted by drilling of the dump, be continued until a definite conclusion is reached concerning the value of the addition of acid to the tailing water.

Reserve Investigation

The basis for the establishment of any mineral resource operation is the probable existance of sufficient available material to permit the economic continuation of the operation. Therefore, to ensure the soundness of any future investment or development based on leaching, the estimate of available material made earlier should be verified using the more accurate figures available within the organization. Such an estimate may require verification by drilling.

Recovery Investigation

Present estimates of ultimate recovery state that 60 percent of the copper present in the dump may be recovered.

It is suggested that application of the results of the investigations just outlined may indicate techniques for improving the ultimate recovery of copper from the dump. Further, the investigations should indicate methods for improving the rate of recovery.

By-Product Recovery

To date, no attempt has been made to recover additional products from the copper-bearing solutions. However, a possible important area of production should not be overlooked in the future.

Molybdenum, in the form of molybdenite (MoS_2), is known to occur in the material being sent to the dump. The mineral form of the molybdenum indicates that leaching of the molybdenum should occur in a manner analogous to the leaching of the copper. Further, there is some indication that molybdenum does occur in the solution. Therefore, study of the possibility of recovering the molybdenum should be instituted.

If a technique is developed to halt deposition of iron in the dump, the iron content of the solution should be expected to increase. To prevent the iron build-up from becoming excessive, some technique will be required to remove the excess iron from solution. Electrolytic recovery of the iron from the solution after removal of the dissolved copper is a possible technique for reducing the iron content of the solution, regenerating the acid content of the solution, and producing a salable by-product in the form of high purity electrolytic iron.

Other by-product possibilities should be investigated.

Chapter VIII

ECONOMIC CONSIDERATIONS

The socioeconomic life of any industrial operation is dependent on its ability to compete with like industrial operations. Therefore it is necessary to examine the possibility of maintaining and improving the competitive position of Chino Mines Division as a producer of copper.

A common basis for comparing the competitive ability of operations is the cost per unit of production. However, this information is not publicly available for Chino Mines Division or for a comparable operation. Therefore, the determination of Chino's relative competitive ability in terms of the ability of other operations is not possible through standard data collection methods.

In the absence of available cost data, a measure of the competitive strength of the operation is the comparison between the present estimated cost per unit production with that estimated cost associated with each unit of production under some reasonable improvement or modification of the present operation.

This discussion will therefore proceed to develop the production unit cost under the present operating plan and under a possible modified operation plan. The discussion will be developed within the constraints of the following assumptions:

(1) The rate of production within the division must not be less than that presently maintained;

(2) Any redevelopment or alteration of the present operation must be made in the context of the present technology;
(3) The present facilities must be used as long as economically feasible;
(4) No major capital investment may be permitted; and
(5) Consideration must be restricted to the operations involved in the physical recovery of copper, i.e. milling and precipitation.

Note that the assumptions just listed restrict the discussion to relatively short-run considerations.

Assumptions

The following data were collected from the literature as aids in the estimation of the necessary cost figures.

MILLING COST DATA

MILL	CAPACITY tons/day	FEED % Cu	COST ¢/ton	RECOVERY %	YEAR
A	1,100	1.52	65.21	91.3	1929
B	2,400	3.29	61.73	95.7	1929
C	4,000	2.15	49.30	91.3	1929
D	6,000	2.63	51.30	(a)	1929
E	10,300	1.25	39.90	87.84	1928
F	17,150	0.716	29.67	78.88	1930

A - Superior Concentrator, Plumas County, California (Nelson, 1932)
B - Cananea Consolidated Copper Company Concentrator, Cananea, Sonora, Mexico (Tye, 1930)
C - Copper Queen Concentrator, Bisbee, Arizona (Whittenau and Cramer, 1931)
D - Nacozari Concentrator, Nacozari, Sonora, Mexico (Rose and Cramer, 1930)
E - Hayden Concentrator, Hayden, Arizona (Garms, 1930)
F - Miami Copper Company Concentrator, Miami, Arizona (Hunt, 1930)
(a) - not reported

Note two interesting relationships;

(1) Recovery and grade of feed appear to be directly related; and
(2) Capacity and cost per ton appear to be inversely related

It was assumed that the relationships do in fact hold; and that the data represent a good sample of the available cost data. Therefore it is possible to construct graphs of recovery versus grade of feed, and capacity versus milling cost per ton. If the data and the resulting curves are taken to represent a norm of operation for copper sulphide mills, an estimate of the expected recovery and milling cost per ton may be developed by reading the curve values corresponding to the projected grade of feed and capacity.

Such curves are presented on the following pages. As a check the reported recovery was compared with the estimated recovery for the grade of feed reportedly received at the mill at Hurley. The two values differ by less than one percent.

On the basis of the indicated accuracy of estimates made by this approach, it is assumed that the estimates of milling cost per ton of ore, recovery, and hence milling cost per pound of copper recovered are valid.

Note that the estimate of milling cost per pound of copper obtained is in terms of 1930 costs and prices. To update the estimate to 1963 costs and prices, the following formula is employed:

$$1963 \text{ cost} = 5 \times \tfrac{1}{2}(1930 \text{ cost}) + 2.1 \times \tfrac{1}{2}(1930 \text{ costs}).$$

The formula is based on the assumption that one-half of the milling costs is labor; one-half, supplies.[a] Com-

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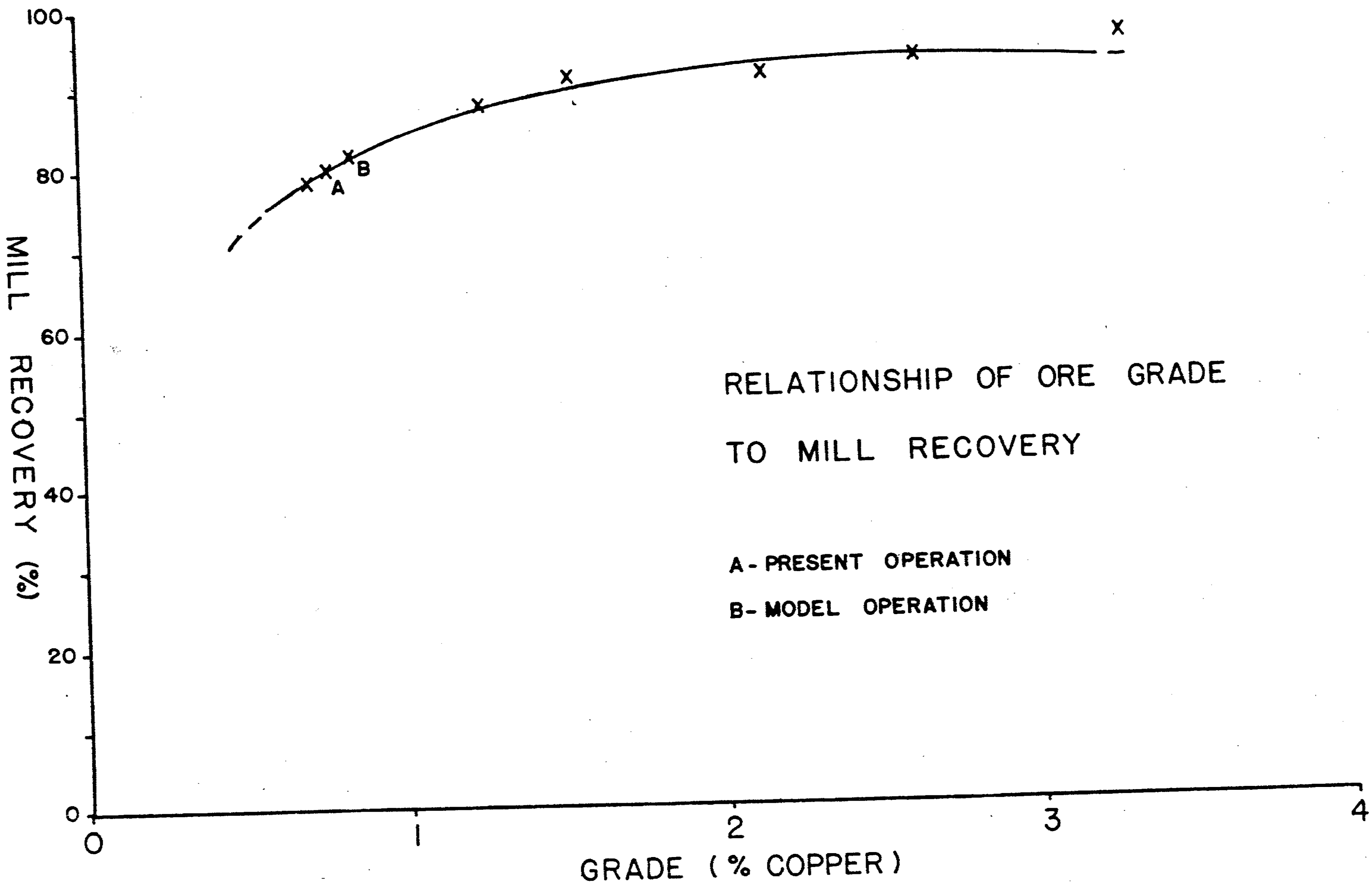
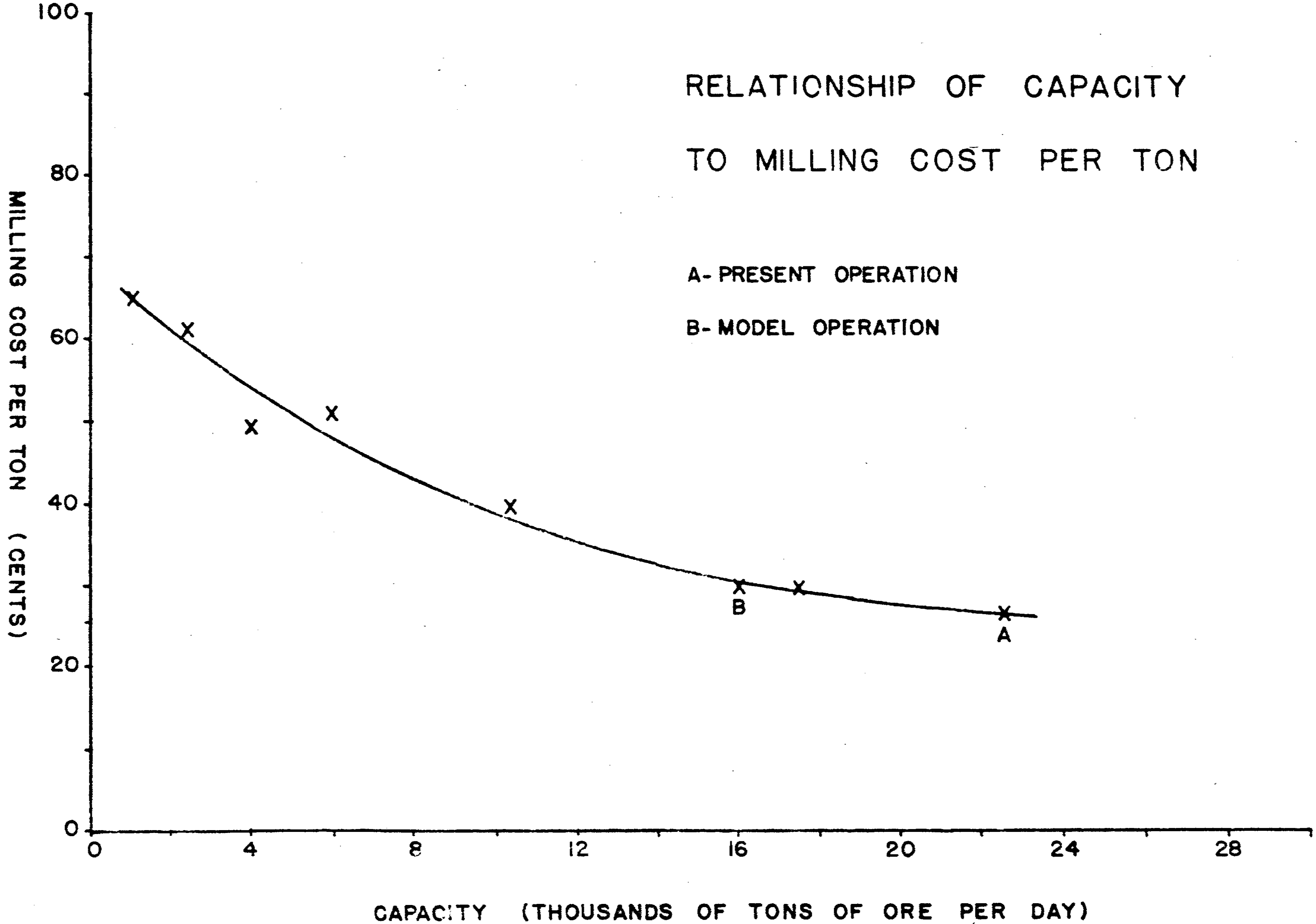
[a] This assumption is an estimate based on data for comparable milling operations. The percent of total cost (continued on following page)


100
80
60
40
20
0
MILL RECOVERY (%)
0
1
2
3
4
GRADE (% COPPER)
A
B
RELATIONSHIP OF ORE GRADE
TO MILL RECOVERY
A - PRESENT OPERATION
B - MODEL OPERATION


RELATIONSHIP OF CAPACITY
TO MILLING COST PER TON
A- PRESENT OPERATION
B- MODEL OPERATION
MILLING COST PER TON (CENTS)
CAPACITY (THOUSANDS OF TONS OF ORE PER DAY)
0
20
40
60
80
100
0
4
8
12
16
20
24
28
A
B

parable day wages have increased from \$4.00 in 1930[a] (Peele, 1941) to \$20.00? in 1963[b]; hence the factor 5. The wholesale price index, the ratio of wholesale prices in the given year to prices in a base year, has increased from 56.1 in 1930 to 119.5 in 1959[c] (U.S. Department of Commerce). If the supplies are purchased at wholesale prices, the increase in supply cost between 1930 and 1963 should be in the ratio of the change in the wholesale price index; hence the factor 2.1.

Present Production Cost

The cost per unit of production may be estimated by using data collected within the Chino Mines Division and cost estimates developed by the technique just described. The cost estimate is shown on the next page.

On the basis of the estimates and calculations made, the cost per pound of copper at a daily production rate of 468,720 pounds is 6.59 cents.

(continued from previous page) attributed to labor at various operations in 1930 is:

Copper Queen	31.44%	(Whittenau and Cramer, 1931)
Superior	40.08%	(Nelson, 1932)

Taggart (1927, page 866) estimates labor cost to be 30 to 50 percent of total flotation milling cost.

a- Miner's daily wage at Ground Hog Mine, Vanadium, New Mexico. Represents a maximum 1930 daily wage for mill personnel other than supervisory staff.

b- Posted contract wage rate for mill personnel; includes some fringe benefits.

c- 1947-1949 = 100

ESTIMATE OF PRESENT PRODUCTION COSTS

MILLING		
Daily tonnage capacity	22,500	tons/day
Grade of feed	0.78	% Copper
Milling recovery	80	%
Cost per pound copper(a)	7.67	¢/pound
Total daily recovery(b)	280,800	pounds
Total daily cost(b)	$21,537	
PRECIPITATION		
Plant flow	7,500	g.p.m.
Copper content of water	20	lbs/1000 gal.
Precipitation recovery	87	%
Total daily recovery(b)	187,920	pounds
Cost per pound copper(c)	5	¢/pound
Total daily cost(b)	$9,396	
TOTAL		
Daily recovery(b)	468,720	pounds
Daily cost(b)	$30,933	
Cost per pound copper(b)	6.59	¢/pound

(a) - estimated
(b) - computed
(c) - from Parsons, 1957

Modification Assumptions

To examine the strength of the present economic situation at Chino, it is necessary to examine the possibility of improving the cost configuration of the operation within the constraints imposed earlier. Because of the conditions noted earlier, it is assumed that the only reasonable improvement possible must be a shifting of relative emphasis between milling and precipitation.

To examine the possible effect of such an emphasis shift on the cost situation, it is necessary to make some assumptions.

They are:

(1) Precipitation recovery can be expanded to compensate for a relative cut-back in mill recovery, i.e., production can be maintained at a constant level; and
(2) The cost of production by precipitation can be maintained at 5 cents per pound regardless of production rate.

Further, some estimate of the ore grade and quantities shipped to the mill must be made. That estimate is presented below. Note that the average grade of such a day's shipment is 0.777 percent copper, while the average shipment grade actually experienced is 0.78 percent copper. The reliability of the estimate can be considered very good.

ESTIMATED GRADE OF ORE SHIPMENTS

INTERVAL[a]	AVERAGE[a]	TONS PER DAY
(b) -0.65	0.6	6,500
0.65-0.75	0.7	5,250
0.75-0.85	0.8	4,000
0.85-0.95	0.9	3,000
0.95-1.05	1.0	2,000
1.05-1.15	1.1	1,000
1.15-1.25	1.2	500
1.25-1.35	1.3	250
	Total shipments	22,500
	Average grade	0.777[a]

(a) - percent copper
(b) - cutoff grade

Modification Model

In the modification model it is assumed that the cut-off grade of mill feed is raised to 0.65 percent copper; and that mill capacity is consequently cut to 16,000 tons per day. The model takes account of all assumptions noted before; and all estimated values are arrived at in the manner described. It has further been assumed that the ore mined will continue to conform to the estimate just made; hence, the 16,000 tons of ore per day that are assumed to be sent to the mill will average 0.85 percent copper.

The estimated cost associated with the modification model, as well as the other salient data, are presented below.

ESTIMATE OF COSTS ASSOCIATED WITH MODIFICATION MODEL

MILLING	
Daily tonnage capacity(a)	16,000 tons/day
Grade of feed(b)	0.85 % copper
Milling recovery(c)	82 %
Cost per pound copper(c)	7.775 ¢/pound
Total daily recovery(b)	223,040 pounds
Total daily cost(b)	$17,341
PRECIPITATION	
Plant flow(b)	10,000 g.p.m.
Copper content of water(a)	20 lbs/1000 gal.
Precipitation recovery(a)	87 %
Cost per pound copper(a)	5 ¢/pound
Total daily recovery(b)	245,680 pounds
Total daily cost(b)	$12,284
TOTAL	
Daily recovery(b)	468,720 pounds
Daily cost(b)	$29,625
Cost per pound copper(b)	6.35 ¢/pound

(a) - assumed
(b) - computed
(c) - estimated

Comparison with the estimate of present production cost made earlier shows that the following advantages may be gained if the modification is instituted:

(1) An overall cost saving per pound of copper produced of 0.23 cent may be developed; and

(2) At constant production a daily saving of $1,308 may be enjoyed.

If mill through-put is decreased to 16,000 tons of ore per day, it is estimated that recovery will increase to 82 percent, while cost per pound of copper is estimated to be 7.775 cents. Increase in precipitation recovery of 57,760 pounds per day will allow daily total production to be maintained. The daily cost saving in physical recovery of the copper is estimated to be $1,308.

Additional Advantages

The mill at Hurley was originally designed to operate at a smaller capacity and higher grade than present operations now make available (Parsons, 1957). A return to conditions more nearly conforming to the design requirements of the mill may introduce recovery and cost improvements in excess of those estimated. Further advantage may be introduced by the reduction of wear on the equipment, thus extending the life of the equipment and reducing maintainance costs.

An advantage that has not been valued is the reduction in heat requirements in the smelting operation. The cement copper recovered by increasing the precipitation production is more desirable for smelting than the concentrate produced by the mill.

Freight Saving

An additional saving is introduced by the modification in that freight costs are reduced.

It is reported that the freight bill for transfer of ore from the mine via the Atchinson, Topeka, and Santa Fe Railway to the mill was $3,000,000 to $4,000,000 in 1962. Assuming that 6,750,000[a] tons of ore were shipped during the year, and that the lower freight figure is correct, the cost of transporting the ore may be computed to be 44.44 cent per ton. A daily reduction in shipments of 6,500 tons can therefore be taken to indicate an estimated daily saving of $2,888.

Additional Production

Thus far the estimates of saving have been based on mill operation days. The mill currently operates six days a week, while the precipitation plant currently operates seven days per week. Therefore, the added precipitation production ability required by the modification would be in operation approximately 60 days each year in excess of mill operation. An added production of 57,760 pounds for 60 days amounts to an added yearly production of 3,465,600 pounds of copper. Assuming that the value of copper to the Chino Mines Division, after deduction of production costs, is 5 cents per pound, the added production would mean an

[a] - based on daily shipment of 22,500 tons and a shipping year of 300 days

added yearly income of $173,280. Based on a mill operation year of 300 days, the added income would amount to $578 per day.

Loss of Molybdenum Production

It is reported that the majority of the molybdenum occuring in the Chino ores is associated with low copper content. By sending 6,500 tons of the lowest grade mill ore to the waste dump instead of to the mill, a considerable reduction in molybdenum recovery may be encountered.

Recent production figures indicate that the mill recovers about 1,100,000 pounds of molybdenum concentrate valued at about 40 cents per pound each year (Anderson, 1957). Based on a 300-day mill year, this indicates a total value of daily production of $1,467. If it is assumed that one-half of the production is lost because of the change in mill feed, the daily loss amounts to $734.

Daily Cost Saving

The estimates of cost savings and income gain and loss just completed may be totaled to show the estimated daily saving introduced by the modification model.

ESTIMATED DAILY SAVING

ITEM	DAILY VALUE
Recovery saving	$1,308
Freight saving	2,888
Additional recovery	578
Gross daily saving	$4,775
Less molybdenum loss	734
	$4,041

Cost of Modification

If the modification used in the model is employed, an expansion of the precipitation facility will be required. The cost of the expansion must be compared with the expected saving to arrive at a decision to employ the modification. It is therefore necessary to make some estimate of the cost of expansion.

The following estimate is hypothetical. However, it is thought that the estimate is conservative; i.e., that the cost expansion is overestimated.

ESTIMATED COST OF EXPANSION PER 1,000 G.P.M. CAPACITY

ITEM	COST
Precipitation facility	$100,000
Pumping facility	40,000
Piping	24,000
Miscellaneous	36,000
Total per 1,000 g.p.m.	$200,000

Earlier it was estimated that an additional daily production of 57,760 pounds of copper would be required from precipitation. Assuming a water capacity of 20 pounds of copper per 1,000 gallons of water, and a recovery of 87 percent, an expansion capable of handling 2,300 gallons per minute is required. Thus an expansion cost of about $500,000 is required.

Alteration of Reserve Situation

In Chapter VI a reserve life of 9 years was computed for the precipitation operation. The present operation requires about 7,440 tons of copper per year from the 'reserve' in the dump. The expanded operation forming a part of the modification model will require additional amounts of copper from the reserve.

Each day of mill operation under the modified plan will require the net recovery of 10,960 pounds of copper from the reserve now in the dump. This is computed on the basis of the addition of 6,500 tons of material containing 0.6 percent copper to the dump, 60 percent recovery by leaching, and a gross increase in recovery of 57,760 pounds of copper per day. In addition, each day that the mill is down but the precipitation plant is in operation, a drain of 57,760 pounds of copper will be placed on the reserve. Assuming that the mill operates 300 days a year and the precipitation plant operates 365 days per year, the net added yearly drain on reserves will be 3,521 tons.

The total yearly consumption of reserves by the present operation and the postulated expansion will amount to 10,961 tons. Earlier it was estimated that the recoverable reserves amounted to 68,420 tons; thus the indicated reserve life of the modified operation will be 6.24 years.

Possible Value of the Modification

An estimated saving of $4,041 per mill operating day has been postulated. If the mill operates 300 days per year, the possible yearly saving amounts to $1,212,300.

Reserve calculations show that such a saving might be enjoyed for about 6 years. The present value of an annuity of $1 for 6 years at 10 percent interest[a] is $4.3553 (Parks, 1957); therefore the present value of such a saving, when considered as an annuity, if $5,259,930. The investment required to secure such an annuity has been estimated to be $500,000. Subtracting the investment required from the present value of the annuity it can be shown that the possible value of the modification is $4,759,930.

Economic Position

If the assumed plan of modification is put into operation, the relative economic position of Chino Mines Division will be improved by reducing the cost of copper production. Any technique for improving the competitive strength of an operation may be important. If not employed immediately, the technique should be kept under consideration for use at such time as the market situation should require.

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a. The interest rate for money forming the capital of a corporation is assumed to be 10 percent, based on reported common practice (Parks, 1957).

Chapter IX

LONG-RANGE DEVELOPMENT

The possible modification of the physical recovery of copper discussed in the preceding chapter represents a short-run or immediate development. This chapter is concerned with a possible pattern of long-range technical and socioeconomic development of Chino Mines Division.

Definition

The principle characteristics of a long-range development are high probability of occurrence, but reasonable uncertainty as to the date of actual implementation of the development. The long-range development plans will be implemented when the accumulated physical or socioeconomic obsolescence makes it impossible for the present copper-producing facility to maintain sufficient competitive strength.

Physical obsolescence may be described as the wearing out of the equipment of the operation. The accumulated physical obsolescence will be indicated by rising maintainance costs; when the cost of maintainance becomes prohibitive, it can be stated that physical obsolescence has overtaken the operation. Hence a new development will be required to continue production of copper.

Socioeconomic obsolescence may be defined as the loss of necessary competitive strength through social or economic change. At any time that the cost of production rises as a result of external events[a], or market return decreases so that the necessary margin between outgo and income is not maintained, it may be said that the operation is obsolete in a socioeconomic sense.

Environment

Before any statement is made concerning the possible long-range development of the physical recovery of copper at Chino Mines Division is made, something must be known about the environment in which the development will take place.

The principle use of copper is in the electrical industry, absorbing about two-thirds of the total output (Parsons, 1957). "The industry has been expanding at a tremendous rate and, no doubt, will continue to expand, not only in the United States and other industrial countries but also, to an increasing extent, in the so-called 'backward' countries. Barring some unforeseen technologic development, copper will be indispensable for generating, distributing, and utilizing electric energy" (Parsons, 1957). Therefore, an expanding demand for copper can be expected.

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[a]- Such external events would include increased supply cost or increased labor costs. These external events represent the changes in cost that are not under internal management or engineering control.

However, the 'unforseen technologic development' has taken place. Aluminum wire, sometimes copper-coated, is now gaining acceptance as a material for use in electrical power distribution. The material has favorable physical characteristics for such use, and is additionally recommended by its lower price.

The laser, a device for the generation of coherent light beams, presents a possible technique for the transmission of electrical power that requires no continuous electrical connection. It is quite possible that within the next few years power transmission lines will become obsolete through application of laser technology.

The postulated expansion in demand for copper will certainly be modified by technologic developments such as these just described. In effect, after decades of enjoying a "natural" monopoly, copper will be placed in cost competition with other materials; and copper technology will be placed in competition with other technologies. If copper is to retain a competitive position, its price will probably have to decrease relative to present levels to ensure an increasing volume market.

On the other hand, the cost of production can be expected to rise. The probable increase[a] in labor cost alone will

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[a]- The change in national average day wage paid in the metal mining industry has been about 200 percent in the past thirty years. Projections of this trend indicate the upper limit of labor cost increase.

be sufficient to offset any cost-cutting developments of the present production technology. Further, the probable trend of supply cost is toward higher expense in servicing and maintaining such operations.

To restate the conclusions:

(1) The market for copper in traditional use areas can be expected to increase;
(2) Competition from other materials and technologies can be expected to increase;
(3) To compete for a share of the market, the price of copper will probably have to be reduced;
(4) The cost of production can be expected to rise, so long as the present methods of recovery continue to be used;
(5) Copper producers can expect an increasingly severe cost-price squeeze unless long-range planning and development can introduce economies in production costs.

Principles

Principles for long-range development are:

(1) In any industrial process it is desirable to minimize handling of the materials involved, and to maximize the ease of handling;
(2) Maximum recovery, and consequent minimum expense due to loss, should be achieved;
(3) It is desirable to obtain the most valuable product possible consistent with the socioeconomic situation;
(4) The simplest process consistent with the foregoing principles is the most desirable.

Assumptions

The long-range development plans here imposed will be restricted to the physical recovery of copper.

It will be assumed at the outset that milling as a technique for the large scale physical recovery of copper will at some point in the future become impractical due to obsolescence; and that long-range development plans will be based on leaching.

The following assumptions are made concerning the Chino Mines Division:

(1) The necessary monies will be available when redevelopment is required;
(2) The copper content of the material mined will decrease on a long-term basis; and
(3) The mineralogy of the copper-bearing material will change, so that chalcopyrite becomes the most important copper-bearing mineral[a].

It will further be assumed that the research outlined in Chapter VII has been conducted, and that the following conclusions have been reached:

(1) Deposition of hydrous ferric iron salts from leach solution may modify the porosity and permeability of the material being leached, but deposition of the salts can be halted by addition of acid to the leach solutions;
(2) Loss of water flow through the bottom of the dumps is in fact occuring, but may be halted by adequate preparation of the dump site;
(3) Artificial addition of oxygen to the material being leached and to the leach solution will improve both the kinetics and the ultimate recovery of the leaching process;
(4) The leach solutions do carry appreciable amounts of material whose recovery is desirable in that it will permit byproduct income.

Areas of Development

Three probable areas of development of the leaching process exist. They are: handling of the material to be leached; the leaching process itself; and the recovery of copper and other products from the leach solution.

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[a] - see Chapter VI

Material Handling

It is assumed that the open pit mining of the Santa Rita ore deposit will continue, and that the dump will increase in size. Further, by assuming that leaching will take the place of milling, it has been implied that all of the material mined will go to the dump.

The first possible development in material handling that comes within the realm of physical recovery of the copper is the preparation of the dump site before material is placed on the dump. Such preparation could end the possibility of loss of copper-bearing water through the bottom of the dump and aid in the leaching process itself.

Water loss through flow through the bottom of the dump may be halted by sealing the dump site in some manner. Such sealing could be accomplished by oiling the ground surface, or by covering the surface with an asphalt pad. However, both techniques have a common failing: settling of the dump material and compaction of the underlying ground would tend to break the effective seal of the material.

Several tests using plastic sheets to stop water loss have been conducted in recent years. After the site is leveled, the sheet is laid down and covered with a thin layer of dirt to hold it in place. The technique has proved most effective in reducing water loss; and is apparently unaffected by loading. The plastic stretches and expands to conform to irregularities introduced by settling and compaction. Site preparation will probably employ such a technique to stop water loss in the future.

A second possible development in site preparation is the laying of concrete or other high-strength low-cost pipe to aid in the leaching of the dump. If such pipe is perforated, and air or oxygen is forced into the pipe after it is covered by the dump, leaching will be promoted.

Earlier it was noted that most of the copper-bearing minerals occurred in fracture planes present in the rock. Further, it was noted that the time of penetration of a rock mass by leaching solution, and hence the time of leaching, is a function of the size of the rock mass. Consideration of these two facts leads to a possible development of material handling.

The fracture planes present in a material represent planes of weakness. If the material is crushed, it will tend to break along the planes of weakness. Since the planes of weakness in the Santa Rita ore material are also the principle sites of copper mineral occurence, crushing of the mined material would tend to expose the minerals to leaching. Further, the decreased rock mass size would promote the rate of leaching of copper minerals within the mass. Thus a kinetic advantage in leaching could be obtained through physical preparation of the material.

Assume that crushing to a certain size exposes most of the copper minerals on the surface of the rock fragments. If the fragments are then subjected to abrasion, the copper minerals would tend to be removed as sand-sized material. Screening of the crushed and abraded material would then tend to produce a physical separation of the copper mineral and the barren rock. The oversize from the screening operation, composed of relatively barren rock fragments, would be sent to the dump for leaching; while the undersize would be sent to a controlled leaching process to be described later[a].

The sizing of the material would aid in leaching of the dump. It is well known that the porosity and permeability of a mass composed of particles of hetrogeneous size. The crushing and sizing would tend to improve the porosity and permeability of the dump, thus improving the rate of leaching.

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a - Note that the operation just described could be applied to the present operation to increase the grade of mill feed. Material mined could be crushed to expose the copper minerals; abraded by rolling in a drum apparatus or passing through a chute; and screened to obtain a physical concentrate. Oversize would be placed on the dump and leached; undersize, sent to the mill for further concentration. Advantage should be obtained in freight and grinding saving, as well as in the increased recovery attributable to increased grade of mill feed.

The possible long-range developments in material handling discussed above are:

(1) Preparation of the dump site by;
 (a) Sealing the dump site with plastic sheeting to prevent water loss;
 (b) Laying pipe on the ground surface before formation of the dump to allow introduction of oxygen during leaching;
(2) Crushing of the material to permit faster penetration by leach solution;
(3) Physical separation of the copper minerals from the barren rock by crushing, abrading, and screening.

Leaching

It has been assumed that the results of research will show that the kinetics and ultimate recovery of leaching can be improved by the artificial addition of oxygen. The oxygen may be added to the material being leached, or to the leach solution.

If perforated concrete pipe is placed beneath the dump, as suggested in the preceding section, oxygen may be applied directly to the dump by forcing air into the pipe and through the perforations. Such application of the oxygen at the base of the dump would also aid in the elimination of 'tertiary' deposition of copper suggested earlier.

Oxygen may be added to the leach solution by passing the solution over a series of riffles. Agitation of the solution in the presence of atmospheric oxygen will allow the absorption of oxygen for use in the leaching reactions.

Addition of sulfuric acid to the leach solution is assumed to be necessary to forestall the deposition of hydrous ferric salts that prevent effective physical contact between the leach solution and the material to be leached.

The addition of acid to leach solution, in the presence of sufficient oxygen, will have an additional beneficial effect. Ferrous sulphate, produced by the leach reactions, is converted to ferric sulphate, a desirable reagent for leaching, by reaction with oxygen and sulfuric acid. Thus the leach solution is maintained as an effective solvent for the copper minerals chalcocite and chalcopyrite by the combined addition of oxygen and sulfuric acid.

As noted earlier, proper preparation of the material will tend to promote physical access of the leach solution to the material. Improved access will promote both faster and greater recovery of copper by leaching.

Long-range developments in the leaching process will therefore probably consist of:

(1) Addition of oxygen to dump and leach solution;
(2) Addition of sulfuric acid to leach solution; and
(3) Crushing of material to improve physical access of leach solution to the copper minerals.

Recovery of Copper From Leach Solution

Recovery considerations will be restricted to examination of cementation and electrolytic recovery techniques.

Other methods of recovery, such as the hydrogen sulphide method (Greenawalt, 1912), sulphur dioxide method (Newton and Wilson, 1942), lime method (Newton and Wilson, 1942), and the ammonia method (Liddell, 1926), will be ignored because they violate one or more of the principles of long-run development mentioned earlier.

Conditioning of Solution

The first matter to be considered is the conditioning of the solution so that it is amenable to processing by either cementation or electrolytic methods.

Both methods of recovery are aided by the removal of ferric iron compounds from the solution. In addition, the acid content should be reduced if the solution is to be treated by the cementation method.

If the mined material is treated as suggested in considering the handling of materials, the technique of conditioning described earlier could be applied. A concentrate containing copper minerals would be placed in tanks and treated with solution withdrawn from the dump. The acid and ferric sulphate in the solution would react with the concentrate and be reduced, while the copper content of the solution would be increased. If air were introduced through the bottom of the tank and allowed to bubble through the concentrate, the reaction would be stimulated, the sulfuric acid and ferric sulphate content of the solution would be reduced, and the copper content of the solution would be greatly increased. Thus the solution would be prepared for cementation.recovery of the copper.

If the copper is to be recovered by electrolytic methods, a high acid content is desirable. If the solution is conditioned by the method just described, without the attendant introduction of oxygen, the ferric sulphate content of the solution will be reduced without altering the acid content. Thus a solution suitable for electrolytic recovery of copper would be developed.

Note that the method of solution conditioning just described is analogous to tank leaching of copper ore as practiced at Chuquicamata and Inspiration (Newton and Wilson, 1942). The conditioning of solution is, in effect, tank leaching of a concentrate. Therefore, examination of tank leaching practice should supply guides for the physical design of the conditioning installation. However, it must be remembered that the chemistry of the ope ation is vastly different since the copper is in the form of sulphide rather than the oxide usually leached by the tank method.

Cementation

One of the principle difficulties with the present cementation process is the separation of the precipitated copper from the metallic iron. Any proposed improvement or development in the cementation technique should include a more satisfactory copper recovery characteristic.

An additional problem in cementation recovery is the supply of a suitable precipitant. Tin cans, the precipitant presently used, are being replaced in consumer use by plastic and paper containers. Since this trend in the container market can be expected to continue, the difficulty in acquiring sufficient amounts of tin cans can be expected to increase.

Sponge iron has been proposed as a substitute precipitant, because of its desirable quality of large surface area for reaction. However, a supply of this precipitant sufficient to meet the demands of cementation recovery has not been assured.

Since the combined supply of the two precipitants will probably be available in the quantities required, it is desirable that any proposed development be capable of using the two precipitants interchangeably.

The conditions noted above lead to the consideration of two techniques for cementation processing of copper-bearing solutions: the vertical column method, and the cone method.

In the vertical column technique, the precipitant is charged to a vertical container such as a mine raise or concrete column, and water bearing copper in solution is passed through the column. Tests of the technique at

Cerro de Pasco indicate that a column 65 feet high and 12 feet in diameter will handle 1,000 gallons of solution per minute with a copper recovery of 95.7 percent. "The vertical column arrangement has much to recommend it in theory but some definite draw-backs could result in practice due to compaction of a long column of (precipitant) leading to inadequate liquor distribution" (Jacobi, 1963, page 6).

If the solution were introduced by reverse flow, i.e. from the bottom of the column, the upward-flowing solution would decrease the tendency of the precipitant bed to pack, in effect agitating the bed. Since the iron at the bottom of the column would be consumed first, a continuous process could be developed by withdrawing metallic copper from the bottom of the column while adding precipitant to the top. Additional advantage would be obtained by the filtering action of the upper portion of the precipitant bed; it would tend to retain the smaller particles of metallic copper, thus eliminating loss in the overflow.

The cone method of cementation recovery of copper employs an apparatus consisting of an inverted cone constructed of suitable material. Copper-bearing solution is admitted at the apex of the cone while precipitant is charged through the top of the reaction area. The operation may be conducted in two ways.

If the precipitant is tin cans, the water is allowed to overflow the cone. The finely divided metallic copper, precipitated by the reaction between the solution and the metallic iron, is suspended in the overflowing water and removed from the water by settling.

When sponge iron is used as a precipitant, a different mode of operation is required. If sufficient sponge iron is added to the cone to form a semi-solid or quicksand bed, the action in the cone will be analogous to that in a heavy media separation unit. The copper precipitate, being more dense than the sponge iron, will be concentrated at the bottom of the cell by the agitating action of the in-flowing solution. Finer particles of copper, suspended in the upwelling water, will be filtered by the upper portion of the sponge iron bed and held until they grow by accretion to a size that will settle in the cone. Overflow water will carry a minimal amount of suspended material, thus requiring a smaller settling capacity. Copper is recovered by withdrawal from the bottom of the cone, while the unit is maintained in continuous operation.

Such a method of recovery is now being examined at the precipitation plant of the Chino Mines Division. However, the method of operation when using sponge iron is somewhat different.

Since no provision has been made in the experimental cone for the removal of copper from the apex of the cone, it has been found necessary to add the sponge iron in a quantity just sufficient to precipitate the copper in the inflowing solution so that continuous operation may be maintained. The copper is collected by passing the overflow to a settling area.

This plan of operation has shown several difficulties. The process control required to permit the addition of just the correct amount of sponge iron is both expensive and delicate. Further, if too much sponge iron is added, the copper precipitate is contaminated by the excess sponge iron carried to the copper settling area by the water; while if too little sponge iron is added the recovery of copper is incomplete.

It is here suggested that this method of operation is inadequate; and that the pilot facility should be modified to examine the plan of operation that calls for use of a sponge iron bed.

Both of the cementation techniques just described may employ several units in series to strip the solution of its copper content, the number of units being controlled by the recovery desired. They have the further advantage of high capacity and continuous operation. Perhaps the greatest advantage of both is the ability to use either sponge iron or tin cans as the copper precipitant.

Production of Iron Precipitant

It is desirable to investigate the possibility of internal supply of iron precipitant for the cementation recovery of copper in the event that external sources of supply should fail.

Since it has been assumed that no mill will be included in long-range development plans, recovery of a source of iron from the mill is eliminated as a possibility. However, iron ore is available in the immediate area; this ore could be converted to a usable precipitant by gaseous reduction techniques employing natural gas, if no other feasible source of iron is available.

However, it was earlier stated that addition of acid to the leach solution would prevent the deposition of iron from the solution during leaching. Thus iron resulting from the oxidation of pyrite and consumption of chalcopyrite will remain in the circuit and build up to possibly harmful levels unless otherwise removed. A process that would both purify the solution by maintaining a desirable iron content and produce a suitable precipitant would be most advantageous.

Electrolytic recovery of iron[a] from leach solution stripped of its copper content provides the most probable solution of the problem. It is theoretically possible that, if the solution is completely stripped of its copper content, the resulting iron will be of a high purity,

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[a]- According to the Cathode reaction:

$$Fe^{++} + 2\,e^{-} = Fe \text{ (metallic)}$$

possibly suitable for sale to outside consumers. The return from the marketing of the high purity, and therefore high value, iron could defray the expense of purchasing sponge iron or tin cans outside the organization.

If the iron contains copper or other impurities it may be reduced to pellet form by melting and granulation through spray cooling; or shearing and cutting may be used to prepare a suitable precipitant for charge to the copper recovery units. This procedure would ensure the recovery of any copper impurity in the iron.

In any event the amount of iron recovered should be in excess of that required for use as a precipitant. Thus a marketable by-product should be recovered to add to the income of the operation.

An additional advantage in the electrolytic recovery of iron from the leach solution is the generation of sulfuric acid at the anode[a].

The reaction should generate enough acid to fulfill the requirements of the leaching process, and thus eliminate the need of outside acquisition of acid. After the solution has passed through the iron recovery process, it is returned to the dump, enriched in sulfuric acid, and is employed in the leaching process once again.

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[a]- Acid is generated by the electrolysis of water at the anode:

$$2\ H_2O = 4\ H^+ + O_2(\text{gaseous}) + 4\ e^-$$

$$2\ H^+ + SO_4^{--} = H_2SO_4$$

Preparation of Cement Copper

The copper product of the cementation process is a fine-grained metallic copper containing appreciable amounts of water. The excess water has heretofore been removed by drying in the open air on drying pads adjacent to settling tanks. The copper has then been sent to reverberatory furnaces or convertors for final processing.

Copper in this form is undesirable for three reasons. First, such copper requires an installation for final processing. Second, "tests on drying cement copper showed, when slowly dried to 15 percent moisture, that 60 to 70 percent of the copper content had been oxidized" (Liddell, 1926, page 1075); thus requiring a second reduction. Last, the copper thus produced is subject to excessive flue gas loss when charged to a reverberatory furnace or convertor.

If the copper were fed to briquetting presses, a form not so subject to flue gas loss would result. Further, the water content of the copper would be greatly reduced by mechanical means, thus resulting in a heat saving during the smelting process. Finally, the oxidation and consequent re-smelting of the product would be avoided. Thus two of the three objections noted above would be avoided.

Another technical ramification of such a process is the direct production of anodes for electro-refining. If the cement copper contains few impurities it could be pressed into anode form for direct shipment to and use in a copper refinery. A considerable saving would result from the elimination of re-smelting and conversion of crude cement copper.

New developments in extrusion forming of metal product shapes suggest a direct use for high-purity cement copper that would avoid all of the objections to cement copper noted before. If a product of suitable purity could be obtained by cementation recovery, its particulate nature would be desirable in extrusion forming of various items.

Electrolytic Recovery

"The most widely practiced method for the precipitation of the copper dissolved in leaching is by electrolysis, using insoluble anodes. This method has one outstanding advantage over all other precipitation methods --- it yields directly a product (cathode copper) which is of the same quality as the cathode produced by electrolytic refining.

"In some respects the process resembles the electrolytic refining of copper --- the tanks and electrical connections are similar to the multiple refining process, the same type of starting sheets are used, and the finished cathodes are treated in the same way as cathodes from electrolytic refining. There are, however, several fundamental differences between the two systems. Briefly, these differences may be summarized as follows:

"1. Insoluble anodes are used, and there is no appreciable corrosion of them, and hence no 'anode mud' is formed.

"2. The copper in the electrolyte comes from the leaching plant, which is in closed circuit with the electrolytic cells, and because there is no copper dissolved from the anodes, the electrolyte becomes depleted in copper and its free acid content increases as it passes through the tank house. In the leaching cycle the opposite effect is found --- acid is used up and more copper is dissolved.

"3. Current efficiency is generally lower, voltage is much higher, and the power consumed per pound of cathode copper is much greater. The cathode current density is less than that used in refining.

"4. Concentrations of dissolved copper and free acid in the electrolyte are generally less than in refinery electrolytes, and the resistance of the electrolyte is greater" (Newton and Wilson, 1942, pages 321-322).

The copper contained in the solution or electrolyte is usually not completely deposited, because as the solution becomes more deficient in copper, the efficiency of the operation becomes lower. In general practice the dilute solution or 'spent electrolyte' is either stripped of the remaining copper content by cementation; or is recirculated into the leaching process on the assumption that the copper will be ultimately recovered.

Leaching as it will probably be practiced at Santa Rita introduces one aspect not commonly encountered in general practice: the high ferric sulphate content of the leach solution. However, a technique for reducing the ferric iron to ferrous state has been described earlier; and "ferrous iron is generally considered harmless" (Newton and Wilson, 1942, page 327).

Note that the solution conditioning process to be employed should be that process described earlier that does not consume sulfuric acid. Sulfuric acid is a desirable constituent of electrolyte because it increases the conductivity of the solution, and hence the efficiency of the process.

Another method of conditioning the solution is the reduction of ferric iron on cement copper. This technique would have the advantage of reducing the ferric iron content of the solution while enriching the solution in copper and thereby increasing the efficiency of the electrolytic recovery method.

The technique just described would necessitate the production of cement copper. The cement copper could be recovered by stripping the 'spent electrolyte' of its remaining copper content before recirculation to the leaching dump.

If the cementation process just described produced cement copper in excess of that required to condition the solution, pressed anodes could be formed by the method described earlier. The anodes could be used in the electrolytic recovery cells in place of the insoluble anodes commonly used in practice; thus enriching the solution in copper, promoting the efficiency of the electrolytic recovery of copper, and avoiding the problems that have been commonly associated with the use of insoluble anodes.

Two recent and relatively unused developments may aid in developing a usable electrolytic recovery process. They are electrodialysis and ion exchange.

Electrodialysis, studied thus far for use in the production of potable water from brines, is a technique used to separate a solution into two products, a concentrated salt solution and a salt-impoverished water stream. The theory and practice of the technique are discussed elsewhere (see Spiegler, 1962; Wilson, 1960); it suffices here to note that the technique might be used to produce a copper-rich solution suitable for use in electrolytic recovery cells and a copper-barren solution suitable for recirculation to the leaching process from the copper-bearing solution withdrawn from the leach dump. The process is as yet only a possible adjunct to an electrolytic recovery system; the present technologic development of the process leaves much to be desired.

Ion exchange represents a better-developed adjunct to an electrolytic recovery system. The technique has been extensively studied, especially for use in the recovery of uranium.

In ion exchange a resinous substance with chemically loosely attached ions is placed in a column and the solution, in this case copper-bearing, is passed through the column. The loosely attached ions are preferentially displaced by the ion whose concentration is desired; in this case copper. The desired ion is recovered by back-flushing the column with a concentrated solution of the ion species originally present in the 'collection sites' of the resin. The ions collected from the original solution are displaced by the concentrated ions in the flush solution, and are present in the flush solution removed from the column. The column is rejuvenated and ready for additional recovery duty, while the flush solution contains a concentration of the collected ion. This flush solution is ready for additional processing to effect the recovery of the desired ion; in this case to allow recovery of copper by electrolytic precipitation.

In theory ion exchange may be a valuable adjunct to the electrolytic recovery technique. However, additional investigation is necessary before the practical value can be stated.

Electrolytic recovery, including its modifications and variations, would be a most desirable method for recovery of copper from leach solutions. The direct production of a marketable form of copper represents a tremendous advantage. Further, the method would produce sulfuric acid, a desirable component of solutions used for leaching.

However, the cost of the process might be prohibitive for use at Chino Mines Division unless a cheap source of electrical power is developed. The incomplete recovery usually associated with electrolytic methods as applied to leach solutions may prove to be an additional handicap.

By-Product Recovery

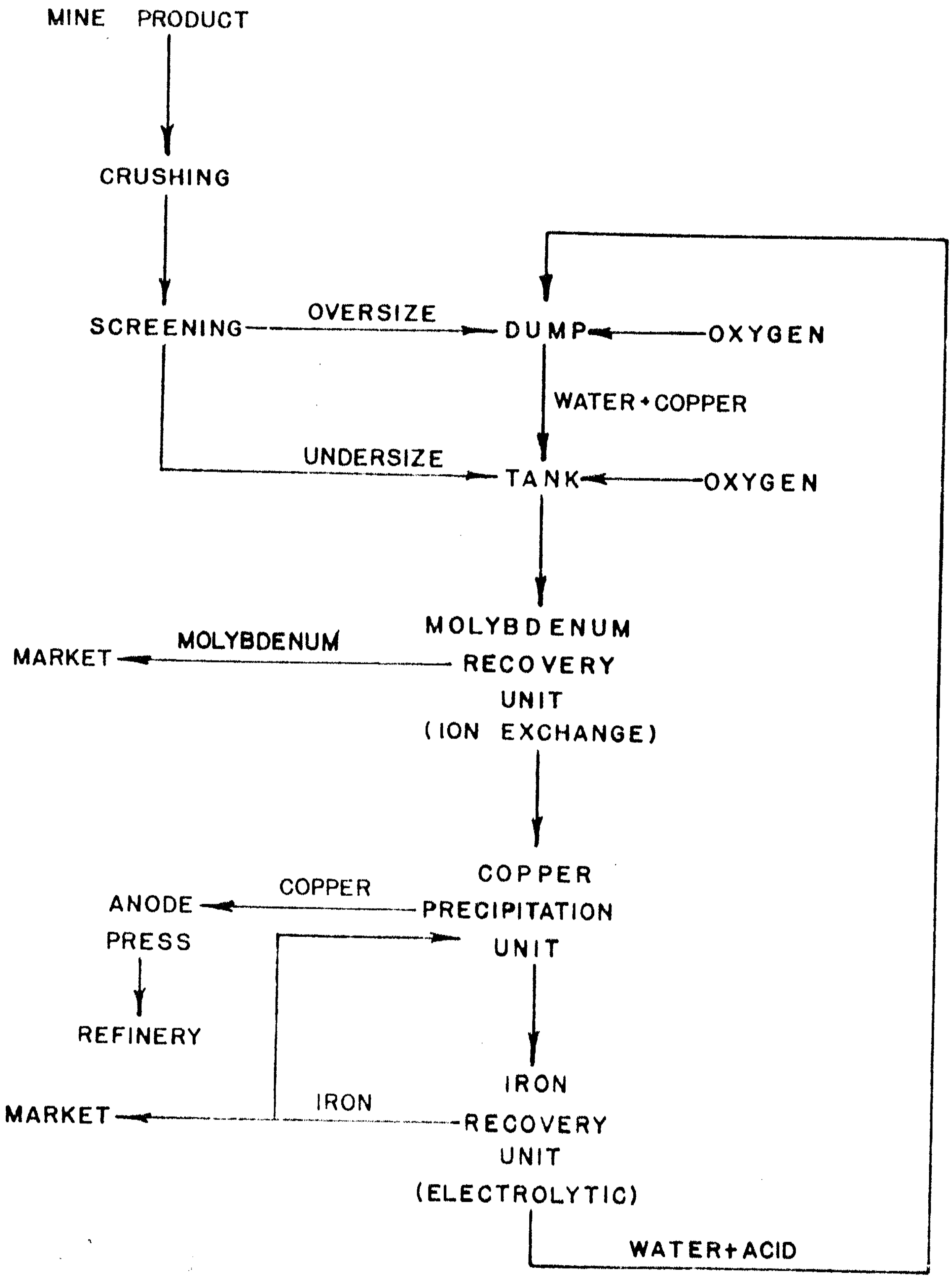
Earlier it was noted that molybdenum probably occured in the leach solution withdrawn from the dump, and that the leaching of molybdenum probably occurs in a manner analogous to the leaching of the copper. If the molybdenum content of the solution is verified, its recovery may be effected by application of ion exchange techniques.

By-product recovery of iron is also possible.

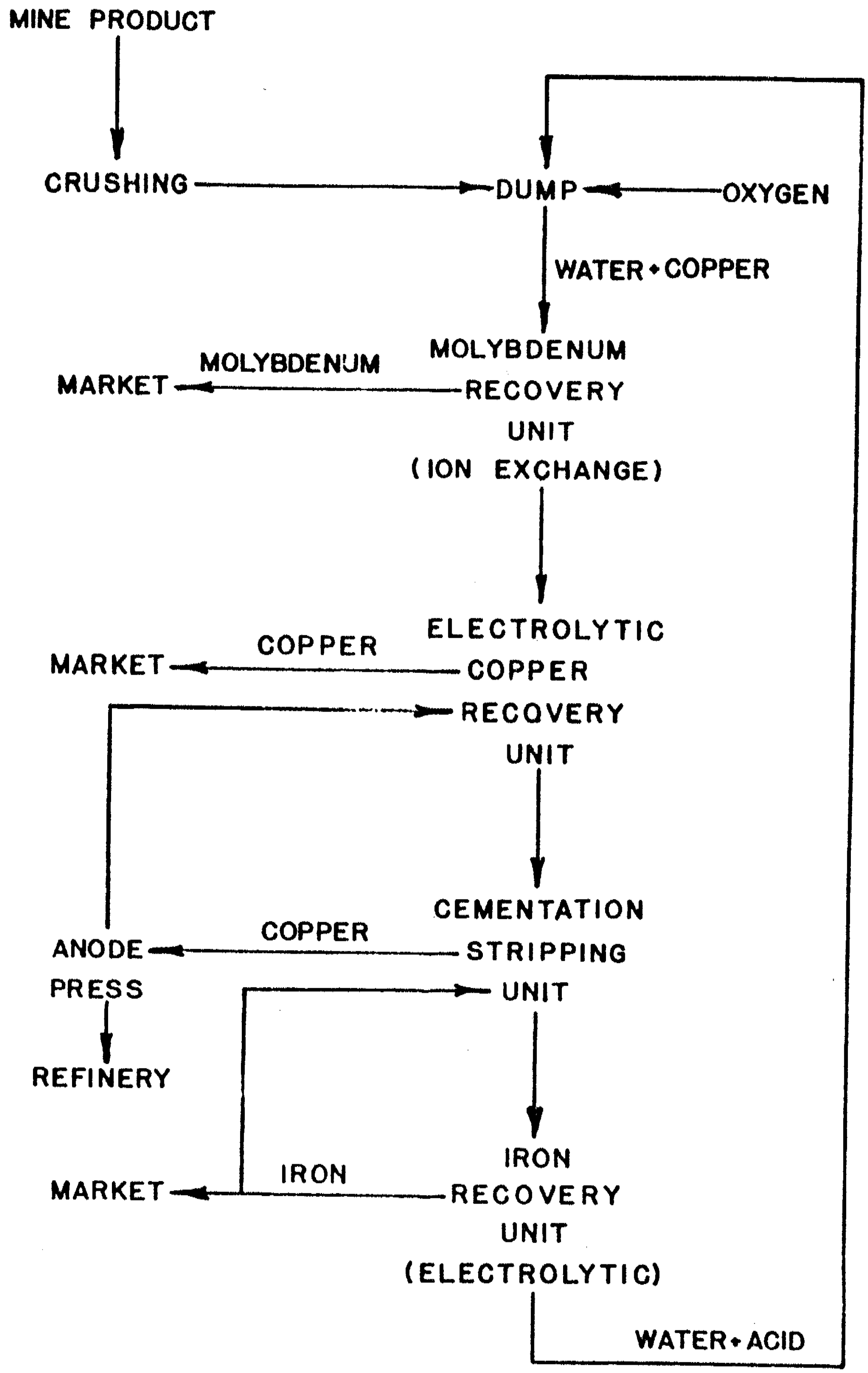
Possible Developments

On the following pages are outlined the flowsheets of two possible developments of the physical recovery of copper from leach solutions. One employs the cementation method; the other, electrolytic recovery.

It is suggested that one of the two flowsheets represents the probable course of long-range development of copper recovery in the Chino Mines Division.



POSSIBLE CEMENTATION RECOVERY FLOWSHEET



POSSIBLE ELECTROLYTIC RECOVERY FLOWSHEET

Chapter X

PENULTIMATE DEVELOPMENT

It is interesting, and may be instructive, to speculate on the nature of the penultimate development of copper recovery at the Chino Mines Division. This development, probably to be followed by the implementation of <u>in situ</u> leaching, will represent the final development of actual mining of the orebody in a physical sense.

Mining will probably consist of some application of continuous mining techniques in which the material is planed from the sides of the pit by mechanical means. The mined product will consist of fine particles.

Material handling and recovery process will probably be combined. After the material is mined, it will be reduced to a size that can be suspended in water, and then slurried with a process solution. The slurry will be pumped from the pit in pipes designed to promote agitation of the slurry so that the chemical components of the solution may act on the entrained particles and leach the desired elements from the particles.

At some point the solution and the suspended material will be separated. The material removed from the solution will be completely stripped of desirable components and will be ready for discard. The solution, containing the desired elements, will be processed to recover those elements, made ready for another round of leaching, and pumped back to the pit to be reused in the transport and processing of the mined material.

The ideas stated here are admittedly futuristic and nebulous. However, it is suggested that they indicate the penultimate development of mineral technology that will be applied at Santa Rita.

CONCLUSION

A study of the leaching process now existing at Santa Rita has been made. Areas of insufficient knowledge and needed research are indicated. Possible plans for immediate development are suggested and long-range development plans are suggested for speculation.

The principle value of this study lies in its effect on the organization and direction of effort in the inductry.

If such a study can be conducted for Chino Mines Division of Kennecott Copper Corporation, it can be conducted for any other copper-producing organization. Therefore, it is entirely possible that such a study has been conducted. Further, it may be suggested that such a study has been placed in use by guiding development in such an organization. Therefore, to maintain competitive strength, some use of this study must be made.

If other studies of this nature have not been conducted, a competitive advantage is introduced by this study. Use of these results and suggestions can lead to a coherent plan of development that will avoid valueless expenditure of effort.

The ultimate value of this study lies in the possible development of an offensive competitive position for Chino Mines Division, rather than continued defensive competitive manuvering. If further study shows that the conclusions reached in this thesis are correct, a tremendous advantage could be gained by rapid but careful implementation of the indicated developments.

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