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IC bureau of mines information circular 8104

MINING, MILLING, AND SMELTING METHODS, SAN MANUEL COPPER CORP., PINAL COUNTY, ARIZ.

By V. B. Dale

DEPARTMENT OF MINERAL RESOURCES
MINERAL BUILDING
FAIR GROUNDS
PHOENIX, ARIZONA

UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

1962

Bureau of Mines
Information Circular 8104

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by

V. B. Dale[2]

SUMMARY AND INTRODUCTION

San Manuel has been a modern prodigy from its inception to its present status as a block-caving operation producing 33,000 tons per day. The deposit was discovered, prospected, and developed with modern techniques, and nearly all planning was done from results of an extensive drilling program through an average overburden of 670 feet.

The Federal Government had a prominent part in the discovery and primary development of San Manuel. The first 17 holes were drilled by the Bureau of Mines, and discovery of the large, low-grade, disseminated copper deposit is credited to the Bureau. A loan fund of 94 million was granted through the Reconstruction Finance Corporation to the San Manuel Copper Corp. for development of the deposit and erection of concentrating and smelting facilities. Then, while the mine was still being developed, the Defense Materials Procurement Agency contracted to buy a large amount of copper at a pegged price.

The ore is low grade for an underground operation, averaging about 0.78 percent total copper, and a low-cost, large-volume production is imperative for a progitable operation. During some periods of development and mining, costs have exceeded those expected, and only meticulous planning and ingenuity have produced an economic operation. The story of cost cutting at San Manuel is an interesting one, and the more important cost-cutting practices at this operation are discussed in some detail.

Metallurgical data used in this paper were collected for the period January 1 to March 31, 1958, inclusive. Tonnages, weights, and measures have since changed, but practices and techniques are essentially unchanged.

Figure 1 is a simplified diagrammatic flowsheet. A resumé of the entire operation follows:

Type, size, and grade of ore deposit:

Type--Disseminated copper.

[1] Work on manuscript completed January 1962.
[2] Former Bureau of Mines mining engineer, Southwest Experiment Station, Tucson, Ariz., now with Bureau of Land Management, Anchorage, Alaska.

Type, size, and grade of ore deposit (Con.):

Host rocks--Quartz monzonite porphyry and quartz monzonite.

Sulfide minerals--Pyrite, chalcopyrite, chalcocite, and molybdenite.

Oxide copper minerals--Chrysocolla, cuprite, native copper, and black oxides.

Ore reserves:
Sulfide ore--367,624,000 tons.
Oxide ore--111,876,000 tons.

Average copper content:
Sulfide ore--0.785 percent.
Oxide ore--.717 percent.

Average molybdenum content--0.02 percent.

Gold and silver--Relatively small value.

Physical characteristics of ore body:

Outcrop--Triangular area, 300 by 400 feet.

Shape of ore body--The size and shape of the ore body are determined by the use of an arbitrary cutoff value based on the copper content of the rock. The ore body thus determined has an irregular, canted, trough shape. In cross section, the irregular U-shape is canted about 55° from the horizontal, pointing to the southeast. The trough bottom, or keel, plunges irregularly to the southwest at about 15°, along the general strike of the ore body. The sulfide ore occurs from about 500 to nearly 2,700 feet below the surface. The ore body has an overall length of about 6,800 feet and a maximum width of about 3,000 feet.

Fracture pattern--Intricate, three-dimensional network.

Mineral occurrence--Disseminated through the host rock.

Overburden or capping:

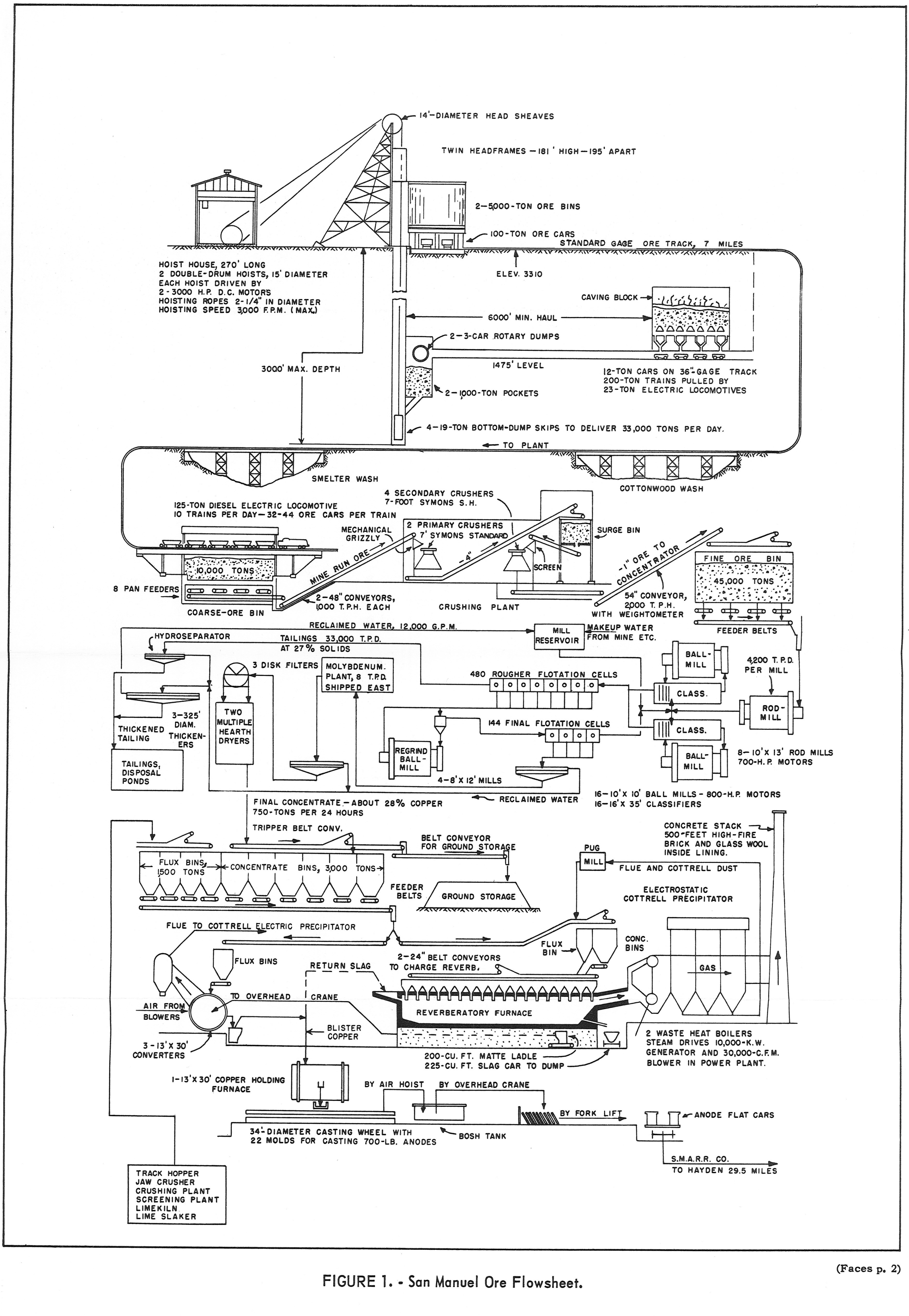
Description--Gila conglomerate and weakly mineralized or leached monzonite.

Average thickness of overburden--670 feet.

Mine openings:

Support of ground--All ground requires support, either timber, steel, or concrete.

Water conditions--Newly opened areas may show appreciable flow. Ore body drains rapidly.



(Faces p. 2)

FIGURE 1. - San Manuel Ore Flowsheet.

Mine openings (Con.):

Temperatures--Moderate.

Mine production:

Capacity:
Daily--33,000 tons.
Annually--12,000,000 tons.

Mining method--Full-gravity block caving with some slusher-gravity caving along fringes.

Underground haulage--Electric trolley locomotive.
Ore-car capacity--12 tons.
Cars per train--15.
Trains per 24 hours--185.

Hoisting ore to surface:
Hoisted through--Two identical vertical shafts.
First hoisting level--1,475 feet.
Second hoisting level--2,075 feet.
Third hoisting level--2,675 feet.
Hoists--Two 6,000-hp. double drums.
Maximum hoisting speed--3,000 feet per minute.
Capacity, each hoist--1,000 tons per hour from first level.
Skips dumped per hour--54 each hoist.
Capacity of skip--19 tons.
Capacity of skip pocket--1,000 tons each shaft.
Run-of-mine ore, maximum size--13.5 inches.

Ore transportation - mine to mill:

Storage at mine loading point--Two 5,000-ton-capacity bins.

Ore moved by--Shuttle service railroad.

Railway construction--7 miles, standard gage, 132-pound rail, level, minimum of curves with liberal radii.

Type of cars--Bottom-dump, air-operated.

Capacity of car--100 tons.

Cars per train--32 to 44.

Ore crushing:

Capacity of ore-receiving bin--10,000 tons.

Ore crushing (Con.):

Two-stage crushing--Two 7-foot Symons[3] standard head cone crushers; four 7-foot Symons shorthead cone crushers.

Capacity--2,000 tons per hour.

Fine ore-bin capacity--45,000 tons.

Concentration of ore:

Capacity of concentrator--33,000 tons per day.

Primary grind in open circuit--Eight 10- by 13-foot rod mills.

Secondary grind in closed circuit with classifiers--sixteen 10- by 10-foot ball mills and sixteen 16- by 35-foot drag classifiers.

Flotation--624 48-inch mechanical cells.

Concentrate regrind--Four 8- by 12-foot ball mills.

Cleaner concentrate--750 to 800 tons of concentrate to molybdenum plant for recovery of molybdenum.

Molybdenum plant products--95± percent molybdenum sulfide concentrate ready to market and 28 percent copper concentrate to smelter concentrate bins.

Smelting of copper concentrate:

28 percent final copper concentrate--To natural gas-fired, side-feed reverberatory furnace, 32 by 100 feet.

Reverberatory products:
Matte--At 32 to 35 percent copper.
Slag--To slag dump.
Waste gases--About 50 percent of contained heat recovered by two waste-heat boilers. Copper-bearing dust recovered from gases by electric precipitator before entering 20- by 500-foot stack.

Matte to converters--Three 13- by 30-foot Pierce-Smith type.

Converter products:
Slag--Return to reverberatory furnace.
Waste gases--Join reverberatory waste gases to electric precipitator and stack.
Blister copper--Delivered to holding furnace where it is poled and then cast into 700-pound anodes for shipment to electrolytic refinery.

---

[3] Reference to specific brands is made to facilitate understanding and does not imply endorsement of such items by the Bureau of Mines.

Power supply:

Outside source--Arizona Public Service by 115,000-kw. transmission line.

Waste-heat boilers, steam--10,000-kw. steam-driven turbo-generator.

Supplemental power--5,500-kw. gas-diesel plant at mine.

ACKNOWLEDGMENTS

The author acknowledges the cooperation and assistance of the management and the operating and engineering staffs of the San Manuel Copper Corp. in compiling and gathering material for this report. Appreciation is expressed for permission to publish the maps and photographs in this report, most of which were supplied by the company.

The company has fully cooperated in the gathering of the operational data.

Many papers have been written by members of the staff. These have been drawn on freely for information, and credit has been given when the information has been used.

LOCATION AND ACCESSIBILITY

The San Manuel mine is largely in secs. 34 and 35, T. 8 S., R. 16 E., in the Old Hat mining district of southeastern Pinal County, Ariz. It is 45 road miles northeast of Tucson, 5 miles southwest of Mammoth, and 7 miles north-northwest of San Manuel townsite. The property is reached from Tucson by traveling north for 23 miles on U.S. Highway 80 to Oracle Junction and then for 22 miles east-northeast on State Highway 77 and 77 alternate (fig. 2).

The concentrator, smelter, and main offices of the San Manuel Copper Corp. are near the San Manuel townsite and may be reached from the mine via a hard-surfaced county road (fig. 3).

A standard-gage railroad was constructed from the mill and smelter to the mine, a distance of approximately 7 miles. A 29.5-mile railroad was constructed between the mill and smelter to a junction with the Christmas Branch of the Southern Pacific Lines near Hayden, Ariz.

PHYSICAL FEATURES

The San Manuel deposit is in a northward-trending pediment from the Santa Catalina Mountains that slopes northeasterly to the San Pedro River. The relief ranges from an altitude of 3,450 feet on Red Hill to 3,000 feet in Tucson Wash west of shaft No. 1, and 2,400 feet at the San Pedro River, a seasonal stream 3-1/2 miles east of Red Hill. The surface is cut by gulches that feed northeast to the San Pedro, which in turn flows north into the Gila River at Winkleman (fig. 2).

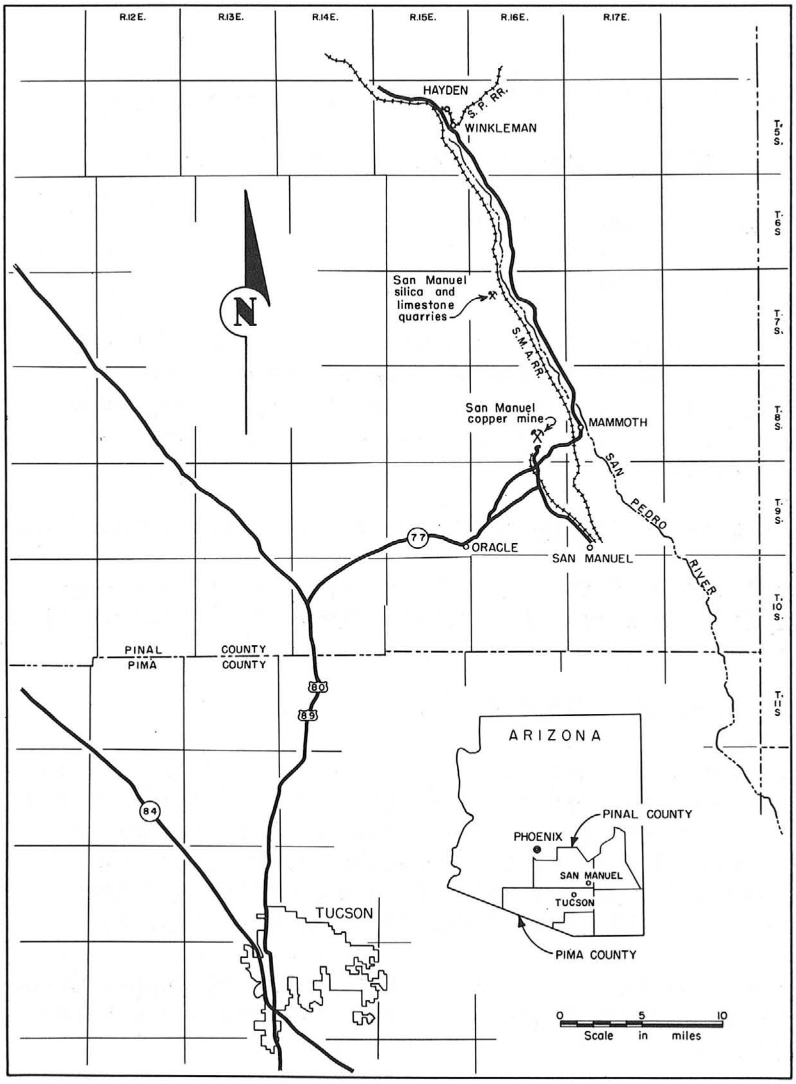


FIGURE 2. - Location Map, San Manuel, Ariz.

Climate and vegetation are typical of southern Arizona. Winters are mild, and summers are hot. Annual rainfall is low, and the water table is quite deep at the mine.

Various cacti grow in the area; giant saguaro and cholla are the most abundant. Creosote bush, ocotillo, medquite, catclaw, chaparrel, and paloverde are common desert growth.

HISTORY

Mining in the San Manuel area began between 1870 and 1880, when the St. Anthony claims, immediately north of the present San Manuel Property, were located and worked for gold. From 1880 to 1901, this area produced approximately $3 million in gold. Later, from 1934 to 1952, the San Manuel area produced substantial quantities of gold, silver, molybdenum, vanadium, lead, and zinc. The Mammoth-St. Anthony mine of the St. Anthony Mining and Development Co. was the principal producer. This mine is located a short distance north of the San Manuel property. (In 1955 all property and assests of the St. Anthony Co. were acquired by the Magma Copper Co., parent company of the San Manuel Copper Corp.)

In 1906, claims were located in the Red Hill area, and a few test pits were dug. Between 1915 and 1917, two churn-drill holes were drilled in a small triangular zone immediately southeast of Red Hill where copper mineralization, principally chrysocolla, outcrops. This work was done by Walter H. Aldridge for the William Boyce Thompson interests that were associated with the Magma Copper Co. The low-grade (0.8 percent) oxidized copper which may have been penetrated in one of the holes was discouraging.

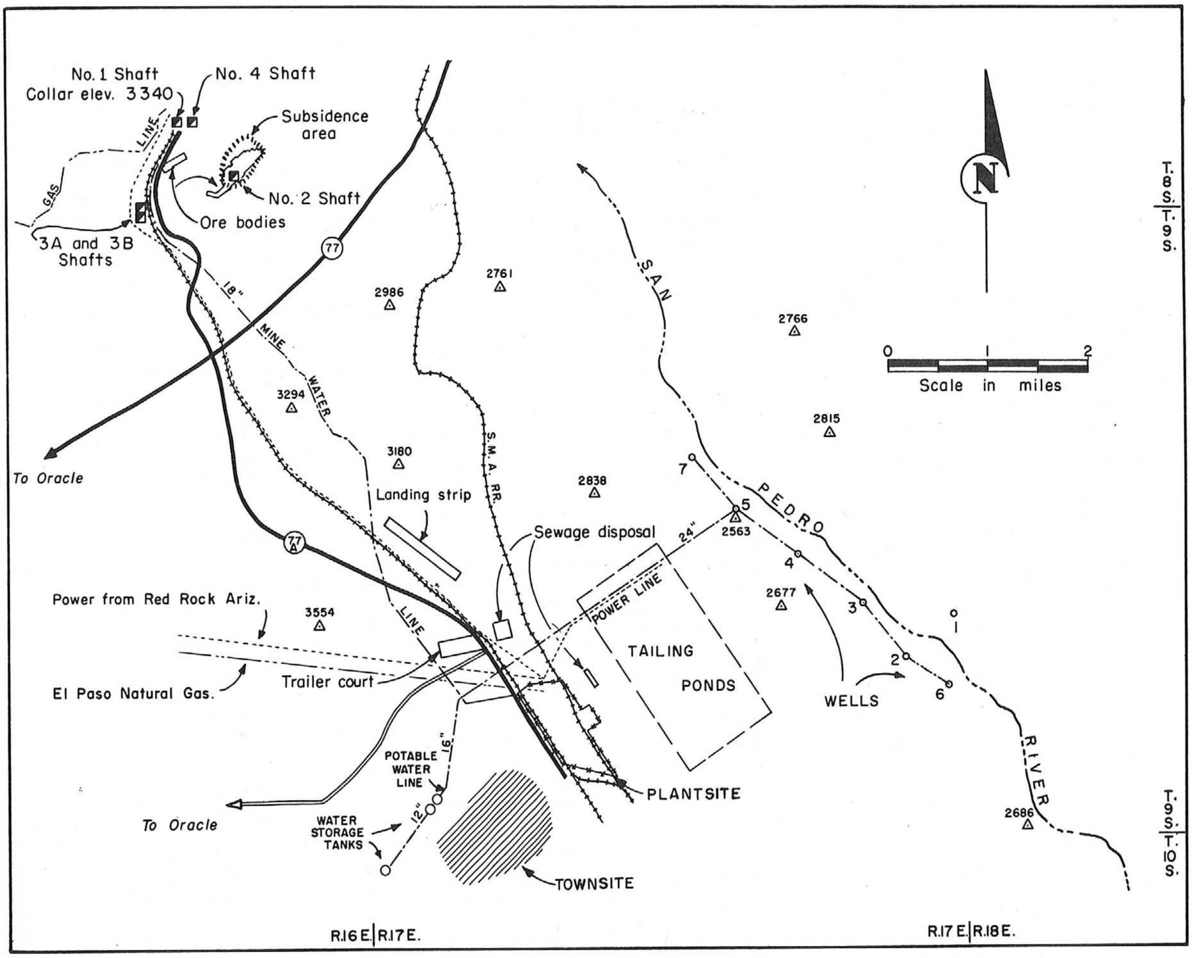


FIGURE 3. - San Manuel Area.

The original five lode claims of the San Manuel group were located in 1925 by Anselmo Laguna. Between 1926 and 1939, James M. Douglas bought two-thirds interest, and in 1939, Douglas and R. Burns Giffin bought the remaining one-third. In 1940, one-fourth interest was deeded to Victor Erickson, and in 1942 Henry W. Nichols acquired one-fourth interest. Douglas deserves credit for maintaining interest in the property. Giffin helped to finance the venture. Erickson earned his interest in the venture with work and knowledge of construction. Nichols submitted an engineering report to the Reconstruction Finance Corporation which aroused the interest of the Geological Survey and the Bureau of Mines.

In 1942, the owners of the San Manuel group applied to the Reconstruction Finance Corporation for a development loan. Nichols' work indicated that the deposit might be of major economic importance. In March 1943, B. S. Butler and N. P. Peterson of the Geological Survey examined the area and recommended exploration. In May 1943, T. L. Chapman and W. D. Hughes of the Bureau of Mines examined the property and wrote a report supporting the Survey's recommendation.

Bureau of Mines Exploratory Drilling Program

The Bureau of Mines commenced exploratory drilling in November 1943. Five shallow churn-drill holes, totaling 1,480 feet, were completed by January 1944. This work disclosed sufficient copper-bearing material to infer 4 million tons of ore containing 0.95 percent oxidized copper. This project showed that copper mineralization extended beyond the 350-foot depth attained in the deepest hole and that the ore body probably had considerable lateral extent. Consequently, the drilling contract was extended, and at its completion on February 2, 1945, a total of 15,839 feet had been drilled in 17 holes with a maximum depth of 2,000 feet. A summary of the cost of drilling and sampling 265 feet of 10-inch-diameter hole, 12,265 feet of 8-inch hole, 3,125 feet of 6-inch hole, and 184 feet of 4-inch hole, including 7,125 feet that was cased and 410 feet that was reamed, is shown in table 1. A total of 2,982 samples were assayed. In addition to costs shown in table 1, $3,387.27 was spent on the project for engineering services, preparatory work and road and trail building.

TABLE 1. - Summary of churn drilling and sampling costs

	Drilling	
	Amount	Per foot
Drilling................	$76,737.00	$4.845
Reaming.................	962.50	.061
Casing..................	718.50	.045
Supervision.............	2,806.22	.177
Extras..................	373.00	.024
Total................	81,597.22	5.152
	Sampling	
Labor...................	7,246.61	0.458
Supplies................	747.85	.047
Supervision.............	2,806.23	.177
Transportation..........	873.76	.055
Analysis................	[1] 2,982.00	.188
Total................	14,656.45	0.925
Grand total..........	96,253.67	6.077

[1] Based on charges of $1.00 per analysis.

Before completion of the last holes drilled by the Bureau, exploration data indicated a block of ore 3,100 feet long by 400 to 800 feet wide and 500 to 900 feet thick which averaged approximately 0.8 percent copper.[4] Subsequent drilling showed that the ore body extended considerably beyond these measurements.

San Manuel Copper Corp.

In September 1944 the Magma Copper Co. took options on the San Manuel group and on adjoining property. The company commenced a churn-drilling program in December 1944. A short time later the Anaconda Copper Mining Co., under the name of Houghton Group, started drilling on adjacent property to the east. In September 1945 San Manuel Copper Corp. was organized as a subsidiary

[4] Chapman, Thomas L., San Manuel Copper Deposit, Pinal County, Ariz. Bureau of Mines Rept. of Investigations 4108, 1947, 93 pp.

of Magma Copper Co. to carry on the exploration. The drilling program was completed by San Manuel Copper Corp. in January 1948, and the sinking of shaft No. 1 was begun in March of that year.

The following ore-reserve estimate was taken from a prospectus issued by the company on February 15, 1949:

	Ore, tons	Total copper, percent	Oxide copper, percent
Sulfide ore:			
North ore body..............	178,762,616	0.784	0.036
Southeast ore body..........	160,522,304	.795	.025
Total sulfide ore..........	339,284,920	.789	.031
Oxidized ore:			
Plus 0.7 percent copper.....	73,165,975	.866	--
Plus 0.5 percent copper.....	123,499,580	.767	--

Higher grade sulfide ore and total copper was calculated as 150,416,100 tons and 0.898 percent, respectively.

Based on information obtained by additional diamond and churn drilling during 1951, the estimate of ore reserves was increased to 479,500,000 tons averaging 0.77 percent copper and consisting of 367,624,000 tons of sulfide ore and 111,876,000 tons of oxidized ore averaging 0.785 and 0.717 percent copper, respectively. These reserve estimates were obtained by 205,536 feet of exploratory drilling.

By late 1955 five shafts had been sunk. Underground development had advanced enough for block-caving operations to begin. The new town of San Manuel, a 33,000-ton concentrator, a smelter, and 29.5 miles of railroad to Hayden had been built.

Production of copper began in January 1956. By yearend, six caving blocks on the 1,415 level had been placed in production, and surface subsidence had occurred prominently over more than 1,382,000 square feet.

In 1957, the San Manuel mine ranked fourth in production among the copper mines of Arizona. On October 16, 1957, the mine attained its scheduled output of 33,000 tons of ore per day. Development of the 2015 and 2075 levels was in progress, and 12 blocks above the 1475 level had been placed in production by the end of the year. Plans were made to abandon No. 2 shaft above the 1415 level because of the expanding influence of the subsidence.

In February 1958 the first block was drawn to completion. The San Manuel mine ranked second in copper production among the copper mines of Arizona in 1958. Production for the year amounted to 149,401,672 pounds of copper.

Production

Annual production of the San Manuel mine from the beginning of operations until the strike interruption in August 1959 is given in table 2.

TABLE 2. - Production from the San Manuel mine

	1956[1]	1957	1958	1959 (7 1/3 months)
Copper pounds..........	78,152,140	119,797,769	149,401,672	92,387,272
Molybdenum sulfide Do..............	591,970	1,452,080	1,872,450	1,411,913
Silver ounces..........	136,074	200,301	253,858	158,907
Gold Do..............	9,719	13,578	16,868	10,306
Ore mined tons............	5,539,581	8,825,130	11,486,300	7,219,860
Tons mined, average per operating day.....	--	--	32,250	33,271

[1] Includes metal recovered from 416,473 tons of development ore, stockpiled on dump of No. 2 shaft.

Financing

Magma Copper Co. spent about $10 million at San Manuel before procuring a Government loan in 1952.

A loan contract between San Manuel Copper Corp. and the Reconstruction Finance Corporation was made on July 14, 1952, wherein a loan fund of $94 million bearing 5-percent interest, was established to develop the mine and construct facilities. The contract stipulated that all property of San Manuel Copper Corp. be pledged as a mortgage, that the company invest an additional $8 million before making withdrawals from the loan fund, and that a townsite of 1,000 housing units with utilities, markets, schools, and so forth be built.

Del E. Webb Construction Co. of Phoenix, Ariz., and M.O.W. Homes, Inc., of Los Angeles, Calif., financed and constructed the new San Manuel town. Magma Copper Co. purchased the entire townsite in April 1955.

According to the Annual Report to the Stockholders of 1958, the Government loan amounted to $76,750,773 million as of December 31, 1958; this was substantially less than the $94 million loan authorized.

Government Sales Contract

The Defense Materials Procurement Agency signed a contract on August 29, 1952, whereby the Government agreed to purchase from San Manuel Copper Corp. 347,500 tons of copper at 24 cents per pound from the first 365,000 tons produced and 15,330 tons of molybdenite concentrate at 60 cents per pound from the first 16,060 tons produced. The prices were subject to adjustment with increased wages and cost of materials. The San Manuel Copper Corp. could sell its prod-

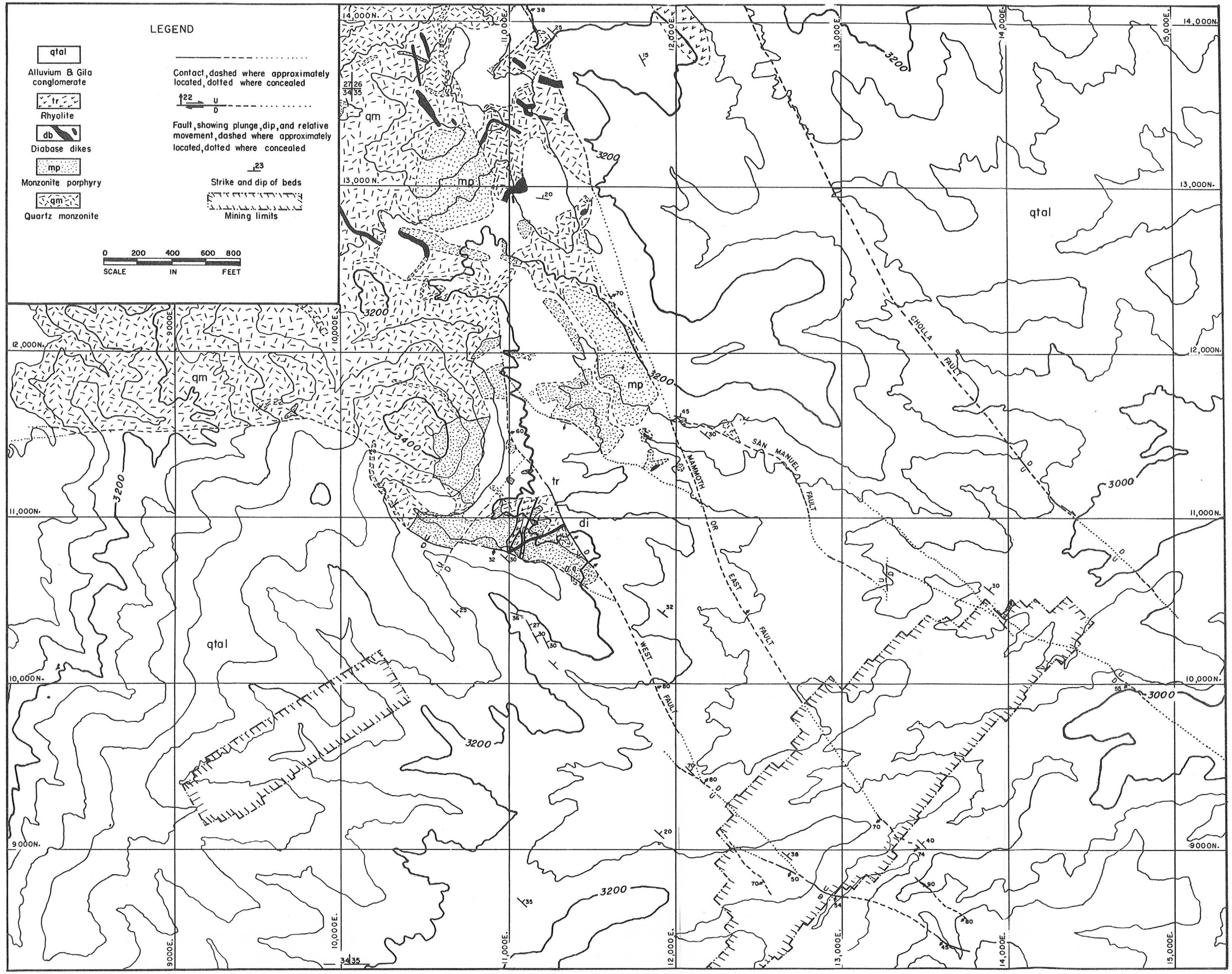


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FIGURE 4. - Surface Geology at San Manuel Mine.

ucts on the open market when prices were higher than the ccntract prices. From January to October 1958, when the contract price was above the market price, San Manuel Copper Corp. sold their copper exclusively to the Government at the negotiated contract price of 27.05 cents per pound.

MINE

Ore Body

Steele and Rubly,[5] Chapman,[6] Goss,[7] and Schwartz[8] provide detailed descriptions of the San Manuel ore body.

The ore body, trending N. 60° E., is 6,800 feet in known length and 3,000 feet in maximum known width. With the exception of a small oxidized outcrop, it is overlain by postmineral conglomerate that ranges in thickness from a feather-dege to more than 1,900 feet. The contact of the conglomerate with the underlying mineralized rock is a low angle fault which is one of the most important structures in the area (Figure 4). The sulfide ore ranges in depth from 475 to 2,665 feet. Much of the upper part of the ore body has been oxidized in varying degrees, mainly to chrysocolla. The bottom of the oxidation is irregular and plunges from a depth of about 400 feet to more than 1,600 feet.

The ore consists essentially of disseminated chalcopyrite in quartz monzonite, monzonite porphyry, and minor amounts of diabase, with little distinction in grade between different kinds of rocks that have undergone similar hydrothermal alteration. The ore mineralization is associated with hydrothermal alteration to sericite, pyrite, quartz, chlorite, kaolinite, and minor amounts of other minerals. Supergene enrichment is relatively unimportant from the standpoint of ore. Neither the enrichment nor the oxidation is related to the present erosion surface or the present water table.

The ore body has an irregular trough shape that plunges to the southwest, whereas in cross section the ore body is roughly U-shaped with the U pointed to the southeast (figs. 5 and 6). The upper part of the ore zone is split into two branches which, for convenience, are called the north and south ore bodies.

The water table before the beginning of mine development ranged from about 300 to 800 feet below the surface. Water in the mine workings has not been excessive. Drainage, water supply, and pumping are discussed in another section of this report.

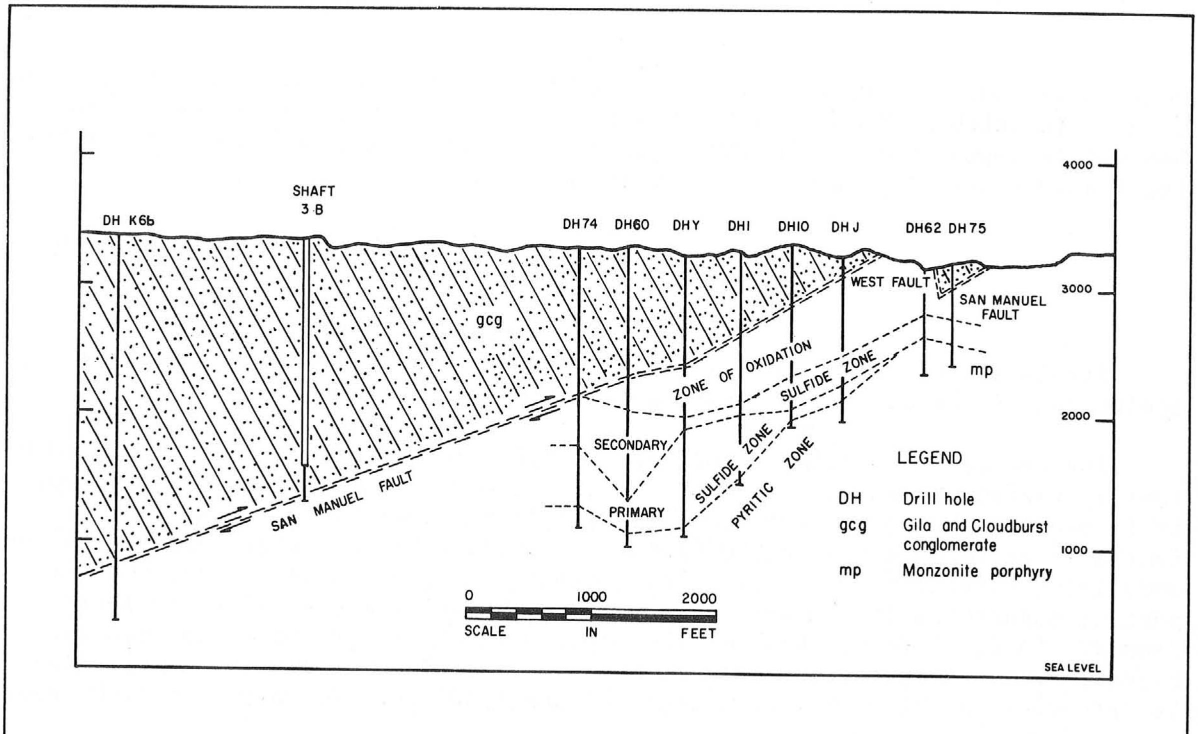
---

[5] Steele, H. J., and Rubly, G. R., San Manuel Prospect: AIME Tech. Pub. 2255, 1947, 12 pp.

[6] Work cited in foot note 4.

[7] Goss, Wesley P., San Manuel Copper Corporation: Eng. Min. Jour., vol. 150, No. 7, 1949, pp. 92-95.

[8] Schwartz, G. M., Geology of the San Manuel Copper Deposit, Arizona: Geol. Survey Prof. Paper 256, 1953, pp. 46-55.



From E. D. Wilson's San Manuel progress report, April 1956 to March 1958, RI 5561

FIGURE 5. - Vertical Section Through Shaft 3B, Looking N. 53° W.

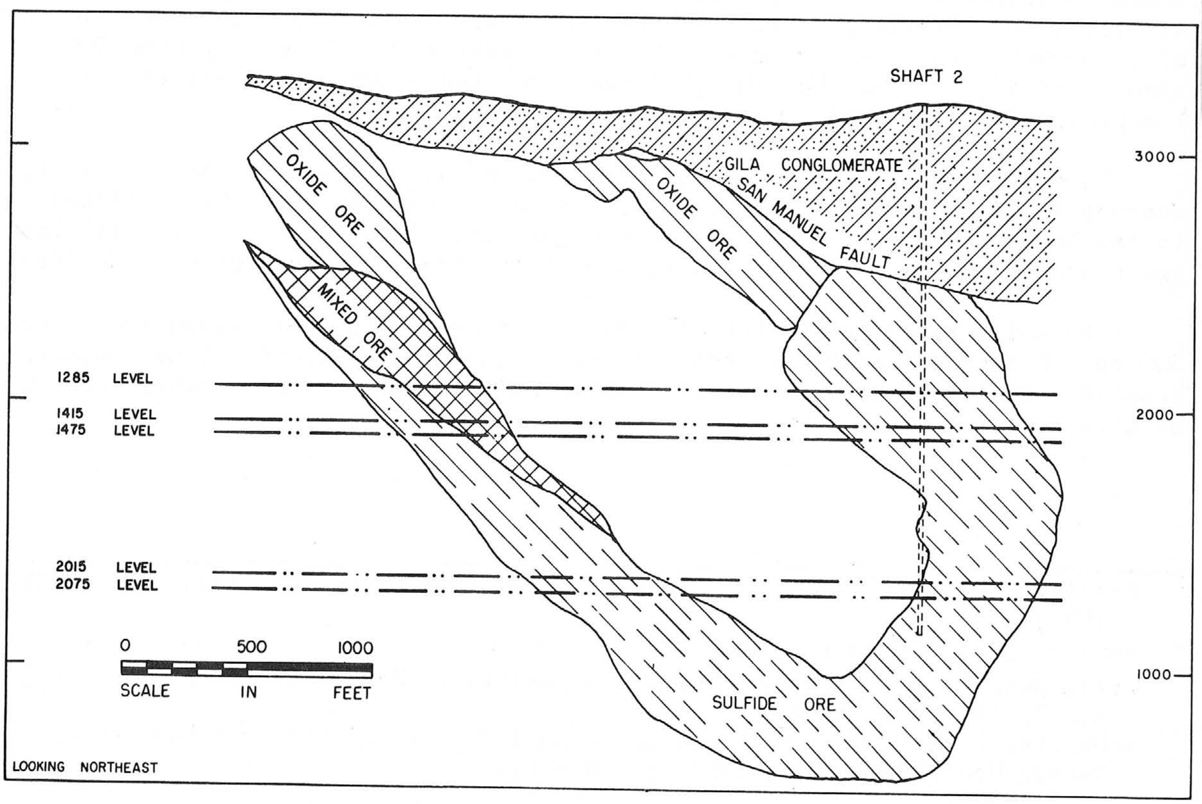


FIGURE 6. - Cross Section Through Ore Body, N. 46° W.

Relation of Ore Deposition to Fractures

In Wilson's latest progress report, which covers the period April 1956 - March 1958,[9] he discusses the relation of ore deposition to fractures.

> As a result of the present study it appears that primary (hypogene) ore mineralization at San Manuel favored those areas in which relatively the largest percentages of the E. - W. and N. - S. fractures are vertical....
>
> Conversely, it seems that where mineralization is poor or lacking, relatively large percentages of the N. 30° W. and N. 60° W. fractures dip 60° to 90° from the horizontal....
>
> In general, the ore tends to be less blocky than the barren rock, as has been noted by H. J. Steele.

Underground Exploration

Underground exploration began soon after the initial exploratory drilling from the surface was completed. The primary purposes of the underground work were to test caving characteristics and verify grade of the ore, to determine if underground waterflow was sufficient to become a major problem during mining, and to collect large samples for metallurgical testing. The program consisted of sinking No. 1 and No. 2 shafts, driving the 1285 level, and core drilling from it.

Shaft No. 1 will be used throughout the life of the mine. Shaft No. 2 was used continuously until 1957, when it was sealed above the 1415 level because of ground movement from subsidence. Below the 1475 level, this shaft will continue to be used for ventilation of lower levels.

No. 1 Shaft

No. 1 shaft was sunk in the footwall in an area that seemed least likely to produce water; it is approximately 900 feet from the nearest mining in the north ore body. The shaft was collared in 1948 before a headframe had been installed. Mucking was done by hand, and hoisting was performed with a tugger hoist. After the headframe was installed, a Riddell shaft mucker was used.

No. 1 shaft initially was sunk 1,643 feet, and its walls were supported by concrete and steel. Later, it was sunk to 2,263 feet to provide the main access for developing the grizzly and haulage levels for the second lift. Steel with timber lagging was used for wall support during the second phase of sinking.

Levels in the mine are designated by their distance below the collar of No. 1 shaft, which is 3,340 feet above sea level.

---

[9] Wilson, E. D., Geologic Factors Related to Block Caving at San Manuel Copper Mine, Pinal County, Ariz. 2. Progress Report, April 1956 - March 1958: Bureau of Mines Rept. of Investigations 5561, 1960, 43 pp.

The shaft has four compartments and measures 7 by 26.5 feet outside the sets. Two compartments are equipped with combination skips and cages, one compartment is equipped with a small service cage, and the fourth compartment has a manway. Electric lines and pipes for transportation of compressed air, water, and concrete are located in the two service compartments. The shaft serves as one of the main intake air courses to ventilate all levels.

During the initial sinking, the shaft crew consisted of six men underground, one topman, one truckdriver, and one hoistman. Their work included mixing of concrete on the surface. Some statistical data pertaining to the sinking of the first 1,643 feet follow:

Depth at start of statistical period	feet	125
Depth at end of statistical period	do	1,643
Advance during statistical period	do	1,518
Man-shifts, sinking statistical period		7,680
Man-shifts per foot of shaft, sinking only		5.06
Man-shifts of station work		2,289
Man-shifts per foot of shaft, sinking plus station work		6.57

No. 2 Shaft

No. 2 shaft was sunk near the center of the south ore body (1) to provide quick access to the ore, (2) to confirm the grade of ore indicated by surface drilling, (3) to test the caving characteristics of the ore, and (4) to measure the flow of water and study the drainage problems.

The shaft has three compartments and measures 7 by 20 feet outside of sets. Treated timber was used to support the shaft and the drifts driven from it. The pocket on the 1285 level was constructed with 8- by 8-inch armored cribbing.

Each crew consisted of five men underground, one topman, one truckdriver, and one hoistman. A time study of the man-hours required to sink a foot in No. 2 shaft is given in table 3.

TABLE 3. - No. 2 shaft sinking time study, man-hours per foot of shaft

	Hand muck, 402 feet	Riddell muck, 1,662 feet	Total, 2,064 feet
Drill and blast	5.10	3.95	4.18
Muck	9.60	9.41	9.45
Timber	7.70	6.81	6.98
Pump	1.00	[1]2.96	[1]2.58
Spile	0	1.26	1.01
Pipe	.74	.92	.89
Manway	.13	.19	.18
Guides	.29	.34	.33
Miscellaneous delay	.70	.36	.43
Total	25.26	26.20	26.03

[1] This figure includes 1,556 man-hours of water-ring construction, or 0.93 man-hours per foot of Riddell mucking, and 0.75 man-hours per foot of total.

1285 Level

The 1285 exploration level, totaling 2,784 feet of drifts, provided access for underground diamond drilling and sampling to determine in detail the mining limits of the south ore body. The level, developed both northeast and southwest of the shaft, was driven approximately through the center of the ore zone 100 feet above the intended undercut. The drift was made to coincide as nearly as possible with the planned boundaries of the first blocks to be undercut so that it could be used later to observe caving action of these blocks. Diamond-drill stations were cut at intervals of 210 feet along the long axis of the ore body, and holes were drilled in a ring pattern around the drift to outline the ore section. Three holes were drilled on each side of the drift, and the lower holes were drilled to intersect the ore limit at the undercut level. Where sharp irregularities occurred in the ore boundary, half-interval sections were drilled.

Diamond-drill, shaft, and drift samples averaged 0.02 percent more copper than the original churn-drill samples.

Completion of the underground exploration program provided adequate information to proceed with detailed planning of the mine and plant.

Planning and Mine Layout

Daily Ore Production

The first step in planning the San Manuel mine was to determine a daily production that would give the greatest efficiency and lowest costs and not overtax the size and potentials of the ore body. Preliminary estimates indicated that a daily production of about 30,000 tons would meet these requirements. The mine layout was planned to give a maximum daily tonnage of 35,000 tons on a 6-day mine workweek or the equivalent of 30,000 tons on a 7-day week for the metallurgical plant. The metallurgical plant was designed to handle 33,000 tons per day on a 7-day workweek. The operation is currently capable of handling more than 33,000 tons per day, 7 days a week. This is roughly 10 percent greater than the planned production.

Mining Method

Exploration furnished accurate detailed information on the grade, size, shape, and position of the ore body.

Two mining methods were considered: Open-pit and block-caving. Detailed cost estimates showed the underground operation to be the most economical because the stripping ratio of waste to ore would be in excess of 6 to 1. The host rock showed desirable qualities for a block-caving operation, but the irregular shape of the ore body and the large amount of tough, heavy capping over the ore were undesirable. The south ore body had a maximum column of sulfide ore 1,800 feet in height with 700 feet of tough, massive conglomerate over it.

Three major mining levels on 600-foot lifts were planned to handle the complete mining operation. The 600-foot caving height seemed excessive when the planning was done; however, experience with concrete ground support has shown that a 600-foot lift is not excessive when considered from aspects of ground weight, erosion, and dilution. The north ore body has a 50° dip, and planning indicated the feasibility of 300-foot lifts to cave this section. Because of thick oxide capping on the north ore body and a resultant short column of sulfide ore above the first 600-foot lift, no sublevel will be required until first-level caving is completed.

The eastern part of the south ore body was chosen as the initial mining area for three reasons: The ore was of better than average grade; the capping was of minimum thickness; and the maximum lateral dimensions were advantageous for withdrawing support from under the thick, tough, massive conglomerate. It was imperative to ensure the failure and caving of the conglomerate capping. The chosen area had two major faults through the ore body and its capping, which were believed to be, and later proved to be, helpful in starting the initial failure of the capping. Starting operations in the eastern extremity of the south ore body allowed a systematic retreat along the strike of the ore body from east to west, and the north ore body could be mined later without conflict.

A checkerboard system of block caving was selected because it gave greater control of the caving by decentralizing operations, and allowed a greater production at a faster rate with more flexibility and greater safety of operation in the designated area. No adjacent blocks are mined at the same time. It was believed that the effect of ground weight and dilution could be better controlled with this method of block caving.

Seven large blocks, each with virgin ground on four sides, were laid out in seven panels to form a checkerboard pattern. The panels were perpendicular to the strike of the ore body and 210 feet wide with 35-foot pillars between panels. It was realized that the pillars would be subject to heavy ground weight but would be required to keep dilution of the marginal-grade ore to a minimum. The length of the blocks ranged from 180 to 270 feet. (Planning of the block sizes was influenced by concern about caving the conglomerate capping).

Both slusher and gravity systems for ore extraction were considered. The two easternmost panels, which have short ore columns, were set up as slusher blocks. The remainder were set up as gravity blocks. Drawpoint spacings of 15 by $17\frac{1}{2}$ feet and $17\frac{1}{2}$ by $17\frac{1}{2}$ feet were used with the latter in areas of anticipated coarser caving fragmentation. Hence, slusher and grizzly drifts were spaced on 30- and 35-foot centers with 12 draw raises on each side along their 210-foot lengths. A horizontal Miami-type undercut was planned, with the floor 25 feet above the grizzly-drift sill. Grizzlies in gravity blocks were placed at each set of draw raises using 9-inch spacing, later opened to $12\frac{1}{2}$ inches. Slusher-block grizzlies were placed at the first and last pairs of draw raises in each line. Originally a 10-inch spacing was used, but it was later increased to 14 inches. An underground crushing facility was considered in relation to wider spacing on grizzlies, but it was discarded as uneconomical because of the high initial expense and the good fragmentation character of the ore.

A transfer raise system to handle ore flow from draw-raise grizzlies to the haulage level was designed for direct ore flow. The haulage level was placed 60 feet below the sill of the grizzly level, the minimum distance considered necessary to protect the haulage level from excessive weight caused by the caving. The short, direct-transfer raise system contributes to an efficient production operation. In gravity blocks, 4- by 4-foot armored, cribbed, transfer raises were laid out on 63° angles from opposite sides of the haulage drift loading station to the No. 1 and No. 4 grizzlies above. Backover raises of the same size are driven from a junction midway on the No. 1 and No. 4 raises to connect with the No. 2 and No. 3 grizzlies. Each loading station, comprising a loading chute on each side of the drift, handles the ore flow from four grizzlies or eight drawpoints. Three gravity haulage drifts on 70-foot centers were required for gravity panels (fig. 7). In slusher blocks, a single 4- by 6-foot armored, cribbed raise on a 63° inclination was laid out directly from one side of the haulage drift loading station to the grizzly at the end of the slusher line. Each station handled the flow from one grizzly or 12 drawpoints. Slusher panels required two haulage drifts on 140-foot centers. Early experimental work on the flow characteristics of the ore indicated that best results were obtained on a 63° angle; production experience confirmed this. The storage capacity of the raise system was important. The present layout allows 55 tons per gravity raise and 62 tons per slusher raise. The storage capacity of the slusher system has proven too small for efficient production.

Conveyor System Versus Track Haulage and Shaft Hoisting

A conveyor-belt system from the block loading chutes to the ore-hoisting pockets for horizontal underground transportation of the ore was given detailed consideration in the early planning. However, it was discarded in favor of rail haulage because of its lack of flexibility, the question of haulage level ground weight and support, the problem of supplying and servicing other operations on the haulage level, the size and character of material to be handled, and the questionable belt life.

A detailed study was made of the use of inclined conveyor belts versus conventional vertical skip hoisting. The length of the inclined conveyor gallery required, the questionable belt life, the size and character of the material to be handled, and the possibility of an underground crushing operation being required indicated the superiority of vertical-shaft skip hoisting.

Rail Haulage

A rail haulage system was designed which would use 90-pound rail in the main haulage drifts and 70-pound rail in panel drifts. The track is laid on 36-inch gage. A 250-volt direct-current overhead trolley system with appropriate bonding and feeder capacity was approved. Power for the system was supplied by three 500-kw. rectifiers operated in parallel and equally distributed around the haulage loop. A manually operated block signal system with ample haulage communication through a centralized dispatcher was included. Box type cars of 12-ton capacity and with two four-wheeled pony trucks were selected in preference to automatic dumping types because of simplicity, low maintenance, and low spillage. The box-type cars required rotary tipples.

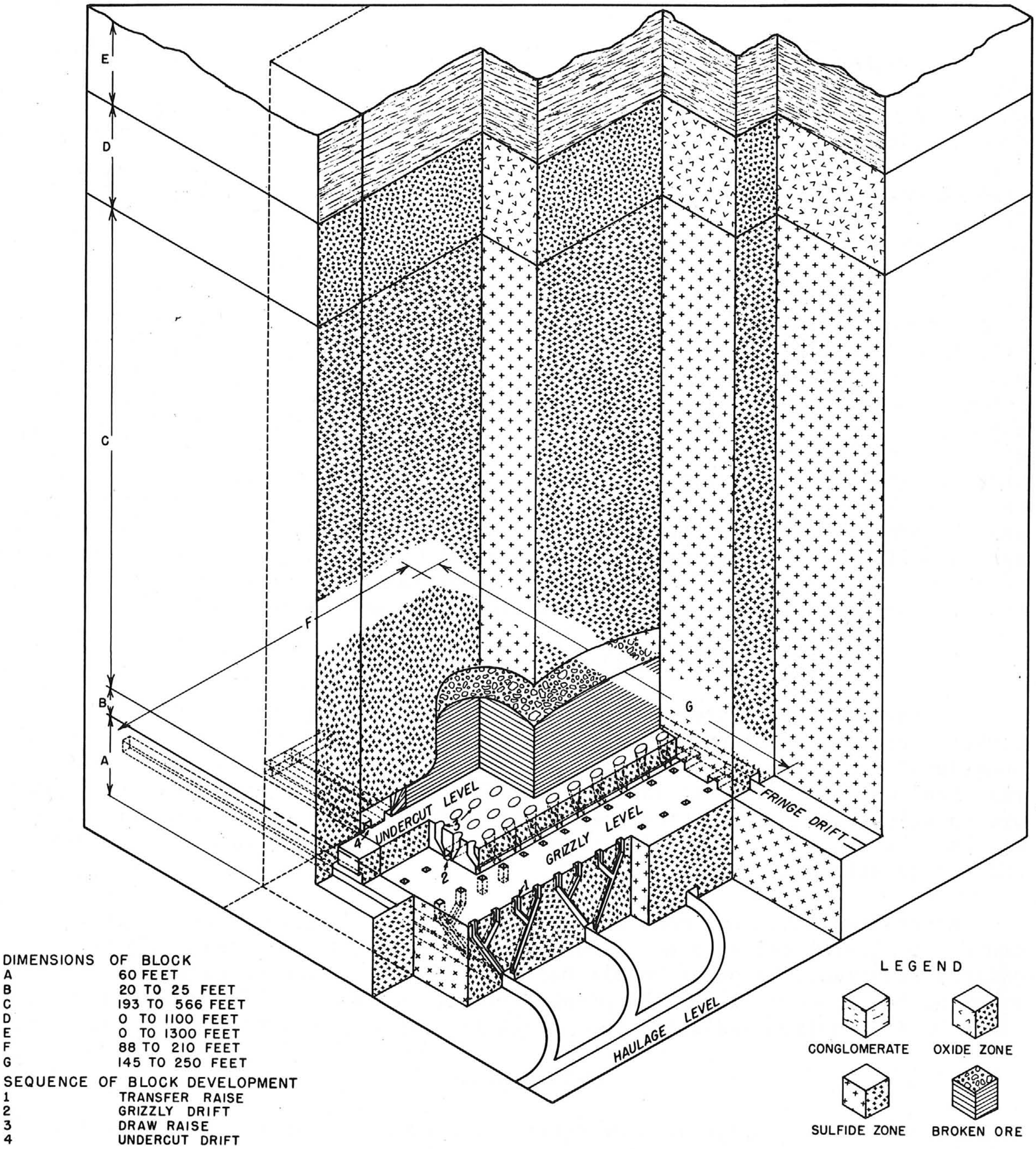


FIGURE 7. - Schematic Diagram of Gravity Block.

Twenty-three-ton, 4-wheel locomotives were chosen to haul 15-car trains. Locomotives with ample power, weight, and braking capacity were specified. It was desired to keep brakes off ore cars, and locomotives were required to brake loaded trains on a minus 0.45-percent grade at 15 miles per hour. The haulage system was laid out to handle 2,000 tons per hour or 12,000 tons during the 6 working hours of a normal 8-hour shift.

Scheduling

In order to coordinate ordering and installation of equipment and supplies and to assure completion of development on time and in sequence, detailed estimates of development, production, and material requirements were determined. Proper scheduling of these requirements was of great importance in bringing the mine into production.

Mine Development

The San Manuel staff considers sinking of shafts, driving of main drifts on both the working and haulage levels, and driving panel drifts on both levels as preliminary mining development.

Twin Ore Hoisting Shafts, 3A and 3B

The hoisting shafts (fig. 8) were laid out as twin, four-compartment shafts, 195 feet apart, each with two hoisting compartments, a manway compartment, and a service-cage compartment (figs. 9 and 10). Each shaft is independent of the other and is a complete unit with its own hoists, headframe, underground rotary dump, 1,500-ton (1,000-ton live load) storage pocket, and loading station. To keep abreast of underground haulage, each shaft handles 1,000 tons of ore per hour. The scope of this hoisting operation requires skips with capacities of 21 (20 dry) tons moving at a maximum speed of 3,000 feet per minute; this is accomplished by two double-drum automatically controlled hoists, each operating in balance.

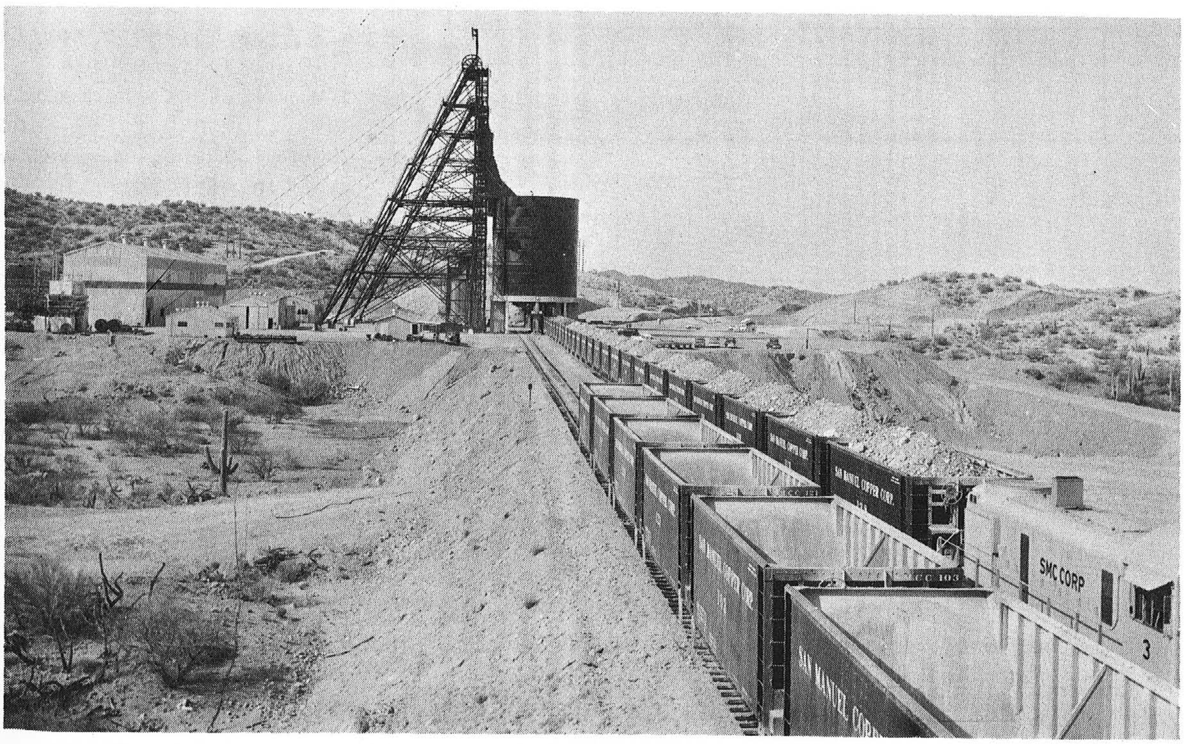


FIGURE 8. - Surface Installations at Twin Ore-Hoisting Shafts, 3A and 3B.

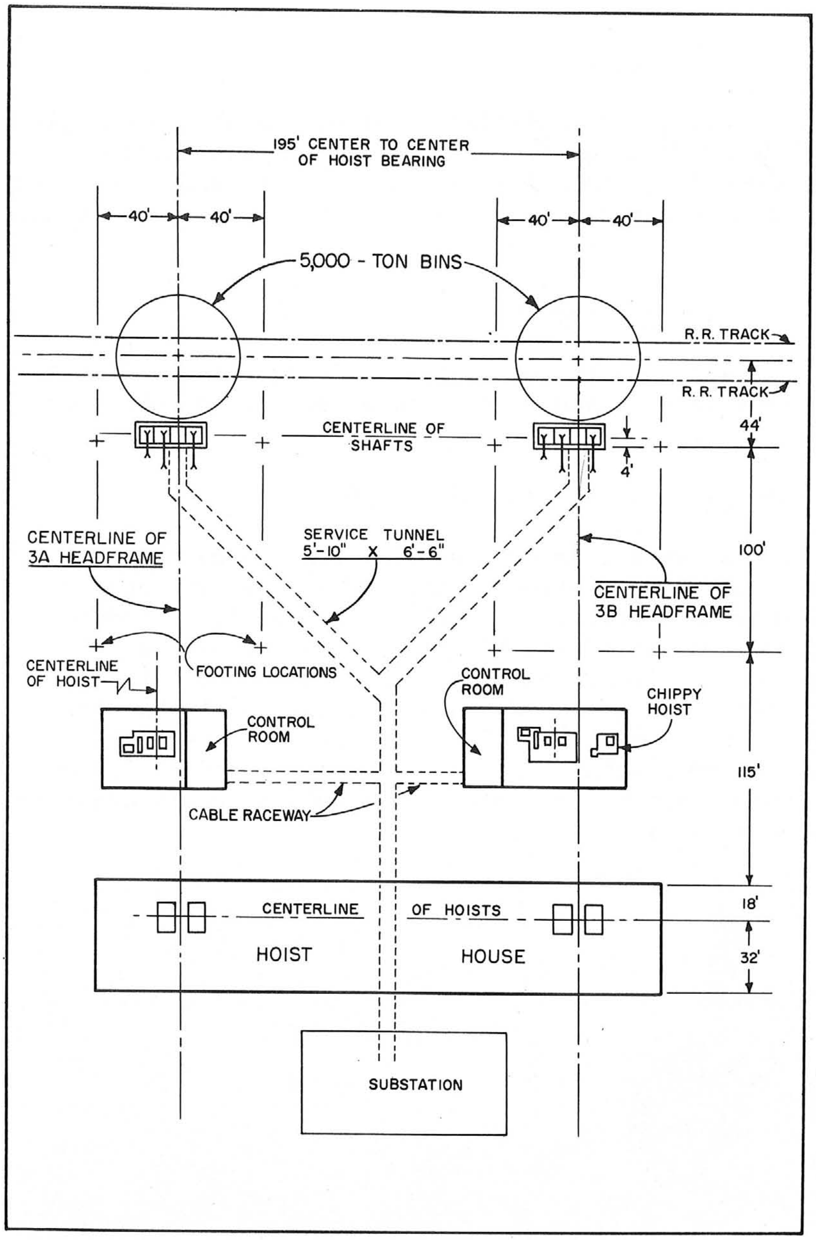


FIGURE 9. - Plan of Surface Installations at 3A and 3B Shafts.

"Jeto" bottom-dump skips (fig. 11) operate on 12 hard-rubber, spring-cushioned wheels running on 6-9/16- by 5-1/16-inch steel box-section guides supported in concrete walls on 3-foot centers. The skips discharge directly into two 5,000-ton-capacity, circular, ore-storage bins, one at each shaft collar. The bins were designed for direct loading into 100-ton-capacity surface railroad cars by air-powered, pivoted, cutoff gates.

The shafts, begun in 1953, are southwest of the ore zone and are beyond the anticipated subsidence perimeter. Each shaft has four $6\frac{1}{2}$- by 7-foot compartments.

The balanced hoists (fig. 12) are double drum, 15 feet in diameter, have 116-inch faces, and accommodate 3,000 feet of $2\frac{1}{4}$-inch wire rope in two wraps with a maximum speed of 3,000 feet per minute. Each hoist is driven by two 3,000-hp. direct-current motors. The direct current to each hoist is supplied by a motor-generator unit consisting of a 4,000-hp. induction motor which drives two 2,500-kw. direct-current generators. The hoists are semiautomatically controlled, as the skip tender may start each hoisting cycle with push-button control from the underground loading stations. The centerline of the sheaves is 181 feet above the ground. The service cages, suspended by 1-1/8-inch wire ropes, are operated with small sinking hoists between each shaft and the main hoist house.

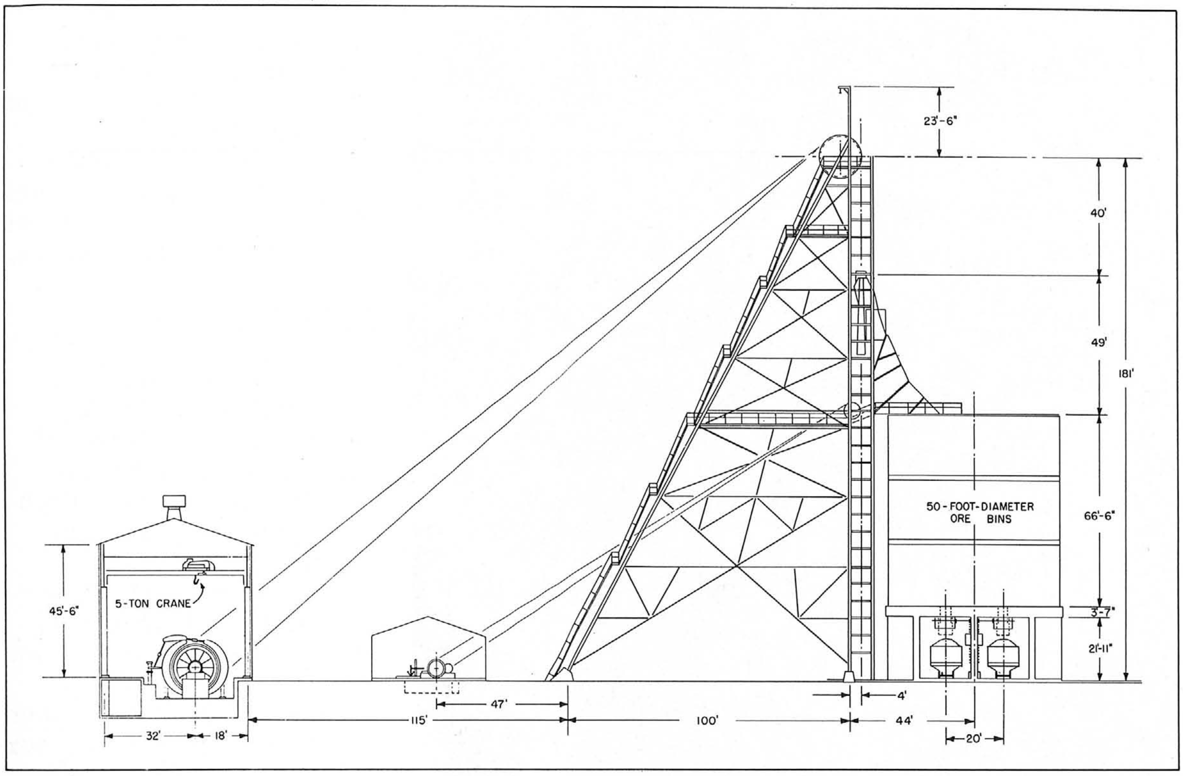


FIGURE 10. - Elevation Through 3A and 3B Shaft Installations.

Both shafts were sunk initially to 1,708 feet. Later, the 3B shaft was sunk to 2,310 feet to develop the 2075 level. Permanent stations were cut at the 1475 and 2075 levels. The 275-foot sumps below the haulage levels provide space for skip loading, for sump pumps, and for beginning future shaft-sinking operations. A small amount of supplies and equipment is handled in these shafts.

The following information on the original shaft sinking is abstracted principally from a description by C. L. Pillar[10], mine superintendent.

Bearing Sets and Headframes

Shaft excavation utilized the permanent headframes, which are of the conventional steel A-frame type (fig. 10). Sinking sheaves were placed 90 feet above the collars, and 180-ton temporary bins were built into the structures.

The shaft collars were installed by excavating 25 feet of unsupported raw shaft with a clamshell. Five 12-inch, wide-flange, 64-pound, 36-foot-long bearer beams were placed across the top of the shaft at the collar elevation.

[10] Pillar, C. L., Progress on Three Big Shafts Reveals Up-to-date Sinking Practice: Min. Eng. vol. 6, No. 7, July 1954, pp. 688-695.

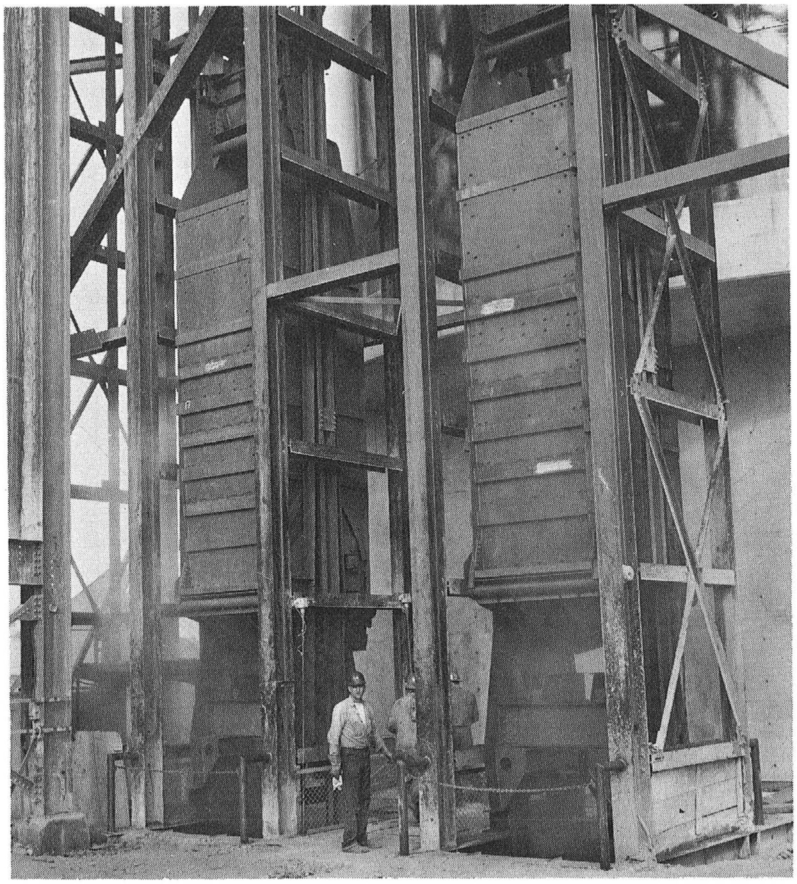


FIGURE 11. - "Jeto"-Type Bottom-Dump Skips.

FIGURE 12. - Interior View of Hoist House at Shafts 3A and 3B.

These were included in the collar fabrication and supported the sets immediately below. After placing the collar concrete, extended ends of bearer beams were cut off flush with the outside of the concrete shaft walls.

Sinking Crew

Crews of six men, employed on contract, worked in the bottom of each shaft on a three-shift-a-day basis. Each crew did all phases of the work, including drilling, mucking, placing steel and concrete, and installing manway and service lines. A top lander, truck driver, and hoistman provided the surface service for each shift. A two-way, blastproof loudspeaker system was installed to allow the shaft crew to talk with the surface operators.

Sinking Hoists

Sinking hoists for both shafts were double-drum clutched hoists driven by 250-hp. motors, with 15,000-pound rope pull and 850-f.p.m. hoisting speed. A 1,500-pound crosshead was attached to each rope, and a 38-cubic-foot sinking bucket was hooked 6 feet under the crosshead. The sinking hoists operated in the first and second compartments of each shaft. The fourth compartment of No. 3B was serviced by a 100-hp. single-drum hoist to handle sinking pump gear.

Drilling and Blasting

Drilling was done with jackhammers of the 55-pound class. An air and water manifold, equipped with line oilers, served as a rack to carry the drills and auxilary equipment to the bottom. Alloy drill steel and one-use, friction-type bits completed the drilling equipment.

The Gila conglomerate was relatively easy to drill but difficult to break. Fourteen rows of four holes, with a relieved V-cut across the center of the second or third compartment, were drilled $6\frac{1}{2}$ feet deep to pull a 6-foot round. Rounds were loaded with 125 pounds of straight 40-percent gelatin dynamite and detonated with regular-delay electric blasting caps.

Shaft Mucking

When the shafts were 25 feet deep, a shaft mucker with counter-weighted 3/8-cubic-yard clamshell (fig. 13) was installed to load broken rock. The mucking track covering the first three compartments was placed and chained to the steel sets. Dividers were omitted from the sets below the mucker and were placed after the machine was lowered, which, under normal conditions, was every 3 or 4 sets. Most of the rock encountered would stand unsupported for 20 to 30 feet. If the ground required support close to the bottom, the mucking track was cut to the two center sets. Then the full steel sets minus the No. 2 divider were placed. However, the open sump at the bottom allowed freer, more efficient use of the mucker. The last few tons of loose muck were hand-loaded or left in the center. The entire crew was used in the mucking cycle.

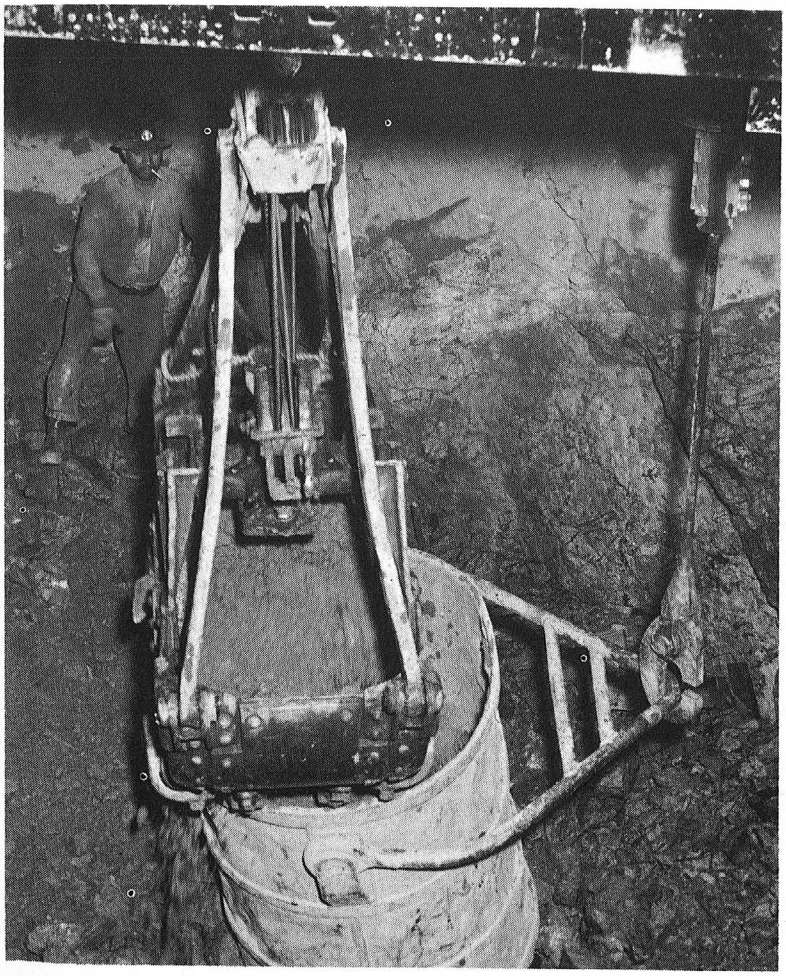


FIGURE 13. - Clamshell-Type Mucker in Operation at Bottom of Shaft.

Muck was hoisted to the surface bin in 38-cubic-foot buckets. The bucket dumps consisted of an underslung car, moved in and out of the shaft by an air cylinder, with a conventional keyhole slot and slide to overturn the bucket with the tail chain. This car completely covered one compartment of the shaft while the bucket was dumped. The complete surface operation, including setting the crosshead on the dumping chairs, was controlled by the top lander.

Ground Support

Steel sets and reinforced concrete provided permanent ground support in the twin shafts. A minimum of 12 inches of concrete was used, with the concrete positioned 6 inches inside the shaft steel. Reinforced concrete curtain walls, 12 inches thick, supported the shaft at the dividers. A 3- by 4-foot arched window in the curtain walls of each set provided attachment space for the guide brackets and for access between the compartments.

Steel sets were hung, alined, blocked, and, where required, lined with 2-inch lagging as sinking progressed. End plates and wall plates of the sets were made of 6-inch, 25-pound, H-section steel. Dividers were 11½-inch box beams built of two vertical 1-inch channels and 3/8- by 4-inch plate strapping. Each set was hung from 10 studdles made of 3½- by 3½- by 5/16-inch angle iron.

At the beginning of the concreting cycle, a set, 48 to 60 feet below the last concrete and approximately 18 to 30 feet above the bottom, was sealed off, and the concrete was placed up the shaft to the previous pour, one set at a time. Reinforcing steel was made of horizontal rings of 3/4-inch rods on 6-inch centers with vertical 3/4-inch rods on 12-inch centers. Reinforcing was welded into panels on the surface; this saved time on the bottom and insured the correct placing of the bars.

Forms for concreting weighed 90 pounds and were comparable to steel lagging 14 inches wide. They were supported and held in place vertically by 6-inch flange walers supported from the dividers and end plates. Each compartment was formed on all four sides with special steel arch forms bolted between the curtain will panels and the outside of the end wall panels to form the necessary windows and recesses for the guide brackets. Form panels were bolted together with speed bolts and blocked in place against the inside flange on the walers, and the concrete side of the forms was brushed with oil prior to pouring. Concrete was permitted to set 24 hours before the main forms were stripped and used to form sets above. Forms for windows in the curtain walls and recesses in the end walls were left in place for at least 7 days.

Concrete was mixed with a 1 : 2 : 4 ratio of cement, washed river sand, and crushed rock, pozzolana was added to give high early strength. Mix was fed from a 40-ton batch plant to a ½-cubic-yard mixer at the shaft collar. Mixed concrete was dropped through an 8-inch pipe into a small bottom gate hopper just above the pouring floor in the shaft. Small rubber-tired buggies were loaded from the hopper and dumped into the forms. This method of placing the concrete reduced segregation of aggregates and brought the pour up evenly in the forms. The six-man shaft crew could place the concrete as rapidly as it could be mixed properly. The concrete was vibrated in the forms by compressed air vibrators.

When water is encountered in shafts, the placing of good concrete becomes considerably more difficult. Best results under heavy water conditions in the No. 1 shaft were obtained by making the forms watertight and allowing the collected water to overflow the forms. This meant that the concrete was placed under water, and that the excess water was displaced and overflowed. This procedure carries some cement and fine aggregate with the overflow, but considerably less than when the water is washing through the forms. In pouring under wet conditions, the cement in the mix was increased 50 percent to offset the overflow loss. The compressive strength of test cylinders of concrete placed in dry zones averaged approximately 3,000 p.s.i. Concrete placed under wet conditions had approximately as much compressive strength when proper precautions were taken.

Weep holes were drilled in the sidewalls of each compartment on 12-foot vertical spacing to relieve hydrostatic pressure in water-bearing ground.

Shaft Dewatering

Water in the bottom of the shafts was handled by an eight-stage deep-well pump equipped with a 75-hp., 440-volt, 1,750-r.p.m. motor, and having a rated capacity of 500 g.p.m. at 420 feet. This unit was close-coupled, 14 feet long, and weighed 4,000 pounds. Similar units handled the water successfully in the bottoms of No. 1 and No. 2 shafts when they were sunk. The pump was connected to an 8-inch discharge line by a 50-foot length of high-pressure hose.

A booster station about 315 feet below the collar was cut to provide for pumps to relay the discharge to the surface. Provision was made for four 100-hp., 440-volt, 3,600-r.p.m., single-stage centrifugal pumps rated at 600 g.p.m. at 425-foot head, with four 500-gallon surge tanks to feed the booster pumps. Pumps were operated automatically by float switches. Additional booster stations were placed at 300-foot intervals as the sinking progressed.

To prevent large quantities of water from falling free in the shaft over vertical distances in excess of 50 feet, water rings were formed by omitting the concrete lining for a vertical distance of 2 feet below the concrete seal-off sets, which were spaced approximately 48 feet apart. A headboard around the bottom of this recess formed a ditch 18 by 12 inches around the perimeter of the shaft. This ditch trapped most of the water falling down the shaft walls from the weep holes and other leakage points. A 300-g.p.m. pump, set below the ring and fed by gravity from it, discharged the water into the sinker columns. As upper rings dried up with shaft advance, the pumps were moved to lower rings. It was possible to connect two or more rings in order to concentrate water in the lower rings for pumping. The rings improved working conditions at the bottom of the shafts and lightened the load on the sinker pumps.

Ventilation

The shafts were ventilated by fresh air coursed through a 24-inch, light-walled vent pipe, the discharge of which was usually kept at the bottom of the concrete lining. One $7\frac{1}{2}$-hp., 440-volt, axial vane fan, rated at 7,000 c.f.m. under 4 inches of water-gage pressure, ventilated each shaft. Experience showed that after shafts became wet, the fans were required only after blasting.

Rate of Advance

Shaft sinking was cycled on 6-foot rounds. Drilling time in shafts 3A and 3B was 4 to 6 hours, and mucking time was 4 to 6 hours. The rate of advance averaged about 115 feet per month of completed shaft.

No. 4 Service and Supply Shaft

The sinking of the No. 4 shaft began in 1953. It was designed to be the main service and supply unit for the operation, and was connected with all working levels. This shaft was sunk with essentially the same technique and procedure as that used in the No. 3 shaft. Steel and reinforced concrete were used for ground support, with mahogany guides for the cages. Men (fig. 14) and supplies were transported in 7- by $13\frac{1}{2}$-foot, double-deck cages that travel at a maximum speed of 1,500 feet per minute. Each cage can hold 100 men or two 8-ton locomotives. The cages are equipped with rubber-tired rollers and guide shoes. The 15-foot-diameter, double-drum hoist is driven with two 700-hp., 600-volt, direct-current motors. The direct current is supplied by a motor-generator unit consisting of a 1,750-hp., 2,300-volt induction motor driving two 600-kw. direct-current generators. The drum face is 90 inches wide and winds 1,500 feet of $2\frac{1}{4}$-inch wire rope in a single layer.

A Cryderman shaft mucker was used to sink from the 1582 level to the sump below the 2075 level.

The No. 4 shaft is divided into two 8- by 14-foot compartments. The ends of the shaft are rounded, giving a total cross-sectional area of nearly 400 square feet.

Table 4 gives statistics of the No. 4 shaft, and table 5 contains data relating to the sinking of all the shafts.

Haulage-Level Drifts

The main haulageway for the first lift is the 1475 level; it was laid out with two main haulage drifts looping around the ore zone and connecting with No. 3A and No. 3B shafts (fig. 15). The double loop provides one-way traffic along two routes and sufficient cross-sectional area for exhaust ventilation; it also prevents congestion of the main ore haulageway. These drifts are parallel and 100 feet apart. They are connected at 1,000-foot intervals with crossovers so that traffic can be routed from one drift to another around a wreck or a section of drift under repair, or for any other reason. A service crosscut provides access from the No. 1 and No. 4 shafts.

Haulage drifts are numbered from north to south, the northernmost being the No. 1 drift. The 1475 level was developed initially by driving 5.29 miles of mainline and 5.79 miles of panel drifts. The panel haulage, or loading, drifts pass through the ore zone at right angles to the main haulage loop. These drifts are connected with the main haulage loop by ladder drifts in order to minimize the turnouts from the mainline.

FIGURE 14. - Shift Change at Collar of Shaft No. 4.

The first haulage level was developed principally from the No. 1 and No. 2 shafts. The double haulage loop was driven to the ore hoisting area before shafts 3A and 3B were down to the level.

The plan of development of the second haulage (2075) level (fig. 16) is similar to that of the 1475 level.

Drilling, Blasting, and Loading

Haulage-level drifts were driven with crews which averaged 3½ men per shift. A two-machine jumbo (fig. 17) mounting 3-inch drifters with 7-foot feeds was used for drilling. The jumbo has boom-type, mechanically powered drill arms. The booms are mounted on a shop-built car with a 1,000-pound counterweight. This car is very stable and requires no blocking or jacks. Thirty to forty holes are sufficient to break a 4½- to 6-foot round. They are loaded with 40-percent straight gelatin blasting powder and are detonated in wet areas by electric caps. In dry areas, for speed and safety, the holes are detonated with thermite connectors and hot-wire lighters.

Overshot loaders are used for mucking. Broken rock is loaded into 100-cubic-foot, A-bottom dumpcars, which are hauled with 8-ton battery locomotives.

When a heading is advancing, a cut for horizontal car transfer is made about every 400 feet. It was determined that 400 feet is the maximum distance that a traincrew can switch cars without the mucking machine waiting for an empty. After being abandoned for car transfer, these wide zones in the drifts are used for material storage, transformer substations, sanitation stations, and so forth.

TABLE 4. - Operational statistics, shaft No. 4[1]

Dimensions, outside	feet	16 x 27
Total depth (initial sinking to first level only)	do	1,582
Total statistical time	days	347
Excavated during statistical period	feet	1,486
Timbered during statistical period	do	1,490
Concreted during statistical period	do	1,424
Average man-shifts in shaft per shift		5.38
Shifts excavating		2,785
Percent of total		49.69
Shifts timbering		1,193
Percent of total		21.29
Shifts concreting		1,483
Percent of total		26.46
Shifts, other		143.4
Percent of total		2.56
Sinking rate, completed shaft	feet per day	4.2
Sinking rate, completed shaft	feet per month	127.5
Man-shifts per foot of excavation		1.87
Man-shifts per foot of timbering		.80
Man-shifts per foot of concreting		1.04
Man-shifts per foot, miscellaneous		.10
Muck per foot	tons	42.9
Overbreak	percent	28.0
Blasting powder per foot	pounds	32.0
Concrete per foot	cubic yards	5.30
Cement per foot	sacks	31.9
Steel sets per foot	pounds	1,178
Reinforcing steel per foot	do	257
Holes blasted per round		83
Advance per round blasted		6.06

[1] Preliminary work period (days pay) 7/19/53 to 12/26/53
Statistical period (contract work) 12/26/53 to 12/28/54

Scheduling is done so that two development headings are driven simultaneously wherever possible; one heading is drilled while the other is being mucked out and timbered. This gives maximum utilization of men and equipment.

Ground Support

Drifts are timbered with 10-foot caps and $10\frac{1}{2}$-foot posts. In most of the drifts 12- by 12-inch treated timbers are used, but in some faulted areas 12-inch 53-pound steel caps are used. All turnouts in the system are driven with 165-foot radius curves and are supported by 12-inch 53-pound steel caps on 4-foot centers. All caps are top lagged with 3- by 6-inch lagging, and sides of the drifts are lagged with 2- by 10-inch planks. In the turnout areas where excessive weight and repair are anticipated, the H-beams are reinforced to the equivalent of 90-pound beams. When two of these heavy ground areas were grouted, the results were so satisfactory and the decrease in ground movement so evident that the grouting program was expanded. Grouting procedure is discussed later in this report.

TABLE 5. - Shaft-sinking data, all shafts

Shaft	No. 1	No. 2	No. 3A	No. 3B	No. 4
Use...............	Development, service, and ventilation.	Exploration and development.	Ore hoisting and ventilation.	Ore hoisting and ventilation.	Supply, service, and ventilation.
Shape and size (inside).........	Rectangular, 25 feet 4-3/4 inches by 6 feet.	Rectangular, 18 feet $6\frac{1}{2}$ inches by 5 feet 6 inches.	Rectangular, 29 feet by 7 feet.	Rectangular, 29 feet by 7 feet.	Rectangular with rounded end compartments, 25 feet 6 inches. by 14 feet.
Number of compartments.....	2 skip hoisting, 6 feet 6 inches by 6 feet; 1 manway, 5 feet by 6 feet; 1 service cage, 5 feet by 6 feet.	2 skip hoisting, 5 feet $7\frac{1}{2}$ inches by 5 feet 6 inches; 1 manway, 5 feet $7\frac{1}{2}$ inches by 5 feet 6 inches.	2 skip hoisting, 6 feet 6 inches by 7 feet; 1 manway, 6 feet 6 inches by 7 feet; 1 service cage, 6 feet 6 inches by 7 feet.	2 skip hoisting, 6 feet 6 inches by 7 feet; 1 manway, 6 feet 6 inches by 7 feet; 1 service cage, 6 feet 6 inches by 7 feet.	2 supply cages, 8 feet by 14 feet; 2 manway and pipeway circular segments, 3 feet 6 inches by 14 feet.
Type of support...	6- to 10-inch structural steel on 6-foot centers, reinforced by 12-inch concrete.	10- by 10-inch treated timber on 5-foot centers. Lines with 2-inch timber lagging.	6- to 2-inch structural steel on 6 foot centers, reinforced 12-inch concrete lining with 12-inch curtain walls.	6- to 12-inch structural steel on 6-foot centers, reinforced 12-inch concrete lining with 12-inch curtain walls.	8- to 12-inch structural steel on 6-foot centers, reinforced 12-inch concrete lining.
Depth shaft sunk during initial sinking period...	1,643 feet	2,064 feet	1,708 feet	1,707 feet	1,582 feet

TABLE 5. - Shaft-sinking data, all shafts (Con.)

Shaft	No. 1	No. 2	No. 3A	No. 3B	No. 4
Average cross section, rock excavation.......	244 square feet	168 square feet	306 square feet	306 square feet	420 square feet
Sinking crew per shift:					
Surface........	1 hoistman 1 toplander 1 truckdriver	1 hoistman 1 toplander 1 truckdriver	1 hoistman 1 toplander 1 truckdriver	1 hoistman 1 toplander 1 truckdriver	1 hoistman 1 toplander 1 truckdriver
Underground....	1 shaft leadman 5 shaftmen	1 shaft leadman 4 shaftmen	1 shaft leadman 5 shaftmen	1 shaft leadman 5 shaftmen	1 shaft leadman 5 shaftmen
Excavation:					
Drilling:					
Equipment.....	Six 55-pound hand-held sinkers	Five 55-pound hand-held sinkers	Six 55-pound hand-held sinkers	Six 55-pound hand-held sinkers	Six 55-pound hand-held sinkers
Type round....	V cut	V cut	V cut	V cut	V cut
No. of holes..	66	50	66	66	83
Average depth drilled......	6.5 feet	5.5 feet	6.5 feet	6.5 feet	6.5 feet
Average depth pulled.......	(1)	(1)	5.33 feet	5.23 feet	6.06 feet
Blasting:					
Type..........	Electric conventional delays	Electric conventional delays	Electric conventional delays	Electric conventional delays	Electric conventional delays
Wiring........	Series Parallel	Series parallel	Series parallel	Series parallel	Series parallel
Powder........	40 percent gelatin	40 percent gelatin	40 percent gelatin	40 percent gelatin	40 percent gelatin
Powder consumption..	(1)	(1)	28.1 pounds per foot	27.8 pounds per foot	32.0 pounds per foot

Mucking:					
Type..........	Riddell single 3/8-yard clamshell	Riddell single 3/8-yard clamshell	Riddell single 3/8-yard clamshell	Riddell single 3/8-yard clamshell	Riddell double 3/8-yard clamshell
Muck hoisted per foot of shaft........	[1]	[1]	27.6 tons	27.6 tons	42.9 tons
Overbreak.....	[1]	[1]	13 percent	13 percent	28 percent
Total Excavation:					
Man-shifts per foot of shaft..	[1]	[1]	1.68	1.86	1.87
Support:					
Structural steel or timber installation, man-shifts per foot of shaft..	[1]	0.87	.87	.92	.80
Steel used per foot of shaft..	[1]	Timber, 350 feet board measure	808.5 pounds	808.5 pounds	1,178 pounds
Reinforced concrete lining:					
Installation man-shifts per foot of shaft.......	[1]	[2]	1.28	1.41	1.04
Concrete placed per foot of shaft.......	[1]	[2]	5.71 cubic yards	5.75 cubic yards	5.30 cubic yards
Rebar placed per foot of shaft.......	[1]	[2]	496 pounds	496 pounds	257 pounds

See footnotes at end of table, p. 32.

TABLE 5. - Shaft-sinking data, all shafts (Con.)

Shaft	No. 1	No. 2	No. 3A	No. 3B	No. 4
Miscellaneous:					
Piping, power, pumping installation, man-shifts per foot of shaft..	([1])	0.49	0.09	0.14	0.10
Total Sinking:					
Average rate of completed shaft per month..........	88.1 feet	110.5 feet	115.2 feet	111.3 feet	127.5 feet
Average advance per man-shift, all labor......	.186 feet	.307 feet	.256 feet	.231 feet	.262 feet
Depth in shaft water was encountered....	1,100 feet	3.7 feet	300 feet	339 feet	1,475 feet
Maximum water inflow encountered....	1,130 g.p.m.	1,300 g.p.m.	25 g.p.m.	10 g.p.m.	748 g.p.m.

[1] Not available.
[2] None used.

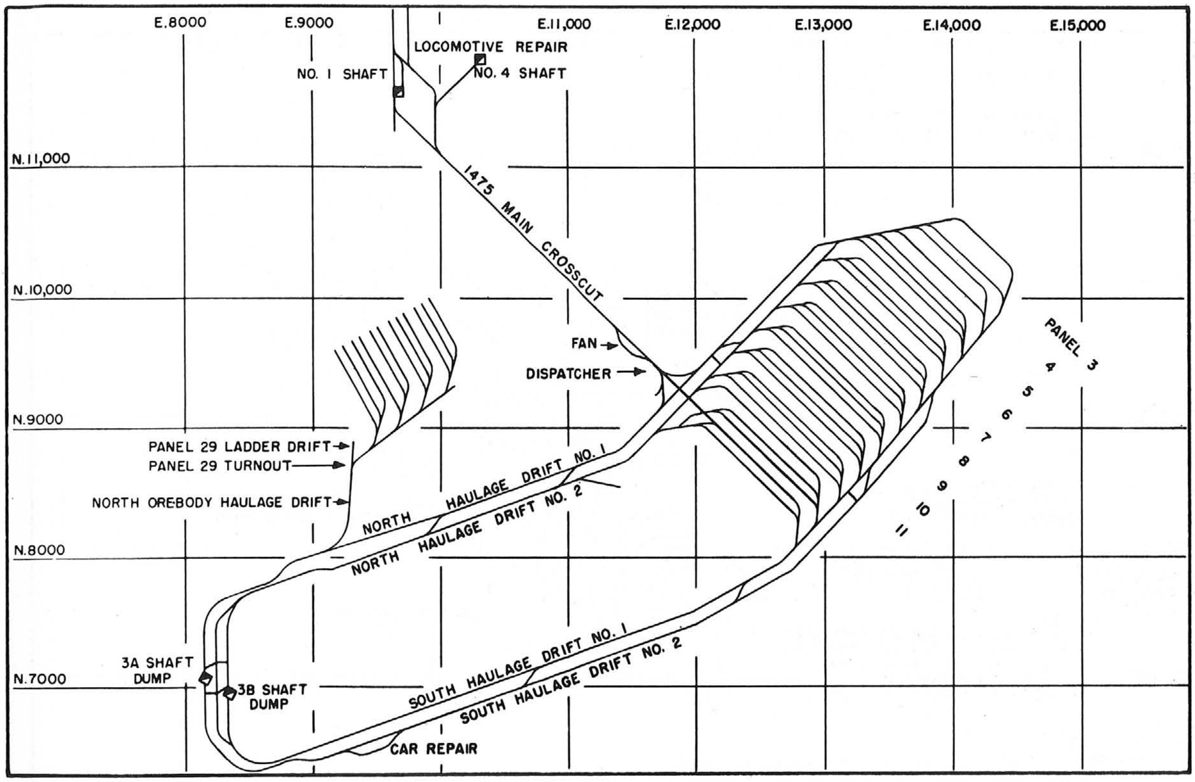


FIGURE 15. - Plan of 1475 Haulage Level.

Panel drifts are driven the same size as the main haulage drifts, and ground support is nearly identical; about the only difference is the location of the track. In panel drifts the track is laid on centerline so that muck may be pulled from chutes on either side. In the main haulage drifts the track is offset to allow for a drainage ditch on one side. Track is not ballastered in panel drifts.

Man-shifts per foot of drift and drift footage at the 1475 level are given for 6-month periods in table 6.

Haulage Track and Retrack

During the driving of the main haulage drifts, a temporary track, consisting of 45-pound rail laid to 36-inch gage, was installed 1 foot below the final track grade. The permanent track, consisting of 90-pound rail laid to 36-inch gage, was installed on 6- by 8-inch by 6-foot treated ties placed on 18-inch centers. The track was ballasted with 1 foot of 2-inch crushed granite, leaving a drainage ditch on one side. The track grade is 0.45 percent in favor of the load.

The 90-pound rail was set in place by a two-man crew with a tugger-powered crane mounted on a track car. At first, the ballast was dropped down the 8-inch

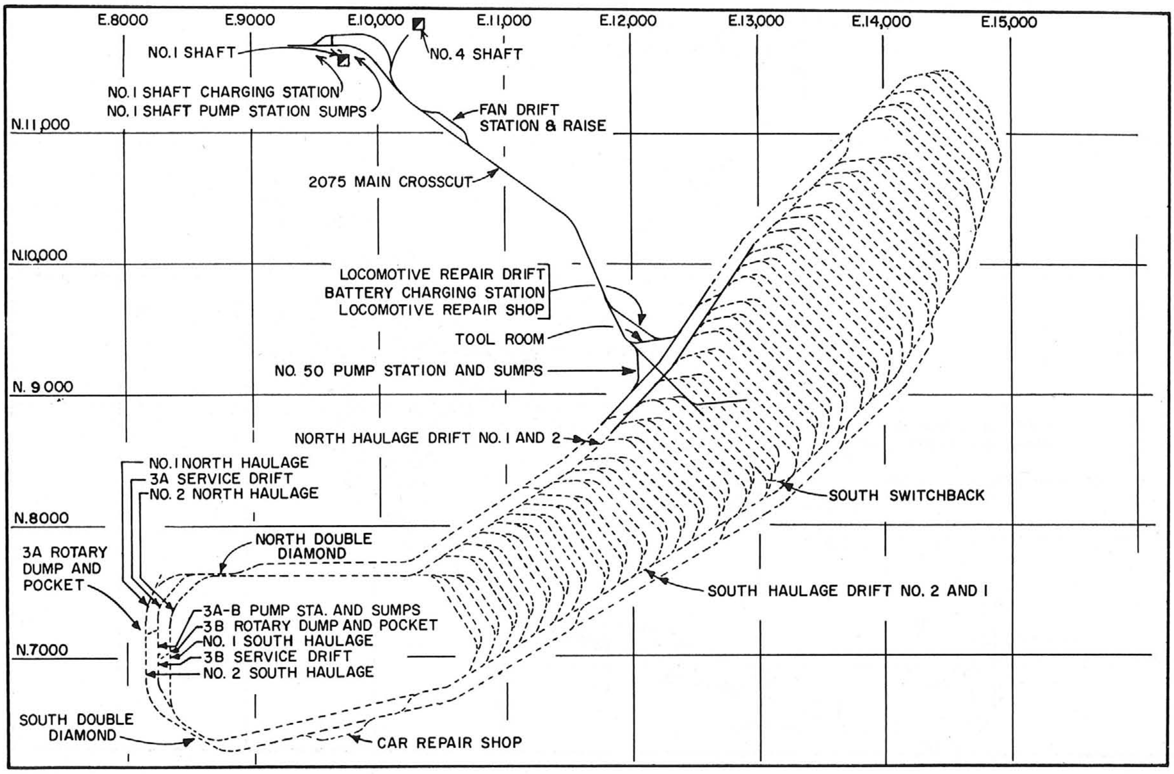


FIGURE 16. - Proposed Development of the 2075 Haulage Level.

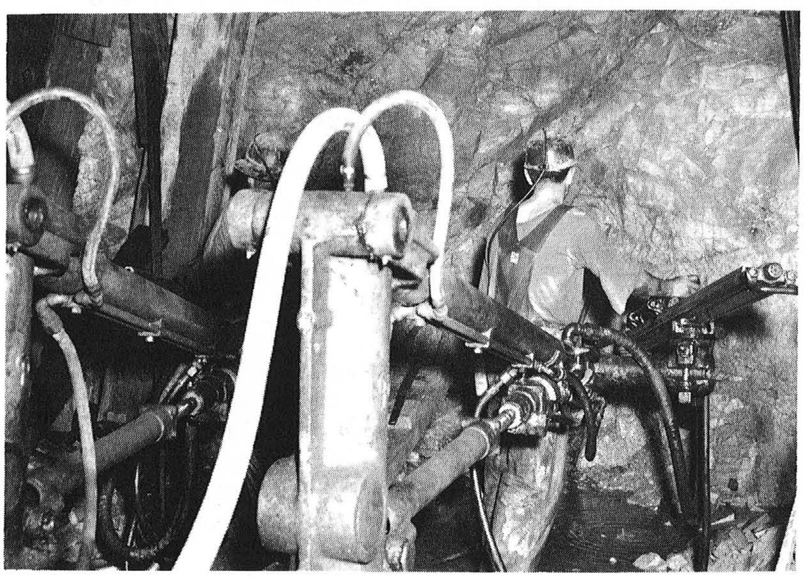


FIGURE 17. - Two-Machine Jumbo at Face of Drift.

shaft pipe normally used for concrete, but after No. 4 service shaft was completed to the haulage level in April 1955, ballast cars were filled on the surface and lowered in cages.

The retrack crew was equipped with an air-powered rail saw; hand-held ballast tampers; acetylene torches; four 5.4-cubic-yard, V-bottom, ballast-spreading cars with adjustable discharge; an 8-ton battery locomotive; and a rocker shovel loader when it was needed.

TABLE 6. - Man-shifts per foot of drift and drift footage, 1475 level (6-month periods)

Period	7/52 to 1/53	1/53 to 7/53	7/53 to 1/54	1/54 to 7/54	7/54 to 1/55	1/55 to 7/55
	Man-shifts per foot of drifts					
Drifts from No. 2 shaft.....	1.03	1.04	--	--	--	--
Drifts from No. 1 shaft.....	--	1.08	--	--	--	--
Level total for period......	1.03	1.05	1.05	1.09	1.23	1.26
Level total to date.........	1.03	1.05	1.05	1.06	1.11	1.15
	Drift footage					
Footage for period..........	3,981	6,062	8,391	10,927	10,424	12,305
Footage to date............	3,981	10,043	18,434	29,361	39,785	52,092

For statistical purposes, switches, crossovers, and so forth were given values in terms of feet of track. The retrack task was then calculated as follows:

38	90-pound switches at 300 feet..........................	11,400
31	70-pound switches at 300 feet..........................	9,300
2	Double-diamond crossovers at 1,000 feet................	2,000
2	90° crossovers at 300 feet.............................	600
	90-pound track...	29,656
	Total..	52,956

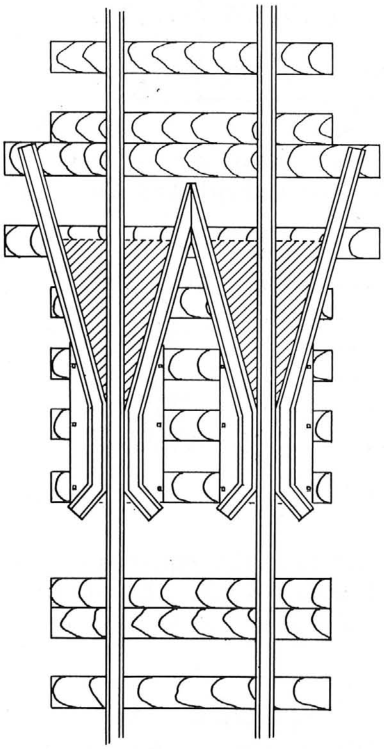
Retrack statistics for 6-month periods are listed in table 7.

Panel haulage tracks were laid permanently with 70-pound rail equipped with a permanently installed rerailer (fig. 18) ahead of each block. Any cars derailed during loading were automatically rerailed as the train advanced through this rerailer.

Automatically operated switches were installed on the loadline. The switches were alined by the front wheels of the locomotive of the loaded train as it left the panel. Electrically operated spring switches were installed on the empty line. They were activated manually by the operator of the empty train on its return trip to the panel drifts. All switch points are protected by guardrails.

TABLE 7. - Retrack statistics (6-month periods)

	10/1/54 to 4/1/55	4/1/55 to 10/1/55	10/1/55 to 4/1/56	4/1/56 to 10/1/56
Advance for period................feet	10,580	28,670	10,300	2,100
Advance to date....................do.	10,580	39,250	49,550	51,650
Man-shifts for period................	2,951	4,533	3,072	---
Man-shifts to date...................	2,951	7,484	10,556	---
Man-shifts per 100 feet of track for period....................	27.89	15.81	29.83	---
Man-shifts per 100 feet of track to date......................	27.89	19.07	21.30	---
Ballast for period.........cubic yards	1,868	3,129	2,716	285
Ballast to date................do.....	1,868	4,997	7,713	7,998
Ballast per 100 feet of track, to date................do.....	17.66	12.73	15.57	15.48



NOT TO SCALE

FIGURE 18. - Rerailer (70-pound).

Two crossovers had double-diamond switches, one located on the approach to the rotary dumps above the shaft pockets and the other on the departure from the dumps. The double-diamond turnout on the approach, or north, haulage was 835 feet in front of the rotary dump, and the one on the departure, or south, haulage was 1,058 feet from the dump. This type of turnout permitted trains coming down either haulage drift No. 1 or No. 2 to be routed into either shaft, and it expedited movement by avoiding delays from congestion in the dumping area. The double diamond on the return haulage loop facilitated the return of empty trains to the panels.

Grizzly-Level Drifts

The grizzly working level for the first lift was the 1415 level (fig. 19). The working areas were connected with No. 1 and No. 4 shafts by a service crosscut and a ventilation crosscut. Two fringe drifts, one along the north edge of the ore zone and the other along the south edge, provided access to the fringe panel drifts, which transected the ore zone across its strike at intervals of 245 feet.

Drifting technique was the same on the grizzly level as on the haulage level.

The fringe drifts and the service drift were timbered with 9-foot 5-inch posts and 10-foot caps. Steel caps were used where it was deemed advisable. The ventilation crosscut was 12 by 12 feet in section.

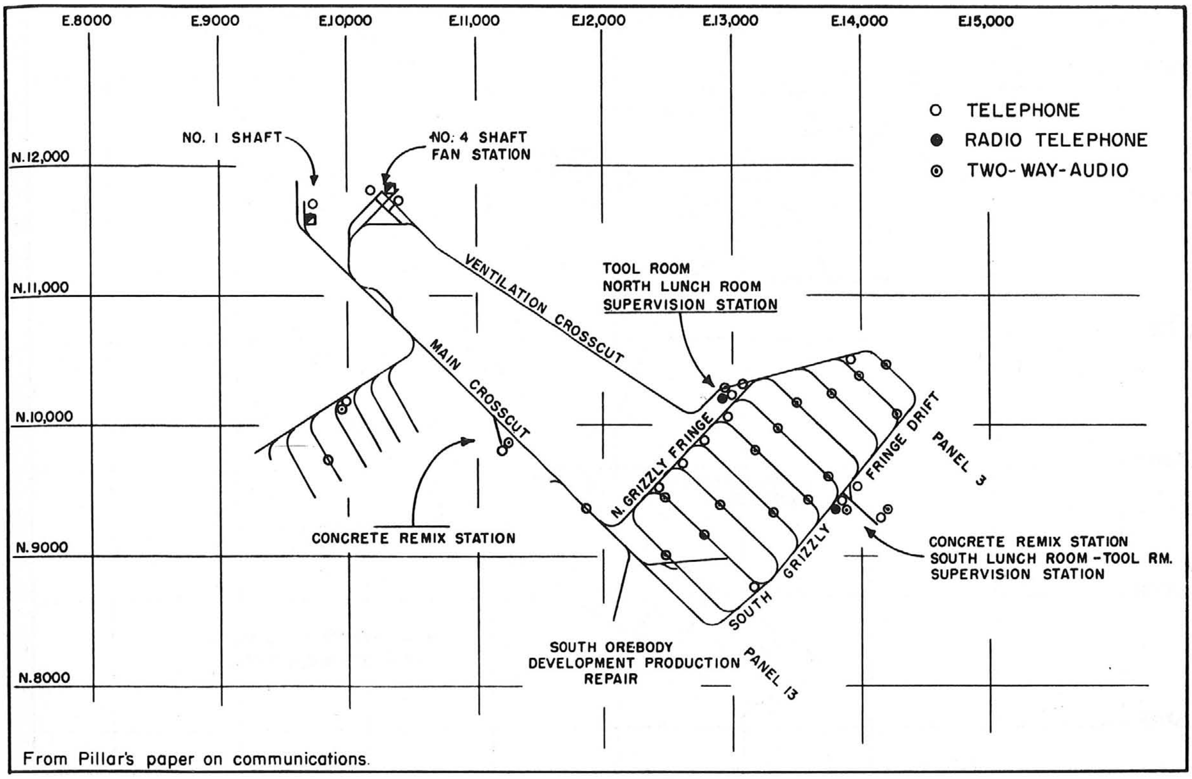


FIGURE 19. - Plan of 1415 Grizzly Level.

It was supported by steel sets with arched caps. In the beginning, fringe panel drifts were timbered with 8-foot caps and 8-foot 5-inch posts. Because of the high timber repair costs in the panel drifts, monolithic concrete was substituted for timber. The results obtained were very favorable, and all grizzly panel drifts were concreted. At first, wooden forms were used, but these were replaced later with collapsible steel forms mounted on a standard-gage track jumbo. The concreting procedure was discussed extensively under Mine Development.

All track on the grizzly level was installed with 45-pound rail. Curve radii on the grizzly level were 100 feet with 105-foot radius turnouts, which was the minimum suitable for the haulage equipment.

The plan of development of the grizzly level (2015) for the second lift was similar to that of the 1415 level (fig. 20).

Man-shifts per foot of drift and drift footages for the grizzly level are summarized in tables 8 and 9.

Drift Stations

Drift crews cut a variety of types and sizes of stations, including those for battery charging, electrical substations, rectifiers, locomotive repairs,

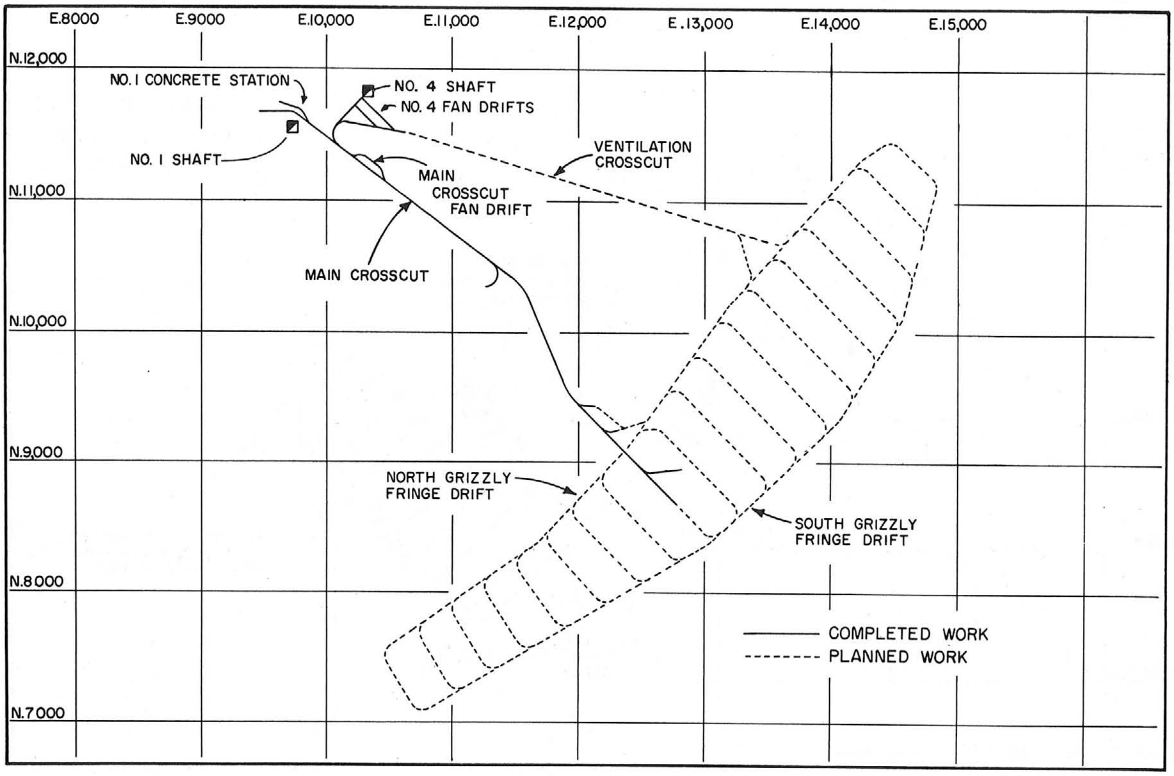


FIGURE 20. - Proposed Development of 2015 Grizzly Level.

car repairs, dumps, dispatcher, lunchrooms, toolrooms, and so forth. All stations are supported with wood, steel, concrete, or combinations of these materials. Timber is usually protected from fire with gunite or plaster, particularly where any electrical equipment is located.

TABLE 8. - Man-shifts per foot of drift and drift footages, 1415 level (6-months periods)

Period	To 11/17/52	11/17/52 to 7/53	7/53 to 1/54	1/54 to 7/54	7/54 to 1/55	1/55 to 7/55
	Man-shifts per foot of drift					
Drifts from No. 2 shaft....	1.92	0.89	0.78	--	--	--
Drifts from No. 1 shaft....	--	1.21	.93	--	--	--
Level total for period.....	1.92	1.00	.83	0.94	0.99	0.76
Level total to date........	1.92	1.05	.91	.92	.93	.91
	Drift footage					
Footage for period.........	146	3,132	6,002	5,352	4,526	4,331
Footage to date............	146	3,278	9,280	14,632	19,158	23,489

TABLE 9. - Man-shifts per foot of drift and drift footages, 1415 ventilation drift, arch steel-supported drift

Period	3/22/54 to 7/55	7/54 to 1/55	1/55 to 5/3/55
	Man-shifts per foot of drift		
For period....	1.22	1.16	1.08
To date.......	1.22	1.18	1.11
	Drift footage		
For period....	598	1,603	1,193
To date.......	598	2,201	3,394

Block Development

To assure a steady production from the block-caving operations, the ore bodies (north and south) were divided into self-contained units (so-called full-protected block caving) that could be developed, caved, and mined gradually, independently, and according to schedule. The bodies were divided by levels in the vertical plane into 600-foot lifts that, in turn, were divided into panels and blocks in the horizontal plane. Since each block had a limited daily output, 8 to 13 blocks had to be drawn for the total daily production; thus, several blocks had to be developed ahead of production. In general, production at San Manuel comes from one level while the next level is being developed.

The main, or south, ore body was laid out in panels numbered from northeast to southwest, and panels were subdivided into blocks numbered from northwest to southeast. Block 6-1 was the most northwesterly block in the sixth panel from the northeast boundary of the ore body. Panels were normally 210 or 140 feet wide; the length of the blocks ranged from 150 to 270 feet, depending on the number and spacing of the grizzly drifts in the block.

In general, the all-gravity caving method was used. On the first lift the slusher-gravity method was used for boundary blocks where the ore column was low. Development costs for the slusher-gravity method were less than for the all-gravity method.

Annual data pertaining to footage of mine workings driven for block development is listed in table 10.

Gravity Blocks

Raise Stations and Transfer Raises

Some time after haulage-panel, grizzly-fringe, and fringe-panel drifts were completed, development of the block began with excavating transfer raise stations in panel drifts on the haulage level. Drift lagging was removed, and the ground was drilled and blasted to make room for the station and to hang temporary (joker) chutes. Quite often it was necessary to grout the arch of the drift before drilling and blasting was done. Details of this procedure are discussed later in this paper.

TABLE 10. - Summary of block development footage per year

Year	Raise stations	Transfer raises, feet	Grizzly drifts, feet	Draw raises, feet	Undercut drifts, feet	Undercut pillars, square feet
1954...........	1	161	323	--	--	--
1955...........	127	14,286	6,154	11,235	6,238	20,535
1956...........	106	17,457	10,223	15,183	14,999	171,914
1957...........	154	15,180	9,488	16,660	11,381	145,270
1958...........	154	22,165	11,653	24,541	20,350	321,897
1959, (first 6 months).....	58	10,936	5,890	12,540	12,460	222,459

The stations are 6-1/2 feet high, 11 feet long, and as wide as the haulage drift. Raises were driven to 2 feet above sill sets on the grizzly level, and backover raises were driven as shown on figure 21. The raises measured 4 by 4 feet inside timber and were lined with 6- by 8-inch cribbing. Where hard, abrasive rock was encountered, or anticipated, 3- by 4- by 3/8-inch angle iron for armoring was installed on the crib rings. The last round was drilled in the transfer and backover raises to form an arch for natural support until the grizzly drifts were driven.

Permanent steel chutes, which were practically free of ground support and may be described as floating, were installed on trunnions supported by the two caps of adjacent drift sets (fig. 22). The undercut guillotine door was operated up and down by an 8-inch-diameter air cylinder with a 36-inch stroke. The door passed through a slot in the bottom of the chute and was held in guides at each side (fig. 23). The door had 3/4- by 2- by 36-inch extensions to prevent the guide from filling with muck. Cylinder and door guides were supported by H-beam and angle iron which was bolted to the bottom of the chute slide. This type of chute, with its 36- by 39-inch opening, gave ample room for barring down or blowing out the chute throat, and the positive undercut action of the door made it virtually impossible for a boulder to block the chute open.

Three men per shift comprised a raise-station crew. Each crew was equipped with an 8-ton battery locomotive, six 5-ton cars, a rocker shovel loader, a stoper, and an air-leg drill.

Each of six two-man transfer-raise crews drove two raises, single shift. Two stopers and two single-drum, air-driven hoists were used by each crew to drive a total of 1,350 feet of transfer raises per month, for an average advance of 4.55 feet per man-shift.

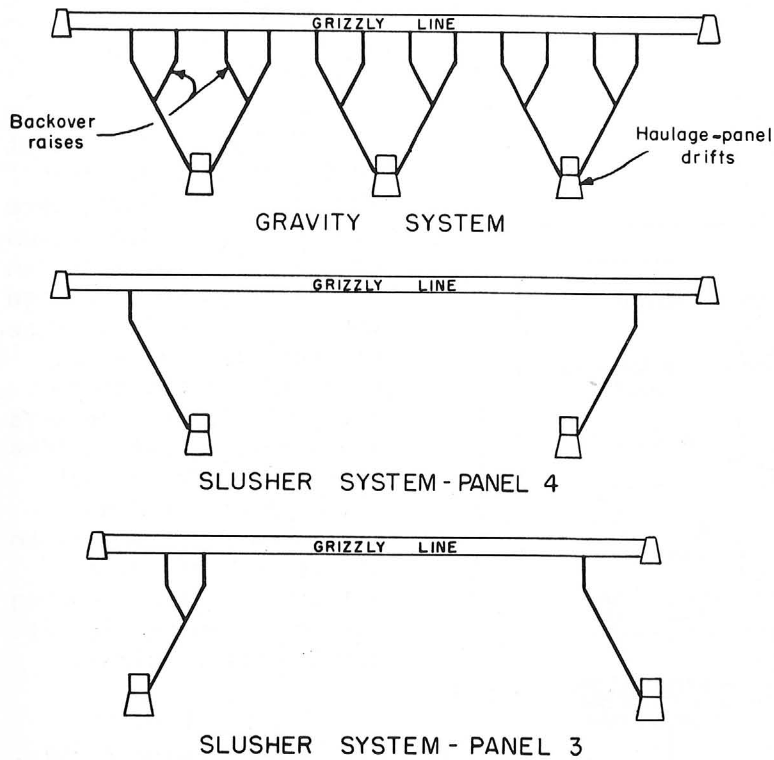


FIGURE 21. - Schematic Sections Contrasting Development of Gravity and Slusher Blocks.

Grizzly Drifts

The grizzly level was 60 feet above the haulage level. Grizzly drifts were driven on 30- to 45-foot centers along the length of the block after the transfer and backover raises were completed. At first, the drifts were supported with 10- by 10-inch and 12- by 12-inch timber sets. Because of high timber repair costs, circular rigid steel sets, circular yieldable steel sets, and monolithic concrete were tested. Subsequently, ground support by concrete lining was adopted as the method of support. Details as to design, results, and costs are discussed in a subsequent section on concreting. Figures 24, 25, and 26 show the various methods used.

The grizzly drifts were driven 8 feet wide by 8-1/2 feet high in rock section (fig. 27). The ground was supported by 6-foot rock bolts and chain-link fencing. Badly fractured ground was supported by light timber sets. Concrete, 1 foot thick, was poured around the top of transfer raises to a depth of 5 feet below sill level, and grizzlies of 45-pound rail with 12-inch maximum spacing were installed. Next, forms were placed in the drifts, and concrete was poured at least 1 foot thick on ribs and 1-1/4 feet thick in the arch. Completed drifts measured 4 feet wide by 6-1/2 feet high (fig. 28). Five two-man crews per shift drove the required 987 feet of grizzly drifts per month, with an average advance of 1.38 feet per man-shift.

Draw Raises

The first undercut levels were placed 25 feet above the grizzly level. This interval was reduced to 20 feet when the drifts were supported with concrete lining. After these grizzly drifts were completed, finger or draw raises were driven to 2 feet above the undercut level. Normal spacing of the raises was 15 by 17.5 feet. They were 5 feet in diameter at the grizzly level, with the last round belled-out to 8 feet in diameter at the undercut level. The raises were lightly timbered. Four two-man crews averaged 1,920 feet per month, with an average progress of 10.85 feet per man-shift. Figure 5 shows the relationship of the draw raise to the grizzly and undercut drifts.

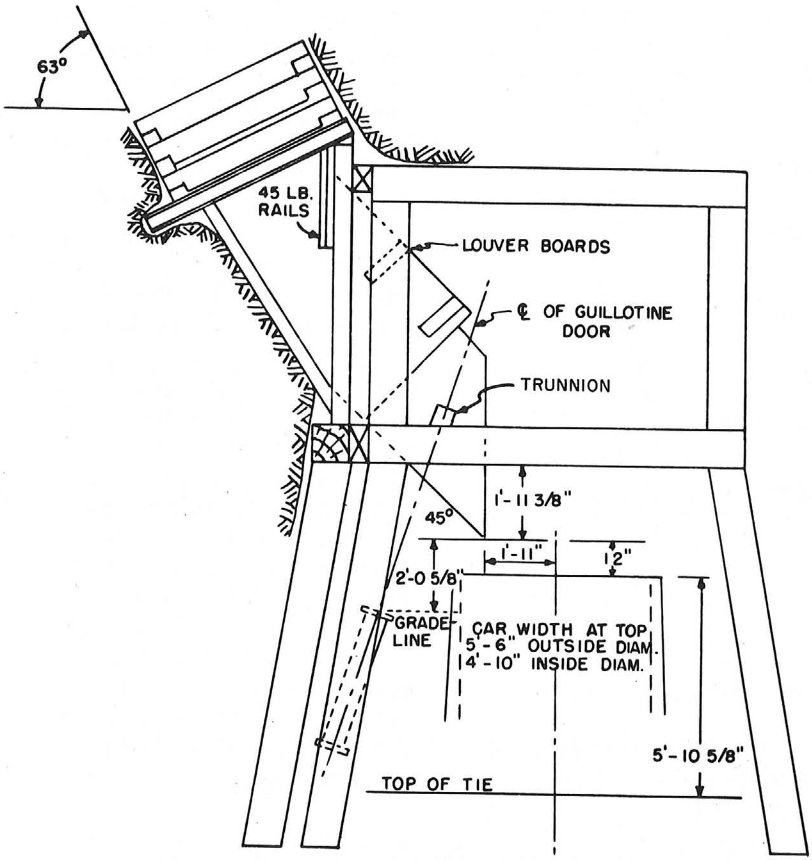


FIGURE 22. - Loading Chute for Transfer Raises.

Undercutting

Blocks were undercut to induce caving. When the sides of a block were adjacent to virgin ground, corner raises were driven 30 feet high to encourage boundary weakening. Three or four draw raises on each side of the stope were selected for manways and supply. Small drifts were driven to connect the tops of the draw raises at right angles to the centerlines of the grizzly drifts. Slabbing was commenced before all drifts were completed.

The width of the drifts was increased to 15 feet, and the backs were slabbed to a height of 15 feet above the undercut floor. This height was considered necessary to take care of the swelling of broken rock and caving action. The slabbing operation was started at one corner of the block and progressed toward the opposite corner. The undercutting was completed by drilling and blasting from one side of the pillars remaining between the drifts. Drawing commenced immediately to give room for good caving action.

Timber was used on the undercut level to assure safe working conditions. Peeled pine poles, 6 inches in diameter and 6 feet 6 inches long, 6- by 8-inch timber, and 2- by 12-inch lagging were used. None of this timber was treated (fig. 29).

Timber was augered, loaded, and shot with the pillars. Delay electric caps, used in pillar blasting, were wired in parallel series, with not more than 25 primers in any one series. Each series within a circuit was tested with a galvanometer, and then the circuit was wired to a No. 12-2 conductor cable from the power source. All blasting lines passed through an interrupter switch and from this switch to the main pillar blasting switch installed on an independent electrical circuit.

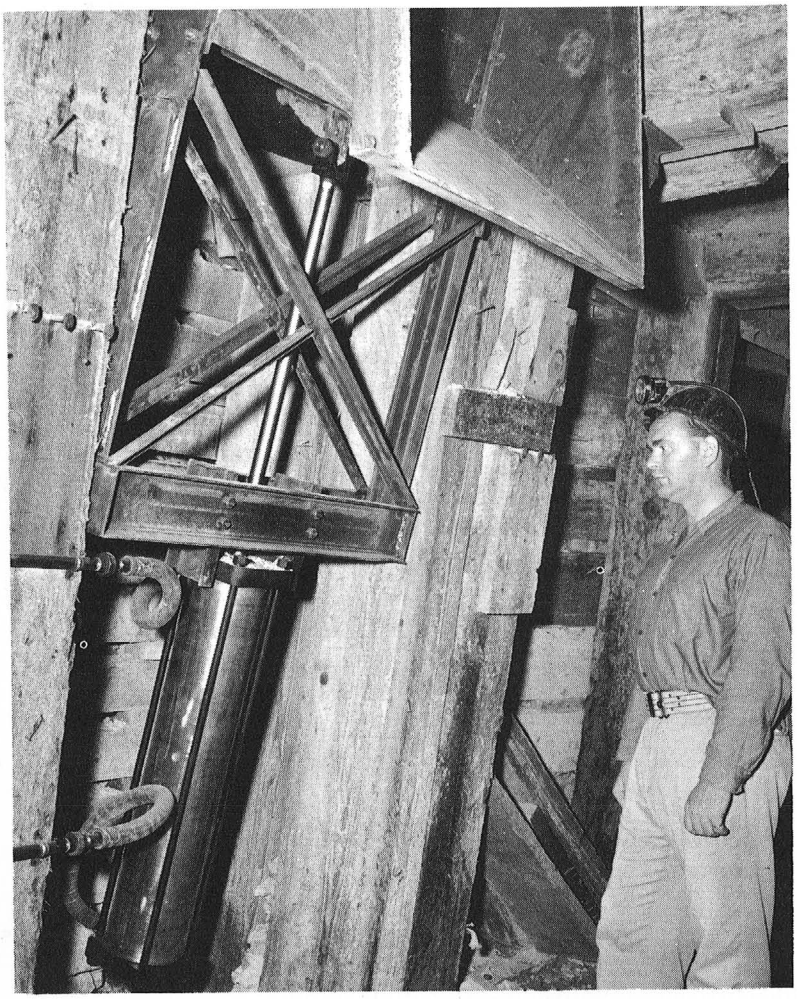


FIGURE 23. - Air Cylinder and Guides for Operating Chute Gates.

After each pillar blast, the broken material was drawn to give room for the next pillar blast and to determine by visual inspection that no part of the blasted pillar was left standing. If any of the pillar remained standing, it was removed before the succeeding pillar was blasted.

Twelve two-man crews, eight men per shift, undercut 30,000 square feet per month.

Slusher-Gravity Blocks

A slusher-gravity system was worked out at San Manuel for use on the first lift where the column of ore is low. Slushers were used in panels 3 and 4, and others are planned. This method requires considerably less development than the all-gravity system, but extraction costs are higher. Figure 7 is a schematic illustration of drifts and raises used in the all-gravity system. Figure 19 shows the differences in transfer raises of the two methods.

Blocks worked with slushers have two haulage drifts; the all-gravity system has three. Less than one-half the transfer-raise footage is required for the slusher-gravity system. Draw raises to the undercut level are spaced 17.5 feet apart, and grizzlies are constructed of 90-pound rail with 13-1/4-inch openings. A single transfer raise, with 4- by 6-foot inside measurements, serves one-half of the draw raises on a slusher-grizzly line. The undercutting procedure is identical to that discussed under the all-gravity blocks.

Double-drum, 2,600-pound slusher hoists powered by 30-hp. electric motors operating on 440-volt, 60-cycle alternating current are used. The drums are 14 inches in diameter. The pull and tail cables are respectively 130 feet of

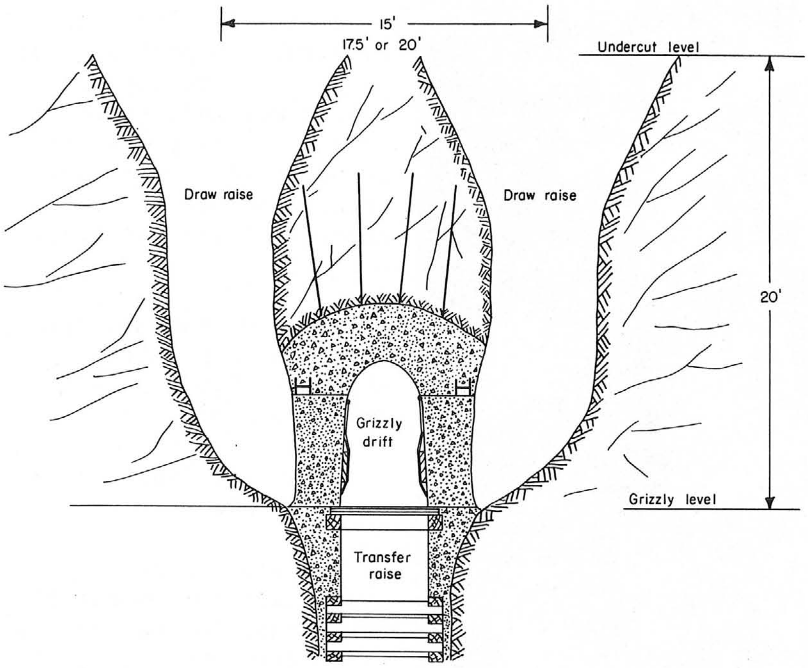


FIGURE 24. - Concrete-Supported Grizzly Drift and Draw-Raise Entrances.

5/8-inch and 250 feet of 1/2-inch wire ropes. The 42-inch scraper is a 1/4 box-type with a capacity of 13 cubic feet (fig. 30).

Ground Control

Heavy ground has been a major problem at San Manuel. Several types of timber sets were tried, and rigid and yieldable circular steel sets were tested. All materials tested were unsatisfactory because of high repair costs. Concrete support tests were started in August 1956. Results were satisfactory, and the decision was made to use concrete support in all future grizzly and panel drifts.

The following information is abstracted from previous papers.[11] [12]

Although the initial cost of concreting is higher than when other materials are used, appreciable savings result because costly drift maintenance is minimized. The following additional advantages are apparent:

1. A safer and cleaner working place is available for the chute tapper, thus increasing the tons per man-shift.
2. The fire hazard within a block is virtually eliminated.
3. Less skilled workmen are required for initial development, and fewer skilled timbermen are needed for repair work.
4. More efficient ventilation is possible since drifts are smooth lined. Most concreted grizzly drifts remain open after a block is completely drawn and can be used selectively for air courses.

The first concrete placing was done with two stationary pneumatic placers located on the 1415 grizzly level. A 1/2-cubic-yard mixer was used for surface

[11] Ward, M. H., Underground Concreting at San Manuel Mine, San Manuel, Arizona: Proc. Arizona Section Meeting, AIME, Tucson, Ariz., December 1958, 11 pp.
[12] Pillar, C. L., The Placement and Use of Concrete Underground at the San Manuel Mine, San Manuel, Ariz.: Proc. Am. Min. Cong., Denver, Colo., September 1959, 15 pp.

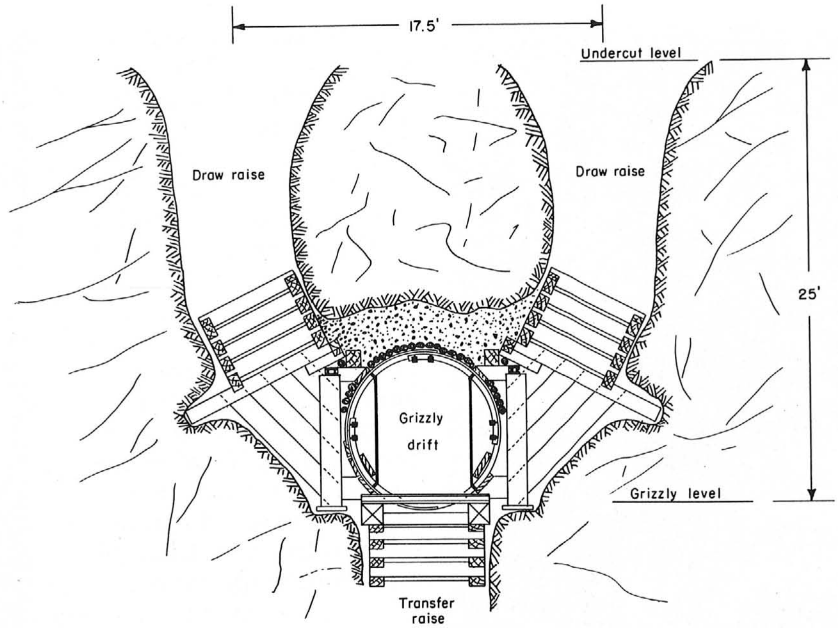


FIGURE 25. - Steel-Supported Grizzly-Drift and Timber-Supported Draw-Raise Entrances.

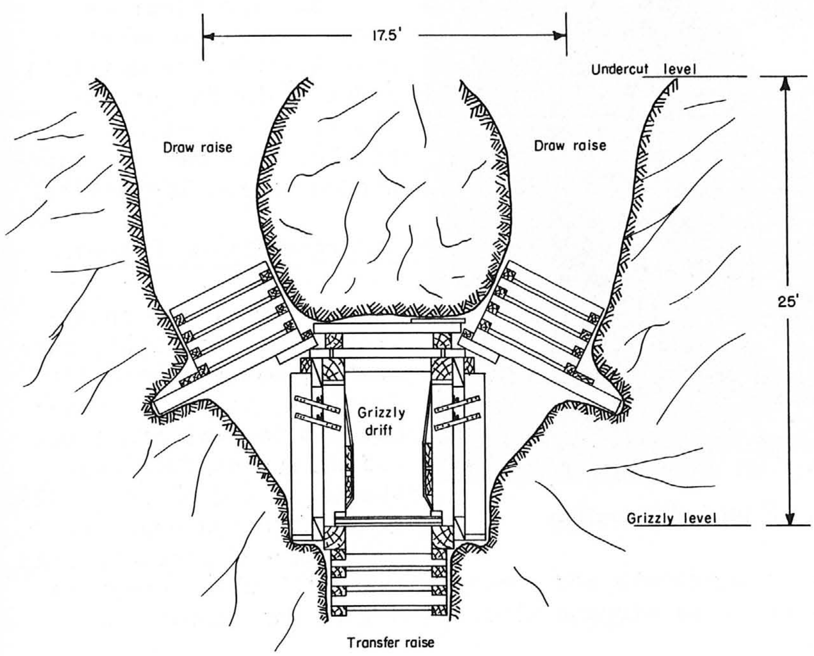


FIGURE 26. - Timber-Supported Grizzly-Drift and Draw-Raise Entrances.

mixing. Sacked cement was hand-opened and dumped into the mixer hopper. Concrete was poured down a vertical exploration churn-drill hole lined with 8-inch casing into a receiving hopper, from which it was chuted one of the two placers. Permanent, 6-inch, Victaulic pipelines were taken off the main feeder line into the panel drifts. Concrete was blown for as much as 2,000 feet.

In August 1957 the stationary placers were replaced by portable placers using the Flocrete process (fig. 31).

Several advantages of portable placers are evident:

1. Less manpower is required. With portable placers, 5.9 cubic yards per man-shift is poured, compared with 2.7 cubic yards per man-shift poured with conventional pneumatic placing.

2. A better concrete is obtained. Disseminated air causes the concrete to flow rather than blast from the discharge pipe. This prevents segregation and honeycombing.

3. The number of plugs in the discharge line is decreased. The maximum pipe length is reduced from 2,000 to 400 feet. If a plug occurs, it is easier to find and break loose. With stationary placers, concrete

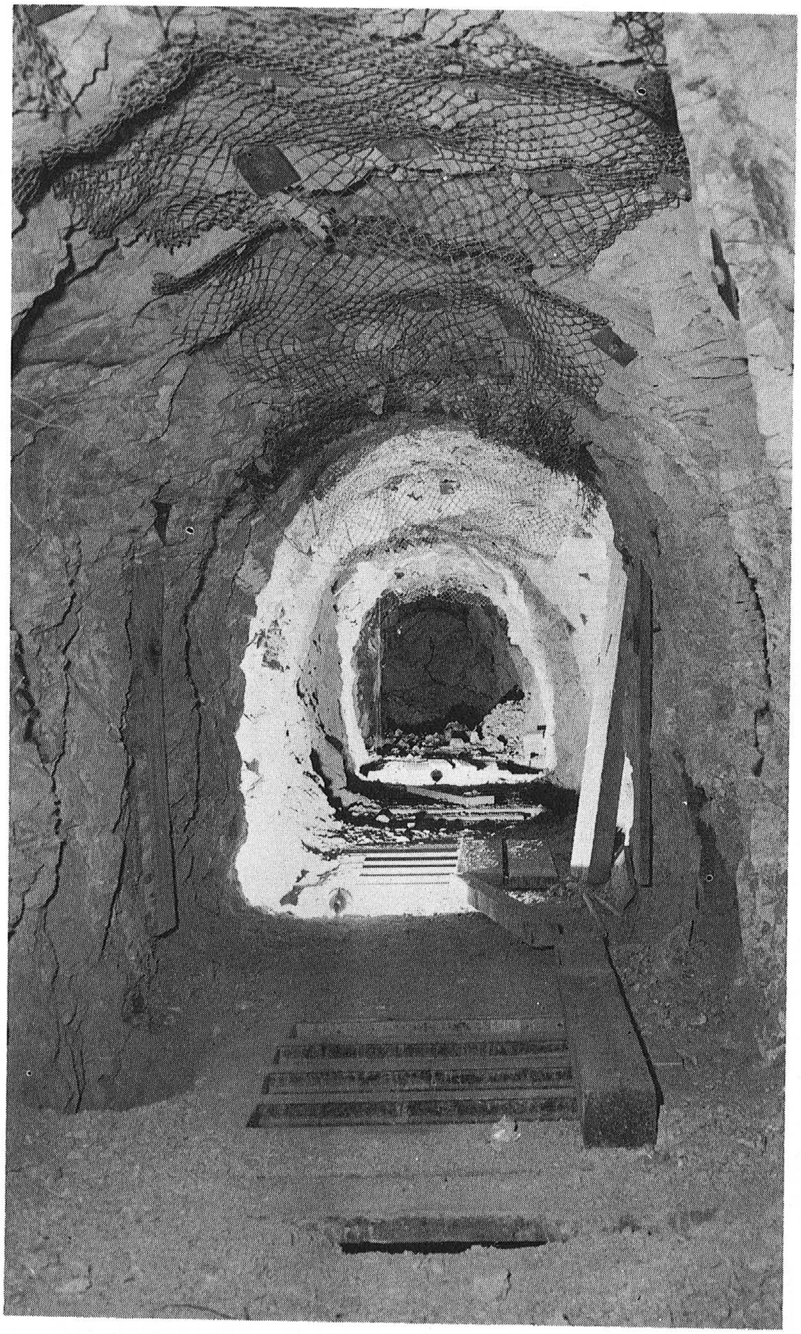


FIGURE 27. - Grizzly Drift Prior to Concreting.

has filled and set up in as much as 1,000 feet of pipe before the plug could be found and removed.

4. Shorter discharge lines are safer. The shorter length has less volume of air, so the resultant of the force of moving concrete is reduced at bends. Most discharge pipe is in grizzly drifts and panels away from mainlines. In case of a break, there are fewer men in the vicinity of the discharge lines.

5. Portable placers are more versatile. Small pours of foundations, floors, stations, and so forth that would not merit stringing pipe from a stationary placer can be poured efficiently.

6. The Flocrete process and equipment have been instrumental in reducing by 25 percent the overall cost of placing concrete in underground extraction drifts.

Concrete Mixing Methods

After it was determined that monolithic concreting was economically feasible, a 300-ton dry-batch plant was constructed on the surface near shafts 3A and 3B (fig. 32). It is situated near the railroad and a county road for convenience of delivery of aggregate and cement. The batch plant consists of elevators to lift aggregates into storage bins and a silo for storage of bulk cement.

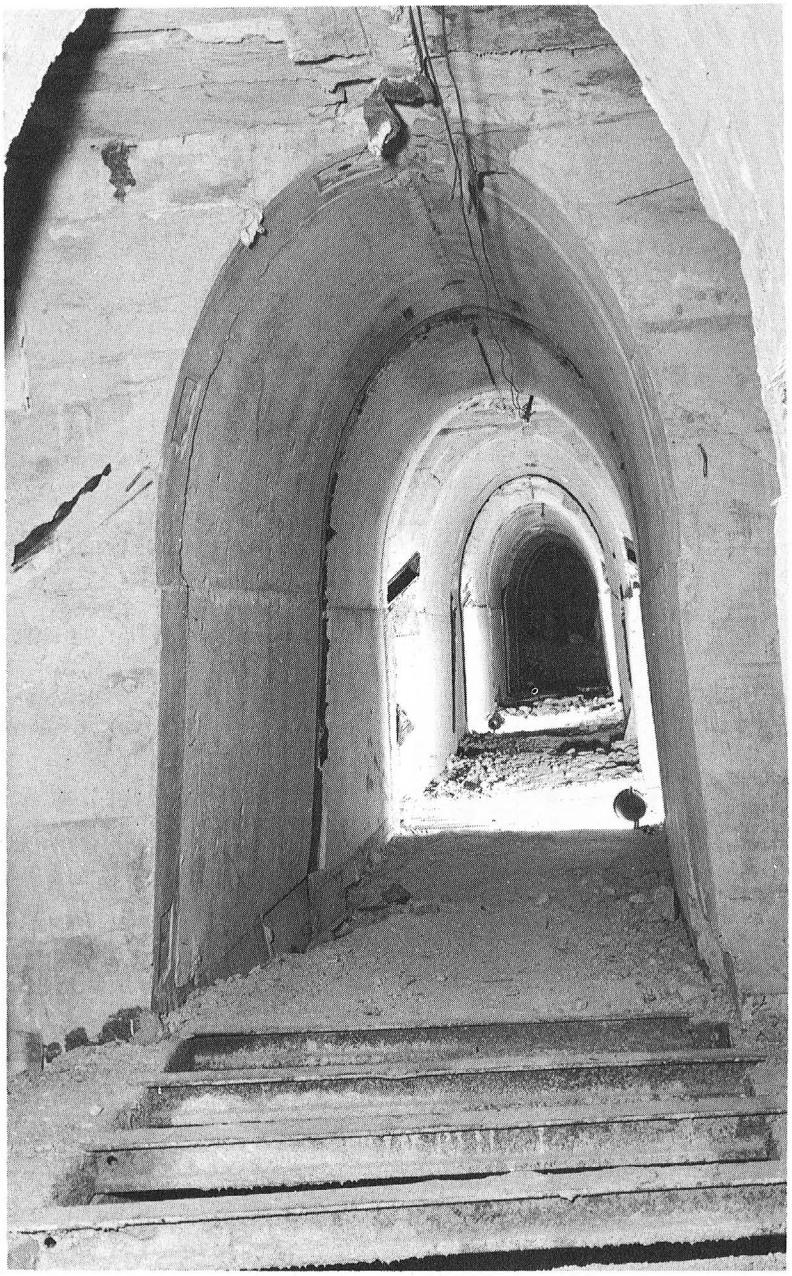


FIGURE 28. - Grizzly Drift After Concreting.

Concrete for the first level (1415) is delivered dry batched to a concrete mixer by three dump trucks, each with four 1-1/4-cubic-yard compartments (fig. 33). The mixer is a 1-1/4-cubic-yard double-drum paver mounted on tracks. At a signal from the underground remix station, concrete is mixed and poured into a hopper mounted over a vertical churn-drill hole with 8-inch casing. The hopper is covered with a 3/4-inch reinforcing rod grid of 3-inch squares, which screens out large rock that may cause plugging. Concrete for the second level is delivered dry batched to the No. 1 shaft, where a track-mounted 1-1/4-cubic-yard mixer located at the shaft collar mixes and pours wet concrete down either of two 8-inch standard pipelines mounted in the shaft's service compartment.

Concrete Mix

A mix that yields 1-1/4 cubic yards of concrete consists of 705 pounds of type 11 portland cement; 1,300 pounds of clean washed sand having 5 percent minus 100-mesh; 2,200 pounds of string cubical minus 1-inch screened coarse aggregate sized within limits of ASTM specifications; 48 gallons of clean, pure water; 14 ounces of air-entraining agent for added workability; and 21 ounces of a retarder and strengthener. The mix yields concrete that permits fast and efficient placing and has an average 28-day strength of 3,500 p.s.i. This mix gives a slump of from 5 to 6 inches at the mixer and 3 to 4 inches in the forms. It is necessary to cool

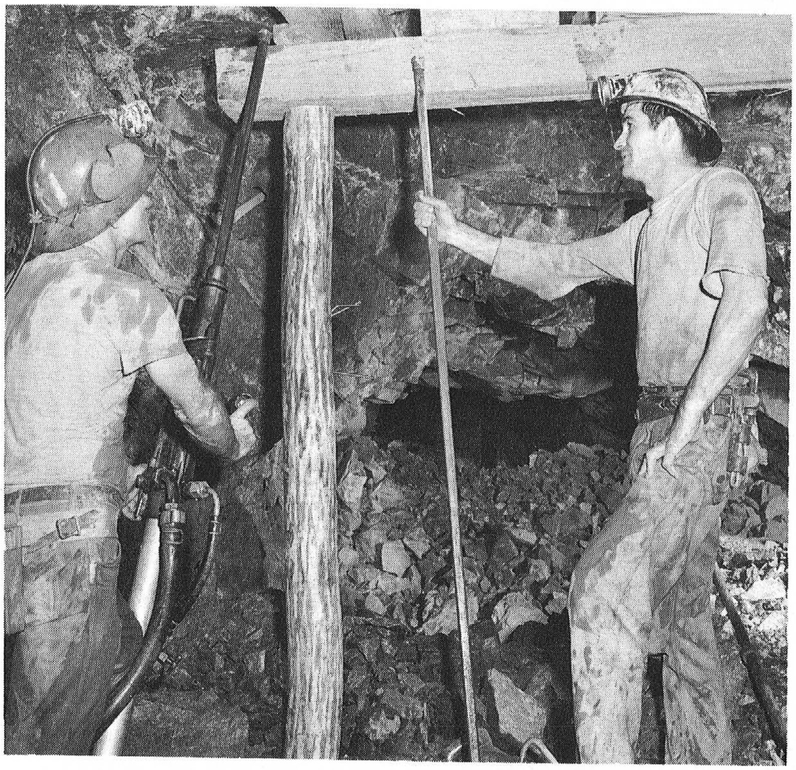


FIGURE 29. - View in Undercut Drift.

the mixing water during the hot, dry summer. The portable, electric-powered, water-chilling plants are capable of producing 20 g.p.m. of water at 40° F. The chilled water reduces the temperature of the aggregate so that concrete temperatures at the mixer are kept at approximately 70° F. The concrete loses considerable water during its journey between the surface mixer and the underground forms (fig. 34). A water-to-cement ratio of 0.58 to 1 is maintained; this keeps the concrete workable but does not allow the accumulation of free water in the forms. The set of the concrete is retarded by the use of additives so that concrete can stay a maximum of 3 hours in the portable placers and be discharged satisfactorily.

Underground Remixing Station

At the bottom of the churn-drill hole most of the wear and shock caused by the falling concrete is taken by a 16-inch-diameter pipe header (fig. 35), which receives and chutes the concrete into a remixer. A replaceable, belt-on bottom for the header has been developed.

Between the surface and the 1415 level the 8-inch casing has several slight bends where above-average abrasion occurs and an occasional hole is worn through the casing. To change the position of wear points, the casing is rotated 90° after each 1,000 cubic yards of concrete is poured. At the same time, an additional 30 feet is welded to the casing at the surface, the casing is lowered, and a corresponding amount is cut off at the remixer station. A method was developed to detect a hole in the casing before escaping concrete can accumulate around the outside of the casing and cement it in place. The 8-inch casing is suspended inside a 12-inch drill casing in which 10 g.p.m. of water is circulated from the surface into a funnel at the remixer station. If a hole is worn in the casing, milky water and particles of rock and sand are observed immediately in the funnel as the concrete descends.

From the pipe leader the concrete flows into a 6-cubic-yard remixer, where it is agitated until loaded into placers (figs. 36 and 37). The remixer is a 5-1/2-foot-diameter tank, 8 feet long with a side section cut out. A spiral agitator built around a 4-inch tubular shaft extends the length of the tank. Each end of the shaft is supported on water-lubricated bearings to prevent cement freezing. The agitator is driven by a chain-connected 10-hp. air motor.

Communications

A separate telephone circuit and light-bell system are used for communicating between remixer and surface mixer. A standardized system of bells is posted for ordering and indicating receipt of concrete. This prevents the pouring of several batches on top of each other should the casing become plugged between surface and remixer station.

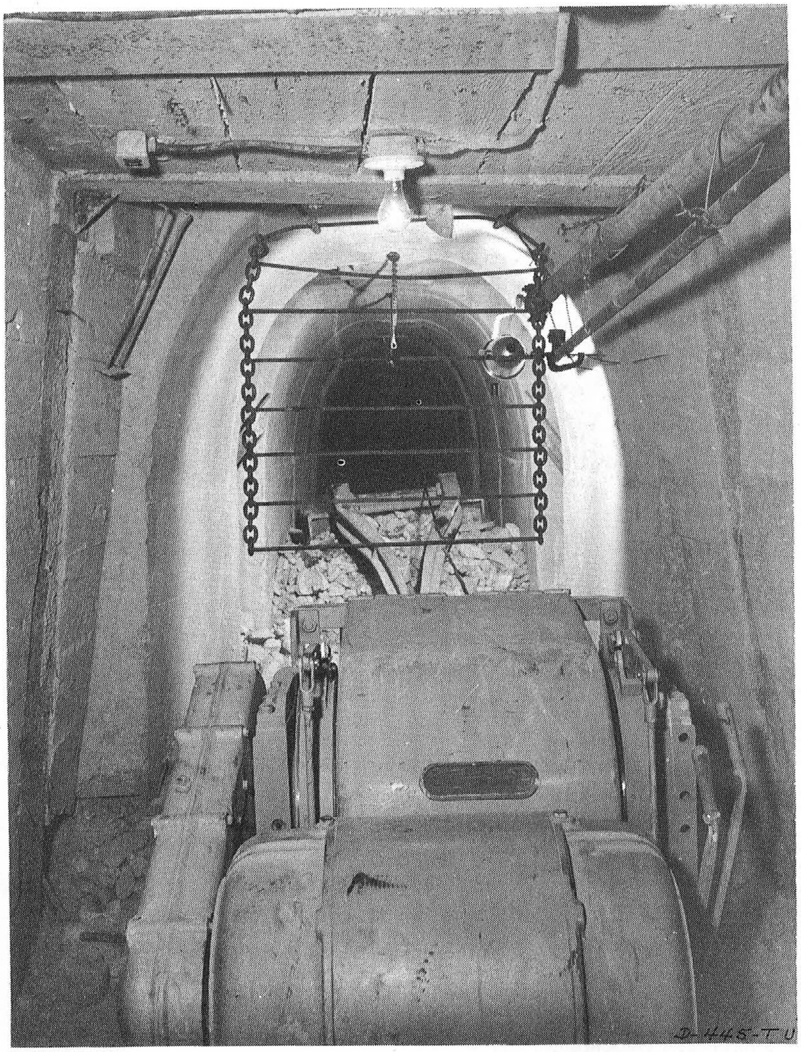


FIGURE 30. - Double-Drum Hoist and Scraper in Concrete-Lined Slusher Grizzly Drift.

Concrete Forms

Wherever possible, drifts that are to be supported with concrete are driven without timber support, using 6-foot rock bolts with wire mesh for pre-concrete ground support (fig. 27). Light timber sets are used where the ground will not stand with rock bolting. The sets have sufficient span for placing of forms between them. The light timber sets are left in place, and concrete is poured around them. When excessive pressures develop before concreting, it is necessary to replace the light timber set with a steel set of 6-inch, wide-flange section. This is done only where absolutely necessary, as structural steel imbedded in the concrete has proven detrimental to concrete support of heavy, moving ground and also where secondary blasting is required. It is the practice at San Manuel to keep as much steel out of the concrete as possible.

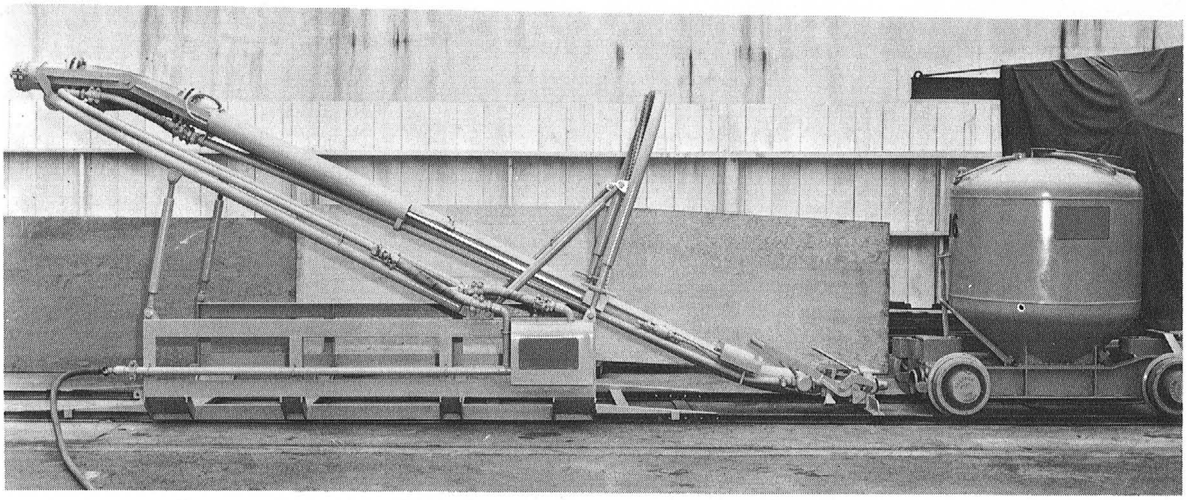


FIGURE 31. - Portable Concrete Placer.

FIGURE 32. - Batching Plant.

Steel imbedded in the concrete is confined to wear beams at the brow of draw raises, eyes, hooks, and brackets for control-board cables, safety chains, pipelines, and slusher sheaves. No reinforcing steel is used in concrete in the mining areas.

In block grizzly and slusher drifts that contain no track and are at right angles to tracked panel drifts, forming is accomplished by using portable, light, strong plywood panels. These forms are made in the mine carpenter shop from 5/16-inch plywood with 2- by 6-inch wooden ribs. The standard panel is 5 feet 10 inches long, which is an even interval for 17-foot 6-inch drawpoint

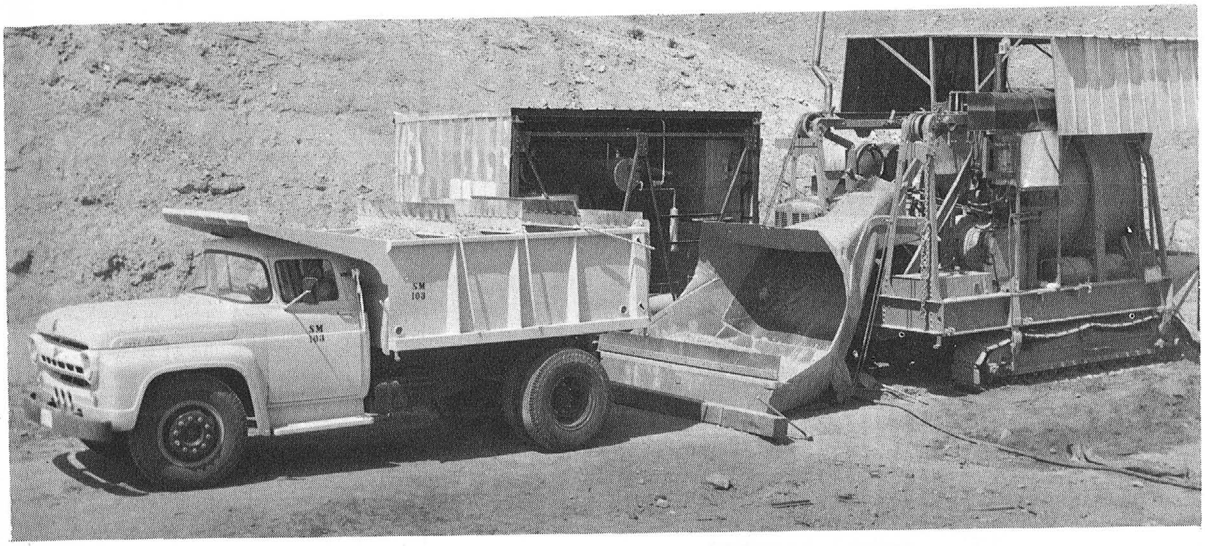


FIGURE 33. - Concrete Mixer at Churn-Drill Hole.

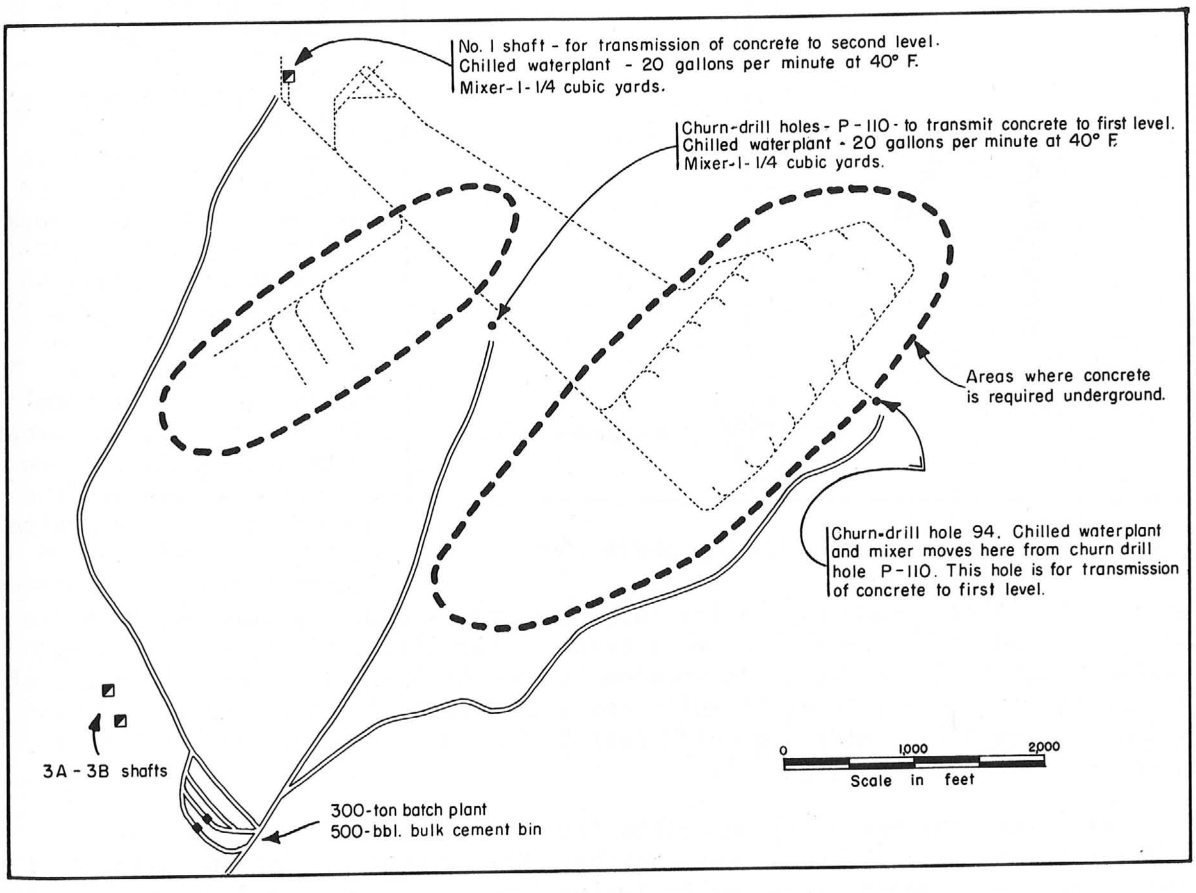


FIGURE 34. - Location of Concrete Batch Plant, Mixing Stations, and Areas Where Concrete is Used Underground.

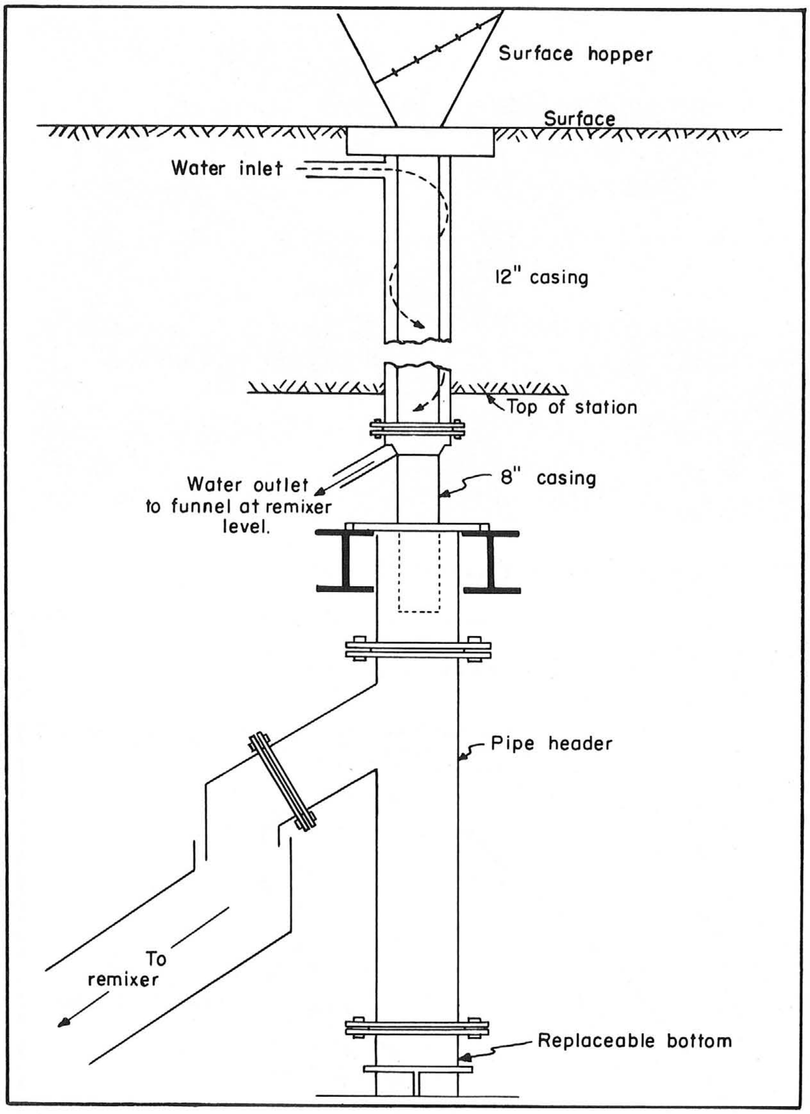


FIGURE 35. - Sketch of Casings and Header.

spacing. Grizzly sections are composed of two straight side panels and two arched back panels bolted together with 5/8- by 4-inch machine bolts. A slusher drift sections uses an additional 12-inch-wide panel bolted between the two arched back panels along the top center of the drift. Properly placed holes in the form panels allow quick placing of eyes, hooks, and brackets that are to be imbedded in concrete. In grizzly drifts, forms are set on leveling sills laid on the raw drift floor.

The drift floor between the side of the forms and the rock walls is dug to solid rock and cleaned of all loose muck. In slusher drifts, a 12-inch concrete floor with imbedded wear rails is poured between the concrete drift walls. A narrow slit in the form allows 1-inch rough lumber to be sectionally placed between the form and the ground to make draw-raise box-outs. These 1-inch lumber box-outs are braced with 3- by 6-inch studding. After forms are set, lined, and leveled, they are stull-blocked to the ground and well braced internally with rough 3- by 6-inch timber. Drifts are driven, and forming is done to allow a minimum thickness of 18 inches of concrete in drift walls and arched roof. Finished grizzly drifts (fig. 28) are 4 feet wide and 6-1/2 feet high, and slusher drifts are 5 feet wide, 6-1/2 feet high (fig. 30).

At first, the panel-fringe drifts (fig. 38) were formed in a manner similar to the grizzly and slusher drifts. Now, circular, jumbo-mounted steel forms are used. These forms are in 10-foot lengths. They are transported and expanded into position by a hydraulic-powered, track-mounted jumbo. In the stripping process, the jumbo contracts the forms, and they are then in a

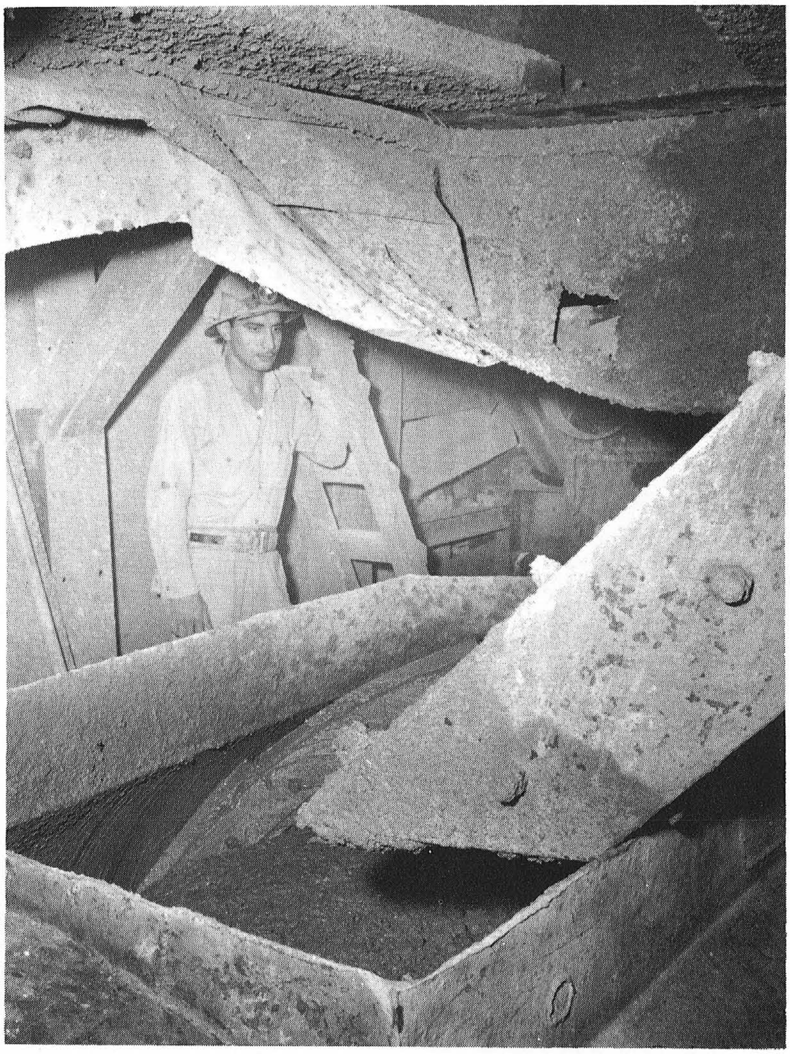


FIGURE 36. - Underground Remixing Station.

position to be transported through the existing forms to any desired location for placing. Light steel bubbles bolted to the outside of the forms are used to form grizzly and slusher drift entrances. Early performance with these forms represents a 50-percent reduction in labor costs over previous methods. Panel drifts a are driven and forms are placed to allow a finished circular section of 8 feet 6 inches, with a minimum thickness of 18 inches of concrete.

Concrete Placing

A train of portable Flocrete placers mounted on trucks is pushed from the remixer station to the forms. Each individual placer is connected o to an underpass discharge spout and emptied (fig. 31).

The placer is a 1-3/8-cubic-yard circular tank with a conical bottom. Compressed air at 90 p.s.i. is introduced through a turnip-shaped valve in the bottom of the placer to force out concrete through a 6-inch bottom-discharge pipe. Air is disseminated throughout the batch as it is gravity-fed to the bottom. A spring holds the turnip valve against its seat until discharge starts. A foot-operated air vibrator is mounted on the outside of the placer and is used to loosen stiff concrete.

The opening at the top of the placer is sealed airtight with a rubber-covered hatch, which is mounted on a swivel and swings aside for loading. If the discharge line plugs, the air seal is broken by opening a pressure-release valve at the top of the tank.

A quick-coupling attachment is mounted on the underpass and is connected to the placer. Air is supplied through a 2-inch pipe attached to a manifold

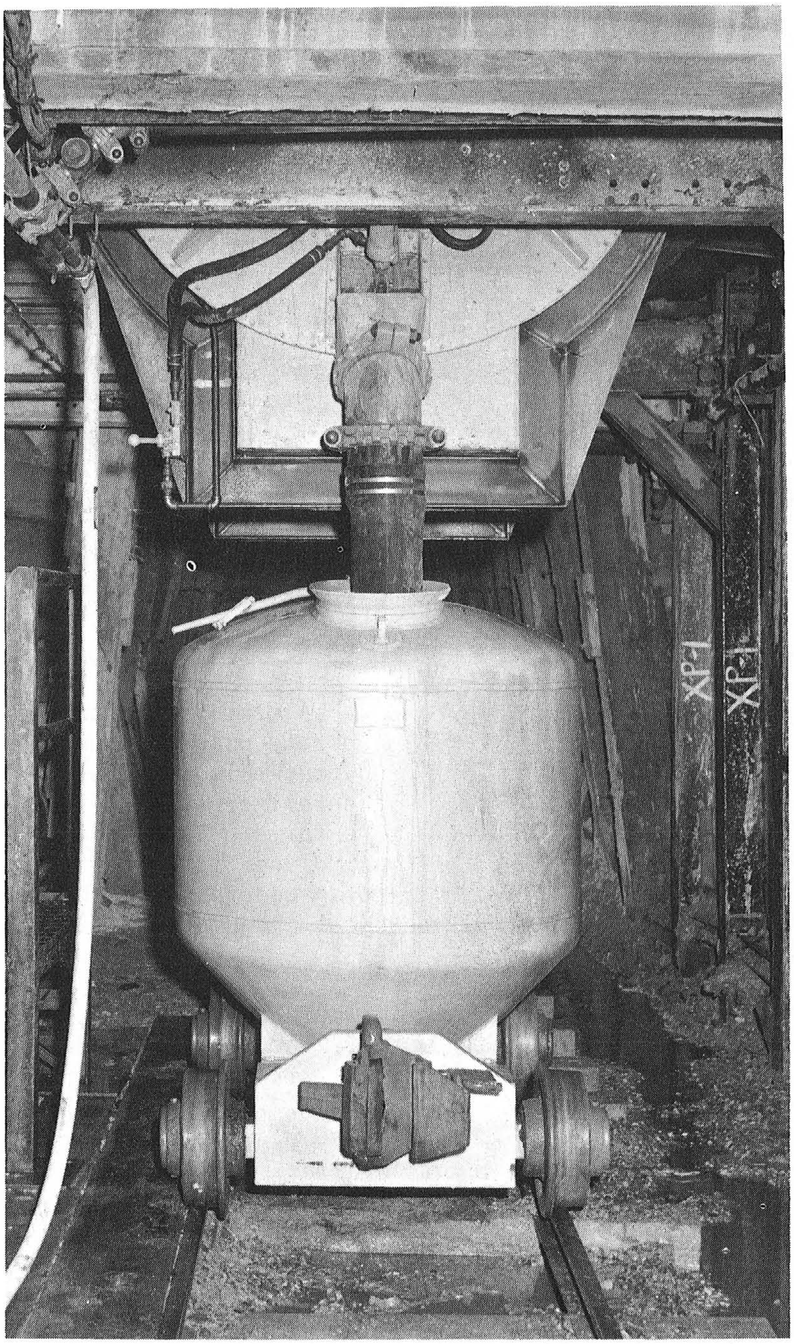


FIGURE 37. - Remixer Discharging Into Placer.

with two 1-1/2-inch protruding tapered fittings, which are sealed against tapered seats of a casting on the placer (fig. 39). Quick-coupling hooks, operated by an eccentric lever, draw the placer into position and lock it for an airtight seal between discharge pipe and air inlets. After each placer is emptied, it is uncoupled from the underpass and pulled back sufficiently to clear the underpass discharge header. The header and pipe are hoisted above placer heights by two double-acting air cylinders (fig. 40). The empty placer is pushed through the underpass and uncoupled from the train. The train is pulled back, the discharge header is lowered, and the next placer is brought into position and emptied (fig. 41).

An 8-ton locomotive pulls a train of four to six placer pots that contain 5-1/2 to 8 yards from the remixer station to the forms. Two trains are used during the placing operation, and two sections of forms are poured simultaneously so that both trains may be run to one form if a plug occurs. The tram distance of placers ranges from 1,000 to 4,000 feet. Telephones and audio phones are located throughout the mine and are used in dispatching concrete to the different drifts.

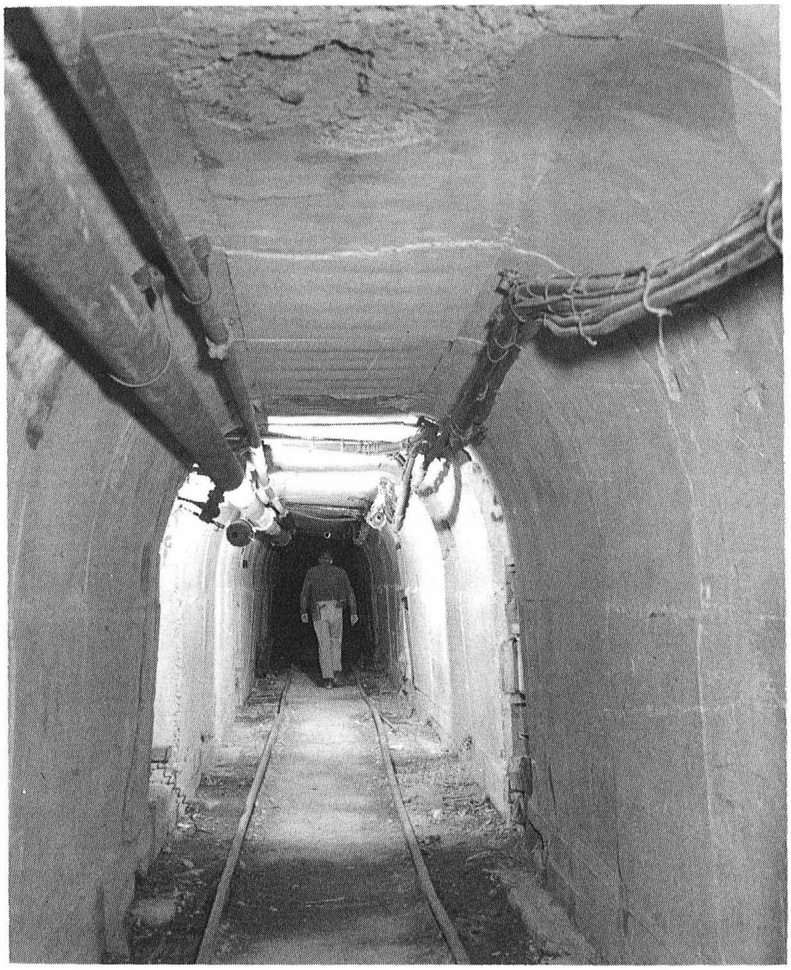


FIGURE 38. - Concrete-Lined Panel-Fringe Drift.

One man works on top of the forms to shift the discharge pipe for even filling. Internal air vibrators are used to consolidate the concrete. Sections of discharge pipe are removed as the form is filled and the pour retreats.

When the pour is completed, a bundle of rags or "rabbit" is forced into a cleanup header. The header is then connected to the underpass; air is introduced through the header, and the rabbit is forced through the discharge line to clean it. Usually two rabbits are run before the line is free of all concrete.

Discharge pipes are made from 6-inch lightweight spiral tubing in 5- and 10-foot lengths. Elbows of 45° and 90° bend and 2-foot radius are fabricated from standard pipe. This smooth bend reduces wear and plugging in the lines. All connections are made with 6-inch Victaulic couplings. The average length of the discharge line is 100 feet around one 90° bend, but pours through 400 feet of pipe and three 90° bends have been made without difficulty.

Concrete Crews

An average of 150 cubic yards of concrete per day is now being placed on the two mining levels. The pouring operation is accomplished on the "A" shift, and the forming operation on both "A" and "B" shifts. The crews for this work are shown in table 11.

This crew has placed a maximum of 230 cubic yards of concrete in one 8-hour shift. The underground placing and forming crew work under a contract bonus system whereby they are paid each week for the work they accomplish.

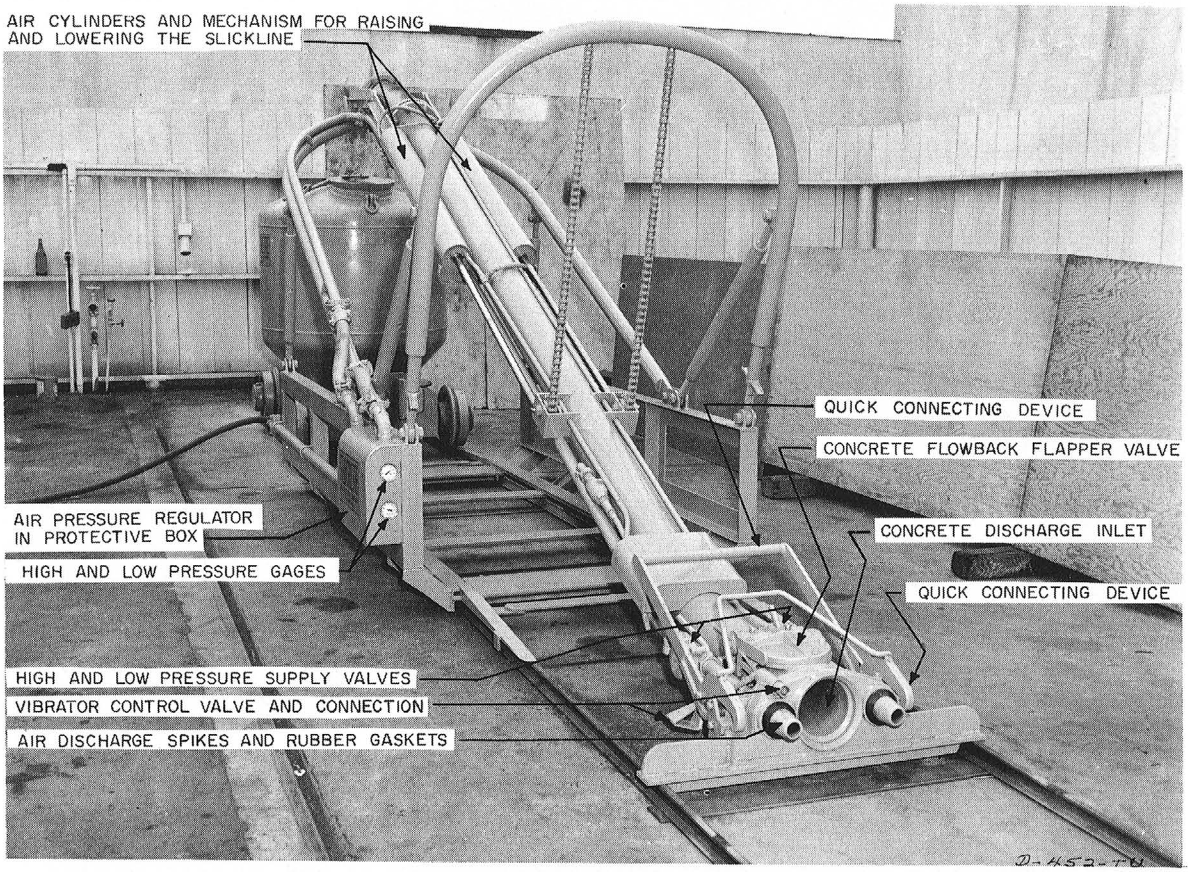


FIGURE 39. - Front View of the Underpass.

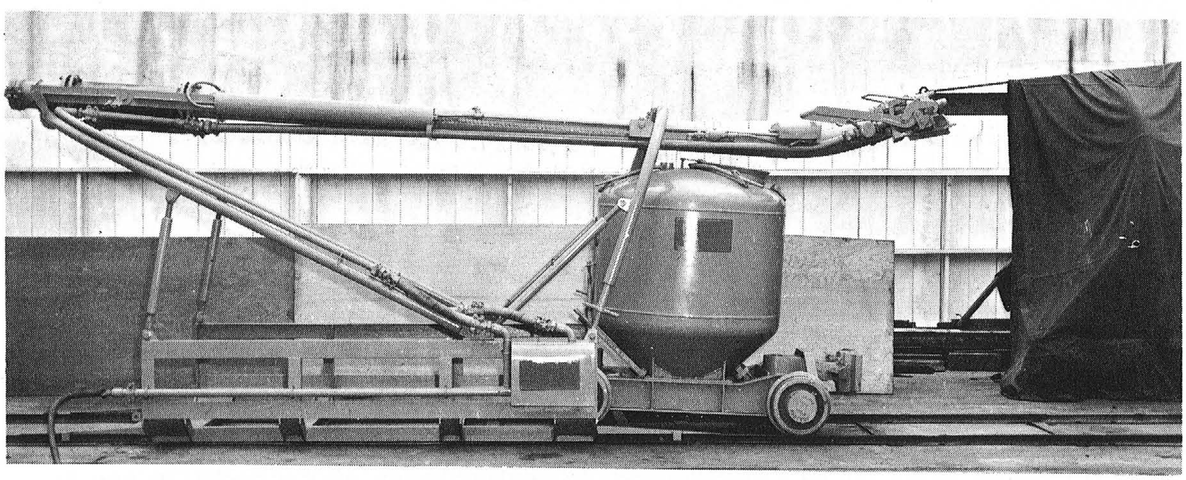


FIGURE 40. - Slickline in Raised Position; Placer Pot Passing Through Underpass.

FIGURE 41. - Coupling Placer Pots to Slickline.

Costs

Costs, as shown in figures 42 through 46, cover all direct supervision, labor, and material, and some equipment maintenance. For a fair and conservative evaluation, these costs were reduced to a 1956 base. This was done by reducing the costs for work accomplished after July 1956 by the same percentage that mine labor has been increased, roughly 3.5 percent a year. It will be difficult to compare these costs with costs from other operations, owing to the different manner in which each individual company keeps its own cost accounts.

TABLE 11. - Crew required for concreting operations

	"A" Shift	"B" Shift
Surface crew:		
Batch-plant operator..........	1	0
Truckdrivers.................	3	0
Mixer operators..............	2	0
Total........................	6	0
Underground placing crew:		
Leadman......................	2	0
Remixer operators............	2	0
Motormen.....................	3	0
Placer operators.............	3	0
Puddlers in form.............	3	0
Pipe riggers.................	0	2
Total........................	12	2
Underground form crew:		
Leadmen......................	2	2
Form men.....................	5	5
Helpers......................	5	5
Total........................	12	12
Grand total..................	31	14

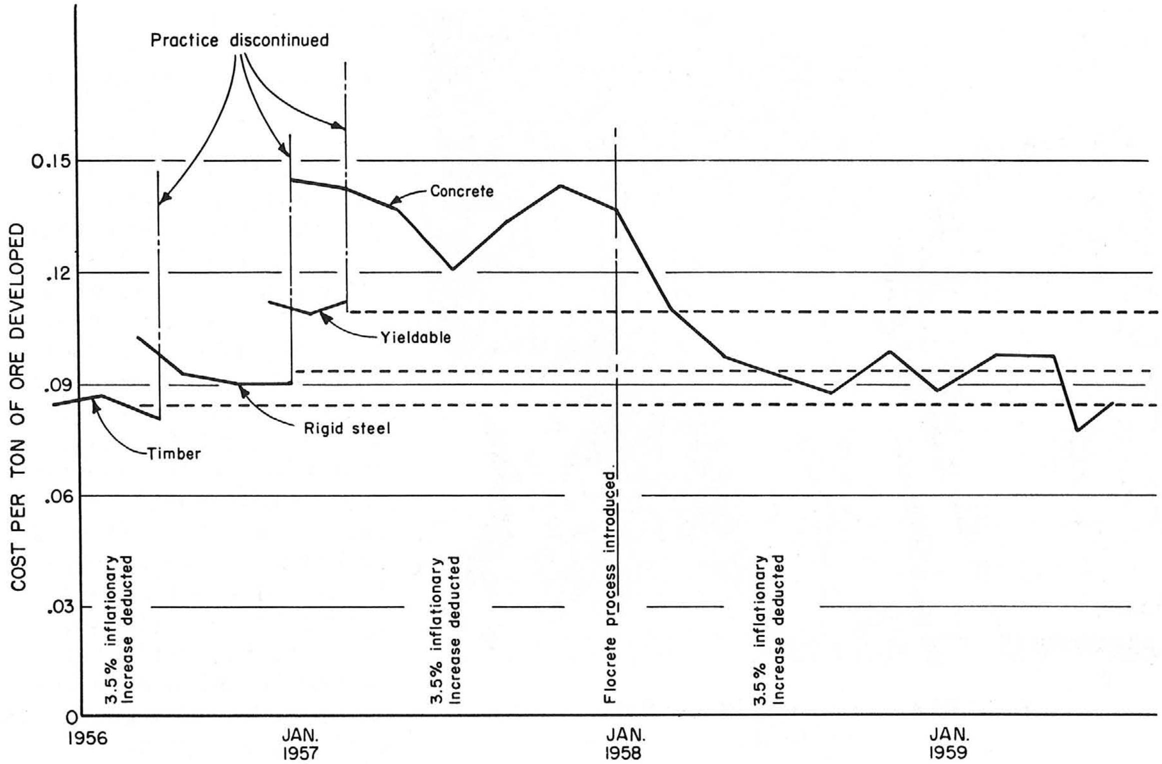


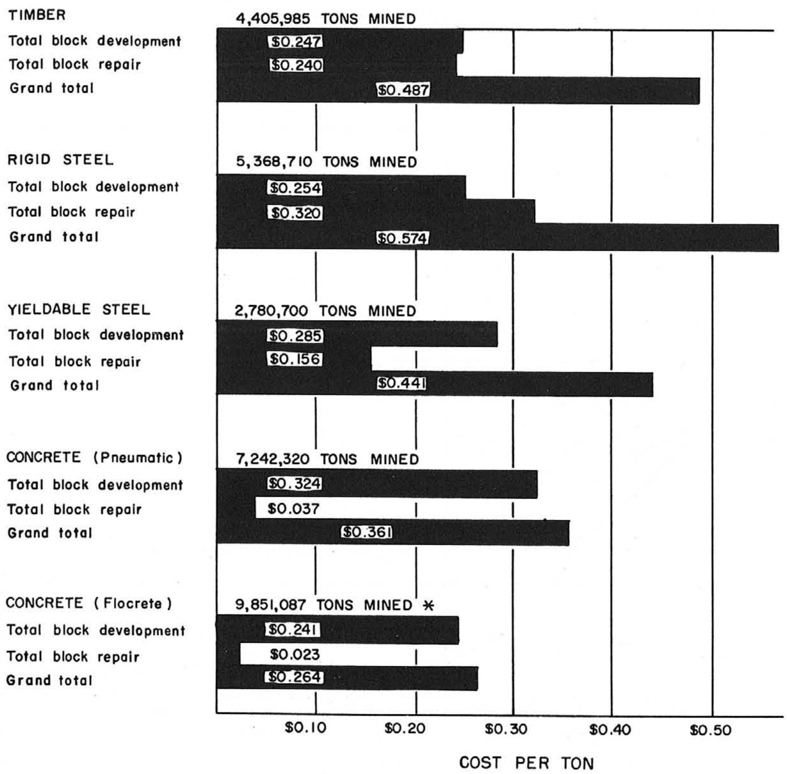
FIGURE 42. - Block Grizzly-Drift and Draw-Raise Costs Per Ton of Ore Developed.

Grouting

Grouting at San Manuel has been very effective on the haulage level in retarding rock movement, preventing timber breakage, eliminating runs of loose material during timber repair, and reducing overbreak during the excavation of raise stations. Grouting was tried first on turnouts where rock was very loose. Good results were obtained, and the practice was extended to timber repair and raise-station excavations.

Grouting is placed where the rock becomes loose or where heavy pressures develop above the caps of timber sets in haulage drifts. It is accomplished by drilling 1-1/2-inch holes in rings on 5-foot centers through timber to depths of 10 and 20 feet in the rock. A 1/2-inch-diameter pipe is inserted in a hole, and a grout pump is attached.

Grout consists of water, cement, and a little bentonite. Bentonite promotes flowability and plasticity while the grout is being placed. Usually grout will plug the cracks in the rock, but occasionally cracks and holes must be plugged with wood or paper cement sacks. The grout is pumped at 300 p.s.i. At first, four to five holes were drilled in a ring, but experiments proved that two holes to a ring were sufficient. Sometimes the 10-foot holes are grouted first at pressures considerably less than 300 p.s.i. The 20-foot holes are grouted last, and always to a pressure of 300 p.s.i.



* Does not include blocks that have not been under full draw for a minimum of 60 days.

FIGURE 43. - Block Development and Repair Costs.

Loose, unconsolidated fault zones in haulage drifts have been grouted before timber repair. The present practice is to pump the grout into the drift backs in such areas and allow it to set. The broken timber is then removed, and the back is excavated for clearance. Thus, no spiling is required. With this repair practice, main haulage drifts are closed for very short periods.

Raises located in loose, unconsolidated material are driven by first grouting the area. After the grout has set, the raise is drilled and blasted to the desired size. Timbering and overbreak are reduced.

Mining

Caving Characteristics

The ore at San Manuel has good caving qualities and generally flows readily from draw raises. After final blasting of undercut pillars, a block is drawn rapidly (as much as 3 feet a day). The general procedure is to draw uniformly from all fingers. After initial caving, the goal is an average daily draw of 18 inches, with an equal draw from all points; however, a draw of 2 feet a day is not considered excessive.

In most cases caving progresses rapidly until it reaches the overlying Gila conglomerate. Visual observations on the 1285 level disclosed that no void existed in the porphyry between virgin ground and the caved material.

Ore fragmentation varies with the hardness of the rock. Ore from the eastern part of the mine is harder and coarser than that from the western part. This difference is apparent in the tonnage broken on the grizzlies. On the east end 7.5 percent was broken, compared to 1.8 percent on the west end. For a short time after undercutting, the fragmentation is coarser than it is after the draw is well established.

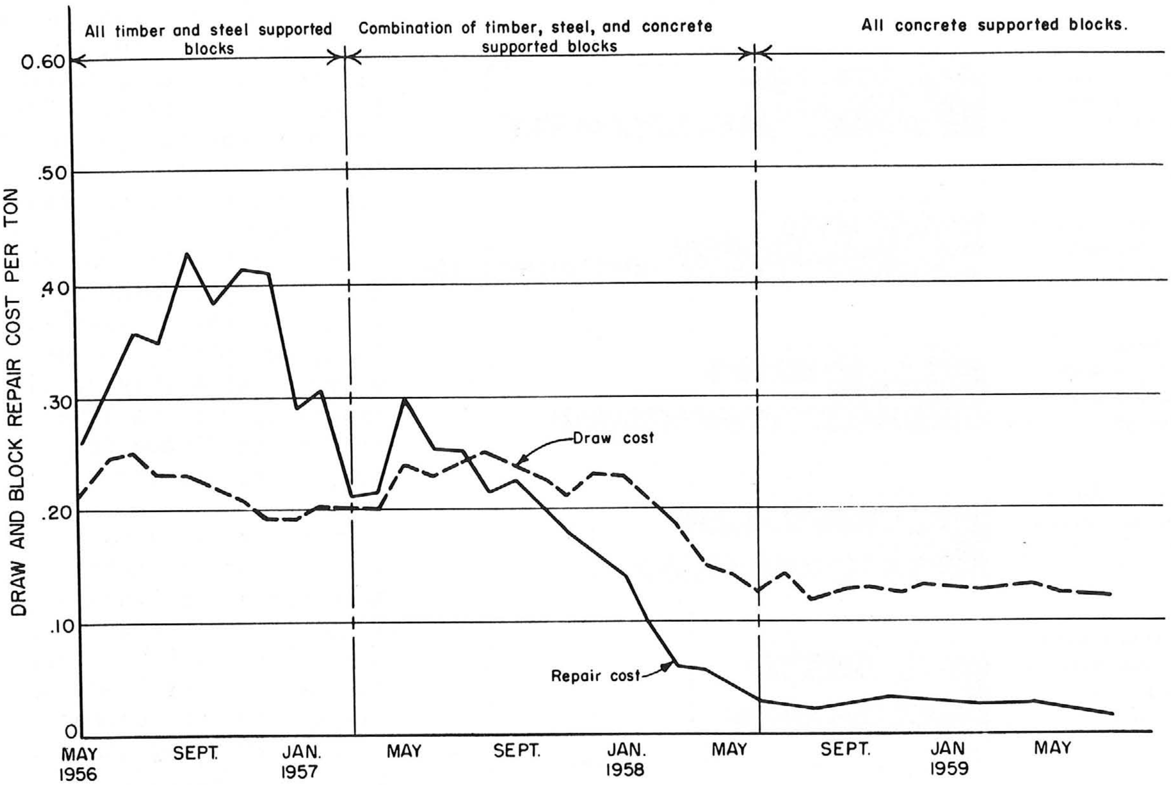


FIGURE 44. - Draw and Block Repair Costs.

Production Planning

Detailed planning is conducted 1-1/2 years in advance by the engineering department. These plans give monthly scheduling for shaft sinking, level development, block development, block sequence, and production.

Ore reserve estimates and block expectancies are compiled primarily from drill logs. Development headings on the grizzly level are sampled every 10 feet; haulage workings are sampled every 20 feet. The reserve estimates are adjusted periodically to the development sample results. The present ore cut-off is 0.55 percent recovered copper, but this figure varies with production conditions and requirements. Tables are used to convert total copper to recovered copper, depending on the oxide content.

The progress of development headings is controlled by a weekly development efficiency report which compares actual progress with quotas based on the development schedule. Block production quotas are set by the mine superintendent, and these are compared with actual production on the daily mine production report.

Both development and production progress are reviewed monthly.

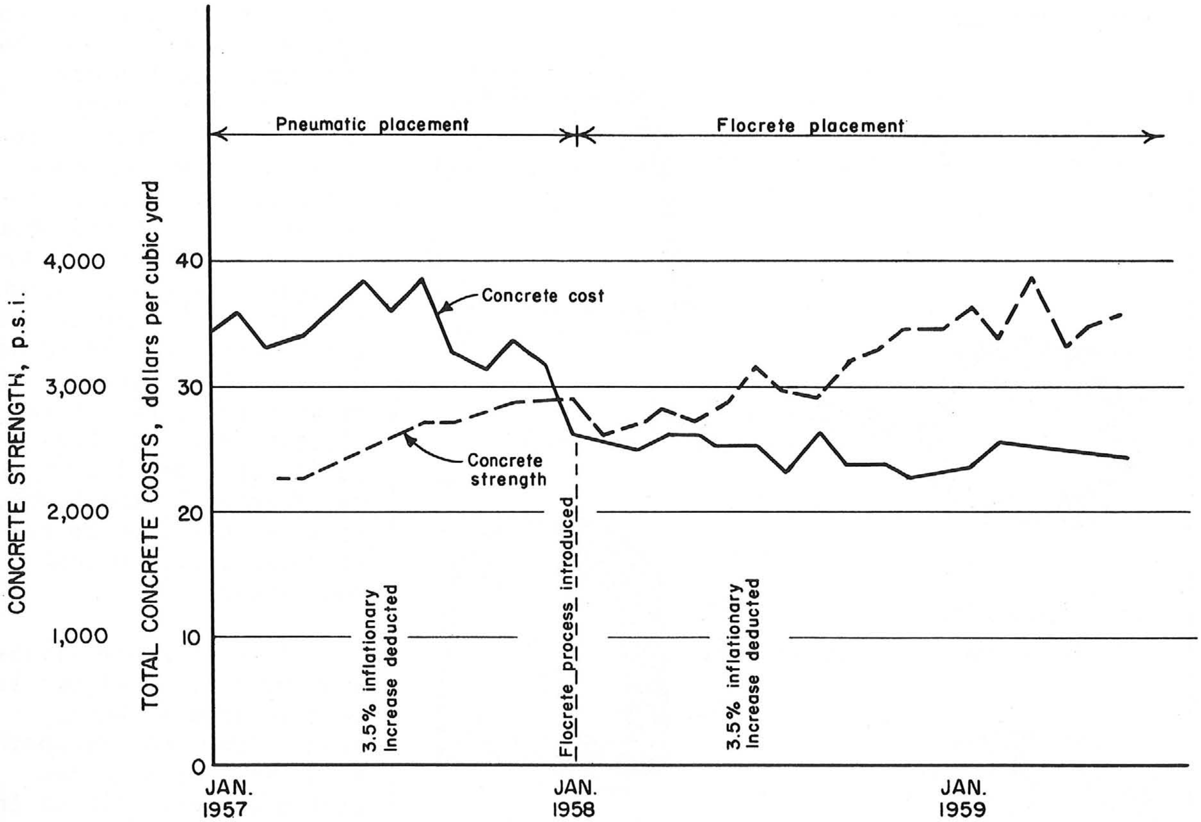


FIGURE 45. - Total Cost of Underground Concrete Forming and Placing, and Compressive Strength of Concrete.

Stope Records

Diamond-drill and churn-drill sample information is plotted on cross-section sheets. From these sheets, longitudinal sections parallel to the long axis of the ore body are prepared on 30-foot spacing.

Stope books are prepared that contain the tonnage and grade for each block and a sheet for each draw-raise line within the block. These sheets show three basic types of information: (1) The profile of the upper limit of ore against which the actual draw may be plotted; (2) the geology of each line, including details of the ore zone and the overlying capping; (3) and a graph where the grade of ore drawn may be plotted against expectancy. Each stope book contains a composite plan showing block layout and its relationship with the haulage level, and a summary sheet of block statistics.

Draw Practice

Draw from a block is controlled as soon as the first undercut pillar is blasted. As pillars above each drawpoint are blasted, ore is drawn rapidly until about 500 tons has been drawn. This permits inspection to ascertain that

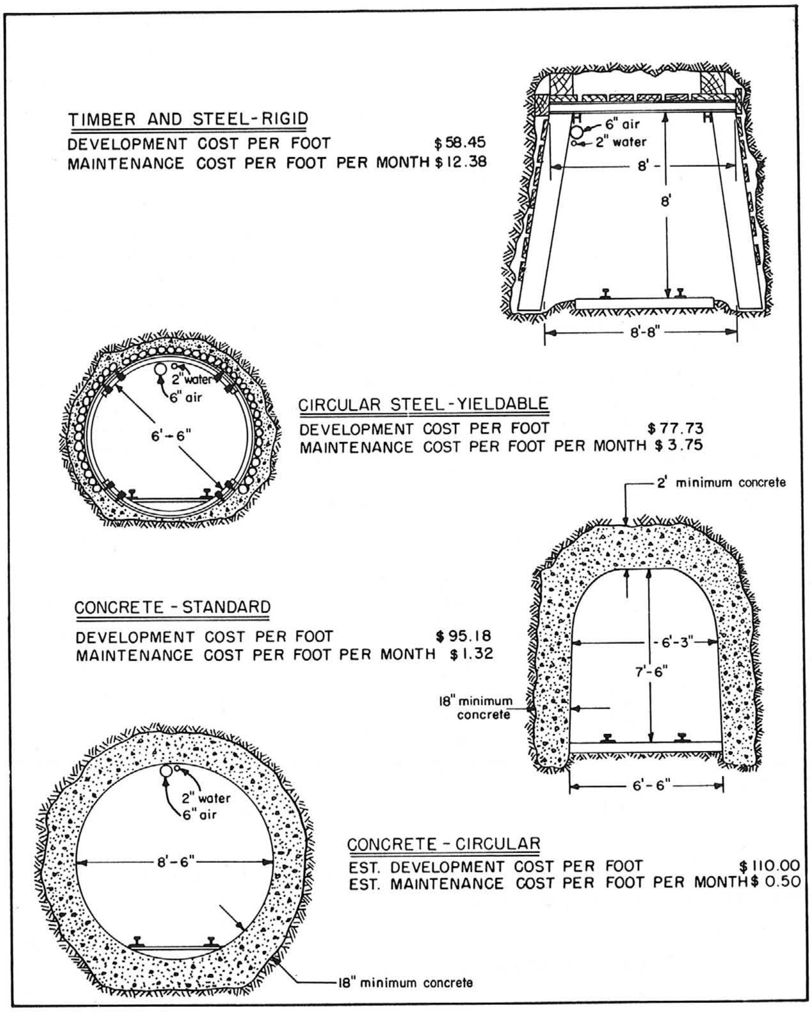


FIGURE 46. - Types and Costs of Stope Panel-Drift Support in Heavy Ground Areas.

all of the pillar has been removed and that space has been made for the ore to cave. When the undercut is completed, orders are given to "even the back" and start all drawpoints running. In a block where the top of the ore column slopes, more ore is drawn from the high side so that the block is completed as a unit. If a back is not drawn evenly, excessive dilution results, and points of excessive weight may develop in the block. Draw is increased to relieve excessive weight in some areas.

After the preliminary draw period, an effort is made to draw an equal amount from each drawpoint. An erratic draw causes piping and ore packing in the various drawpoints. If a drawpoint is not pulled regularly, the ore packs in it.

Rate of Draw

Experience has shown that an average rate of draw of 18 inches of ore in place, or about 35 tons per draw raise per day, is the most effective rate. In order to produce 33,000 tons of ore per day, it is necessary to have 8 to 10 blocks in full production, and to have this output supplemented by tonnage from blocks closing out or coming into production.

Chute Tappers and Blasters

Chute tappers and blasters fill the transfer raises according to their draw orders. Broken ore usually is measured by filling the raises before trains are loaded. Tonnage is estimated if trains are loading while ore is running into the transfer raise from draw raises. The capacity of an all-gravity transfer raise is 35 tons in the main branch and 20 tons in the back-over. In slusher blocks the tonnage drawn is estimated by counting scraper loads. When transfer raises above a haulage panel drift contain a trainload of ore, the dispatcher is notified, and he sends a train into the loading chutes. Tonnage drawn is cross-checked with the car reports of haulage loaders.

Chute blasters do all of the ordinary secondary blasting in the blocks with block holes and bombs of 45-percent gelatin dynamite, and instantaneous electric blasting caps.

High hangups and high packs in draw raises are brought down by specially trained "high-pack" crews. Sometimes a crew can free a hangup by climbing ladders and placing charges of explosives directly against the boulders. At other times it is necessary to erect timber stagings from which the boulders can be drilled and blasted.

Blasting is done only at lunch time and at the end of the shift. The shift foreman is responsible for blasting from a central location after an accounting of all personnel has been made. This blasting procedure does not delay operations and allows time for the smoke and dust to clear before men reenter the blocks.

Modifications in Draw Practice

To better cope with existing conditions, several modifications of the original plans have been made. At first, excessive time was consumed in breaking boulders, clearing flooded grizzlies, and opening small draw raises. Later, the 8-pound hammers used by the chute tappers were replaced by 16-pound hammers. Grizzly rail spacing was increased from 9 inches to 12-1/2 inches; this allowed about 90 percent of the boulders to pass the grizzly. The size of the transfer-raise opening was increased and the grizzly was lengthened by 2 feet. The change in grizzlies virtually eliminated flooding.

In the original timbered blocks, draw raises were cribbed to a height of 25 feet above the grizzly sill, and the unsupported part of the draw raise was driven with a diameter of 3 feet. This cribbing took weight after undercutting closed the raises. Later, the cribbing was eliminated, and the draw-raise diameter was increased to 5 feet. The draw raises were belled out at the undercut level to a diameter of 8 feet. This has greatly reduced the hangups in the raises.

The vertical distance between the floor of the grizzly drift and that of the undercut level was reduced from 25 to 20 feet. This brought the hangups in the draw raise closer to the raise mouth and hence made them more accessible for blasting. The shorter raise reduced the block development cost. No appreciable increase in weight on the grizzly level has been noted.

Based on the caving experience of other mines, it was orginally planned to draw 12 inches of ore in place per day. After modifications were made on the grizzly level, the rate of draw was increased to 18 inches per day. This increased rate of draw reduced the active blocks required to meet the production quota. It also increased fragmentation of the ore and indicated better copper recovery from blocks initially drawn. Pillar blocks are drawn at a slower rate than those with four sides in virgin ground.

Draw Control

Procedure

The chief engineer and planning engineer maintain a $1\frac{1}{2}$-year tonnage and grade production forecast. This is corrected periodically to actual tonnages drawn. A monthly tonnage quota for the various blocks is set by the mine superintendent, who, with the general mine foreman and stope engineers, works out an allotted tonnage for the three shift foremen. A balanced tonnage per shift is maintained.

The stope engineer gives draw orders, compiled from the quotas, to the draw boss. The chute tapper is assigned to a grizzly line and makes a copy of the tapper report. He tries to fill his quota. The tonnage drawn is entered on the stope engineer's draw-order sheet and submitted to the tonnage department. The drawn chute tonnage is adjusted to the tram report once a week. When a block is being undercut or sealed off, the stope engineer may request an adjusted drawsheet at more frequent intervals.

Drawpoints are sealed off temporarily for chute repairs and to break up piping to control dilution. They are sealed permanently when the ore reaches the cutoff grade of 0.55 percent recovered copper. Recovered copper is a computed value equal to the total copper assay less the sum of the oxide copper and tailing loss.

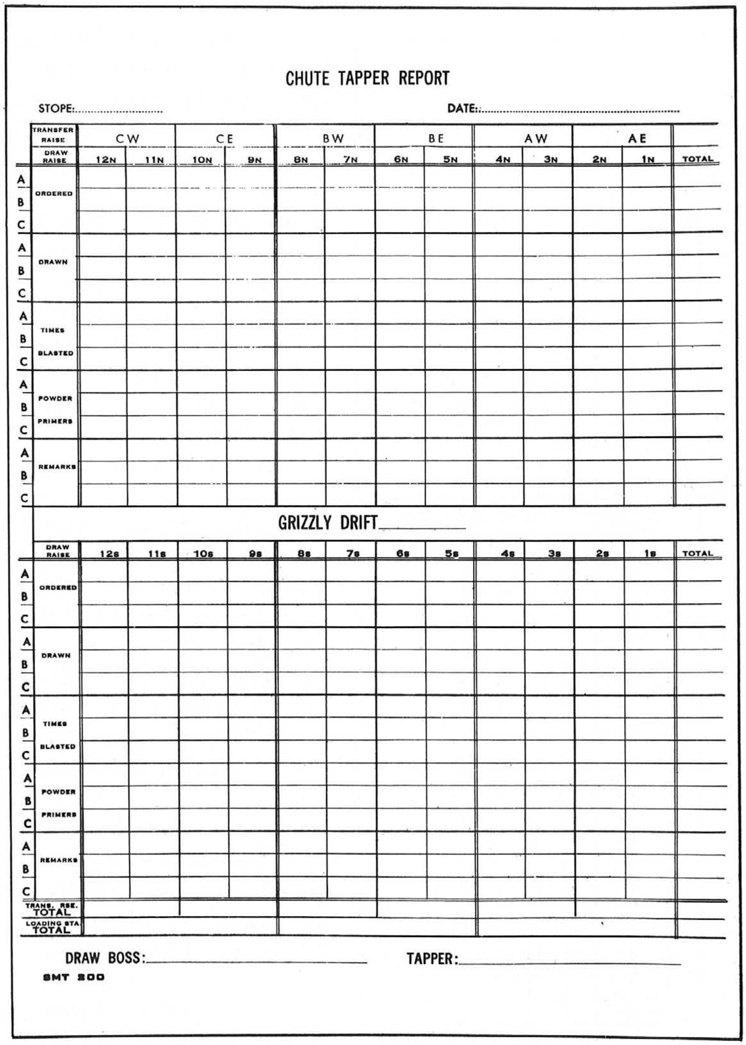
Reports

From various daily mine shift reports (figs. 47 through 53), the production department prepares daily and monthly tonnage reports (figs. 54 through 56), assay compilations (fig. 57), draw-tram comparisons, and special studies of production records requested by the mining department.

Sampling and Grade Control

Ore reserve estimates are compiled from drill logs. To help delineate the ore, primary development headings on grizzly levels that are entering or leaving the ore zone are sampled at intervals of 10 feet. When in ore, development headings on haulage levels are sampled at intervals of 20 feet. The interval is reduced to 10 feet where the headings approach the limits of the ore body. The results of the sampling of development headings are used when the estimate of ore reserves is adjusted periodically.

For grade control of the blocks, a 5-pound sample is taken at each drawpoint from each 100 tons drawn. Sample results are given regularly to the stope engineer. He may request that samples be taken from each 50 tons of ore drawn when the grade at any drawpoint approaches the cutoff point. When the grade reaches the cutoff point, he compares the physical appearance of the rock with the assay and decides if the raise should be sealed or held for subsequent sampling.



CHUTE TAPPER REPORT

STOPE:............................ DATE:............................

	TRANSFER RAISE	CW		CE		BW		BE		AW		AE		
	DRAW RAISE	12N	11N	10N	9N	8N	7N	6N	5N	4N	3N	2N	1N	TOTAL
A	ORDERED													
B														
C														
A	DRAWN													
B														
C														
A	TIMES BLASTED													
B														
C														
A	POWDER PRIMERS													
B														
C														
A	REMARKS													
B														
C														

GRIZZLY DRIFT__________

	DRAW RAISE	12S	11S	10S	9S	8S	7S	6S	5S	4S	3S	2S	1S	TOTAL
A	ORDERED													
B														
C														
A	DRAWN													
B														
C														
A	TIMES BLASTED													
B														
C														
A	POWDER PRIMERS													
B														
C														
A	REMARKS													
B														
C														
	TRANS. RSE. TOTAL													
	LOADING STA. TOTAL													

DRAW BOSS:______________________ TAPPER:______________________

SMT 200

FIGURE 47. - Chute-Tapper Report Form.

"Hold" signs are painted on boulders in drawpoints which have been overdrawn, show early dilution, or are very low grade and ready to be sealed permanently. Drawpoints are sealed after the assay returns from three consecutive samples fall at or below the grade cutoff. At the discretion of a stope engineer, temporarily sealed drawpoints are reopened occasionally to test for grade improvement.

In low-grade block-caving operations, large tonnage and close grade control are essential for economical operation. The stope engineer attempts to draw the block evenly without cutting tonnage.

Underground Transportation

Transportation of a large tonnage at high speed from a single level at San Manuel requires a very flexible haulage system to give maximum access to stoping areas, minimize delays, and provide for safety. With these requirements as a guide, the haulage was laid out as a simple double loop with one-way traffic.

The following is abstracted from a paper[13] presented by C. F. Cigliana, general mine foreman at San Manuel.

Equipment

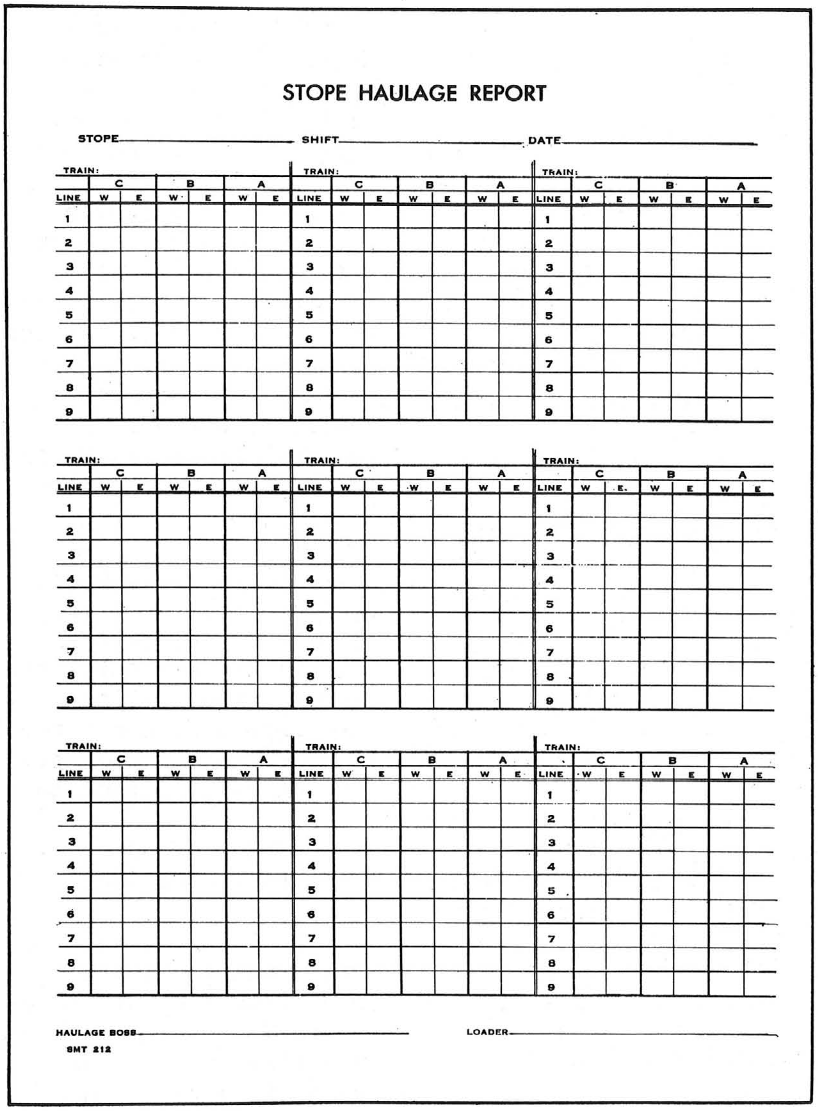
Haulage equipment consists of fourteen 23-ton locomotives and two 12-ton locomotives that operate from a 275-volt direct-current trolley line and

[13] Cigliana, C. F., Ore Transportation at San Manuel: Min. Eng., vol. 10, No. 5, May 1958, pp. 573-576.

thirteen 8-ton battery locomotives used for development, repair, supply, and service. Full voltage is maintained by three 50-kw. rectifier stations; they are parallel so that if any station fails, power can be maintained by the other two. One station is on the north haulage, one is on the south haulage, and one is at the hoisting shaft.

The 23-ton trolley locomotives (64 inches wide, 56 inches high, and 21 feet 6 inches long) are driven by two centrally hung 120-hp. motors. This arrangement provides the balance necessary for best track performance for long hauls at high speed. At both ends of the locomotives there are automatic couplers, each equipped with an air-operated uncoupler.

All locomotives have three types of braking on all four wheels--air braking, hand braking, and dynamic braking--and are also equipped with overspeed and deadman controls.



STOPE HAULAGE REPORT

STOPE________________ SHIFT________________ DATE________________

TRAIN:							TRAIN:							TRAIN:						
	C		B		A			C		B		A			C		B		A	
LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

TRAIN:							TRAIN:							TRAIN:						
	C		B		A			C		B		A			C		B		A	
LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

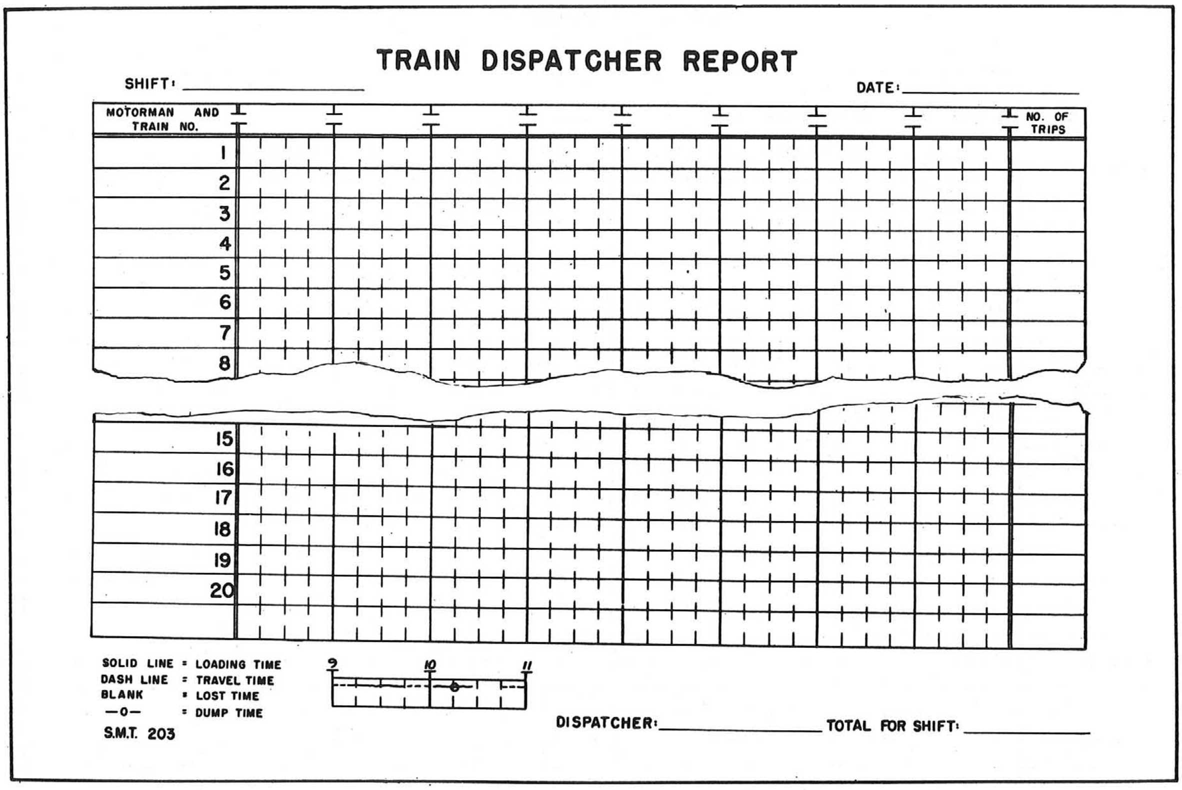
TRAIN:							TRAIN:							TRAIN:						
	C		B		A			C		B		A			C		B		A	
LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E	LINE	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

HAULAGE BOSS________________ LOADER________________

SMT 212

FIGURE 48. - Stope-Haulage Report Form.

Control is set up for selective series or parallel operation of traction motors in either direction. Acceleration is obtained by moving the main drum clockwise, and dynamic braking is provided by moving the main drum counterclockwise from the off position. The controller is equipped with a deadman tip-up handle, which must be held down to apply power to the locomotive; if it is released, the circuits are opened, and power is withheld from the traction motors. At the same time emergency airbrakes operate. If the locomotive exceeds 15 m.p.h., the overspeed contact will automatically open, removing power from the traction motors, and brakes will be applied through an emergency air valve in the run or brake position. When the motor speed has been sufficiently decreased, power again can be applied.


TRAIN DISPATCHER REPORT

SHIFT: ______ DATE: ______

MOTORMAN AND TRAIN NO.									NO. OF TRIPS
1									
2									
3									
4									
5									
6									
7									
8									
15									
16									
17									
18									
19									
20									

SOLID LINE = LOADING TIME
DASH LINE = TRAVEL TIME
BLANK = LOST TIME
—o— = DUMP TIME

9 10 11

S.M.T. 203

DISPATCHER: ______ TOTAL FOR SHIFT: ______

FIGURE 49. - Train-Dispatcher Report Form.

The handbrake is applied by a 12-inch wheel operating a square-threaded brake screw. Airbrakes exert pressure on the brake linkage in the same way as the handbrake. A pressure cylinder is connected to the standard hand-operated brake linkage, and a governor cuts the compressor into the circuit when the pressure is too low to be effective.

Eleven trains of fifteen 12-ton cars each are needed to haul the ore. Each car is equipped with one stationary coupling and one rotary coupling and a local innovation--a safety coupling used to prevent cars from uncoupling in transit. The box-type steel cars have rubber-cushioned draft gear and are mounted on two four-wheel roller-bearing trucks with 14-inch-diameter gray chilled cast-iron wheels.

The entire haulage operation is controlled by a dispatcher from whom clearance must be obtained before trains are moved into the haulage loops.

Train Loading

There are three phases in ore transportation--loading, traveling, and dumping. In the first phase, trains are loaded against the track grade to take

POCKET DUMP REPORT

SHAFT:__________ SHIFT:__________ DATE:__________

ESTIMATED TONS BEGINNING SHIFT:__________

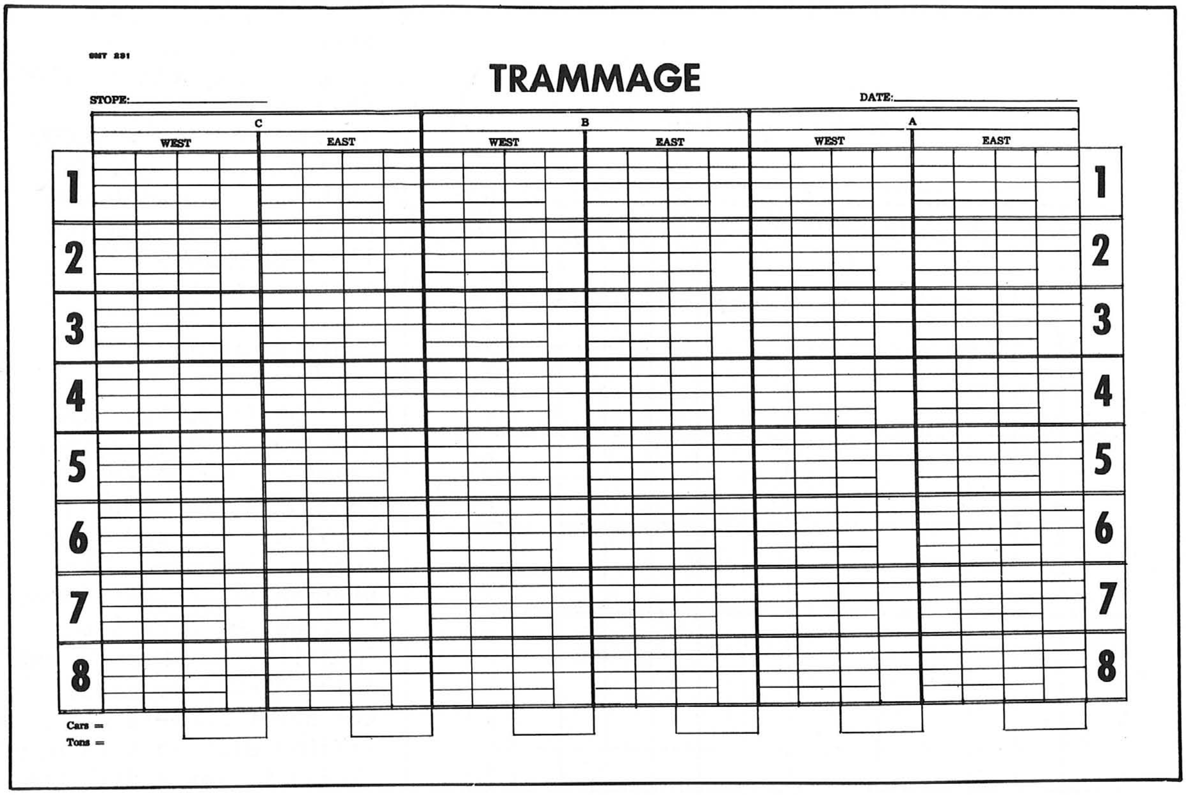
		CARS DUMPED					CARS DUMPED		
TRAIN	TRIP	ALLISON	MAGOR	DEVELOPMENT	TRAIN	TRIP	ALLISON	MAGOR	DEVELOPMENT
TOTAL					TOTAL				

SKIPS HOISTED:__________

SKIP TENDER:__________ DUMPMAN:__________

SMT 204

FIGURE 50. - Pocket-Dump Report Form.


TRAMMAGE

STOPE: DATE:

	C		B		A		
	WEST	EAST	WEST	EAST	WEST	EAST	
1							1
2							2
3							3
4							4
5							5
6							6
7							7
8							8

Cars =
Tons =

FIGURE 51. - Trammage Form.

all slack out of couplings, making it easier and smoother to spot cars under the loading chute. Since trains are approximately 300 feet long, a system of signals is used to spot cars at the chutes. Trains being loaded are stopped or moved by the operator when signalled by men using hand-operated lights at the loading stations (fig. 58).

Ore flows into the cars from transfer raises through steel chutes equipped with undercut guillotine gates (fig. 23).

The mine operates on a three-shift-a-day basis, but blocks are drawn on a staggered basis of two shifts, leaving the third shift for maintenance and repair work. Cleaning the drifts of spillage from chutes during loading is one of the jobs done on the third shift. The spillage is loaded with a rocker shovel and a specially constructed track cleaner that draws power from the trolley line (fig. 59).

Hauling and Dumping

When a train is loaded, the operator obtains clearance and hauling instructions from the dispatcher and proceeds into the north haulage, clearing the panel drift and closing the haulage block signals. He then observes posted speed limits and block signal procedure while traveling the haulage loop to the rotary dump.

Record of Sampling

Stope No.____________ Date____________

	DRAW POINTS												
Line	12	11	10	9	8	7	6	5	4	3	2	1	
1													N
													S
2													N
													S
3													N
													S
4													N
													S
5													N
													S
6													N
													S
7													N
													S
8													N
													S
	C				B				A				

SMT 230

FIGURE 52. - Record of Sampling Form.

Cars are dumped three at a time in a tipple that rotates 180° (fig. 60). The ore is discharged into a 1,000-ton-capacity concrete pocket, lined with 60-pound rail, from which it is hoisted to the surface by skips.

Shaft Pockets

Shaft pockets and stations on the haulage level in No. 3A and No. 3B shafts are supported by steel sets and reinforced concrete. The storage ore pockets have a 1,000-ton live-load capacity. Weak rock and heavy ground pressures in the shaft pockets and loading station area precluded larger underground storage areas.

The skip loading arrangement is laid out to measure the required volume of ore from a storage pocket through an air-powered undercut-guillotine-type gate into a measuring basket or skip loading pocket. The pocket chute door is operated by a 16-inch air cylinder with a 56-inch stroke. The skip loading pocket (fig. 61) discharges its load directly into the skip through an air-powered overcut arc-type gate. Provision was made by planning engineers for the installation of skip loading chairs, but these were unnecessary.

Hoisting is semiautomatic in that the automatic controls that move the skip from the loading position to the dumping position are actuated by the skip tender at the loading pocket.

Skip Loading

Two men load the skip to avoid the hazard that may occur when especially wet ore is handled. Electrical signal lights are used to coordinate the loading operation. One man draws the ore from the storage pocket into the measuring pocket. The other man loads the skip from the measuring pocket and then sets into motion the semiautomatic hoisting cycle. Automatic controls take the skip from the loading position to the dumping position. Ore skips are run in balance.

Personnel and Performance

A typical haulage shift is organized as follows:

Haulage-shift boss	1
Jigger boss	1
Dispatcher	1
Rotary-dump operator	4
Skip tenders	4
Operators, 23-ton locomotives	11
Operators, 12-ton locomotives	2
Chute blasters	11
Chute tappers	11
Cleanup man	1
Total	47

This organization is capable of hauling a maximum of 13,124 tons in a single shift. The maximum for three shifts is 35,321 tons. The average ore tonnage per man-shift on haulage only is 245.

A breakdown of the average time consumed in the four phases of the haulage operation shows the loading time to be 24.43 minutes, hauling time to the rotary dumps, 12.94 minutes; dumping time, 4.64 minutes; and return trip, 12.12 minutes. This totals 54.13 minutes per train, and each train averages 6.4 trips per shift.

Shaft Stations

No. 4 shaft stations on the haulage and grizzly levels are 22 feet wide by 11 feet high, rising to 17 feet at the shaft. All stations are supported with steel sets and reinforced concrete. The stations can handle the largest items of equipment and supplies used in the mine. Stations in other shafts are large enough to handle the equipment needed for development headings driven from the shafts and to provide room for pumping equipment needed in sinking.

Underground Equipment Maintenance

A regular schedule is followed for all maintenance and lubrication of underground operating equipment. Electric slushers are inspected and lubricated each day. Spare slushers are kept in stock to replace worn machines. When necessary, steel chutes are repaired on each shift. Haulage cars are inspected daily, and 15-car trains are sent to the maintenance shop for lubrication and preventive maintenance. A cycle of these cars is completed each month. The cycle for the smaller 5- and 7-ton cars is 2 months. About 1,800 pounds of lubricants is required for a cycle of all mine cars. Rotary dumps, loading equipment, and hoists are inspected and lubricated daily.

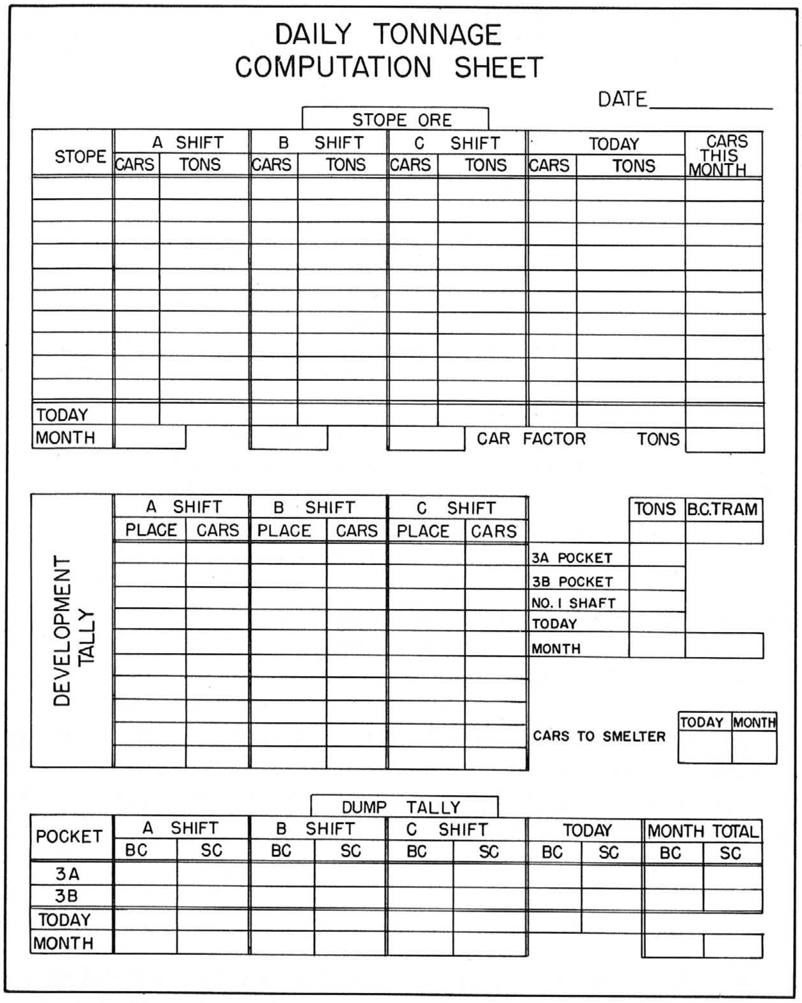


FIGURE 53. - Daily Tonnage Computation Sheet.

Three hours each Monday and Friday are set aside for minor preventive maintenance. Major maintenance is done during holiday periods, and emergency repairs are undertaken as needed. A large locomotive shop is situated on the haulage level in the No. 1 shaft area. Three large haulage locomotives are rotated through the shop at a time; one locomotive goes through for major overhaul, and the other two are exchanged periodically for preventive maintenance. The cycle requires about 1 week.

Communications

Good, dependable communication is one of the key factors in a safe, high-speed, efficient underground mining operation. To coordinate a working force of approximately 900 men and its equipment, three methods of communication are used: A standard telephone system, a two-way audio system, and a radiotelephone system.

The following information about the communications system is abstracted from a paper[14] by C. L. Pillar, mine superintendent.

Telephone System

The standard telephone system is mentioned briefly because it is highly conventional and has been in standard use in industrial operations in this country for the past 50 years. The main function of this system is to connect underground operations with surface operations and service facilities.

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14 Pillar, C. L., Mine Communication System at San Manuel: Proc. Ann. Metting AIME, San Francisco, Calif., February 1959, 9 pp.

Daily Mine Production Report

DATE________________

BLOCK	"A" SHIFT				"B" SHIFT				"C" SHIFT				BLOCK QUOTA	DRAW TOTALS								TOTAL MAN SHIFTS	AVERAGE TONS M.S.
	TONNAGE		MAN SHIFTS	TONS M.S.	TONNAGE		MAN SHIFTS	TONS M.S.	TONNAGE		MAN SHIFTS	TONS M.S.		ALL SHIFTS	% COPPER			MONTH TO DATE	% COPPER				
	QUOTA	DRAWN			QUOTA	DRAWN			QUOTA	DRAWN					TOTAL	OXIDE	NET		TOTAL	OXIDE	NET		
STOPE TOTALS																							
													DEVELOPMENT ORE										
													TOTAL MINE PRODUCTION										

UNDERGROUND CAR FACTOR =

TONS NET CU/DAY =

ORE HOISTED

SHAFT	"A" SHIFT		"B" SHIFT		"C" SHIFT		DAY TOTALS		MO. TO DATE TOTALS	
	SKIPS	TONS	SKIPS	TONS	SKIPS	TONS	SKIPS	TONS	SKIPS	TONS
1										
2										
3A										
3B										
TOTAL										

SMT 211R

ORE SHIPPED · CAR FACTOR FOR DAY =

LOADED AT	"A" SHIFT		"B" SHIFT		"C" SHIFT		DAY TOTAL		MO TO DATE TOTALS	
	CARS	TONS	CARS	TONS	CARS	TONS	CARS	TONS	CARS	TONS
RAMP										
3A BIN										
3B BIN										
TOTAL										

FIGURE 54. - Daily Mine Production Report Form.

STOPE ________ DRAW RAISE TONNAGE PERIOD ________

	DATE	C																B				A								LINE TOTALS				
		12 W				11				10 E				9				7			6	3			2									
		TONS	TOT	OX	NET	TONS	TOT	OX	NET	TONS	TOT	OX	NET	TONS	TOT	OX	N	OT	OX	NET	TONS	TOT	OX	NET	TONS	TO			T	TONS	TOT	OX	NET	
N																																		N
1	ADJ'D																																	1
	FWD.																																	
	To date																																	
S	ADJ'D																																	S
	FWD.																																	
	To date																																	
N																																		N
8	ADJ'D																																	8
	FWD.																																	
	To date																																	
S	ADJ'D																																	S
	FWD.																																	
	To date																																	
	ADJ'D																																	
	FWD.																																	
	To date																																	

NOTE:
☐ UNADJUSTED DRAW FOR PERIOD

Date	TOT %	OX %	NET %	Date	TOT %	OX %	NET %	Date	TOT %	OX %	NET %	Date	TOT %	OX %	NET %

FIGURE 55. - Draw-Raise Tonnage Form.

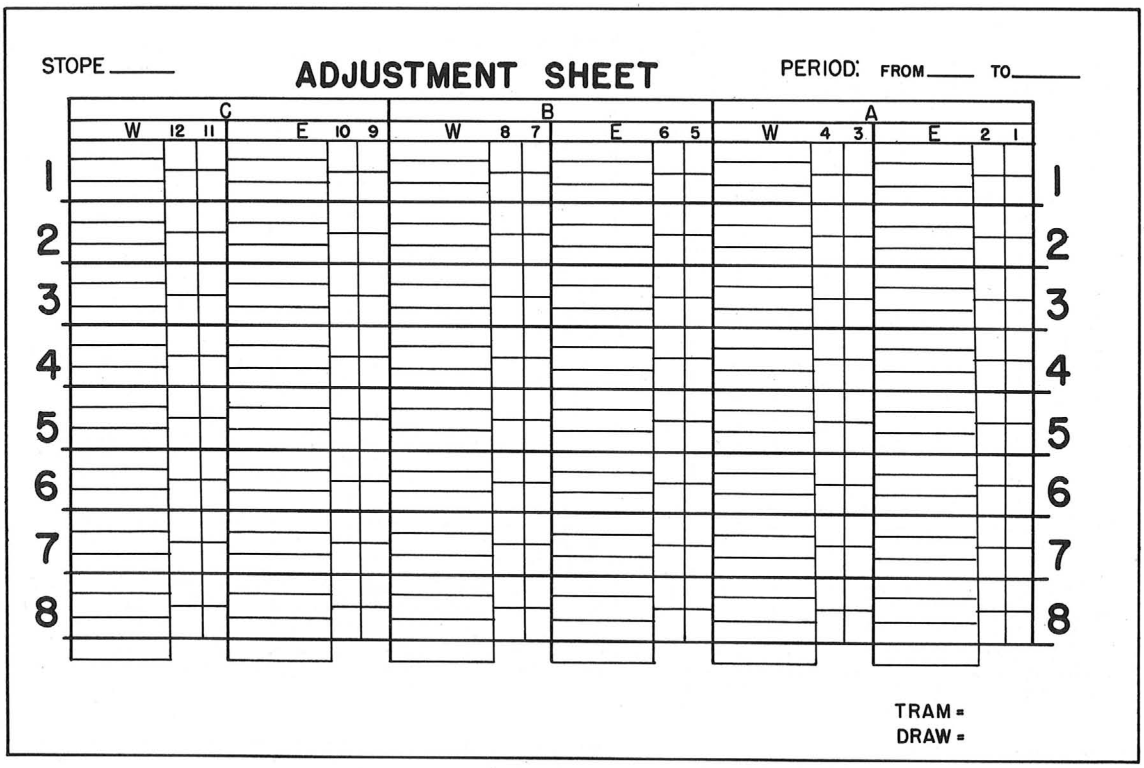

FIGURE 56. - Adjustment Sheet Form.

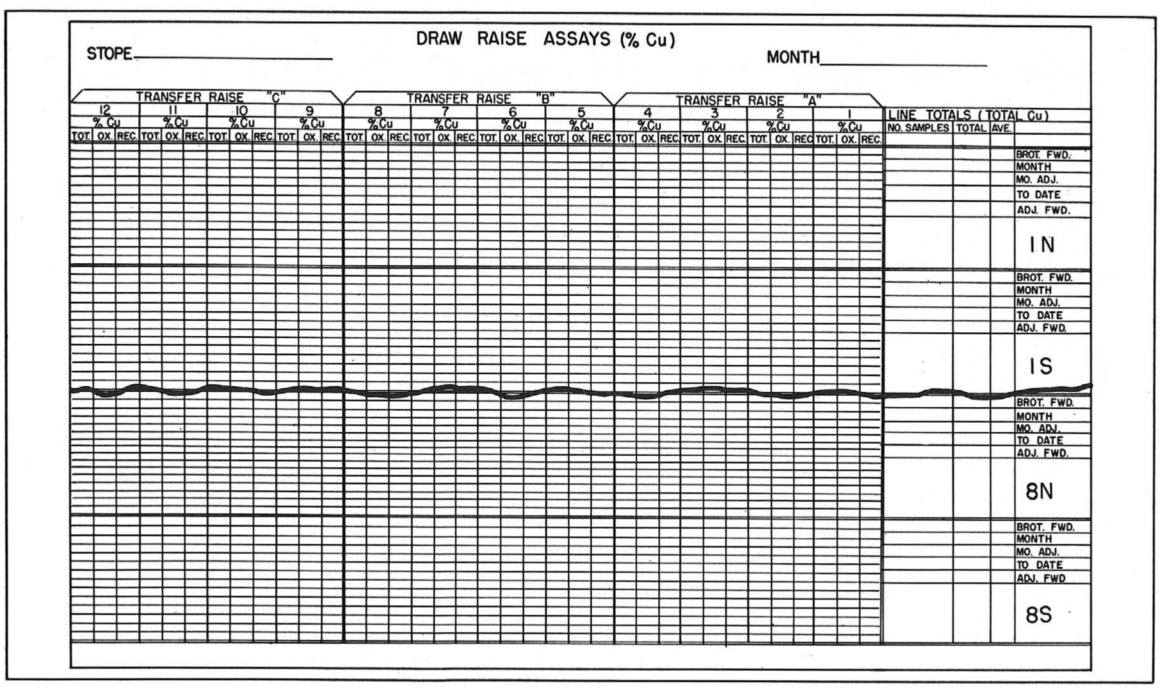

FIGURE 57. - Draw-Raise Assays Form.

FIGURE 58. - Train Being Loaded in Haulage-Panel Drift.

The telephone system is divided into seven separate circuits. Each circuit operates like a rural telephone system; that is, a speaker gives a designated ring on his circuit to call a particular station. The system is built so that from each phone, by pressing the proper button, any ring can be made on any circuit. Each of the four shafts has its own circuit; it includes hoists, headframe, collar, level stations, dumping stations, electric substations, and spill pockets. The fifth circuit includes all surface shops and supply and service facilities. The sixth circuit covers all phones in the interior part of the mine for the first lift operations, which covers both grizzly and haulage levels and is too far away from the mine shafts to be practical to include them in the shaft circuit (figs. 62 and 63). The second-level operation requires a seventh circuit for its interior telephones. This type of communication is used by relatively few men and is used mainly to coordinate hoisting and maintenance operations at the shafts and to order supply, maintenance, and service functions from the surface for underground operations. Its scope is limited to personnel who are normally stationed by a telephone. It cannot be used by a large, flexible operation that is in constant motion.

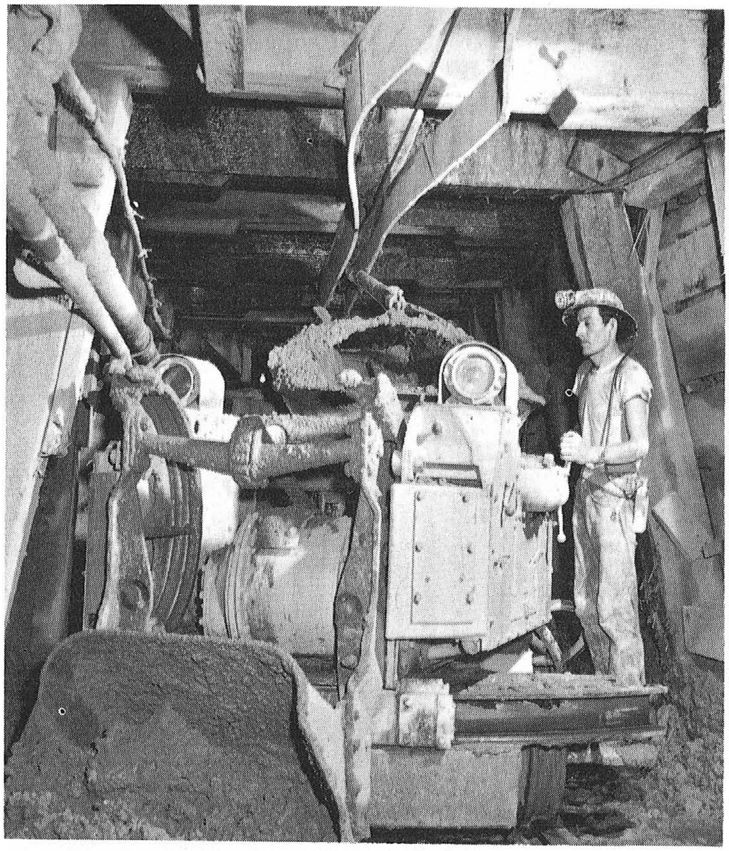

FIGURE 59. - Track Cleaner Modified to Take Power From Trolley Line.

Two-Way Audio System

The second method of speech transmission used is referred to as a two-way audio system. Each audio station is a rugged, compact, wall-mounted unit with a speaker equipped with volume control, microphone, and talking switch; it is wired directly through an amplifying unit to all other units in the system. Two two-conductor individually shielded circuits connect all units of the system. The microphone transmits the voice over the system to a control room that houses proper amplification equipment, which, in turn, relays the amplified voice back through the system to the volume-control-equipped speakers. This is accomplished by pushing in the talking switch and speaking into the mouthpiece of the microphone. When the talking switch is pushed in, it cuts out the individual loudspeaker in that particular station. As soon as the button is released, the loudspeaker is automatically ready to receive replies from any other station. This method allows two or more persons to talk to each other at the same time and also broadcasts their conversation over the entire audio system (fig. 64). It is superior to a conventional telephone primarily because it can be used to locate almost immediately patrolling supervision and service crews in order to transmit necessary orders to them. Another big advantage is that the conversation, orders, and instructions are broadcasted over the entire system and shortly become known by a large majority of the working force.

The audio system is used on grizzly and haulage levels in the mining area where 50 audio stations are strategically located to coordinate the concentrated operations in the area (figs. 61 and 62). The system is channeled through a dispatcher and all supervision stations in the mining area. The dispatcher acts as a clearing house for all information fed into the system. He is responsible for seeing that the proper people receive the information.

The following examples indicate what kind of information is transmitted over the system, to whom it is relayed, and for what purpose. The examples represent a few of the more common usages of this communication equipment.

Originator	Information	To whom relayed	Action taken
Draw boss.....	Amount of ore in block-transfer raise system.	Dispatcher......	Routing of correct number of trains under block.
Do......	Special blast clearance.	Repair and haulage crews in area on haulage level.	Area properly cleared and guarded for the blast.
Haulage boss..	Mechanical trouble in loading chutes, pipelines, dust sprays, and so forth.	Mechanical boss and crew.	Mechanical crew made trouble area its next stop.
Do......	Bad-order track, switches, and so forth.	Track boss and crew.	Track boss dispatched necessary crew to correct the trouble.
Do......	Bad-order electrical equipment, locomotives, trolley line, loading lights, block signals, and so forth.	Electrical boss and crew.	Electrical crew made trouble area its next stop.
Do......	Sizable spillage from loading chute temporarily blocking panel haulage drift.	Dispatcher......	Cleanup crew with mucking equipment immediately dispatched to trouble area.
Development boss.	Transfer raises full and need pulling to provide room for rock broken by stope preparation crews.	.......do.......	Haulage train dispatched to block area to pull required chutes.
Do......	The amount of concrete required in block areas.	Concrete remixer.	Proper number of concrete trains dispatched to required areas.
Do......	Additional supplies and equipment not covered on routine supply orders immediately required in development areas.	Supply boss and crew.	Necessary supplies and equipment delivered to required areas at first opportunity.

Originator	Information	To whom relayed	Action taken
Repair boss...	Special material not covered on routine supply orders--immediate demand.	Supply boss and crew.	Special material dispatched to required area at earliest opportunity.
Do......	Removal of electrical or mechanical equipment required to accomplish repair.	Electrical or mechanical boss and crews.	Necessary crew assigned and dispatched to required area to perform work.
Production foreman.	Reassigning of draw crews during the shift to compensate for change.	Draw bosses.....	Required men immediately transferred to other areas so maximum production could be maintained.
Development foreman.	Reassigning of work and location of development crews during the shift to compensate for unforeseen changes in regularly scheduled work.	Development boses.	Required men immediately reassigned with minimum delay and work stoppage.
Shift foreman.	Changes in regularly scheduled production and development operations during the shift to compensate for any radical change in operating conditions.	Production and development foreman.	Orders to individual bosses immediately transmitted to place desired operational changes in effect.

For wrecks or accidents this system of communication has proven invaluable in getting necessary personnel to the trouble area in minimum time and for obtaining the proper tools and equipment to handle the job at hand.

It is also used to account for all personnel in the entire mining area during the scheduled central electric blasting at the lunch and quitting times on each shift.

In general, this system of communication allows supervisors to quickly change routine scheduled operations and to correct sources of trouble rapidly and efficiently.

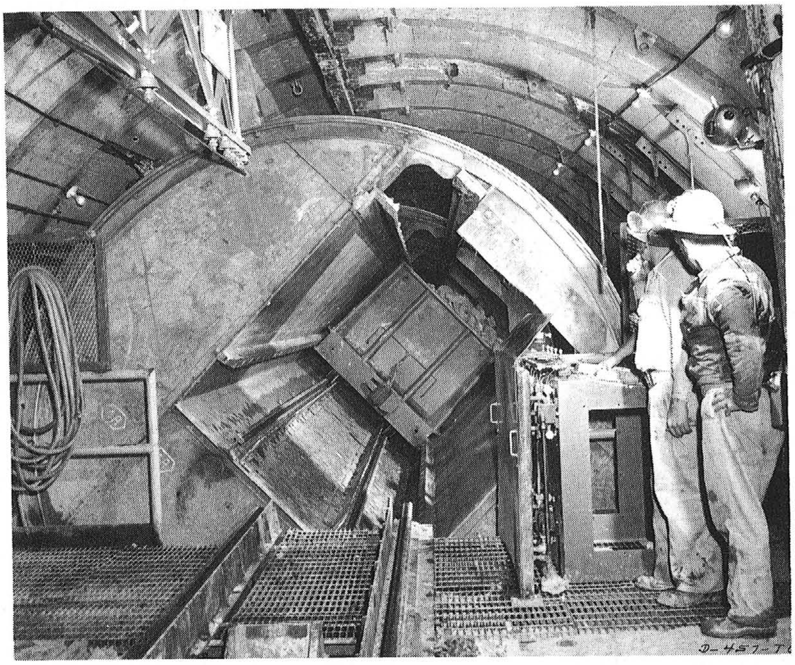

FIGURE 60. - Rotary Tipple Dumps Three Cars at a Time.

Radiophone System

The third method of speech transmission used generally is referred to as a radiophone. Its major advantage over other types of communication is that it allows voice transmission to moving trains and locomotives. All conversation is carried on a single 88-kc. frequency-modulated system. With this equipment, orders can be sent instantly and simultaneously to all stations on its frequency. By pressing a button on the microphone and speaking directly into it, a speaker transmits to all other stations. When the speaker releases this button, the loudspeaker is automatically ready to receive replies from any other station.

Radiophones on locomotives operate on electricity from locomotive batteries; stationary units operate on 110-volt alternating current (figs. 65, 66). Trolley locomotive radiophones use 250-volt direct-current power direct from the trolley line. Battery locomotive radiophones use a dynamotor to raise the 96-volt battery potential to 250 volts. This allows a common 250-volt radiophone unit to be used in either type of locomotive. A resistor unit reduces the available direct-current power to that required for operation of individual locomotive phone stations. To protect equipment and operator, a combination power cutoff and fuse is placed in the line between the power source and resistor. The equipment is completely insulated. The frequency-modulated transmitter is matched to a carrier line or the mine trolley and transmits with sufficient range to cover the entire haulage level, a maximum distance of 2 miles. The receiver takes the message from the carrier or trolley line through a 4-foot airgap and relays it to a heavy-duty loudspeaker equipped with volume control. The microphone is a single carbon button-type unit with a molded neoprene case and is equipped with a length of "coiled cord" sufficient to make it self-retracting when the microphone is returned to its hanger.

Some difficulty has been experienced with good, clear transmission from battery locomotives without the use of an aerial along the top of the locomotive. This is not too practical, but the problem was solved by using a small, light, highly insulated "pigtail" which the battery locomotive engineer

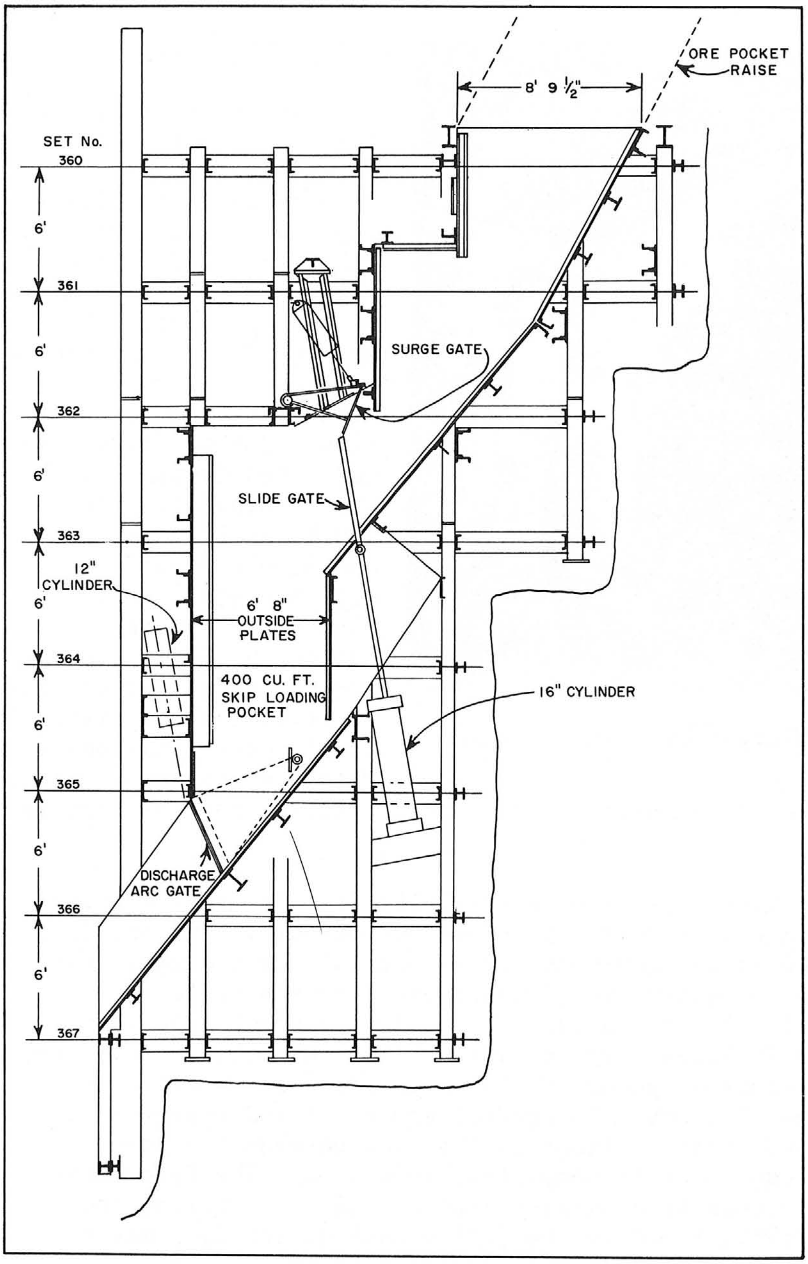

FIGURE 61. - Section Through Skip Loading Pocket.

hooks over the trolley wire when he transmits a message. Thus, when he is talking into the system, he cannot be in motion. However, this has not been a serious handicap to haulage operations, for battery locomotives receive all communications while in motion clearly and with good volume without the use of the pigtail.

The radiophone is installed in all locomotives operating on the haulage level. In addition, radiophones are installed on the haulage level in the dispatcher's office, locomotive repair shop, car repair shop, electric maintenance shop, rotary dump stations, and supervisory stations for haulage, development, repair, supply, and service operations.

The radiophone is used mainly to coordinate the haulage system. All trains on the level are moved only on direct verbal orders from the dispatcher, who becomes the nerve center of the operation. He knows the location of all trains at all times. To implement his mental picture of the haulage operation, a diagram of the haulage level is mounted on a steel panel in front of him (fig. 67). Small, numbered magnetized blocks are placed on the diagram and are moved to correspond with trains operating on the level.

Locomotive engineers periodically report to the dispatcher while in transit; this enables the dispatcher to trace train movements. It should also be pointed out that a complete block signal system covers the entire haulage level.

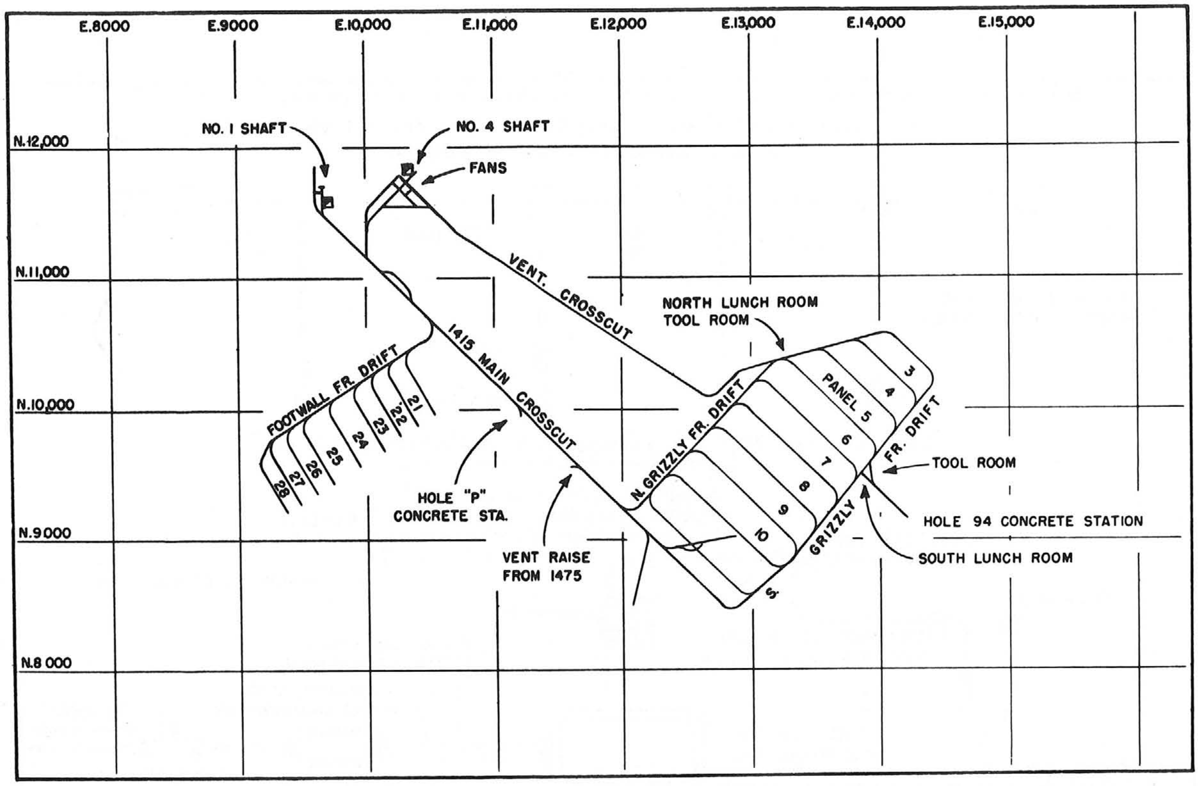

FIGURE 62. - Location of Communications Sets on 1415 Grizzly Level.

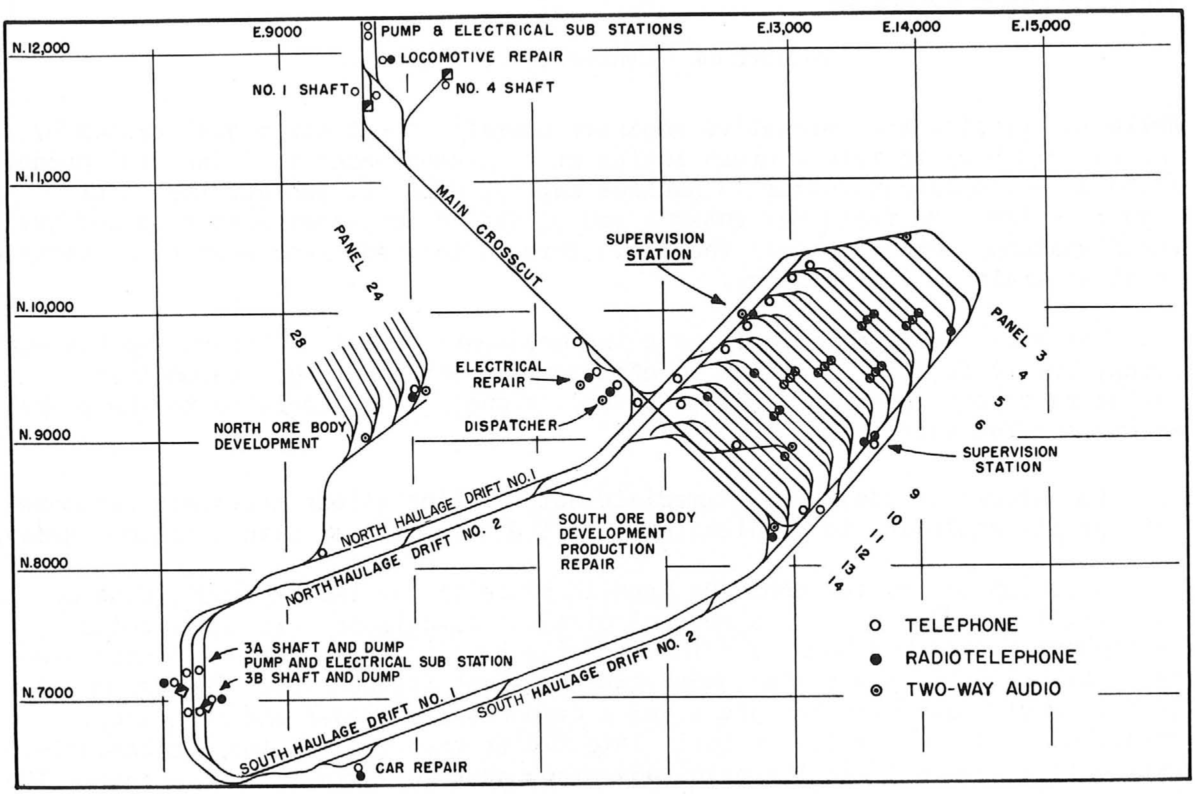

FIGURE 63. - Location of Communication Sets on 1475 Haulage Level.

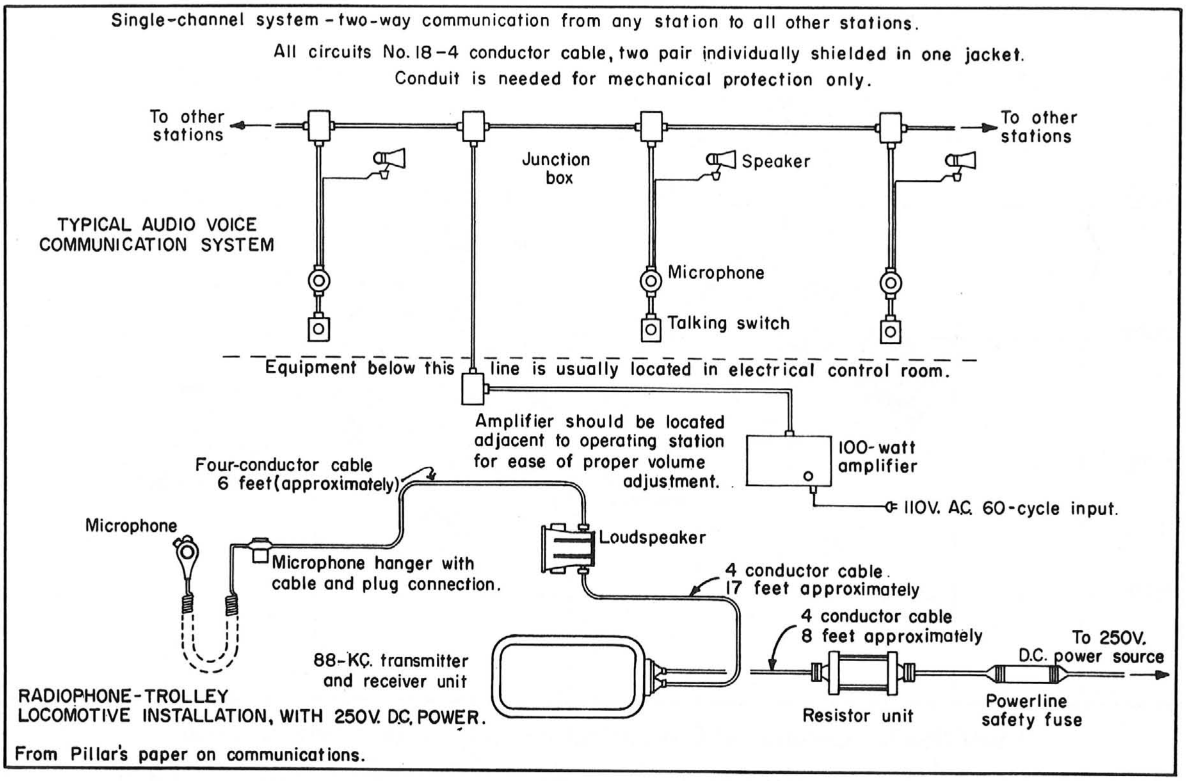

FIGURE 64. - Communication Diagrams.

While in transit, the locomotive engineer operates the block signal system by raising his hand to trip a brush switch as he passes under it. The radiophone, although an important factor in haulage safety, does not replace the block signal system. No train may enter a red block; the engineer must stop and call the dispatcher, who can clear the train through the red block when he is certain no other train is in the block.

The radiophone not only directs the movements of all trains on the haulage level, but it is also important in efficient troubleshooting. Locomotive engineers report bad-order track, trolley, signals, and pipelines to the proper supervisor for correction.

For wrecks or accidents, immediate notification allows necessary personnel with proper equipment to be dispatched to the trouble area with a minimum delay.

A radiophone system could be used in place of the two-way audio system. However, the audio system has several distinct advantages. In the event a radiophone system had been used in lieu of an audio system, it would have been necessary to split the system under two different frequencies. The volume of traffic would have been too great for a common system under one frequency. When the radiophone system is split into two or more frequencies for communication in a common area, its major advantage over an audio system is lost. The two systems work together very well.

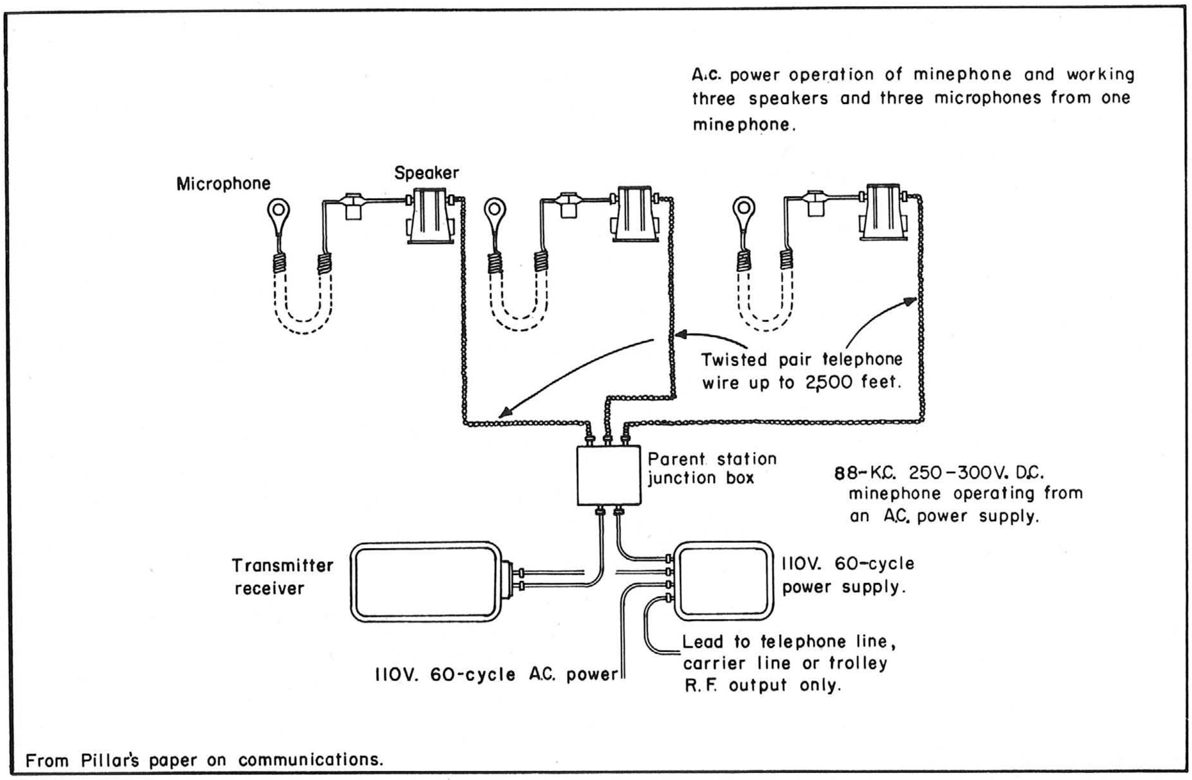

FIGURE 65. - Radiophone Stationary-Station Installation With 110-Volt Alternating-Current Power.

The audio system and radiophone system have been in operation approximately $2\frac{1}{2}$ years. The required maintenance on both systems has been moderate. An electronic technician and a helper, working 6 days per week on day shift, keep abreast of all routine maintenance on both systems, the movement of stations, the installation of new units, and the rebuilding and repairing of bad-order equipment. On the whole, the equipment has proven very reliable. In $2\frac{1}{2}$ years only one serious communication failure has been experienced. The radiophones were out for 6 hours on a graveyard shift. During that shift only 7,000 tons of ore was hauled. This tonnage reduction was not due to a wreck, but because the tempo and efficiency of the entire haulage system had been substantially reduced.

Costs and Results

During 1958, the cost of installing new communication equipment for the expanding development of the mine and for maintaining all communication systems over the entire mine was approximately $0.01 per ton of ore produced.

Each system has its advantages and limitations, but the combination of the systems, channeled through a dispatcher and the supervision stations, gives highly satisfactory results. Experience with the relatively new electronic communication equipment has been most satisfactory. It is dependable and practical and has a good performance record. Maintenance has not been a serious

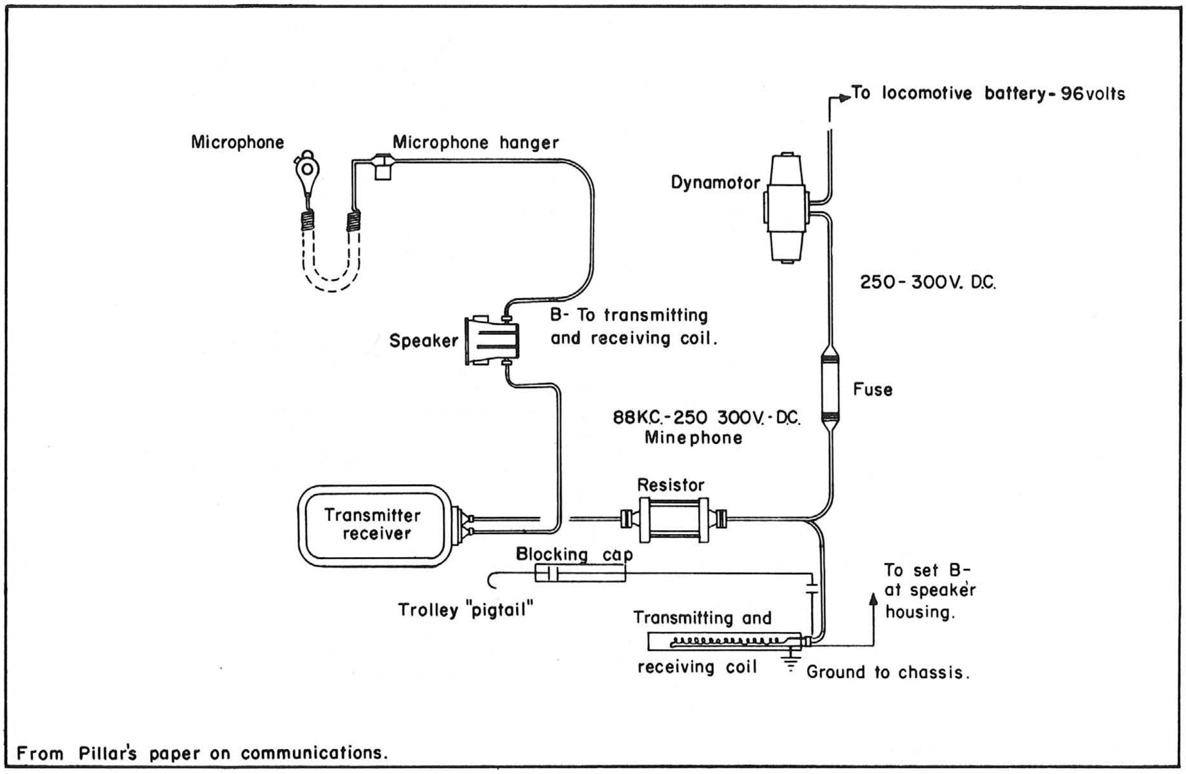

FIGURE 66. - Radiophone Battery Locomotive Installation with 96-Volt Direct-Current Power.

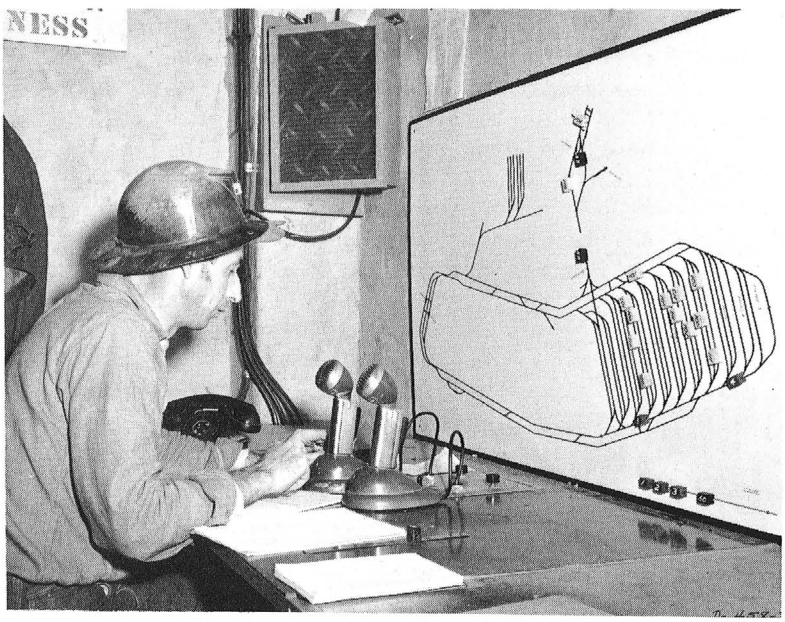

FIGURE 67. - Train Dispatcher's Office on Haulage Level.

problem. An electrician can be trained in a short time to properly handle all required maintenance on this type of equipment.

The use of electronic communication systems by the mining industry in this country today is laying the foundation for the future automation of mining operations. It is now possible, by the expansion and modification of a basic electronic communication system, to completely operate equipment for pumping, ventilation, ore transportation, ore dumping, ore loading, and hoisting by remote automatic control.

Waste Disposal

In earlier stages of the mine development, waste from crosscuts and haulage-loop drifts was hoisted through No. 1 and No. 2 shafts. No. 2 shaft was abandoned in the later part of 1958, and now all waste is hoisted in 4-ton-capacity skips through the No. 1 shaft. The rock is dumped into bins on the surface; from there it is hauled by trucks to the waste dump. The storage yards and many buildings of the surface plant are on the common dump of the No. 1 and No. 4 shafts.

Extraction

To date, both tonnage extraction and ore grade have been good. By August 1959, 23 blocks had been drawn to completion. The results are shown in table 12.

Most of the 23 blocks were surrounded by virgin ground, and overall extraction from a completed level of the mine may not be as high as first indicated. From experience to date, San Manuel engineers have estimated that the total extraction at completion of the first level will be approximately 103 percent of the expected undercut tonnage at 92 to 93 percent of the estimated ore grade.

Dilution

The first dilution is expected to appear after about 35 percent of the estimated tonnage has been drawn from a block. It has appeared after as little as 5 percent and as much as 65 percent of the tonnage has been drawn. Iron oxide and copper oxide minerals make up the first dilution material. As the draw progresses, the overlying conglomerate also becomes a part of the diluting material.

Uneven stope draw often causes dilution. If a drawpoint is continually overdrawn, a pipe is formed that draws waste into the ore column.

The engineering department has estimated that dilution has averaged 10.7 percent in the 23 blocks drawn to completion as of August 1959.

Production Rates

The tons per man-shift for tappers, blasters, and slusher operators at the 1415 level in 1956, 1957, and 1958, was 105.5, 124.1, and 203.1, respectively.

Production for various classifications of labor during the first 7 months of 1959 was as follows:

Gravity blocks:	
Tons per man-shift (blasters and tappers)	310.4
Tons per repair-shift	3,555.2
Slusher blocks:	
Tons per man-shift (slusher operators)	165.0
Tons per repair-shift	9,668.0
Haulage (Haulage men, skip tenders, and rotary dump operators), tons per man-shift	224.6
Mine production, tons per man-shift	54.0

TABLE 12. - Summary of draw data for first 23 blocks extracted

Type block[1]	Type support[2]	Date under draw: Block	Date under draw: From	Date under draw: To	No. of virgin sides	Area, square feet	Average ore column	Average rock column at undercut date	Average subsidence at undercut date	Draw-raise spacing	Tonnage: Expected	Tonnage: Drawn	Tonnage: Extracted, percent	Net sulfide, percent: Expected	Net sulfide, percent: Drawn	Net copper, percent extracted	Dilution, average percent	Average draw rate, inches per day per drawpoint
G	T	7-1	1-24-56	2-12-58	4	37,800	398	1,115	0	15x17½	1,210,278	1,993,720	164.7	0.761	0.741	97.4	12.6	13.0
G	T	9-1	2-4-56	4-15-58	4	51,450	566	1,125	0	17½x17½	2,329,400	2,412,265	103.6	.710	.671	91.3	12.4	10.1
G	R	6-2	3-28-56	7-11-58	4	58,400	467	1,100	0	15x17½	1,888,450	2,617,900	138.6	1.020	.914	89.6	12.8	12.6
G	YR	8-2	4-27-56	7-7-58	4	51,450	512	1,105	0	17½x17½	2,106,792	2,473,040	117.4	.850	.811	95.4	8.1	10.2
G	R	5-3	6-15-56	5-28-58	4	50,400	329	1,085	0	15x17½	1,326,224	1,858,450	140.1	.750	.708	94.0	12.1	11.1
S	R	4-1	11-16-56	7-14-58	4	37,800	267	1,060	0	15x17½	808,209	892,360	110.4	.696	.660	94.5	8.5	9.9
S	C	3-2	4-4-57	4-5-58	4	36,750	273	1,063	0	15x17½	800,666	765,620	95.6	.690	.658	95.4	6.6	12.2
G	C	W7-4	5-23-57	7-30-58	4	15,750	454	1,018	38 - 62	15x17½	572,450	761,580	133.0	.840	.823	97.9	5.7	30.5
G	C	E7-5	6-28-57	7-9-58	4	18,900	264	1,047	18 - 48	15x17½	399,180	474,635	118.9	.750	.744	99.2	8.6	28.0
G	C	W9-4	8-9-57	7-18-58	4	15,750	412	1,160	25 - 75	15x17½	518,868	635,480	122.5	.790	.744	94.2	8.2	27.3
G	Y	E9-5	9-23-57	5-2-58	4	15,750	213	1,119	0 - 25	15x17½	267,935	307,660	114.8	.790	.722	100.6	6.5	22.9
S	C	4-2	12-6-57	8-10-59	3	37,800	345	1,060	1 - 67	15x17½	1,041,369	1,319,685	126.7	.888	.803	90.4	14.2	12.1
G	C	E10-2	1-30-58	12-31-58	4	22,050	534	1,145	46 - 67	17½x17½	941,437	1,065,550	113.2	.744	.664	86.6	7.5	36.3
G	C	5-1	2-8-58	7-15-59	2	37,800	359	1,002	67 - 121	15x17½	1,084,350	1,446,185	133.4	.933	.798	85.5	20.3	20.9
G	C	W10-1	2-22-58	5-31-59	4	25,200	493	1,093	0 - 55	15x17½	994,000	1,516,710	152.6	.623	.620	99.5	4.8	31.3
G	C	6-0	3-22-58	3-15-59	3	31,500	400	1,050	37 - 120	15x17½	1,008,000	1,120,410	111.2	.740	.705	95.3	10.6	27.2
G	C	W10-4	4-11-58	11-13-58	3	15,750	348	1,110	4 - 43	15x17½	438,900	535,645	122.0	.727	.730	100.4	8.4	39.8
G	C	W10-5	4-24-58	8-16-58	3	15,225	193	1,130	5	12½x17½ 15x17½	234,700	260,335	110.9	.725	.712	98.2	7.8	27.8
G	C	6-4	6-20-58	1-7-59	2	31,500	308	980	35 - 120	15x17½	775,600	725,730	93.6	.744	.708	95.2	11.0	30.2
G	C	11-3	7-9-58	7-15-59	4	25,200	484	1,164	14 - 82	15x15	976,600	247,825	127.8	.749	.703	93.9	8.2	26.8
G	C	7-2	9-20-58	6-16-59	3	25,200	410	939	155 - 190	15x17½	826,500	828,010	100.2	.834	.801	96.0	13.5	28.1
G	C	E7-4	10-7-58	8-8-59	2	15,750	464	935	155 - 185	15x17½	584,000	455,400	78.0	.855	.748	87.5	15.0	20.6
G	C	W7-5	10-28-58	6-9-59	2	18,900	252	982	85 - 124	15x17½	381,500	283,455	74.3	.744	.698	93.8	12.4	19.1
Total...........			1-24-56	8-10-59	-	684,075	393	1,072	95	-	21,515,408	25,907,650	120.4	.793	.748	94.3	10.7	19.0

[1] G = gravity.
S = slusher.

[2] T = timber.
R = rigid steel rings.
Y = yieldable arch.
C = concrete.
YR = yieldable arch and rigid steel rings.

Ventilation

The ventilation system was designed to circulate 400,000 c.f.m. of air through the mine. All of this air will not be required until production begins from the second level. Air enters the mine through the No. 1 and No. 4 shafts and is coursed through the 1475 main crosscut and the 1415 main ventilation crosscuts by four 5-foot axial vane mine fans, each powered by 200-hp. electric motors. Near the south end of the 1475 main crosscut the air passes through a raise to the 1415 level. Thus, all air enters the production area on the grizzly level. The air passes down through the transfer raise network to the 1475 level, where it is exhausted to the No. 3A and No. 3B shafts through the main haulage drifts.

Air distribution through the blocks is controlled by doors at the entrances to the fringe panel drifts. Additional doors needed to control airflow in case of fire are installed in drifts where needed.

The 1415 ventilation crosscut was lined with steel sets to gain greater cross-sectional area than could be obtained by using timber. The major ventilation passages were designed large enough so that smooth lining was not required.

One fan, similar to those on the 1415 level, has been installed on the 2015 and 2075 levels. On the two lower levels, air is introduced through the No. 1 shaft and exhausted through the No. 2 shaft to the 1415 level, where it enters the air circuit.

Each block that is being worked has one or two auxiliary fans on the grizzly level to exhaust contaminated air from the working areas through transfer raises to the haulage level. Fifteen axial vane fans, driven by 20-hp., 1,750-r.p.m. electric motors, are installed in the south ore body, and six axial vane fans, driven by 10-hp., 3,450-r.p.m. electric motors, are installed in the north ore body.

As of December 1959, 73 fans had been installed in the mine and 9 were awaiting installation.

Dust is controlled partly by adequate ventilation. In addition, about 2,000 water-spray nozzles are installed in the mine. Transfer raises have sprays directed into them from the grizzly drifts. Each slusher has a spray bar of six nozzles mounted in front of it. Foot-operated sprays between each drawpoint are controlled by the slusher operator.

Each chute on the haulage level is equipped with spray nozzles. Water spray is directed into the chute throat, and another nozzle mounted beneath the chute directs spray into the car being loaded.

Grizzly-fringe drifts are sprayed at the beginning of the shift and after blasts. Fringe-panel drifts are wet down with hoses during each shift.

The blasting time is rigidly controlled. Dust and powder smoke are vented from the working areas during the lunch period.

Every man at San Manuel is issued a free, approved respirator, which he is encouraged to wear. A replacement is available when necessary.

Mine Drainage

Considerable water was anticipated in the fractured rock underlying the conglomerate, and tentative plans were made to protect the mine against unexpected flooding. Pressure doors were installed off the No. 2 shaft station in the 1285 exploration drift. Water encountered in the No. 1 and No. 2 shafts was not excessive, nor did it have a high head. Plans for water doors were abandoned in favor of pilot diamond-drill holes in advance of exploration and development headings. In addition to giving advance warning of excessive water, pilot holes have partly drained the rock in advance of headings, thereby improving drifting conditions. Pilot holes continue to be used in drifts where water is expected.

No sealing off or grouting was done in the shafts, as it was essential that the ore body be drained before caving operations began.

All drifts are drained toward the No. 1 or No. 3 shafts.

Two main gathering sumps, with capacities of 85,000 gallons each, were constructed on the 1475 level at the No. 1 and No. 2 shafts. Each sump is equipped with two 18,000-gallon settling sumps to desand the mine water. Only one settling sump is in operation at a time; thus, time is provided for cleaning the other. The No. 1 shaft was equipped to pump 3,600 g.p.m. in two stages, and the No. 2 shaft was equipped to pump 2,400 g.p.m. in three stages. If it had been necessary, the capacity could have been doubled. At first, water collected in the No. 3 shaft area was pumped to the No. 1 gathering sump. Later, the system was modified to meet changing conditions caused by abandonment of the No. 2 shaft. A schematic diagram illustrates the current pumping practice (fig. 68).

Mine water is pumped to storage tanks at the reduction plant through an 18-inch, 3,000-g.p.m. pipeline, with the excess being discharged into an arroyo south of the subsidence area.

Table 13 contains data on pumping.

A valve is installed on the 1415 level at No. 1 shaft to turn mine discharge water into the drill waterline, thus providing adequate water and pressure in case of fire.

Potable Water

Potable water for use at the mine was pumped originally from the Mammoth-St. Anthony mine to storage tanks above the No. 1 shaft yard, a distance of about three-fourths of a mile. Continued pumping from the lower levels of the

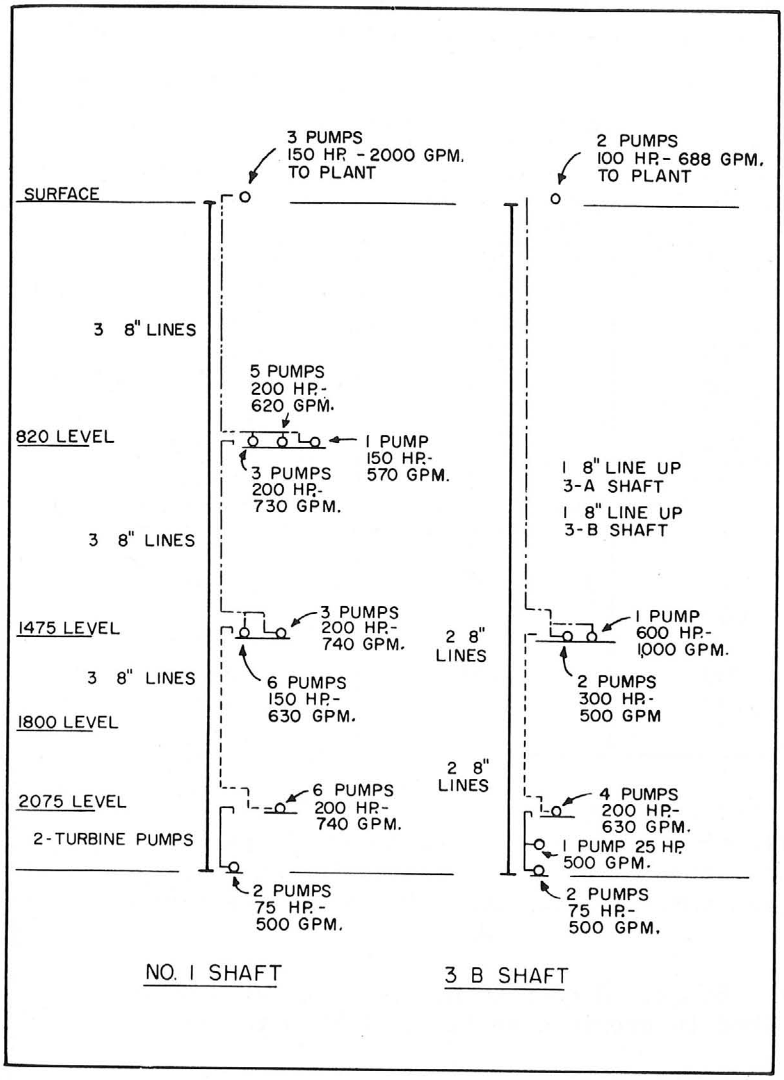

FIGURE 68. - Schematic Diagram of Current Pumping Practices.

San Manuel mine drained the Mammoth-St. Anthony mine, making it necessary to develop another source of drinking water. This was accomplished by horizontal diamond-drill holes, each a few hundred feet long, drilled from the loading station below the 2075 level in the No. 1 shaft. Sufficient water also was developed to be used for drilling, spraying at grizzlies, and wetting down drifts and working areas. The water is collected and chlorinated at the loading station and then pumped to a surface storage tank.

Control of Surface Drainage

The mine is in an area where flash floods result in heavy runoff. To protect the mine from this water, a large arroyo that coursed the subsiding area was dammed west of the mine, and a large canal was dug to divert surface drainage to a wash south of the subsiding area. Smaller washes that drain into the caving area have been diked to catch and evaporate as much water as possible.

Surface Installations

Most of the surface structures at the mine are on the common dump of the No. 1 and No. 4 shafts. They are steel-frame structures covered with sheet metal, except the mine office building, which is stucco covered. The dump also serves as storage yards for bulky supplies such as timber, rail, pipe, cables, and wire netting used with rock bolts. The surface plant is serviced with roads for delivery of supplies by trucks and with a railroad spur that extends from the twin hoisting shafts to a warehouse located between the collars of the No. 1 and No. 4 shafts.

TABLE 13. - Drainage, gallons per minute

1959	No. 1 shaft, water to mill	No. 1 shaft, potable water	No. 1 shaft, total
January........	3,051	400	3,451
February.......	3,032	400	3,432
March..........	2,846	350	3,196
April..........	2,799	400	3,199
May............	2,808	450	3,258
June...........	2,828	450	3,278
July...........	2,334	450	2,784
1949	**No. 3 pump, station to mill**	**Discharge, Mammoth wash**	**No. 3 pump station, total**
January........	--	450	450
February.......	--	540	540
March..........	1,000	--	1,000
April..........	450	300	750
May............	700	300	1,000
June...........	742	300	1,042
July...........	253	300	1,553

A system of tracks, with the same gage as used in underground transportation, extends from the collar of the No. 4 shaft to many of the buildings. A gasoline-driven engine pushes cars loaded with supplies to the collar of the shaft, where a small single-drum, air-operated hoist pulls the cars onto the cages.

Hoisting facilities at the 3A and 3B shafts and concrete batching and mixing plants have been described in previous sections of this report.

Mine Office

The building that houses the mine office consists of rooms built around a courtyard. The outside dimensions of the building are 85 by 105 feet. A covered walkway in the court provides access between rooms. The building contains offices of the superintendent, chief mining engineer, planning engineer, mine research engineer, mine geologist, mine surveyors, and tonnage and department clerks. The building also contains drafting rooms, a room for making blueprints, and a vault for storing maps and records.

Changehouse

The changehouse is 339 feet long by 60 feet wide and has two stories of changerooms and a one-story office section. It is situated southwest of the No. 4 shaft. The changerooms are equipped with lockers, overhead drying racks, toilets, and showers. The office section houses the mine time office, foremen's offices, safety department, first aid dispensary, and a lamproom with racks for storage and charging electric-cap lamps.

Machine Shop

The machine shop is 176 feet long by 60 feet wide. It is equipped for repair work and some fabrication. The equipment consists of two lathes, two drill presses (including one radial drill), a shaper, three bandsaws, a shear, a bending machine, and numerous smaller pieces of machinery. Air-cooled radiograph cutting and burning machines are used.

The machine shop operates on a three-shift basis. All reinforcing steel for underground use is fabricated in the shop.

Drill Repair Shop

A drill repair shop, 30 feet wide by 35 feet long, is adjacent to the machine shop. Major repairs that are not made in underground repair shops are made in this shop. Drills from the quarries are also repaired here.

Blacksmith Shop

The 30- by 60-foot blacksmith shop is adjacent to the drill repair shop. Picks, timber axes, and scaling bars are sharpened in the blacksmith shop. Drill steel for use with detachable throwaway bits in the mine and that for use with detachable tungsten carbide bits used with wagon drills at the quarries is made up here.

Truck Repair Shop

The 40- by 40-foot truck repair shop is used to maintain and repair the gasoline- and diesel-powered equipment. This equipment consists of trucks, bulldozers, graders, shovels, cranes, and forklift loaders. The equipment is lubricated and fueled on the job by a service truck.

Electrical Equipment Repair Shop

The electrical repair shop is 96 by 60 feet. Its equipment includes a 5-ton overhead hoist, a drill press, a coil winder, paper shears, a small lathe, a small powersaw, and a grinder. Repairs are made on electric motors, including locomotive motors but excluding the motors on ore hoists at the No. 3 shaft.

Carpenter and Paint Shops

The carpenter and paint shops are in a building, 40 by 60 feet, situated in the southwest part of the surface plant area at a considerable distance from many other buildings and the collars of the No. 1 and the No. 4 shafts. Underground concrete forms, ladders, signs, and other items of lumber are made in this shop.

Timber Treatment Plant and Framing Shed

The timber treatment plant and framing shed (fig. 69) are in the southwest part of the surface-plant area near the carpenter shop.

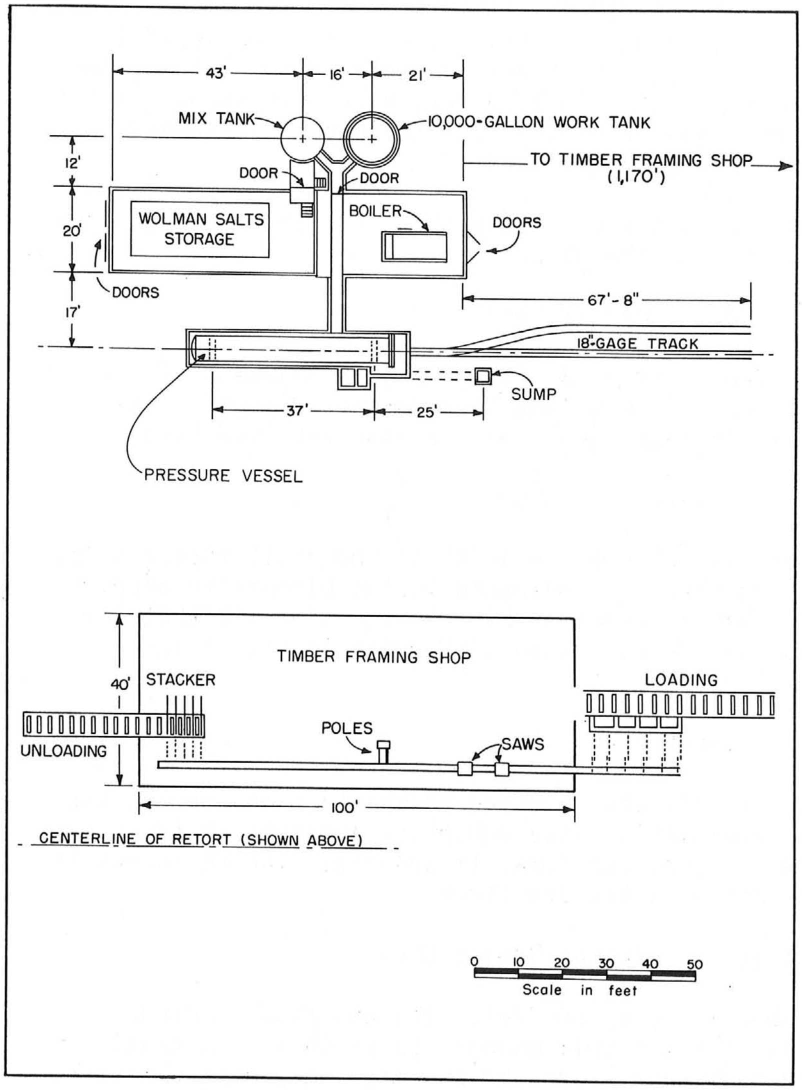

FIGURE 69. - Timber Treatment Plant and Framing Shed.

Timber for permanent installation in the mine is pressure-treated with a preservative composed of 35.6 percent sodium chromite, 23.8 percent sodium fluoride, 23.8 percent disodium hydrogen arsenate, 11.8 percent dinitrophenol, and 5 percent inert matter.

Timber is treated as follows: The timber is placed on cars, which are placed in a horizontal pressure vessel. This vessel is sealed and charged with a solution of preservative at 80° C. A pressure of 150 p.s.i. is maintained for about 6 hours. The solution is then drained, and the treated timber is removed. A 5,500-bd.-ft. charge uses about 700 to 800 gallons of solution; two shifts use about 1,500 gallons.

Timber is framed in a shop northeast of the treatment plant. The shop is equipped with a rip saw, a 30-inch-diameter hydraulically operated cutoff saw (fig. 70).

Compressor Building

It was estimated that about 10,000 cubic feet per minute of compressed air would be used when full production would be attained from the first level of the mine. It was anticipated that requirements for compressed air would increase somewhat when development of the second level began.

The compressor building, 48 by 162 feet, is close to the collar of the No. 1 shaft. The building houses seven compressors, motor-generator sets, a motor-control center, and a toolroom. It is equipped with an overhead crand with a 5-ton capacity. Data on compressors and driving motors follow:

Compressors

No. of units	Size, low pressure cylinders, high pressure cylinder, and stoke, inches	Capacity, c.f.m.	Pressure, p.s.i.g.
1	24 x 13 x 14	1,596	100
1	26 x 15 x 18	1,936	100
5	32 x $18\frac{1}{2}$ x 22	3,500	100

Motors

No. of units	Horse-power	Volts	Phase	Cycles	Revolutions per minute
1	300	2,300	3	60	277
1	350	2,300	3	60	200
5	600	2,300	3	60	200

Most of the compressed air enters the mine through a 12-inch line in the No. 1 shaft. This is supplemented by a 6-inch line in the No. 4 shaft and two 6-inch lines in the ore-hoisting shafts. These two 6-inch lines are fed by an 8-inch line from the compressor plant.

Warehouse

The warehouse, 60 by 120 feet, is in the central part of the mine plant area. It contains a complete inventory of supplies and repair parts that are withdrawn by requisition. Electrical supplies, such as wire, cable, transformers, and powerline fixtures, are stored in a fenced yard near the warehouse.

Standby Powerplant

The standby powerplant is in the extreme northern part of the plant area at some distance from other buildings. It houses diesel-driven generators with a rated output of 5,500 kv.-a. Water for cooling the equipment is treated in a water softener installed in the building.

Explosives Storage Magazines

Explosives are stored in a magazine built on an isolated hill about one-half mile from the mine storage area. The site is enclosed by a high-wire net fence. The walls and roof of the magazine (fig. 71) are built of concrete blocks. The holes in the blocks and the spaces between them are filled with sand. The door is made of laminated wood covered with steelplate. The magazine has a capacity of 84 tons (3,360 cases of 50 pounds each). A 24-hour reserve supply of dynamite is dispatched once a day to magazines on each grizzly and haulage level.

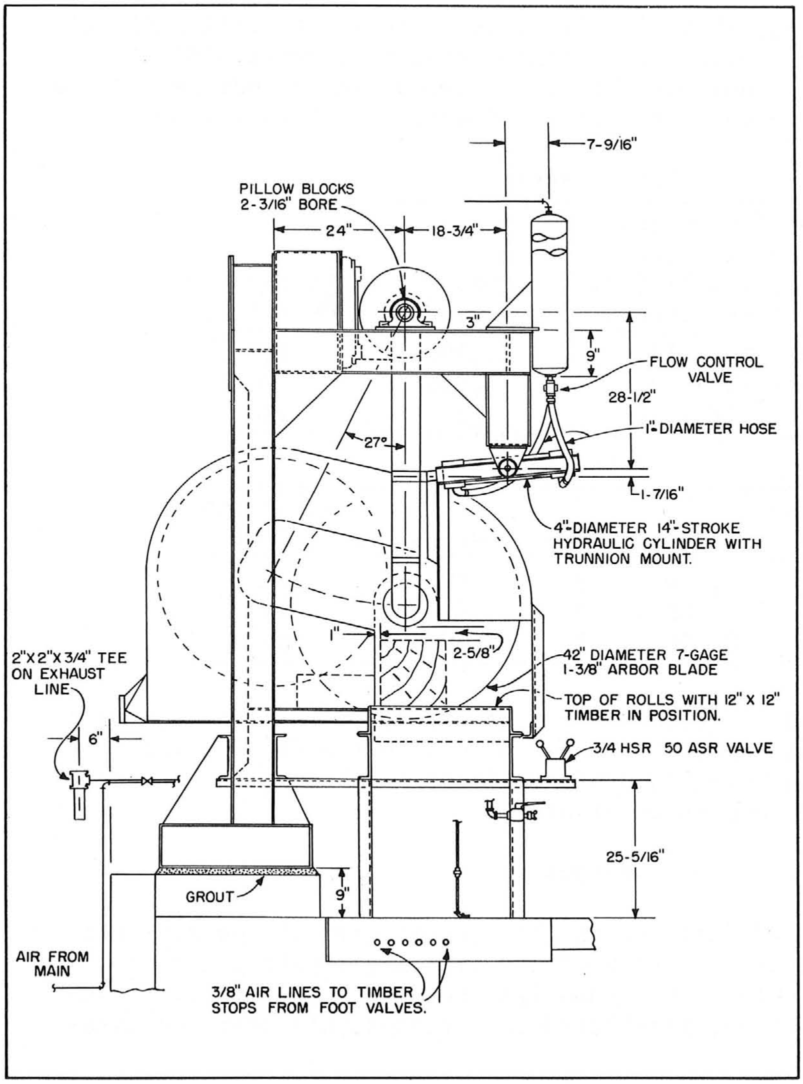

FIGURE 70. - Hydraulically Operated Saw.

Ammogel and 40-percent straight gelatin explosives are used underground. Bag powder, for use at the lime quarry, is stored in the main powder magazine and is transported to the quarry as needed.

Detonators for use with fuse, standard- and millisecond-delay electric caps, and detonating fuse are used. The detonators are stored on the surface in a magazine beneath the dump of the No. 4 shaft, and a day's supply is stored in magazines on each level. Detonating fuses are used in blasting hung transfer raises and in blasting at the quarries.

Supplies and Materials Handling

A large quantity of supplies and materials must be procured and distributed for the 33,000-ton-per-day mining operation. Approximately 50 carloads of timber per month have been used so far. Four thousand cubic yards of concrete are placed per month. About 8,000 sacks of cement per month are used for grouting. Anticipated materials for the next 2 years include 2,200 tons of steel, exclusive of special jobs; 40 miles of pipe; and 12 miles of track.

One supply train is sent underground to each level during every shift.

Two 5-ton and one $7\frac{1}{2}$-ton forklifts are used for loading materials and supplies onto flat-rack mine cars. Specially built cars for transporting dynamite and primers are bolted to mine car chassis. Materials over 12 feet in length are slung beneath the cages for transporting to the working levels.

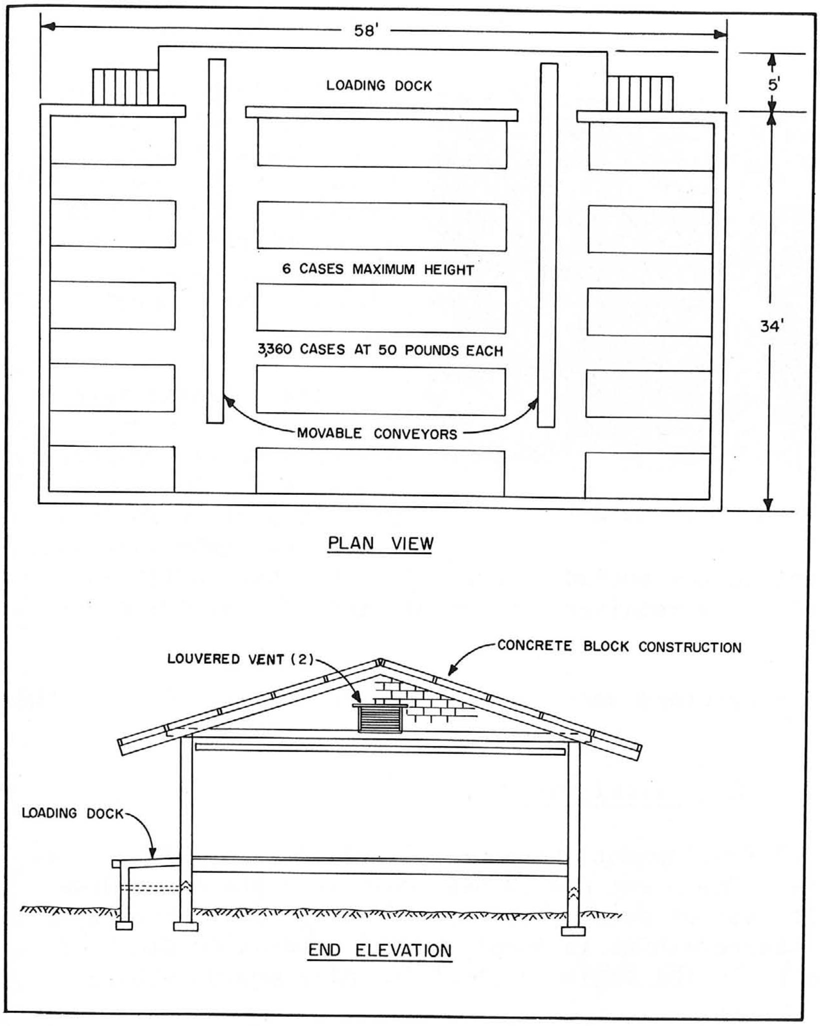

FIGURE 71. - Sketch of Explosives Magazine.

A toolroom on each level serves as a warehouse for supplies and parts. Rail and pipe are stored alongside the track in the heading where they are to be used.

Timber

Timber is purchased directly from mills in Oregon and Washington, and is delivered to the mine in railroad cars. Timber used in undercutting and in temporary repair work is untreated, but timber for permanent or semipermanent installation is treated at the mine.

Except for large posts and caps, much of the timber is purchased in a standard cuts and baled for underground transportation at the lumber mill or mine (fig. 72). Where possible, bales are transferred to mine cars and transported directly to the heading where they are to be used.

Steel

Steel is purchased in standard sizes. The machine shop is equipped to fabricate reinforcing, and other types of steel. However, some fabrication is contracted to private firms. The work force for steel fabrication is kept constant, and work is contracted outside when required. Much of the new shaft and station steel fabrication is contracted. Drill steel is made up in the blacksmith shop.

Track and Pipe

Rail and pipe are slung under cages for transporting underground. Rail in 90-, 70-, and 45-pound sizes is used.

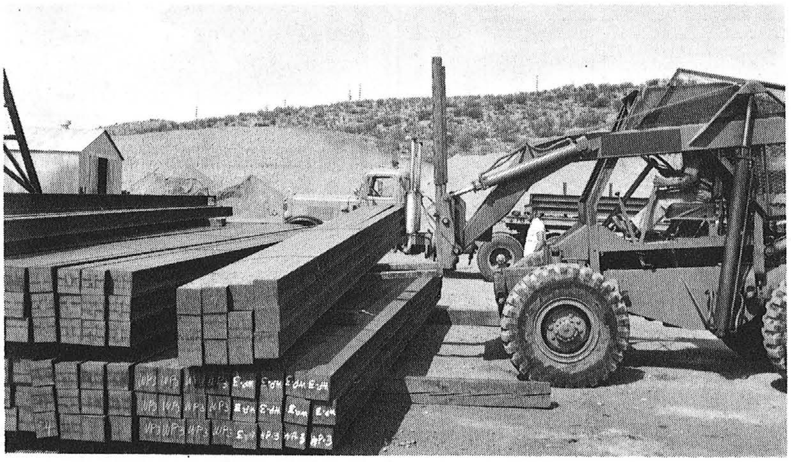

FIGURE 72. - Forklift Moving Baled Timber.

Salvage

A salvage crew removes rail, electrical fittings, pipe and fittings, and the angle iron on raise cribbing from mined-out blocks. Only usable items are recovered. No timber is salvaged.

The salvaged material is sorted underground into two grades. The material in good repair is sent to working places for immediate use. The rejected material is sent to the surface, where it is further sorted as scrap or repairable material. The repairable items are sent to various shops to be reconditioned.

All salvage work is done by crews working on a contract basis. Considerable material is recovered.

Electrical Power

In the initial stages of development, power was supplied by a 6,875-kv.-a. generating plant at the mine. The plant now is used only as a standby source of power to operate pumps in case of emergency. About two-thirds of the power consumed is purchased, and the remainder is supplied by a generating plant at the smelter. It was estimated in the beginning that the mine would require about 30,000 kv.-a.

Substations, with a combined capacity of 45,000 kv.-a., were installed at the power house and compressor house and at each shaft, where the power is transformed from 46,000 to 2,400 volts for underground distribution. Most of the power used underground is transformed from 2,400 volts to 440 or 120 volts alternating current or converted to 250 volts direct current, depending on its use. The main substations underground are located at pump installations near the shafts, at three points around the haulage system for convering trolley-line voltage, and at other locations where power is needed for fans, slushers, and so forth.

Wages and Bonuses

Manpower Distribution

At the San Manuel mine, more than 900 men work underground on development, stope preparation, ore extraction, haulage, and so forth. Approximately 450 additional men work on the surface and at underground supporting operations.

Table 14 gives the distribution of employees by work categories. Figures 73 and 74 are organization charts of the staff.

TABLE 14. - Manpower distribution (average for June 1959)

Work category	Men per shift			
	Shift A	Shift B	Shift C	Total
Mine Surface:				
Surface construction	12	--	--	12
Surface maintenance	65	28	10	103
Lime and silica flux pits	6	--	--	6
Total mine surface department	83	28	10	121
Mine Electrical:				
Electrical construction underground	13	--	--	13
Electrical construction surface	2	2	--	4
Electrical maintenance	68	13	15	96
Total mine electrical department	83	15	15	113
Mine Mechanical:				
Mechanical construction underground	16	--	2	18
Mechanical construction surface	13	5	5	23
Mechanical maintenance	98	21	15	134
Operating - compressor house	2	1	1	4
Operating - pumping and hoisting	14	10	9	33
Total mine mechanical department	143	37	32	212
Mine Underground (1st level):				
1415 level development	4	2	2	8
1475 level development	8	8	8	24
Stope preparation	52	53	33	138
Drawing	37	39	40	116
Hauling	42	41	41	124
Hoisting	4	4	4	12
1415 level repair[1]	20	15	15	50
1475 level repair[1]	38	32	30	100
Block repair	4	6	5	15
Construction	2	--	--	2
Salvage	--	--	2	2
Track	30	7	6	43
Service, supply, and miscellaneous	36	21	20	77
Total	277	228	206	711
Mine Underground (2d level):				
Shafts	10	11	11	32
2015 level development	7	9	7	23
2075 level development	27	22	22	71
Track	7	7	--	14
Concreting	4	--	--	4
Forming	2	2	2	6
Service, supply, and miscellaneous	14	15	12	41
Total	71	66	54	191
Total first and second levels	348	294	260	902

[1] Concreting and grouting practices have resulted in an approximate decrease of 30 percent in the repair force on the 1415 and 1475 levels.

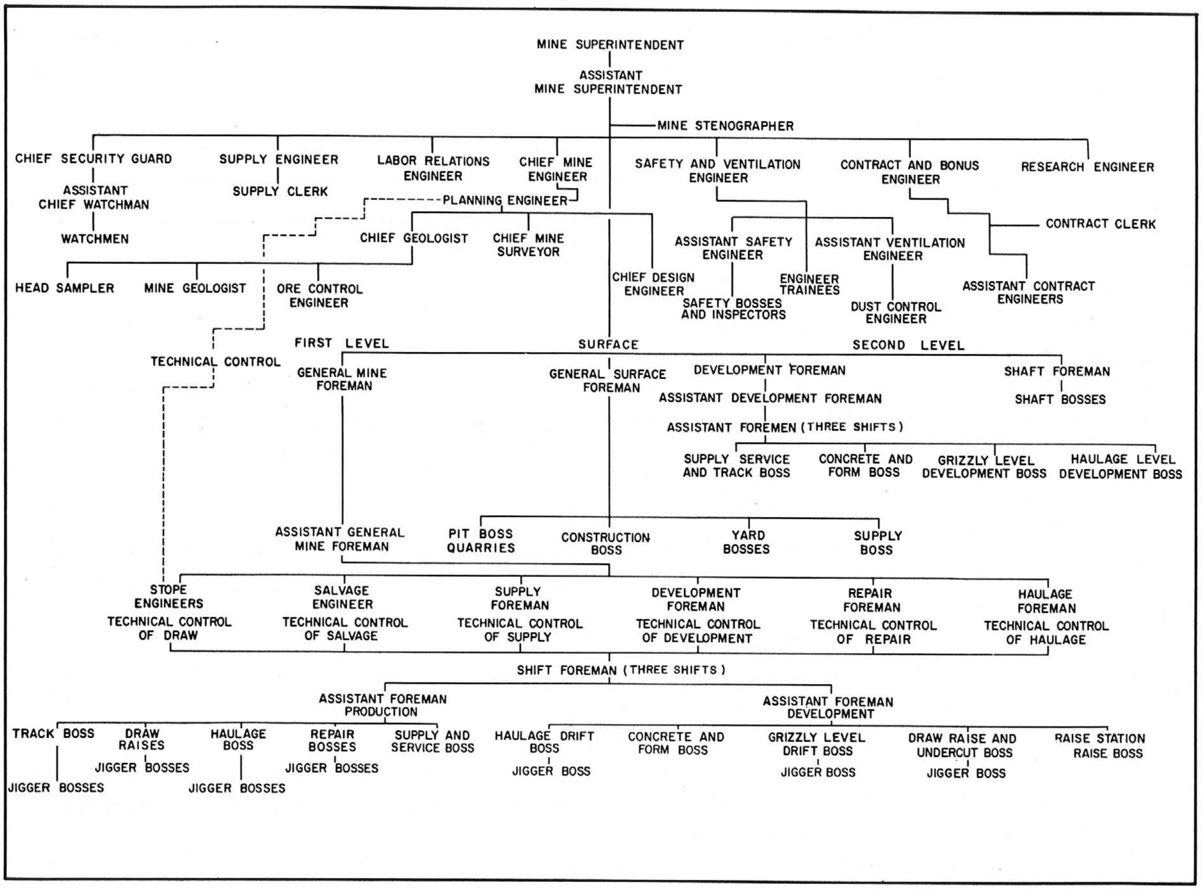

FIGURE 73. - Organization Chart.

Wages

Members of the staff are paid salaries, and other employees are paid day wages. Employees doing certain types of underground work are paid an incentive bonus in addition to guaranteed wages. Pay rates are negotiated between members of the company staff and representatives of labor unions.

Incentive Contract Bonus System

Approximately 45 percent of all underground employees are on the incentive contract bonus system and have wages guaranteed. The system is applied to all development work and to some maintenance. It includes shafts, drifts, raises, stope preparation, timber and concrete repair, concrete forming and pouring, and salvage of materials. No incentive bonuses are paid in the production phase.

Three engineers in the contract department set the rates of the incentive bonuses with the approval of management. Unit rates are posted before work starts on any contract. Measurements are made once a week, and earnings are computed by the contract and accounting departments.

This incentive plan is considered instrumental in maintaining high work efficiencies.

Safety

San Manuel has an effective safety program administered by a safety engineer, two assistant safety engineers, a fire marshal, and two mine shift bosses who act as safety inspectors on 2-month tours. The safety department include fire protection, engineering, maintenance of emergency equipment, first aid supplies, and rescue apparatus.

The introduction of new safe procedures and equipment and the issuing or revising of rules is done by consultation and concurrence with the heads of the department affected by the changes.

New employees spend their first half day under the supervision of the safety department. Following a lecture, they are given copies of current procedures covering their job assignments. Then, after a short tour of the mine, they report to the shift foreman underground.

Supervisory safety meetings are held twice weekly. These meetings are held after each shift and are attended by all supervisors who are currently on the day shift. The subject matter, prepared well in advance, includes material the supervisors will present to their men in the employees' safety meetings.

Employees' safety meetings are conducted monthly by the supervisors. At other times, safety films are shown in the underground lunchrooms after lunch. Employees are encouraged to offer practical suggestions at these meetings. If found to be practical, the suggestions are forwarded to the safety department for consideration and processing. Written answers to these suggestions, and suggestions turned in from suggestion boxes, are returned to the employee through his boss.

A monthly report is distributed to all foremen. This report shows the frequency and severity of accidents, each supervisor's record, and comparisons with State and Federal Bureau of Mines accident rates for underground mining.

Mine rescue training classes are conducted twice monthly. Twelve 2-hour oxygen breathing apparatus and five Chemox units are maintained in the safety office, and eight Chemox units are installed in various parts of the mine underground. Six mine firetrucks, complete with tools, fittings, bentonite, and so forth, are located on car transfer stations at strategic locations underground. An ethyl mercaptan mine fire stench warning system is installed in the compressed air system. The men are shown fire exits from their working places on a regular schedule.

An emergency procedure has been set up, and supervisory personnel have been assigned duty stations in case of emergency.

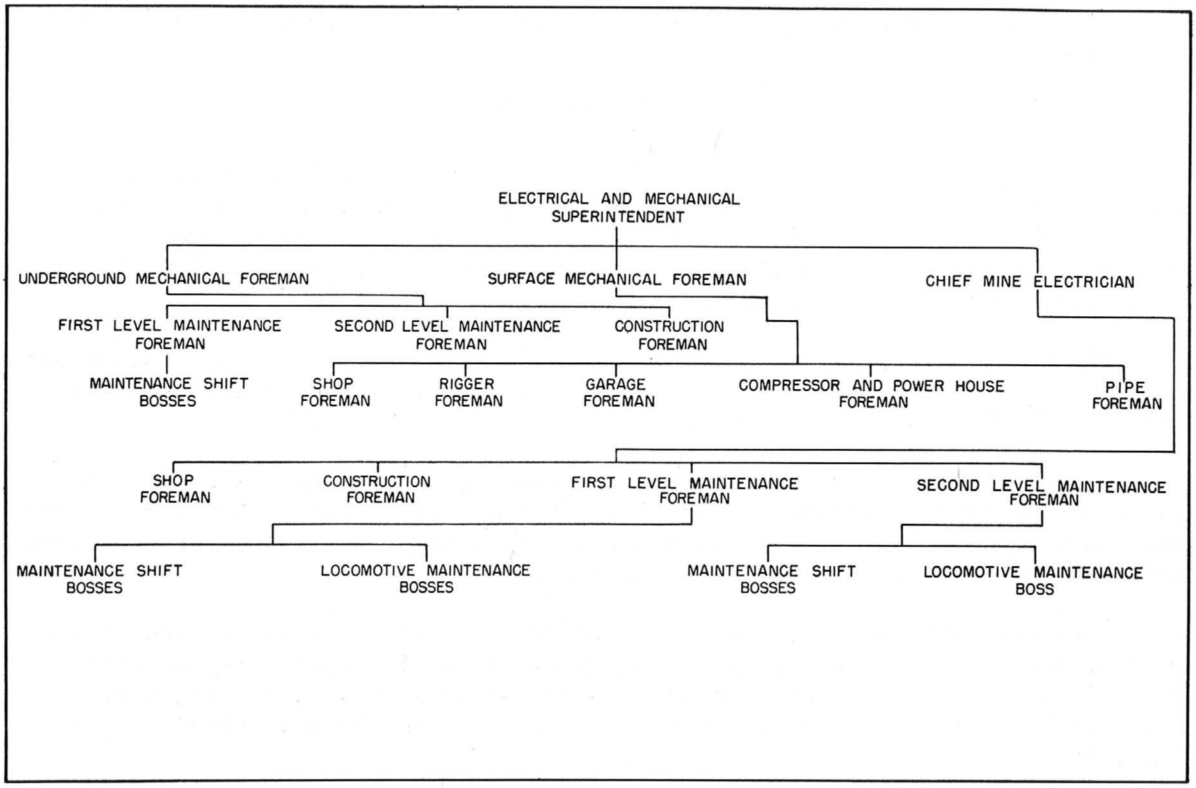

FIGURE 74. - Organization Chart of Mechanical Department.

San Manuel Copper Corp. is a member of the Globe-Miami Mine Rescue and First Aid Association. In the autumn of 1917, an idea was conceived for combining the resources and equipment of various mines in the Globe-Miami mining district in order to render greater and more effective service in mine rescue and first aid.[15] Since its organization, the Association has provided member mines with a source of competent, well-equipped rescue teams.

Employees' safety records are kept current and are made available to supervisors. Pictures and diagrams are used to provide detailed instructions on new safety procedures. Safety cartoons drawn by an underground employee are used to show how accidents occurred. Lights over the changeroom door show the day of the month. A green light signifies an accident-free day, and a red light shows days on which accidents occurred. Daily safety inspections cover most of the operations in progress. The safety inspectors' reports and recommendations are processed through the immediate foreman.

Dust surveys are made once a month by the ventilation department. Every 3 or 4 months, the State Dust Engineer makes a survey which requires about a week.

[15] Van Fleet, L. A., and Look, Allen D., Central Mine Rescue Station, Globe-Miami District, Mine Rescue and First Aid Association, Globe, Ariz. Bureau of Mines Inf. Circ. 7577, 1950, 20 pp.

Adequate measures have been taken to initially control fires in the mine. Airflow to the caving panels is controlled by steel fire doors. The air may be stopped, reduced, or diverted in all panels. This control may be used to stop air supply to a fire area and to maintain a fresh-air supply to control operations near a fire area. Airflow at the shaft collars may be shut off if necessary.

The pumps in shaft No. 1 may be connected into the mine water distribution system. The main water system has firehose connections every 300 feet and hose stations every 1,000 feet.

There are four underground firefighting trucks equipped with hoses, nozzles, ladders, and miscellaneous handtools for firefighting (fig. 75).

FIGURE 75. - Equipment Carried in Underground Firefighting Cars.

All electrical equipment installations are equipped with carbon dioxide extinguishers. Maintenance shops and pump stations are concreted or fireproofed with gunite or transite board.

Water sprays with automatic controls are installed in every toolroom in the mine and in the locomotive repair shop. Manually controlled water sprays have been installed at the shaft stations.

LIMESTONE AND QUARTZITE QUARRIES

Limestone and quartzite, used in the concentrator and smelter, are quarried approximately 14 miles north of the reduction plant. The pits are about one-half mile apart, and each is approximately three-fourths of a mile from a loading ramp on a spur from the San Manuel Arizona Railroad.

The limestone contains about 50 percent CaO, 3.5 percent MgO, and 3.5 percent insoluble. The quartzite consists of 95 percent SiO_2, 1.5 percent Al_2O_3, and 3.5 percent unclassified material. The average monthly production for the first 7 months of 1959 was 5,453 and 1,284 tons of limestone and quartzite, respectively.

The limestone is mined from beds of Martin limestone of Devonian age. The beds strike northwest and dip approximately 35° northeast. Their maximum thickness is 60 feet.

At present limestone is mined from five benches; each is 20 feet high. Blastholes are drilled with two wagon drills, supplied with compressed air by two air compressors rated at 300 c.f.m. each. The limestone is blasted with bag powder. The maximum size of the fragments is 2 feet. The broken rock is loaded into 10-ton-capacity end-dump trucks by a track-mounted, diesel-powered shovel with a $1\frac{1}{2}$-cubic-foot capacity. The stripping ratio of overburden to lime rock is 1 to 2.5.

Quartzite is scraped from talus slopes with a bulldozer. The rock is loaded and hauled with the same equipment as that used in the lime quarry.

The quarries operate on a one-shift basis under supervision of members of the mine staff. The crew consists of a foreman, mechanic, shovel operator, bulldozer operator, two truckdrivers, and four drill operators.

RAILROADS

Ore is hauled from bins at the twin hoisting shafts to the crushing plant over a company-owned, standard-gage railroad. This branch is approximately 7 miles long and has curves of large radii and no appreciable grades. The equipment consists of three 130-ton diesel-electric locomotives, forty-five 100-ton bottom-dump ore cars, and twenty 30-yard lime rock cars. Supplies for the mine, except cement aggregates and structural reinforcing steel, are transported on this track from the terminus of the San Manuel Arizona Railroad at the town of San Manuel.

Supplies for the mine, mill, and smelter are hauled from Hayden, Ariz., a station on the branch line of the Southern Pacific lines, to San Manuel by the San Manuel Arizona Railroad Co. The railroad branch is 29.5 miles long and is downgrade from San Manuel. Silica and limestone are hauled by this railroad from quarries in the San Pedro Valley to bins near the concentrator. Copper anodes are hauled from the smelter over the same route to Hayden, where they are sent to refineries.

According to the annual report to the stockholders of 1958, that year the San Manuel Arizona Railroad Co. hauled 114,107 tons of inbound freight, including 68,904 tons of limestone and quartzite, from San Manuel's quarries and 76,521 tons of outbound freight, compared with 104,437 tons of outbound freight in 1957.

SAN MANUEL TOWNSITE

San Manuel townsite (fig. 76) was built for those who work for the San Manuel Copper Corp. and for others in supporting activities such as merchants, doctors, police officers, clergymen, and so forth. The town is about 7 miles southeast of the mine. The first residences were completed in December 1953, and there are now 1,050 housing units in the town.

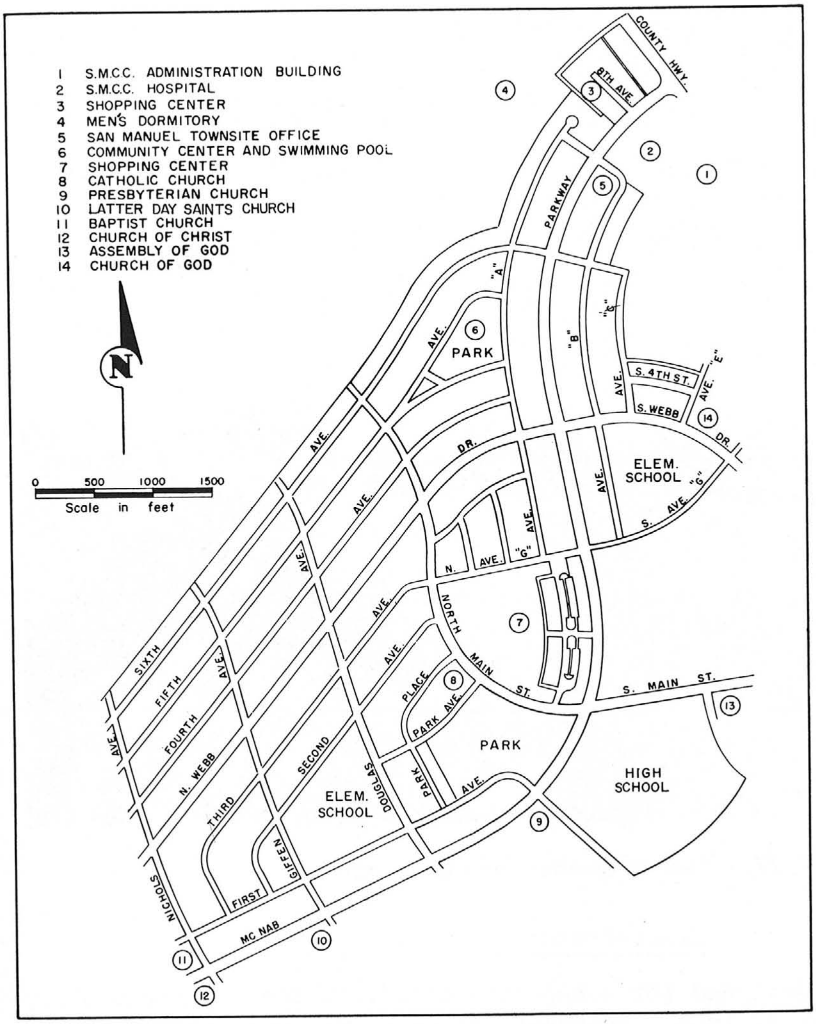

FIGURE 76. - San Manuel Townsite.

San Manuel Copper Corp. donated a 40-acre plot for the site of a high school and two 15-acre plots for sites for elementary schools. School buildings and equipment cost $870,785.

San Manuel hospital is a half-million dollar institution which features the latest equipment and the ultimate in construction design. It includes a surgical wing, a 30-bed patient wing with two isolation rooms, and two smaller wings. The surgical wing includes a completely equipped operating room, two obstetrical rooms, a nursery, an emergency room, an X-ray and X-ray developing room, and a reading room. The hospital is staffed with a chief surgeon, supervisor, supervisor of nurses, four doctors, eight nurses, and four aides.

San Manuel has two shopping centers, a park, a swimming pool, and a community center. There are 38 business establishments in San Manuel, all operated by private enterprise. Thirty of the buildings are leased from the San Manuel Townsite Co., and eight are privately owned.

Natural gas and electricity are distributed by the Arizona Public Service, a public utilities corporation. Water is supplied by the Arizona Water Co., a privately owned enterprise.

REDUCTION PLANT

The concentrator and smelter (fig. 77) are situated near the town of San Manuel, approximately 7 miles southeast of the mine. A lime and flux plant and machine shop are in the immediate vicinity of the concentrator. A tailing storage damsite is east of the concentrator and downslope toward the San Pedro River.

FIGURE 77. - View of Smelter and Concentrator.

Concentrator

The concentrator was designed for a capacity of 33,000 tons of ore per day. Operations began on October 4, 1955. No major changes have been made in the equipment, and the metallurgy has been changed only slightly as experimentation has dictated. The flowsheet of the concentrator is illustrated by figure 78 and its legend (which follows the illustration). Information has been abstracted from a paper by the late E. V. Given,[16] mill superintendent.

Table 15 is illustrative of the metallurgical data obtained at the San Manuel concentrator.

[16] Given, E. V., Milling Methods at the Concentrator of the San Manuel Copper Corp.: Proc. Arizona Section Meeting, AIME, San Manuel, Ariz., May 1958, 45 pp.

Crushing

During unloading, ore is dumped into a coarse-ore bin at the rate of 4 cars (simultaneously) every 3 minutes. The bin, which has a capacity of 10,000 tons, is a flat-bottomed structure supporting the double-track railroad bridge (fig. 79) and surmounting panfeeders and a conveyor system. The maximum size of ore delivered from the mine is limited to that which will pass through a 12-inch grizzly system at the mine. The ore contains approximately 6.2 percent moisture, and the angle of repose in the bin is between 50° and 60°. The angle of repose approaches 80° or more when ore containing as much as 8.5 percent moisture is received. A crew of bin blasters reduces the angle when necessary to help maintain a suitable live load.

Primary crushing is divided into two sections; each includes four panfeeders, four 48-inch and one 72-inch conveyors, one vibrating grizzly, and one 7-foot standard cone crusher (fig. 80). Withdrawal of ore from storage to the conveyor is regulated by the feed of the feeders. To protect the cone crushers against tramp iron, each conveyor system is equipped with an iron detector and a large fixed magnet suspended over the ore stream at the head pulleys of the conveyor feeding the 72-inch picking conveyor. Bar grizzlies with $2\frac{1}{2}$-inch spacings remove undersize from the cone feed.

Primary cones are equipped with bowls and mantles suitable for coarse crushing. They operate with a $2\frac{1}{2}$-inch setting on the closed side and yield a product which is less than 3 inches. The combined cone discharge and grizzly undersize from both crushing sections constitutes the final product of this crushing stage.

The product from the cone crusher is advanced by conveyors to a four-compartment steel surge bin with an 800-ton capacity. Its pyramidal bottom is above and adjacent to the secondary crushers.

The secondary crushing installation consists of four sections; each employs a belt feeder, a vibrating screen, and a 7-foot short head cone crusher in open circuit. The screens are automatically fed by ore drawn from the surge bin onto belt feeders. Usually screen cloth with 11/16- by 6-inch openings is used and is mounted with the shortest dimension of the openings normal to the fall of ore. The screens are set at an inclination of 25°, and screening action is effective on ore containing less than 7 percent moisture. Screen oversize is fed to short head cone crushers, which comprise the final crushing stage. The cones operate with 5/8-inch setting on the closed side and yield a product minus $1\frac{1}{4}$ inch (nominal). Cone discharges join the screen undersize and are routed by conveyors to the fine ore storage bin; they pass over a weightometer in transit. Typical screen analyses of feed and product are given in table 16.

The crushing plant operates on a 24-hour basis. Most of the crushing is done on afternoon and graveyard shifts. This time is adequate to permit the production of 36,000 tons of millfeed of desirable size. The day shift is thus available for making essential repairs and replacements and for crushing some ore.

An oil-circulating system is conveniently located to provide lubricating oil for the cones. This system includes a centrifuge for cleaning oil at periodic intervals. The cone crushers are equipped with water seals to prevent entry of dust into the lubrication system.

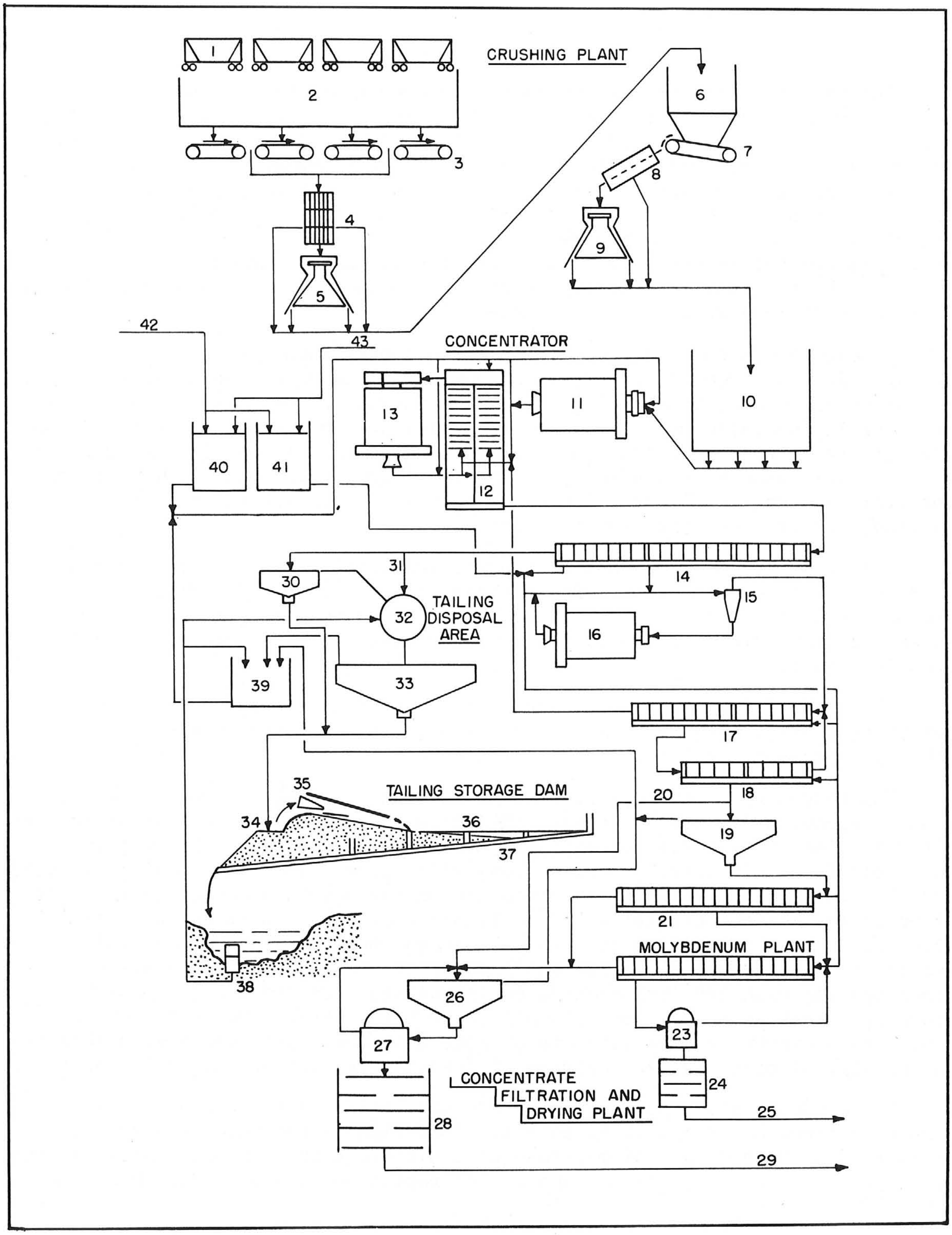

FIGURE 78. - Flowsheet of Concentrator.

Legend - San Manuel Concentrator Flowsheet.

1. Bottom dumpcars from mine (71 cubic yards - 100-ton capacity).
2. Coarse-ore bin (10,000-ton capacity).
3. Eight panfeeders (4 feet wide).
4. Two vibrating grizzlies ($2\frac{1}{2}$-inch opening).
5. Two 7-foot standard cone crushers ($2\frac{1}{2}$-inch discharge openings).
6. Surge bins.
7. Four belt feeders (variable speed).
8. Four vibrating screens (11/16- by 6-inch openings).
9. Four 7-foot shorthead cone crushers (5/8-inch discharge openings).
10. Fine-ore bin (45,000-ton capacity).
11. Eight 10- by 13-foot rod mills.
12. Sixteen rake-type classifiers (16- by 35-foot).
13. Sixteen 10- by 10-foot ball mills (grate discharge).
14. Four rougher flotation sections (120 cells per section).
15. Four regrind cone classifier sections (five 12-inch diameter cones per section).
16. Four 8- by 12-foot regrind mills (trommel screen discharge).
17. Four primary cleaner flotation sections (28 cells per section).
18. Four final cleaner flotation sections (8 cells per section).
19. One concentrate thickener (100-foot diameter).
20. Alternate flow for final concentrate.
21. One molybdenum rougher flotation section (16 cells).
22. Eight-stage molybdenum cleaner section (54 cells).
23. One molybdenum filter (2 disks, 4-foot diameter).
24. Molybdenum concentrate dryer (4 hearths, 4-foot diameter).
25. Drums of molybdenum concentrate to storage.
26. Concentrate thickener (100-foot diameter).
27. Three copper concentrate filters (8 disks, 6-foot diameter).
28. Two copper concentrate dryers (4 hearths, $23\frac{1}{2}$-foot diameter).
29. Copper concentrate to smelter by conveyor.
30. Hydroseparator (55-foot diameter).
31. Alternate flow for tailings.
32. Distributor box.
33. Three tailing thickeners (325-foot diameter).
34. Transite pipe for tailing distribution to the dams.
35. Cyclones for berm building and tailing distribution at dams.
36. Four tailing dams (300 acres storage area).
37. Dewater Ducts.
38. Two settling ponds.
39. Two mill water tanks (100,000-gallon capacity each).
40. Reclaimed water storage head tank (1-million-gallon capacity).
41. Fresh water storage head tank (1-million-gallon capacity).
42. Fresh water from domestic storage.
43. Mine water.

TABLE 15. - Summary of metallurgical and operating data for the period from January 1 to March 31, 1958, inclusive

Total tons of dry ore treated	2,643,686
Percentage of moisture	6.2
Dry tons per operating day	30,134
Dry tons per grinding section per 24 hours	3,767
Solids percent:	
Rod-mill discharge	66.2
Ball-mill discharge	69.9
Classifier overflow	23.3
Tailing to dams	37.7
Screen analysis, percent:	
Rod-mill feed plus 1-inch	5.17
Rod-mill feed minus ½-inch	59.26
Rod-mill discharge plus 10-mesh	16.58
Rod-mill discharge minus 10- plus 65-mesh	48.44
Regrind product minus 325-mesh	91.40
Final tailing plus 65-mesh	6.00
Final tailing minus 200-mesh	65.84
Assays percent[1]:	
Feed:	
Total copper	.762
Oxide copper	.046
Molybdenite	.018
Gold, ounces per ton	.0022
Silver, ounces per ton	.033
Iron	3.6
Sulfur	2.1
Copper concentrate:	
Copper	27.98
Molybdenite	.60
Gold, ounces per ton	.074
Silver, ounces per ton	.99
Insoluble	8.8
Iron	27.4
Moisture	
Filter cake	14.4
Dryer discharge	9.7
Tailing:	
Total copper	.107
Oxide copper	.046
Molybdenite	.004
Ratio of concentration, copper	42.55
Extraction, percent:	
Total copper	86.26
Sulfide copper	91.68
Molybdenite	78.80
Gold	77.25

[1] Except gold and silver.

TABLE 15. - Summary of metallurgical and operating data for the period from January 1, to March 31, 1958, inclusive (Con.)

Silver	70.59
Iron	17.90
Power, kilowatt-hours:	
Crushing	1.07
Grinding	10.58
Flotation	2.34
Regrind	.97
Molybdenum plant	.19
Concentrate dewatering	.14
Lime and flux plant	.09
Tailing disposal and reclaimed water	1.41
Miscellaneous	1.30
Total	18.09
Steel consumption, pounds per ton of ore:	
Crushing	.010
Grinding:	
Rod-mill liners	.053
Rods	.580
Ball-mill liners	.035
Balls	1.292
Reagents, pounds per ton of ore:	
Collectors:	
Minerec "A"	.031
Xanthate Z-11	.010
Stove oil	.018
Frother, methyl isobutyl carbinol	.049
Lime	2.791
Labor:	
Man-shifts per day	224
Dry tons treated per man-shift	134.53
Safety:	
"Lost time" accidents	3
"No lost time" accidents	16

A crane and craneway ample to handle any crusher part are available as required to facilitate repair operations in a repair bay or at the crusher stand.

Dust-control facilities have been provided at three principal locations; a wet collection method is used. All conveyor loading and transfer points, crushers, and feeders are hooded or housed and connected with adequate dust systems at various collectors. Small dust-control units are located in remote sections of the plant where they are needed.

Nineteen men per shift, including a shift boss, maintain the operation and do most of the cleanup. A central panel for the plant is operated by one lead

FIGURE 79. - Train of Ore at Coarse Ore Bin.

operator and a suboperator with the aid of a public address system, telephone, and a signal-horn system connected to each phase of the operation.

Grinding

The fine-ore storage bin is designed to receive the crushing plant product from a conveying system that includes a tripper conveyor unit that spreads the feed in the bin. The flat-bottomed steel bin extends the length of the grinding department and has a rated capacity of 45,000 tons.

The grinding department (figs. 81 and 82) is divided into eight sections; each has a nominal rating of 4,200 dry tons a day. They yield a product for flotation feed of which about 6 percent remains on 65-mesh. Two-stage grinding is employed; one rod mill in open circuit on a primary grind is used and the pulp is discharged into two classifiers in a closed circuit with two ball mills for the secondary grinding.

Feed to the rod mill is drawn from the bin by two or more of four feeders and is discharged into a collecting belt. Two of the feeders are drawn by variable-speed drives. Ore on the collector belt is transferred to a conveyor that feeds the rod mills. This conveyor has a weightometer that records the tonnage and regulates the feeders at a predetermined tonnage rate.

The rod mill feed varies in maximum size from 1.25 to 1.5 inch, and approximately 5 to 6 percent is retained on a 1-inch screen. Screen tests made on shift composites dictate changes to be made at the crushing plant to control any excess in the amount of oversize.

The rod mill discharge is a suitable product for feed to the secondary grinding circuit, as primary slimes are removed by rake-type classifiers and clean sand results for ball mill feed. Rod mill and ball mill discharge densities are maintained from 65 to 68 and 68 to 72 percent solids respectively. The classifier overflow is between 22 and 24 percent solids.

Operating requirements, such as tonnage, densities, pH control, and so forth, are conformed to by operators; thus, erratic changes in operation are minimized. Metallurgical results indicate that these conditions are beneficial throughout the plant. Some reagents are added in the grinding circuit to

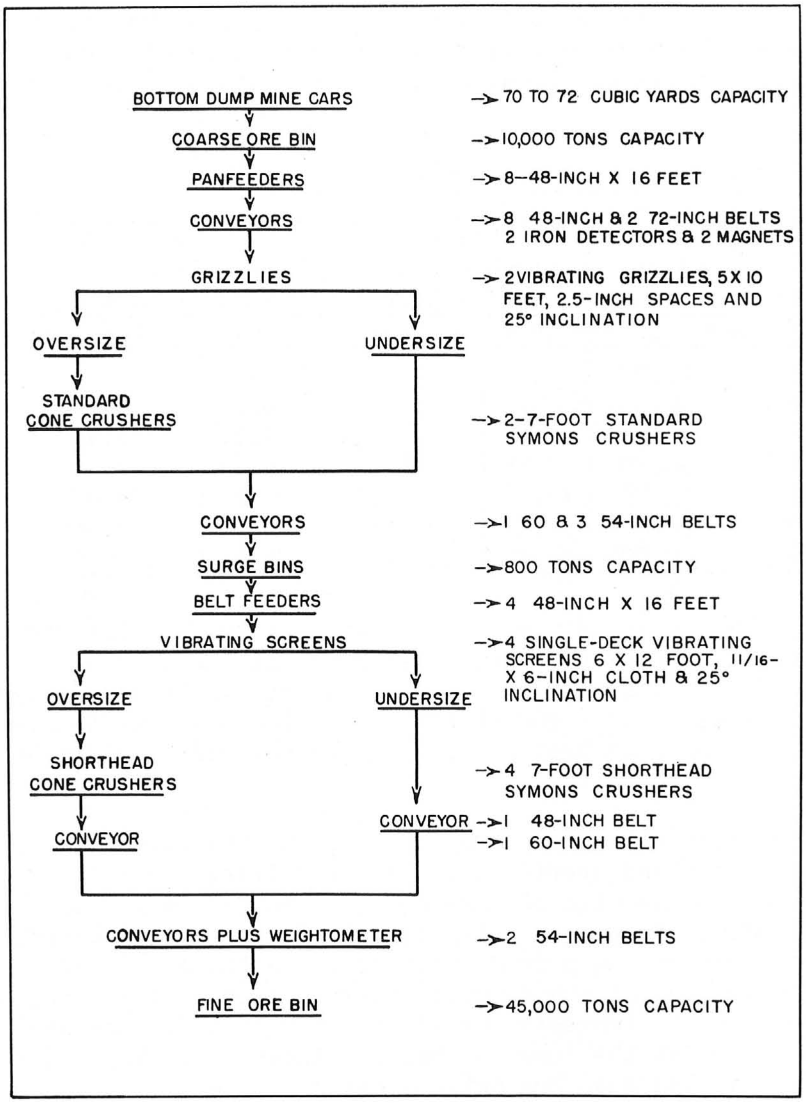

FIGURE 80. - Flowsheet of the Crushing Plant.

control alkalinity, to condition pulp, and to collect the valuable minerals. At present, milk of lime is added to the ball mill feedbox, and the standard collector, Minerec "A", is added to the classifier overflow box. The classifier overflow ranges from 10.3 to 10.7 in pH value.

A routine check of the rod load in the mills is made once a week, and the mills are charged wihh $3\frac{1}{2}$-inch rods to a load level of 40 to 45 percent of the total volume. The ball mills are charged daily with a ration charge of 50 percent 2-inch and 50 percent $1\frac{1}{2}$-inch forged alloy balls. Monthly inspections of the ball mills reveal the ball load levels which are to be corrected if not within 2 to 4 inches below the mill's centerline.

Eleven men per shift, including a shift boss, are assigned to the operation. There are an operator and helper for each of two sections and two laborers for chute blowing, feeder inspection, and general cleanup. The operator controls the operation from a central control panel (fig. 83) on the upper rod mill deck, and the helper cuts samples for a pH and density check on all mills and classifiers and the flotation pulp distributors. Typical screen analyses of the grinding-circuit products are shown in table 17.

Flotation

The flotation operation includes rougher flotation, regrinding of rougher concentrates, and cleaner flotation (figs. 84 and 85). Each phase is divided into four sections, and under normal operations the sections are operated independently. However, they are designed to bypass and distribute the rougher concentrates from any one of the flotation circuits to all four regrind circuits or to any three of the regrind and cleaner circuits. Therefore, repairs on the regrind mills or cleaner circuits can be accomplished without serious effect on recovery operations.

TABLE 16. - Typical screen analyses of crusher products

Size	Primary crushing section product, weight - percent		Screen undersize product, weight - percent		Final product (millfeed), weight - percent	
	Unit	Cumulative	Unit	Cumulative	Unit	Cumulative
Plus 1.50 inches...	22.87	22.87	--	--	--	--
Plus 1.25 inches...	7.46	30.33	--	--	0.25	0.25
Plus 1.00 inches...	7.19	37.52	0.32	0.32	1.91	2.16
Plus 0.75 inches...	12.78	50.30	2.95	3.27	16.02	18.18
Plus 0.50 inches...	11.46	61.76	12.74	16.01	26.69	44.87
Plus 10-mesh.......	19.12	80.88	37.14	53.15	30.53	75.40
Minus 10-mesh......	19.12	100.00	46.85	100.00	24.60	100.00

The rougher flotation sections are divided into six banks of 20 cells each. The cells are 48-inch units equipped with submerged airports at the base of impellers. The air volume is controlled by gate valves on the air header serving each cell and by butterfly valves at each end of the bank. The gate valves are adjusted only occasionally; the main control comes from the butterfly valves at the discharge end of each bank of 20 cells.

The feed to each rougher flotation bank is apportioned by the pulp distributor as received from the combined overflows of two classifiers in the grinding section. In addition to the mild of lime and Minerec "A" collector added to the pulp in the grinding circuit, a frother, methyl isobutyl carbinol, is introduced to each compartment of the pulp distributor. Xanthate Z-11 is added in small amounts and stage-fed at the junction boxes between cells 6 and 7 (two boxes) 12 and 13 (one box) to increase the flotability of middlings and molybdenite. Frother MIBC is fed at the junction boxes between cells No. 6 and 7, and stove oil is added at the junction box between cells 12 and 13.

Rougher flotation concentrate is collected in a common launder on one side of each bank of cells. Countercurrent froth flow is used to decrease the volume and eliminate some insoluble in order to maintain the desirable grade of rougher concentrate. Rougher concentrate is classified with cyclones in closed circuit with the regrind mills that produce a concentrate cleaner circuit feed of about 90 percent minus 325-mesh.

The cleaner flotation circuit is composed of two-stage concentrate cleaning. The first stage, the primary cleaner circuit, includes two banks of fourteen 48-inch cells each, and the second stage, the final cleaner circuit, has one bank of eight 48-inch cells.

The grade of final concentrate produced in the second stage of cleaning is held at about 28 percent copper. The concentrate includes chalcopyrite,

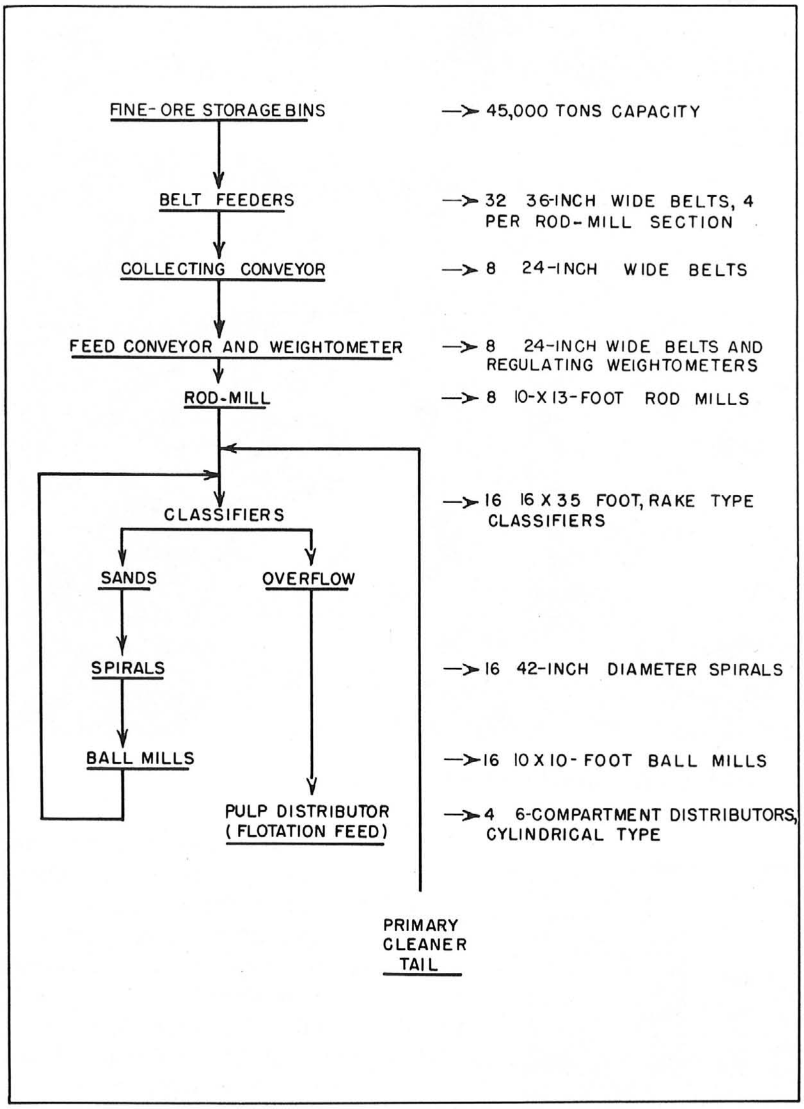

FIGURE 81. - Flowsheet of the Grinding Circuit.

molybdenite, gold, and silver, and is subjected to further treatment for recovery of molybdenite and gold.

Tails from the final cleaner stage are returned to the primary cleaner, and the primary cleaner tails are pumped to their respective rod mill discharge launders feeding the classifiers.

Five men per shift execute the duties of the various stages from rougher flotation to cleaner flotation; they include one operator for each of the two sections of roughers, one operator for regrinding rougher concentrates, one cleaner flotation operator, and one flotation laborer. A shift boss supervises this phase of operations and the molybdenum plant, concentrate handling, and tailing thickeners.

Molybdenum Plant

In general, treatment of the copper concentrate feed containing small amounts of molybdenite and other sulfide minerals with a process similar to the one at San Manuel is covered by patent 2,559,104[17]. In this process, concentrate feed is treated with a weak solution of sodium hypochlorite to partially oxidize flotation reagents generally employed in the recovery of copper minerals from ores. With the copper concentrate pulp in a suitable condition following the addition of the hypochlorite solution, other reagents are added; these include sodium ferrocyanide to depress the copper, stove oil to act as a collector, and rough conditioner and methyl-isobutyl-carbinol frother or antifoam, depending on froth conditions.

[17] Arbiter, Nathaniel, and Orel, E. Young: Flotation Recovery of Molybdenite: U.S. Patent 2,559,104, July 3, 1951.

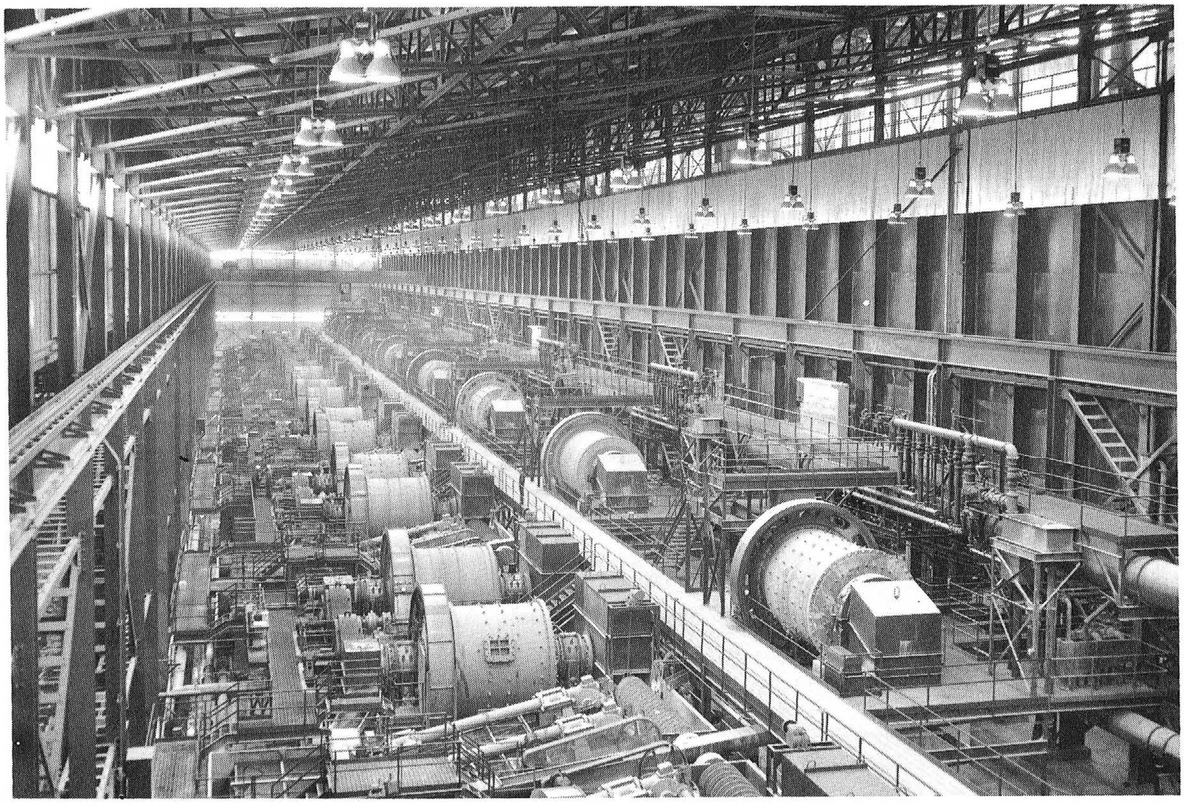

FIGURE 82. - Grinding Section.

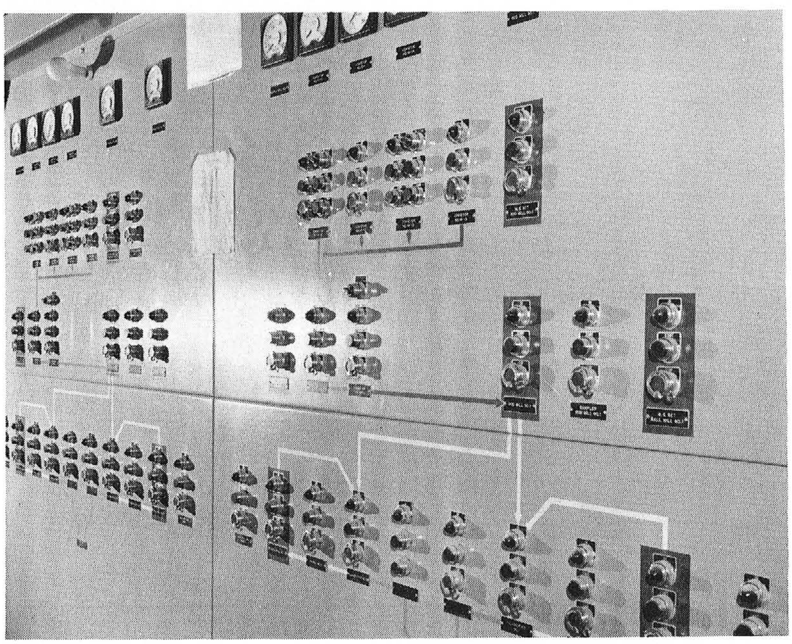

FIGURE 83. - Central Control Panel, Grinding Section.

The flowsheet of the molybdenum plant is shown in figure 86. Details of the equipment are given in the legend that follows the flowsheet. The final concentrate from the copper plant is pumped to a thickener and, after thickening to about 45-percent solid, is pumped by diaphragm pumps to the molybdenum plant rougher flotation circuit for treatment. The tailing from the rougher circuit is pumped to the thickener in the copper concentrate filtration and drying circuit. Eight stages of cleaner cells process the rougher concentrate and produce a molybdenite concentrate for shipment. The tails from the first two cleaners are combined with the rougher flotation tailings, and the tails from the subsequent cleaning stages are progressively reverted back to the previous cleaners.

TABLE 17. - Typical screen analyses of concentrator products, percent

Mesh	Rod-mill discharge	Classifier sands	Ball-mill discharge	Classifier overflow
Plus 4....	1.12	1.19	0.28	--
Plus 6....	2.98	2.05	.40	--
Plus 8....	8.02	4.78	.72	--
Plus 10....	11.02	6.63	.95	--
Plus 14....	12.66	8.80	1.71	--
Plus 20....	10.40	8.65	2.60	--
Plus 28....	8.12	9.30	4.56	--
Plus 35....	6.09	11.01	7.85	--
Plus 48....	5.03	14.80	13.36	--
Plus 65....	3.44	13.95	15.74	6.42
Plus 100....	3.92	9.66	15.21	10.79
Plus 150....	2.64	3.26	7.89	9.15
Plus 200....	2.48	1.72	6.52	9.90
Minus 200...	22.08	4.20	22.21	63.74
Total....	100.00	100.00	100.00	100.00

	Rougher concentrate	Primary cleaner tailing
Plus 200....	22.65	--
Plus 325....	10.40	8.19
Minus 325...	66.95	91.81
Total....	100.00	100.00

The molybdenum concentrate is pumped to the cyanide treatment plant for the recovery of gold and the removal of some copper. The concentrate is then filtered in a two-disk unit producing a pulp with a moisture content of 15 percent. The filter cake receives further treatment in a multiple-hearth drier to decrease the moisture content to about 2.5 to 3.0 percent. The concentrate is fed into drums, weighed, and stored for shipment.

Illustrating the quantative results of the flowsheet described in the foregoing paragraphs, the analyses of the feed, tailings, and concentrate at different stages of the operation are shown in table 18.

The operation depends greatly upon a smooth operation of the copper plant; usually the plant operates continuously. Table 19 shows the metallurgical data resulting from a 24-hour period of operation.

Part of the molybdenum plant operation is the making of a sodium hypochlorite solution. An automatic unit for mixing chlorine, sodium hydroxide,

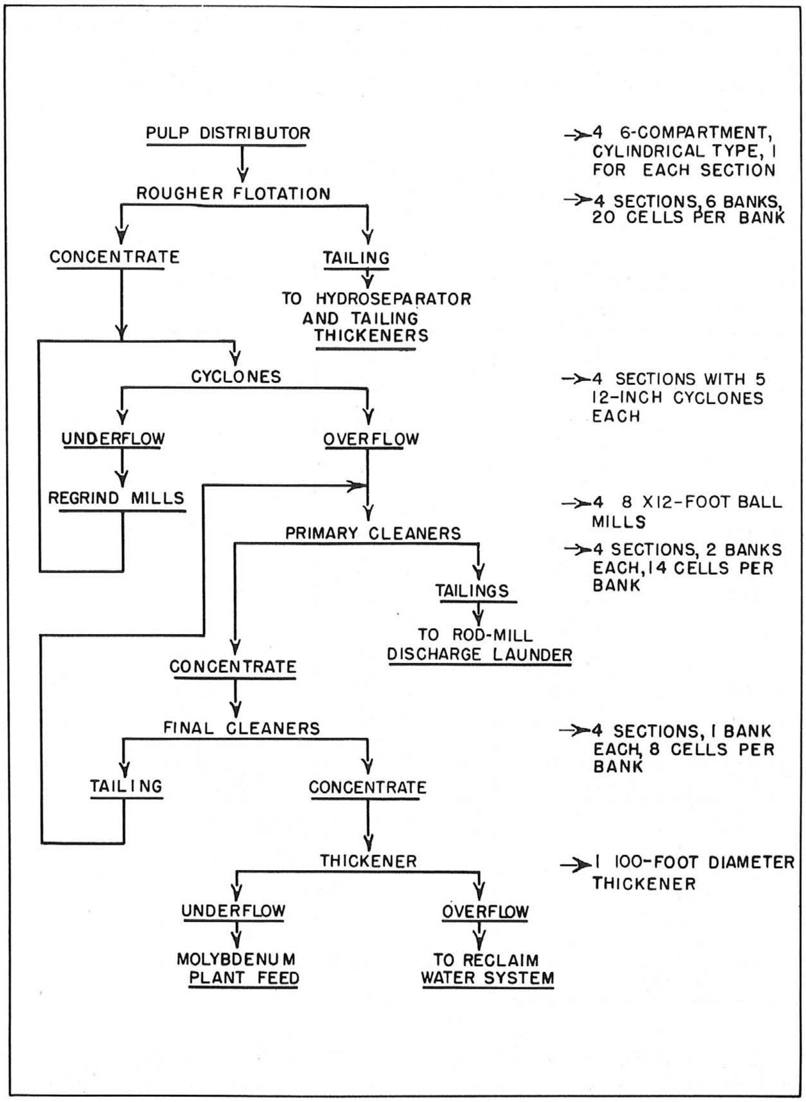

FIGURE 84. - Flowsheet of the Flotation Circuit.

and water operates continuously, producing a solution with a strength ranging from 1.60 to 2.00 percent NaOCl. Two men per shift, an operator and a helper, are assigned to the operation. By constant surveillance, the operator maintains standard conditions in the molybdenum and hypochlorite plants. The helper informs the operator as to pH readings and densities of the pulps. One laborer is assigned to the day shift for general cleanup duty at the two plants. A day foreman supervises the operation on the day shift, making the necessary circuit adjustments, and inspects and controls the loading of molybdenite concentrate for shipment.

Gold Recovery From Molybdenum Concentrates

Some of the gold in the copper concentrates is floated with the molybdenum sulfide in the molybdenum flotation process. The molybdenite concentrate is pumped to a cyanide treatment plant, where gold is recovered and some copper is removed. After treatment, the molybdenum sulfide is pumped to the molybdenum plant for filtering, drying and storage. The pumps that move the materials as slurry to and from the cyanide plant operate continuously in closed circuits to prevent loss of prime and settling of solids. The cyanide plant is treating approximately one-half of its designed daily capacity of 8 tons of molybdenum concentrates. Approximately 25 tons of pregnant solution is treated daily.

The flowsheet of the cyanide plant (fig. 87) is comparatively simple. The concentrates are filtered before treatment to remove flotation reagents that remain in the slurry and prevent accumulation of water is the leach circuit. The filtrate is returned to the feed thickener of the molybdenum plant, and the

FIGURE 85. - Flotation Section.

filter cake passes to the agitator section, where it is agitated in tanks in three stages. The tanks are 10 feet in diameter and 10 feet deep; pulp depth is maintained at about 8 feet. The tanks are equipped with pumps so that each can be operated as a batching unit, but currently they are operated in tandem. Barren solution and a strong solution of calcium cyanide are added to the pulp in the first agitator. After being agitated for the required time, the slurry passes to the second agitator, where sufficient cyanide solution is added to maintain the cyanide strength. After the required period of agitation, the slurry passes to the third agitator, where the process is repeated. The agitation cycle requires 72 hours.

The slurry is filtered after agitation. The filtrate or pregnant solution is pumped to storage tanks. The cake is washed in an agitator, and this slurry also is filtered. The filtrate is pumped to the pregnant-solution storage tanks.

The pregnant solution is fed to a clarifier tank, where lead acetate is added as a precipitating aid. Sludge is removed periodically from the tank.

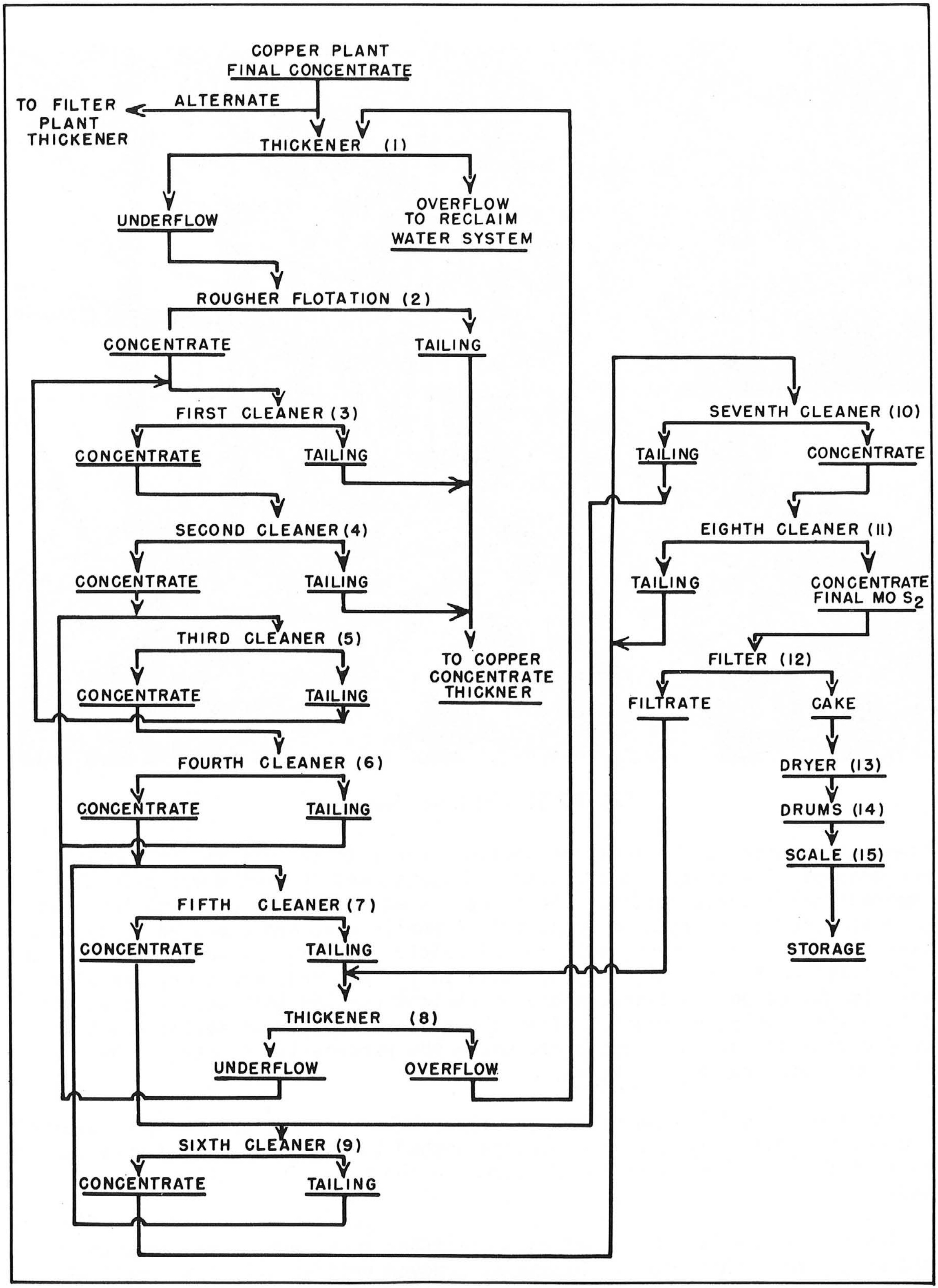

FIGURE 86. - Flowsheet of the Molybdenum Plant.

Legend - Molybdenum plant flowsheet

1. One 100-foot diameter thickener.
2. Sixteen rougher flotation cells (36 inches).
3. Twenty first-cleaner cells (36 inches).
4. Twelve second-cleaner cells (36 inches).
5. Ten third-cleaner cells (36 inches).
6. Six fourth-cleaner cells.
7. Two fifth-cleaner cells.
8. One 30-foot-diameter thickener.
9. Two sixth-cleaner cells.
10. One seventh-cleaner cell.
11. One eighth-cleaner cell.
12. One 4-foot-diameter two-disk filter.
13. One multiple-hearth concentrate dryer (4-foot diameter).
14. Galvanized steel drums (55-gallon capacity) for MoS_2 concentrate storage.
15. One scale with conveyor (steel roller type).

The solution passes to a deaeration tower for the removal of air by a vacuum pump. Zinc dust is added to the solution to precipitate gold and other metals. The precipitate and liquor are pumped to a bag-type precipitate filter. The precipitate is sent to the smelter, where it is fed to the holding furnace in the anode casting department. Most of the filtrate or barren liquor is pumped to tanks from where it is fed to the cyanide mixer and the agitators. A little liquor flows to the zinc cone tank. Some of the liquor must be discharged periodically into the tailrace from the concentrator in order to prevent excessive fouling with copper and zinc and an accumulation of excess water in the circuit.

TABLE 18. - Typical analyses of the various stages of the molybdenum plant circuit, percent

Flotation stage	Feed		Tailing		Concentrate	
	MoS_2	Cu	MoS_2	Cu	MoS_2	Cu
Rougher.........	0.55	27.13	0.06	24.80	1.44	32.05
First cleaner...	--	--	.34	31.90	4.40	32.85
Second cleaner..	--	--	.29	33.40	7.24	32.25
Third cleaner...	--	--	1.30	33.70	18.66	27.95
Fourth cleaner..	--	--	1.25	33.90	--	--
Final cleaner...	--	--	--	--	94.90	.85
Final tailing...	--	--	.17	27.62	--	--

TABLE 19. - Metallurgical data of a 24-hour operation, molybdenum plant[1]

Time operated..	hours	24
Feed...	dry tons	760
Assay, percent:		
Feed:		
MoS_2..		.55
Cu...		27.13
Concentrate:		
MoS_2..		94.90
Cu...		.85
Tailing, MoS_2.....................................		.17
Concentrate produced.................................	dry pound	6,404
Moisture...	percent	3.0
Contents, pounds MoS_2:		
Feed (indicated).....................................		8,360
Concentrate..		6,077
Tailing (indicated)..................................		1,292
Recoveries, MoS_2, percent:		
Indicated assays.....................................		69.21
Actual...		72.69
Reagent consumption, pounds per ton of feed:		
Caustic..		14.76
Chlorine...		10.53
Sodium ferrocyanide..................................		1.37
Stove oil..		1.37
Antifoam...		.52

[1] The recoveries indicated by assays will check the actual recovery within 0.5 percent at the end of given period of a month.

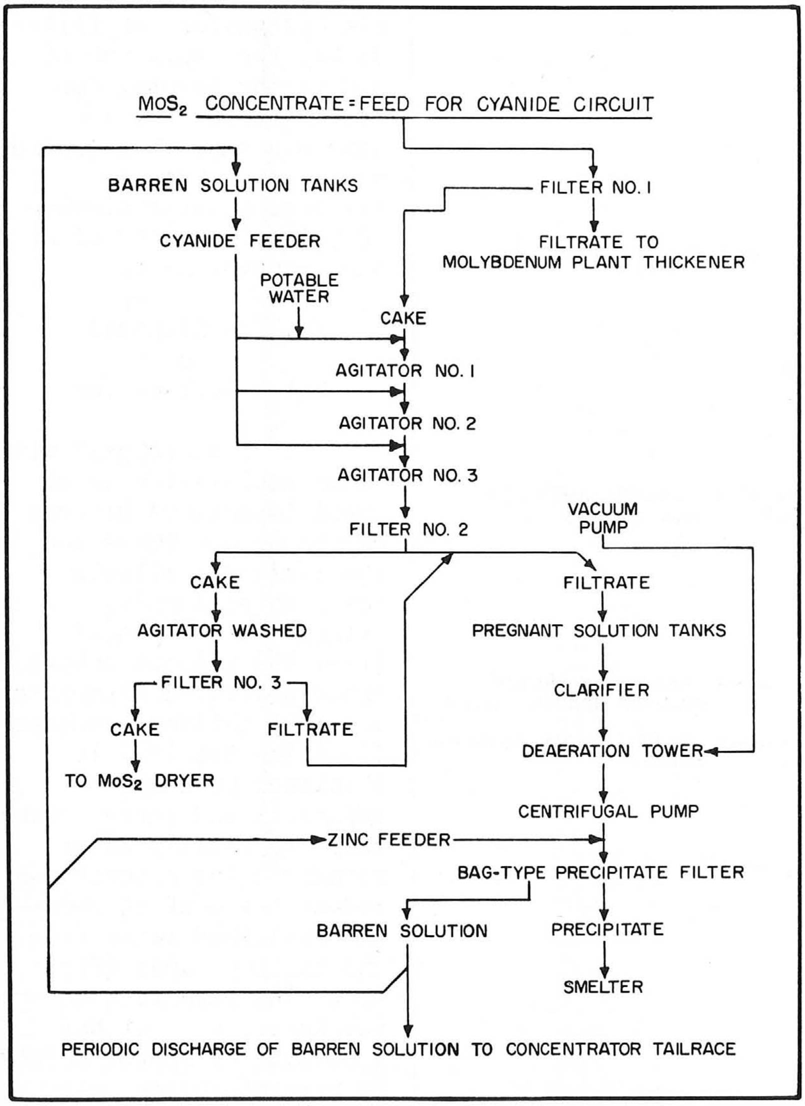

FIGURE 87. - Flowsheet for Cyanide Treatment of Molybdenum Concentrate.

Approximately 75 pounds of calcium cyanide is required to treat a ton of molybdenum concentrate. This is believed to be the result of poor aeration rather than the presence of copper in the feed. Plans are being made to increase aeration in order to reduce cyanide consumption.

Concentrate Handling

The flowsheet of filtration and drying of concentrates is shown in figure 88. Practices in this department present no unusual features, expect the additional treatment of drying the molybdenum filter cake in multiple-hearth dryers. The copper concentrate flow, which is the molybdenum plant tailings, is pumped to a thickener and thickened from 14 to 15 percent solid to 50 percent solid. The underflow is pumped to the top floor of the filter plant, and under normal operation, it is distributed to two or three filters. The filtrate is returned to the thickener, and the filter cake drops onto the top part of the dryer or onto a bypass conveyor. The moisture content of the filter cake and the dryer discharge ranges from 14 to 15 and 9.5 to 10.0 percent, respectively. The dried concentrate is delivered by conveyor to the smelter concentrate bins, passing over a weightometer in transit.

Round-shoulder sector covers of cotton cloth are used on the filters. Their operating life averages between 6 and 7 days. A fine regrind circuit product and a high concentrate grade cause very severe blinding.

The operating crew consists of one operator and helper per shift. Their duties, in addition to the operation of the equipment, include the changing of

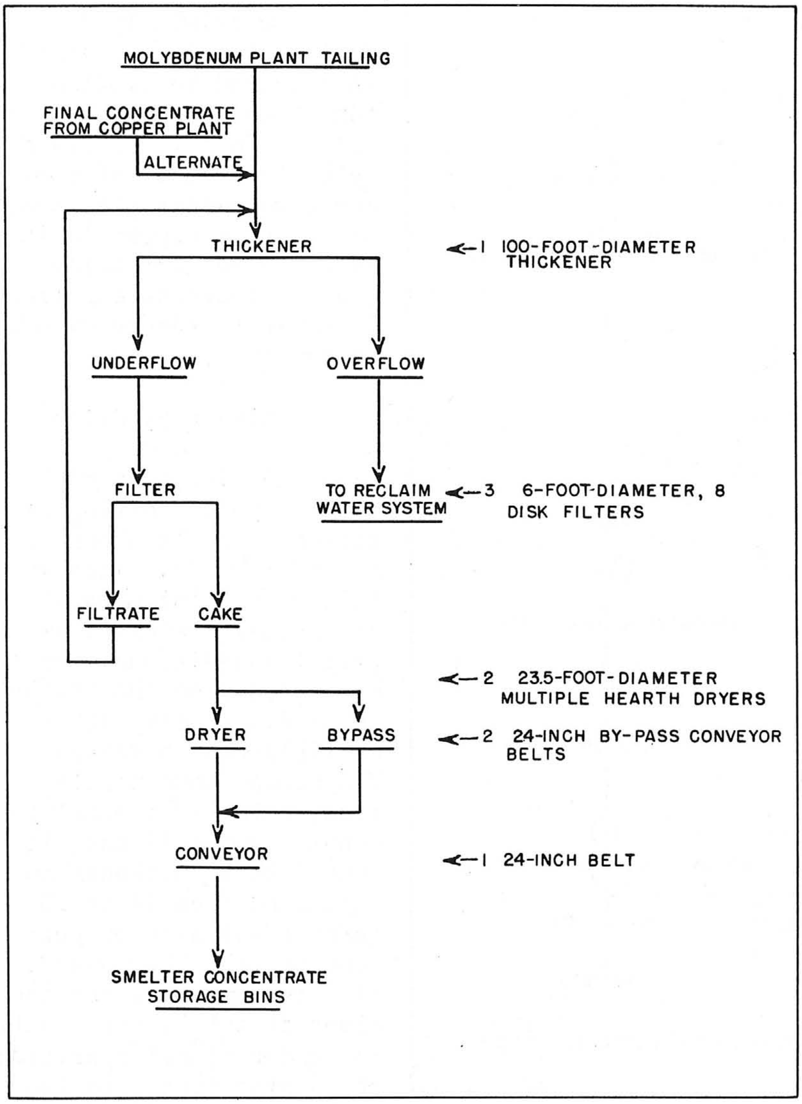

FIGURE 88. - Flowsheet for Filtration and Drying of Concentrates.

blanket sectors on filter disks, the replacing of old sector covers, conveyor inspection, and some cleanup. One laborer on the day shift is assigned general cleanup duty in the filter plant and conveyor ways.

Tailing Disposal and Water Reclamation

Tailing disposal with water reclamation is of great importance because of the arid climate and the limited available fresh water supply. Methods at San Manuel (fig. 89) are not unlike those practiced elsewhere. The mill tailing (rougher flotation tailing) is thickened by a hydroseparator and three thickeners to reclaim water sooner in the circuit and reduce the cost of pumping reclaimed water from the tailing sands (fig. 90). The feed to the thickeners circuit has a density of approximately 22 percent solids, and the thickened discharges from the hydroseparator and the thickeners average 40 to 48 and 39 to 42 percent solids, respectively. The thickener overflows are conveyed to a pumping station adjacent to the thickener area and are returned to the mill's reclaimed water circuit. Under normal operations, about 75 percent of the mill tailings is fed to the hydroseparator. Underflow from this unit is mixed with thickener underflow discharges at a spigot house, and the overflow is combined with the remaining mill tailings and distributed to the thickener.

The tailing storage damsite is located about 1 mile east of the tailings thickeners down the slope toward the San Pedro River. The average slope of the terrain is about 5 percent. The location is free of large, deep washes or ridges which might cause difficulty in water reclamation or tailings disposal.

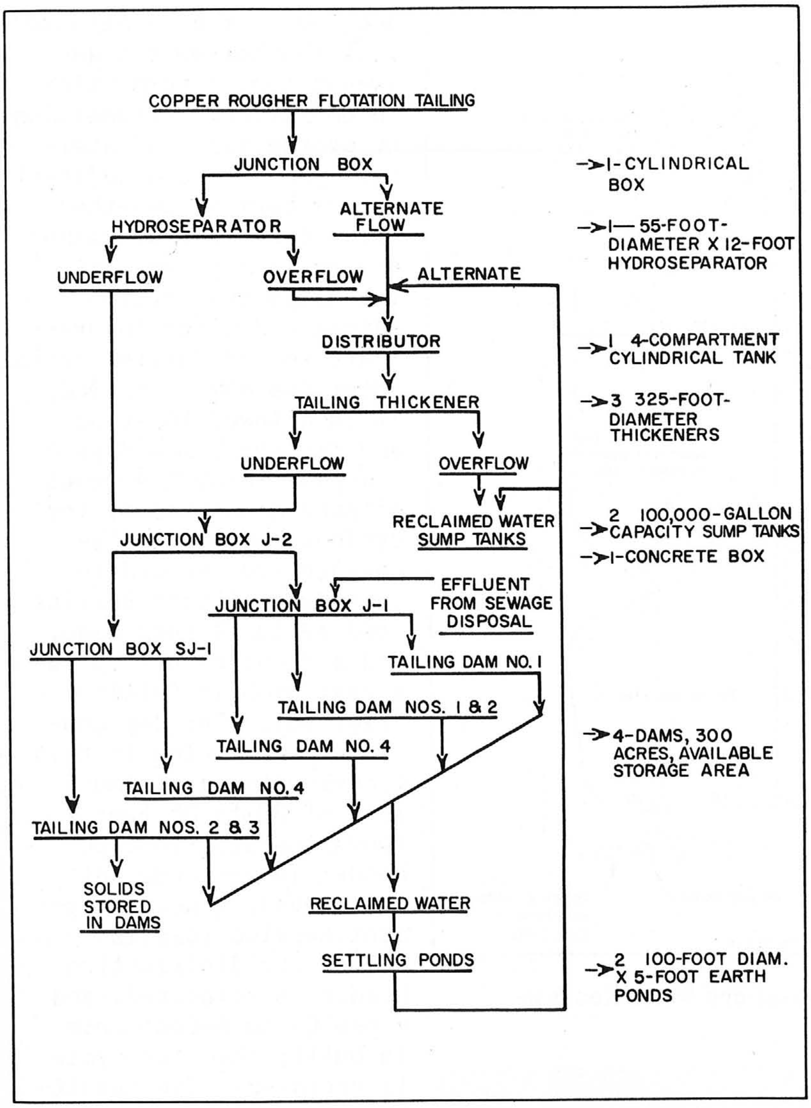

FIGURE 89. - Flowsheet for Tailing Disposal and Water Reclamation.

Thickener discharges are piped from the spigot house to a concrete junction box which has a suitable butterfly valve by which the tailings can be diverted to other junction boxes located near the different storage dams (fig. 91 and 92). The pipes are of wooden stave or cement-asbestos composition and are laid on a 0.5 percent slope. Wooden stave drop boxes are placed at intervals in the line where the slope of the ground exceeds 0.5 percent. At a location above the dams, concrete or wooden stave junction boxes with butterfly valves distribute the tailings to the various tailing distribution headers located around the periphery of the dam berms. The original earth dam berm and distribution headers are shown on figure 93.

The tailing distribution headers are equipped with 4-inch, round-port lubricated-plug valves at 26- or 52-foot centers, depending on the area of the dam (berm to dewater ducts). A light-gage steel pipe connects the header with a 12-inch cyclone operating under a pressure of between 2 to 5 p.s.i. The cyclone is placed on a lightweight stand oriented with the overflow discharge 15° below the horizontal and facing the dam. An extension (4 feet long) spills the overflow away from the stand, and the cyclone overflow discharges the slimes through a 4-inch line for at least 40 feet into the dam (fig. 94).

This method of tailing disposal has advantages over the old method of wooden-trestle structure supports for tailing distribution headers. The advantages are: The need for fewer skilled personnel for maintenance and repair, less consumption of structural material, less downtime between dragline berms;

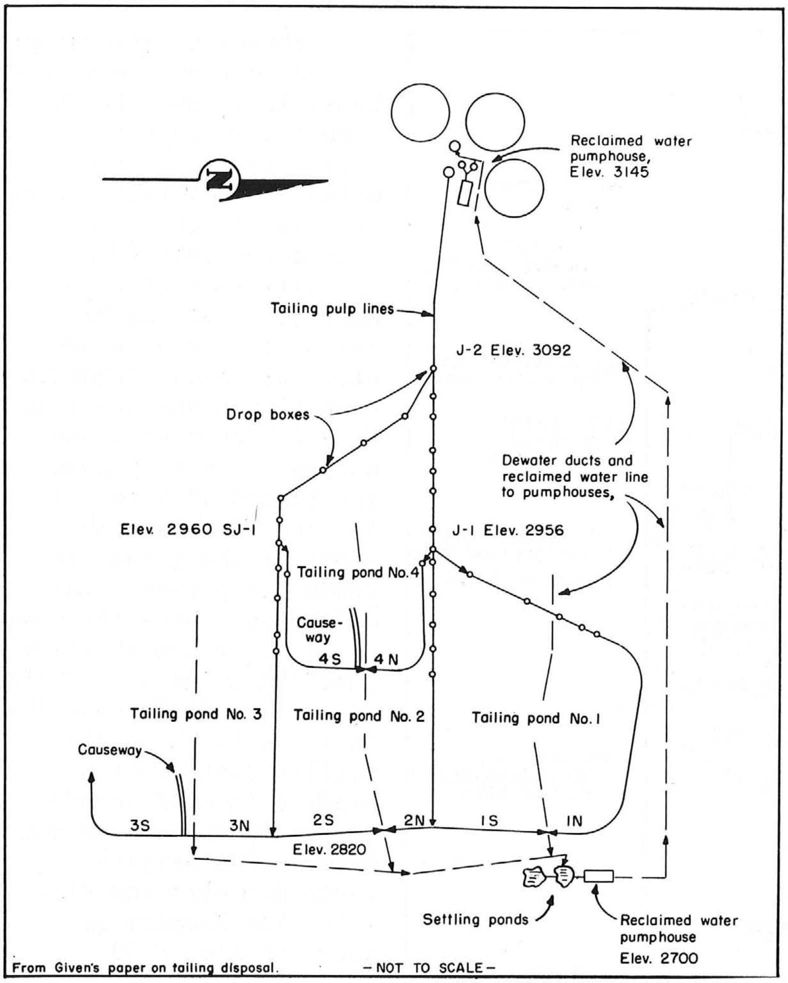

FIGURE 90. - Tailing Disposal and Water Reclamation Area.

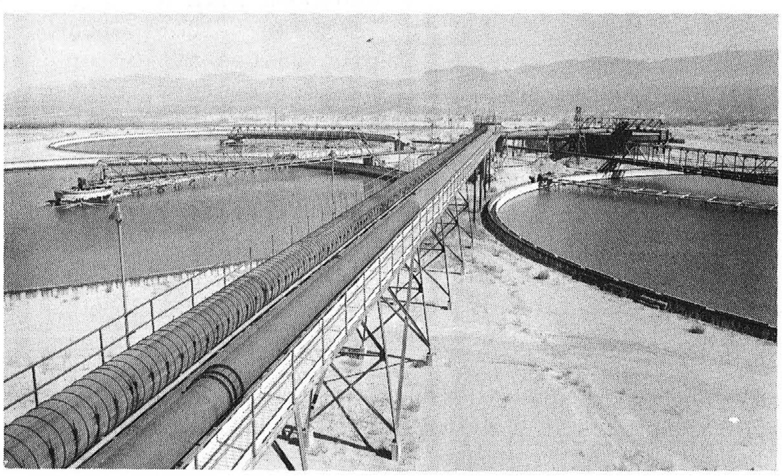

FIGURE 91. - Thickeners.

and the lack of a need to fill the borrow pit adjacent to the berm prior to operation. Disposition of coarse tailing material (sand) on and adjacent to the berm allows the berm to be free of slimes and to form a gradient that aids the tailing slimes and water to gravitate to the dewater ducts. After the dam is filled, it is allowed to stand and dry for a few days or a week. Then, the cones of sand deposited by the cyclone underflow discharges are leveled to form an equipment service road 12 to 14 feet wide, and a dragline berm 5 to 6 feet high is built (fig. 95). The dam continues to develop in this manner until a maximum lift of 30 to 35 feet above the distribution header is reached. At this level, a new equipment service road is built, the distribution header is relocated, and a new 6- to 8-foot berm is built; then the cycle is repeated. The resulting slope of the dam face will be 35° or less. For water reclamation, all dams are equipped with a dewater duct system that extends down the slope under the base of the dam to a large collection line. To provide for removal of water, standpipes at 50-foot centers on the duct system are extended as slime level rises in the ponds.

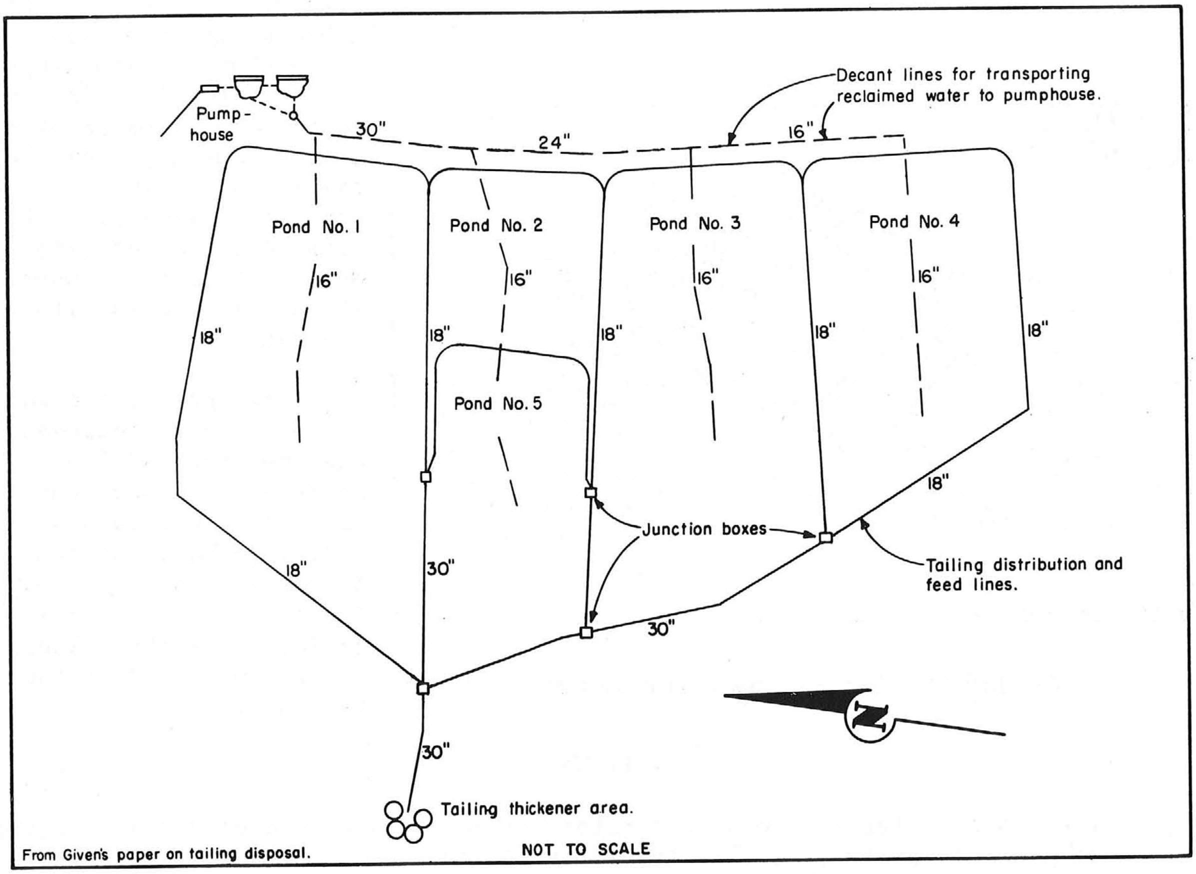

FIGURE 92. - Operation of Tailing Ponds.

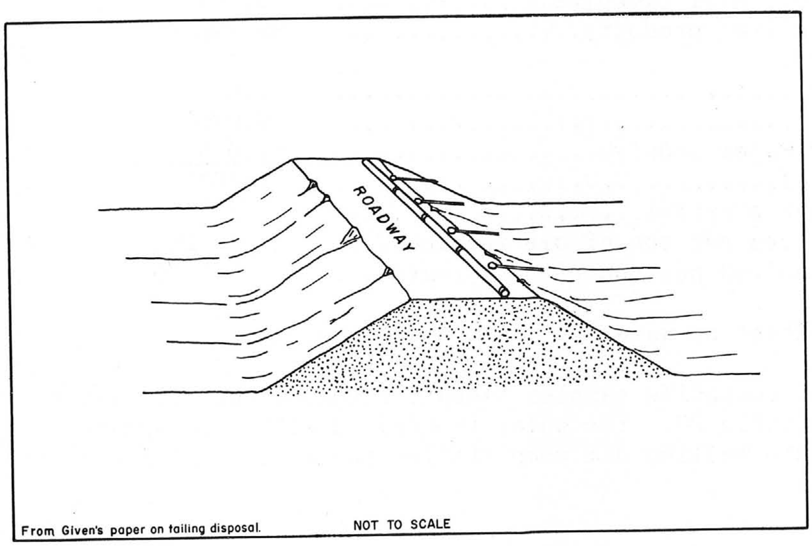

FIGURE 93. - Original Earth Berm Showing Installation of Header and Discharge Pipes Into Storage Area.

Each dam is contained by a wing, or continuation of the dam face, at each end; it extends up the slope and at approximately right angles to the dam face. The location of the water pool can be controlled from the wings and the face of the dams by tailing discharges through cyclones.

On small dams, a causeway near the center and running parallel to the wings is maintained for more flexibility and as a safety measure against possible dam

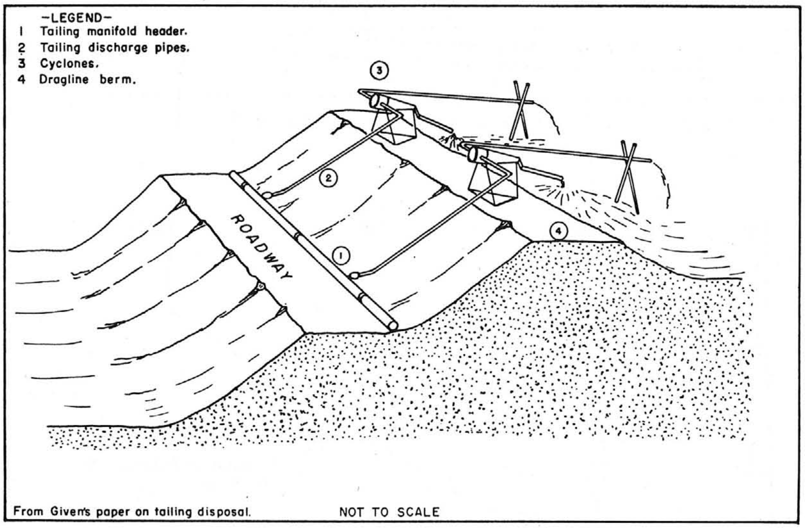

FIGURE 94. - Cyclone Operation From Berm.

breaks. In the event of a break, the causeway would check, or possibly hold the tailings in the dam on the opposite side of the causeway from the break. A cut in the causeway allows water to flow to the dewatering duct. After it is used, the cut is dammed with sandbags.

The operating crew for each shift includes one operator and two helpers. The operator controls the amount of reclaimed water to be withdrawn from the dams for mill use and helps or instructs the helpers in the operating of the cyclones.

Mill Water

The use of water at the concentrator for an average 24-hour period (32,148 tons of ore was treated) is illustrated as follows:

	Tons
Water used in all mill circuits..............	115,599
Water reclaimed from products.................	88,892
New water added:	
From smelter................................	1,026
From mine...................................	19,020
From potable water supply...................	7,029
Total.......................................	26,707
Loss due to tank overflow....................	368
New water required per ton of ore treated.....	.83
Total water required per ton of ore treated...	3.60

Figure 96 is a flowsheet of water at the concentrator.

Chemical analyses of composite samples taken throughout the mill water circuit are tabulated in table 20. The water is treated with a phosphate sequestering reagent at the tailing dam pump station and at the tailing thickener pump station.

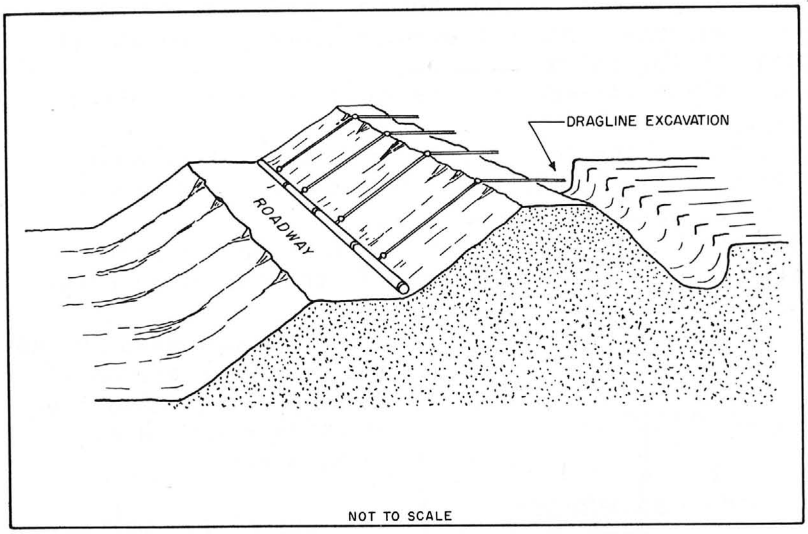

FIGURE 95. - Dragline Berm and Method of Raising Tailing From Header Over Top of Berm Into Tailing Dam.

Metallurgical Control

Control of the rod mill feed size dictates a daily check of the crusher plant product from each of the various grizzlies and screens, and from both primary and secondary crushers. Ore is weighed by weightometers enroute to the fine ore storage bin and at the feed end of the individual rod mill grinding sections. Individual samples of rod mill feed are cut by hand, one each for the moisture and screen analyses. The rod mill discharge (mill head sample) is sampled once each hour at the various mills.

The tails from each rougher flotation section are sampled by automatic samplers every 20 minutes. The primary cleaner flotation tail is sampled by hand every 30 minutes, but regrind feed and final concentrate samples are cut by automatic samplers.

Most of the samples in the molybdenum plant are cut every 20 or 30 minutes by automatic samplers. Filter concentrates are hand-sampled every hour. This sample is for mill control only. The weighing of concentrates is practiced and used for mill control.

Desk-type pH meters are widely used on the grinding floor, and in the cleaner flotation section and molybdenum plant. Recorders and meters are located on the tailing thickener overflow water launder and at the hypochlorite plant.

General

Crusher and mill repairs are performed by a crew assigned to each of the two units. The mechanical department, electrical department, and plant construction department are available for repair jobs beyond the scope of the mill repair department.

Other personnel not mentioned in the foregoing text are listed under the following jobs:

1. Work in coarse and fine-ore bins--three bin blasters to clean bins and blast choke-ups in bins, feeders, and chutes.

2. Mixing of reagents--one operator and one helper to mix and prepare all mill and molybdenum plant reagents, unload reagent shipments, and assist in loading molybdenum concentrate for shipment.

3. Charging of balls to ball mills--one helper to charge the required balls to each ball mill daily.

4. Maintenance of tailing dam--one foreman, one crew boss, four equipment operators, one truckdriver, two helpers, and nine laborers to maintain the tailings dams and reclaim all possible water.

5. General labor--one labor boss and a crew of 18 to 24 men to perform general cleanup work and fill vacancies that may occur in the operating crews.

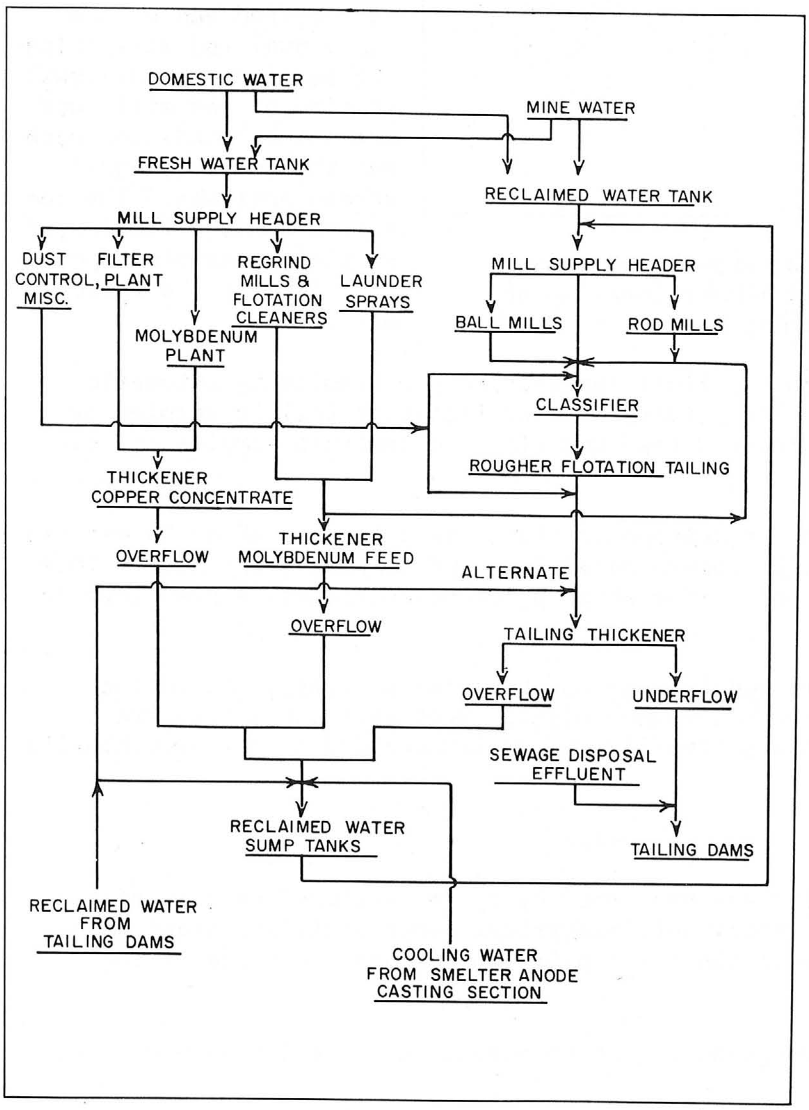

FIGURE 96. - Flowsheet of Plant's Water Supply.

The employees in the mill dividion are on a 5-day workweek, and the plant is operated on a 7-day basis.

The mill, smelter, and mine sample preparation and analyses are completed by an assay office located near the concentrator. The work is supervised by a chief chemist.

Supplies of all types are available through the warehouse at the concentrator, which maintains an adequate stock.

Electrical power for the mill division is furnished by the Arizona Public Service Co. The powerplant, which was constructed for the initial operation at the mine, is available on a standby basis. Waste heat from the smelter is used to generate a small amount of power; this is fed into the Public Service substation. Power for the crushing plant and mill is delivered to the substation at 12,500 volts, where it is transformed into 2,400- and 440-volt current for plant usage.

TABLE 20. - Chemical analyses of the mill circuit waters, parts per million

Analysis	Source of water composite samples[1]							
	1	2	3	4	5	6	7	8
Total dissolved mineral solids - weighed residue...	244	404	316	982	982	256	570	932
Water - pH..................	8.0	8.0	7.9	10.4	7.6	7.6	7.4	9.7
Alkalinity as $CaCO_3$:								
Phenolphthaleim...........	(2)	(2)	(2)	146	(2)	(2)	(2)	22
Methyl orange.............	134	212	176	180	106	140	246	52
Hydroxide.................	-	-	-	116	-	-	-	-
Hardness as $CaCO_3$...........	48	170	106	304	86	48	98	298
Calcium, Ca.................	14	49	23	95	-	18	27	119
Magnesium, Mg...............	3	12	12	17	-	.5	7	(3)
Bicarbonate, HCO_3...........	163	259	215		129	171	249	7
Carbonate, CO_3..............	-	-	-	38	-	-	-	26
Hydroxide, OH...............	-	-	-	39	-	-	-	
Chloride, Cl................	10	25	16	169	404	10	36	134
Sulfate SO_4.................	24	61	43	328	-	30	142	428
Phosphate, PO_4:								
Ortho.....................	(3)	(3)	(3)	1	(3)	(3)	26	1
Poly......................	1	1	2	6	1	6	22	3

[1] The numbers in the boxheads correspond to the items which follow:

1. Domestic well water.
2. Mine water.
3. Fresh water (mixture of domestic and mine water).
4. Reclaimed water in mill supply header (mixture of reclaimed and new water from reclaimed waterhead tank).
5. Concentrate thickener overflows.
6. Smelter cooling water from anode casting section.
7. Sewage effluent water.
8. Tailing dam return water from pump station.

[2] None.

[3] Trace.

Safety

Good housekeeping is practiced in the mill division and has been instrumental in improving the safety record. Table 21 shows the improvement of the safety record for the first quarter of 1958 over that for the same period in 1957.

Lime and Flux Plant

As the name implies, the lime and flux plant has a twofold purpose, supplying flux to the smelter and milk of lime to the grinding, flotation, and tailings circuit of the concentrator. The flowsheet of the lime and flux plant is shown in figure 97.

TABLE 21. - Comparison of lost-time and no-lost-time accidents - 1957 versus 1958 (concentrator only)[1] [2]

Month	Hours of exposure	Accidents: Lost time	Accidents: No lost time	Accidents: Total	Frequency rate for month	Cumulative frequency rate
1957:						
January.......	46,960	0	6	6	0	0
February......	43,784	3	6	9	68.62	33.06
March.........	49,656	1	4	5	20.14	28.49
Total for quarter.....	140,400	4	16	20	-	28.49
1958:						
January.......	53,368	0	8	8	0	0
February......	48,992	2	5	7	40.82	19.54
March.........	53,376	1	3	4	18.74	19.26
Total for quarter.....	155,736	3	16	19	-	19.26

[1] The first quarter of 1958 shows a 32.4-percent improvement in frequency rate.

[2] There were no fatal accidents during either quarter.

Quartzite and some mine ore and limestone are required for smelter flux, and milk of lime is required in mill circuits. These materials are delivered from the mine or from local quarries in bottom- and side-dump cars.

The various materials are dumped into an ore pocket as needed and are fed to the primary crusher in open circuit. The ore pocket is made of concrete with a pyramidal bottom to eliminate a dead load. This facilitates the cleaning of the pocket between crushing operations of various types of flux. The ore pocket surmounts a panfeeder that feeds a jaw crusher. The jaw crusher is operated with a 4½-inch discharge opening, and the resulting products are stored in their respective bins (one bin each for limestone and quartzite and two bins for mine ore). A common conveyor system and tripper conveyor unit feed the various products to the bins.

Each bin is equipped with a panfeeder to feed a common conveyor system arranged to feed either of two shorthead cone crushers in open circuit. One of the secondary crushers handles quartzite and mine ore, and the other crushes limestone. Quartzite or mine-ore feed is advanced by a conveyor system to a double-deck vibrating screen. Screen cloth on each deck is of square mesh, 1½-inch on the top deck and 3/4-inch on the bottom deck. Oversize passes through the cone crusher set at 5/8-inch on the closed side and yields a minus 1-inch product. The combined cone discharge and screened undersize constitutes the final product to be conveyed to smelter-flux storage bins. Limestone is similarly processed in the other cone-crusher system. The final product is conveyed to a single-deck screen having a 5/8-inch mesh cloth. The screen undersize drops into the conveying system connected to the smelter flux bins, where it is sent to the proper bin by a tripper conveyor unit. The screen oversize spills into the kiln feed storage bin.

Kiln feed passes from the storage bin onto a conveyor and then over a variable feed belt to a kiln at any predetermined rate from 2 to 7 tons per hour. The normal rate of feed is about 5 tons per hour. The residence time of the limestone in the kiln is 3 to 4 hours, depending on tonnage. The kiln is fired with natural gas and is operated between 2,350° and 2,450° F. The resulting burned lime from the kiln can be fed directly to the lime slaker or stored in the bin for use during down periods of kiln operation.

The lime-slaker operation is controlled by the demand from the mill. It operates on the average of 16 hours per day at the rate of 3 tons an hour. Burned lime is sent to the slaking compartment by a drag-chain conveyor from the storage bins, or directly from the limekiln. The first method is preferred because of the limited capacity of the slaker. The operating temperature for an efficient operation is from 190° to 200° F. The temperature is controlled by the addition of water as the feed of burned lime is held constant. Lime slurry is agitated slowly by a mechanical agitator to insure complete slaking of the lime, this, in turn, will free the insolubles.

The slaker assembly includes a reclassifier attached to the slaking compartment. A small opening near the bottom of the compartment allows the thick lime slurry to discharge to the reclassifier. The slurry is diluted to maintain the resulting classifier overflow density at 6 percent solids. Under normal operations, the solids in the milk of lime will pass through a 65-mesh screen, but on occasion the grade of limestone will drop and cause the solids in the milk of lime to increase to 3 or 4 percent on 65-mesh at the normal operating density of 6 percent.

The classifier overflow is pumped to the milk-of-lime storage tanks for use in the mill circuits. The classifier sands are elevated to a small bin and later hauled to the tailings dam. The milk of lime is circulated through the mill in one header in closed circuit with the storage tanks. Outlets are conveniently located and are operated manually by diamond quartz valves. This type of valve has given satisfactory results in that the accumulation of solids behind the valve is nil and the valve can be opened and turned to its operational setting with ease.

Table 22 reviews the metallurgical data and results of operations from January 1 to March 31, 1956, inclusive.

A crew on the day shift operates the crushing unit and performs the necessary cleanup. The crew consists of one operator, one helper, and two laborers. The limekiln and the slaker-classifier unit are operated on each shift by one kiln burner and one suboperator. One foreman is responsible for supervising the operation and training the personnel.

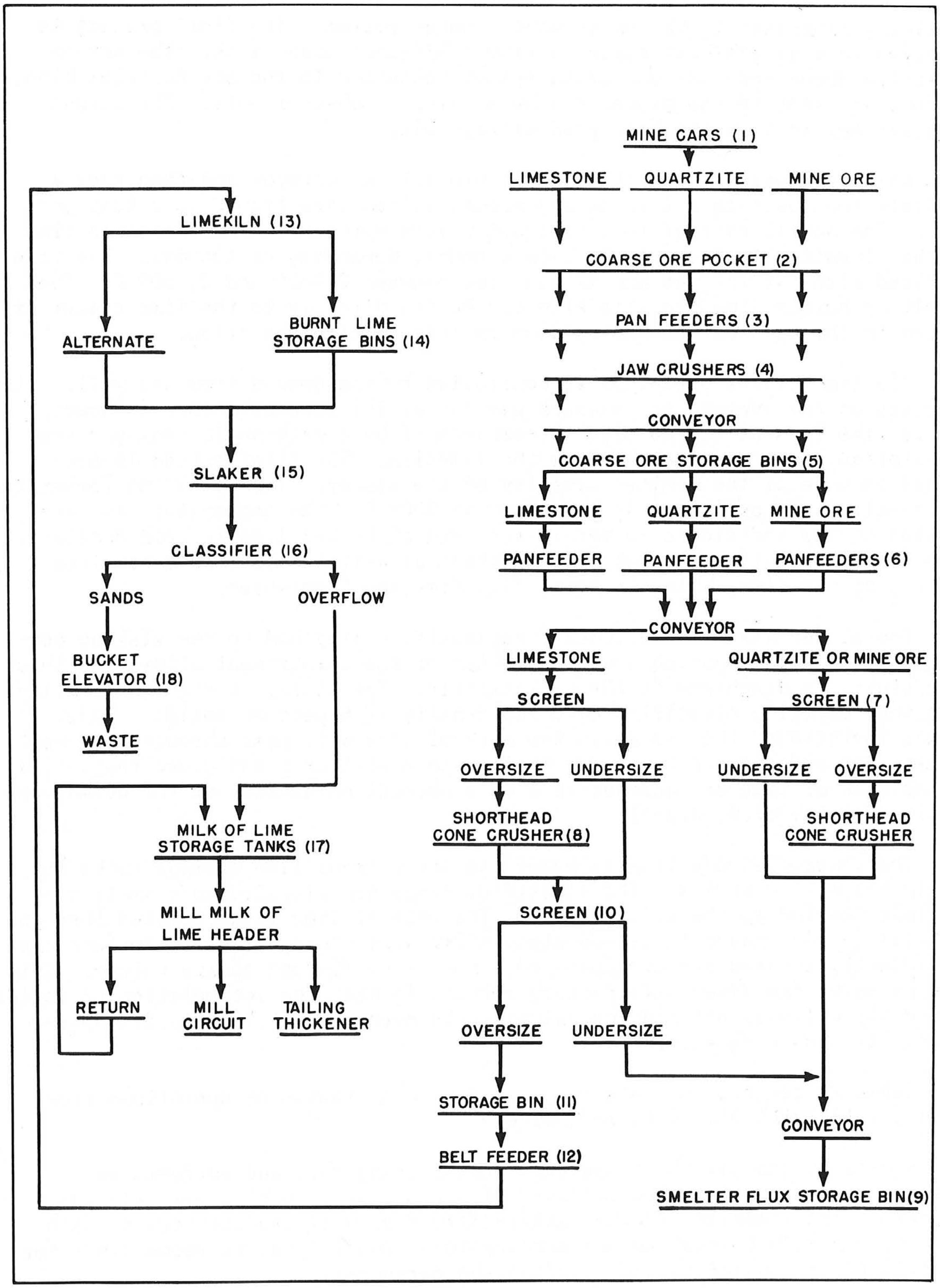

FIGURE 97. - Flowsheet of Lime and Flux Plant.

Legend - Lime and Flux Plant Flowsheet

1. Bottom- and side-dump cars (71 and 40 cubic yards capacity).
2. Coarse-ore pocket (300-ton capacity).
3. One panfeeder (5 feet wide).
4. One jaw crusher (30 by 42 inches, $4\frac{1}{2}$-inch discharge opening).
5. Four coarse-ore storage bins (500-ton capacity).
6. Four panfeeders (4 feet wide).
7. Two double-deck screens ($1\frac{1}{2}$- and 3/4-inch square mesh screens).
8. Two $5\frac{1}{2}$-foot short-head cone crushers (5/8-inch discharge openings).
9. Four smelter flux storage bins (360-ton capacity).
10. One vibrating screen (5/8-inch square mesh openings).
11. One limestone storage bin (500-ton capacity).
12. One belt feeder ($25\frac{1}{2}$ by 67 inch variable speed).
13. One limekiln (7 feet 6 inches by 180 feet - 1.8° inclination).
14. Two burned lime storage bins (500-ton capacity).
15. One lime slaker (3 to 4 tons per hour capacity).
16. One rake-type classifier in combination with the slaker mentioned in item 15.
17. Two milk of lime storage tanks (90,000-gallon capacity).
18. One bucket elevator (31-foot center to center on sprocket drives).

TABLE 22. - Summary of metallurgical data of the lime and flux plant[1 2]

Sulfide ore:		
Amount crushed	tons	14,606
Assays, percent:		
Cu		.75
SiO_2		65.4
Fe		4.2
CaO		1.6
Al_2O_3		15.7
Quartzite:		
Amount crushed	tons	2,766
Assays, percent:		
SiO_2		93.3
Al_2O_3		2.5
Limestone:		
Amount crushed	tons	11,863
Assays, percent:		
CaO		48.0
MgO		4.2
Insoluble		5.8
Kiln feed:		
Limestone	tons	6,470
Assays, percent:		
CaO		49.8
Insoluble		2.5
Kiln discharge:		
Burnt lime	tons	3,693
Assays, percent:		
Total CaO		88.7
Available CaO		83.3
Insoluble		2.5
Gas consumption per ton of burnt lime	thous. and B.t.u.	9,705

1 Material received and crushed by the primary crusher at the lime and flux plant for the year ending March 31, 1958.

2 The limekiln was operated for 1,676.92 hours during the year ending on March 31, 1958.

Smelter

Much of the description of the smelting process is abstracted from a paper[18] by R. C. Wilson, smelter superintendent.

Plant facilities at the smelter were designed to produce approximately 12 million pounds of copper each month. The selection of equipment and its general arrangement were based on well-established smelter practices, modified only to meet requirements unique to conditions at San Manuel. The absence of impurities in the ore body and the slight variation of copper content in the concentrate make the smelting metallurgy relatively simple.

18 Wilson, R. C., The San Manuel Smelter: Proc. Ariz. Section Meeting, AIME, November 1957, 13 pp.

Concentrate Handling and Storage

Partly dried concentrate, containing 9 to 10 percent water, is conveyed from the mill to a receiving bin with a capacity of 3,000 tons. Six belt feeders under the bin discharge the concentrate onto a 24-inch conveyor belt for transportation to the main smelter building; there it is charged into the reverberatory furnace.

Fluxes

Limerock is the only flux added to the concentrate; it is fed onto the same belt that conveys the concentrate to the reverberatory. The flux is added in an amount equal to 7 percent of the concentrate. About 3 tons of quartzite a day is used for fettling of the reverberatory burner wall.

Converter flux is a mixture of sulfide ore and quartzite proportioned to contain 70 to 72 percent silica. It is conveyed from the flux plant to the smelter, where it is stored in receiving bins above and behind the converters.

Reverberatory Furnace

Concentrates are not roasted but are smelted directly in the reverberatory furnace. The prepared charge is received in two hoppers with 500-ton capacities. The hoppers are on each side and above the furnace at the uptake end. The hoppers discharge out individual tripper conveyors that discharge into 24 fettling hoppers on each side of the furnace. Fettling hoppers are not used for storage but to direct the charge into the furnace and to collect the dribble from the return side of the conveyor, thus reducing the accumulation of dust on the furnace roof.

The reverberatory furnace (fig. 98) is 32 by 102 feet inside the brickwork in the combustion zone; the bottom of the roof is 11 feet above the hearth line. About 40 feet from the burner end, the roof slopes downward until it is 9 feet above the hearth line; this dimension is maintained to the uptake end. The walls of the crucible are 4 feet thick and are corbeled back to an 18-inch-thick sidewall. In the smelting zone, or the first 72 feet from the burners, the sidewalls are topped with steel cans filled with magnesite grains. The cans are 27 inches high, 20 inches wide, and 8 inches thick. They form the junction between the top of the sidewall and the furnace roof. The cans facilitate repair to the sidewall at places of rapid deterioration.

The interior face of the crucible and sidewalls is lined with steel-encased basic brick. The outer veneer of the walls is made of silica brick. The entire roof of the furnace and sidewalls of the uptake are of suspended basic tile.

Water jackets, made of copper, are used at the slag line beneath the burners and around the settling zone. They are not used where walls are protected by charge piles.

Matte can be drawn from the furnace through two tapholes on each side of the furnace at the uptake end. Matte ladles are transported to the converter

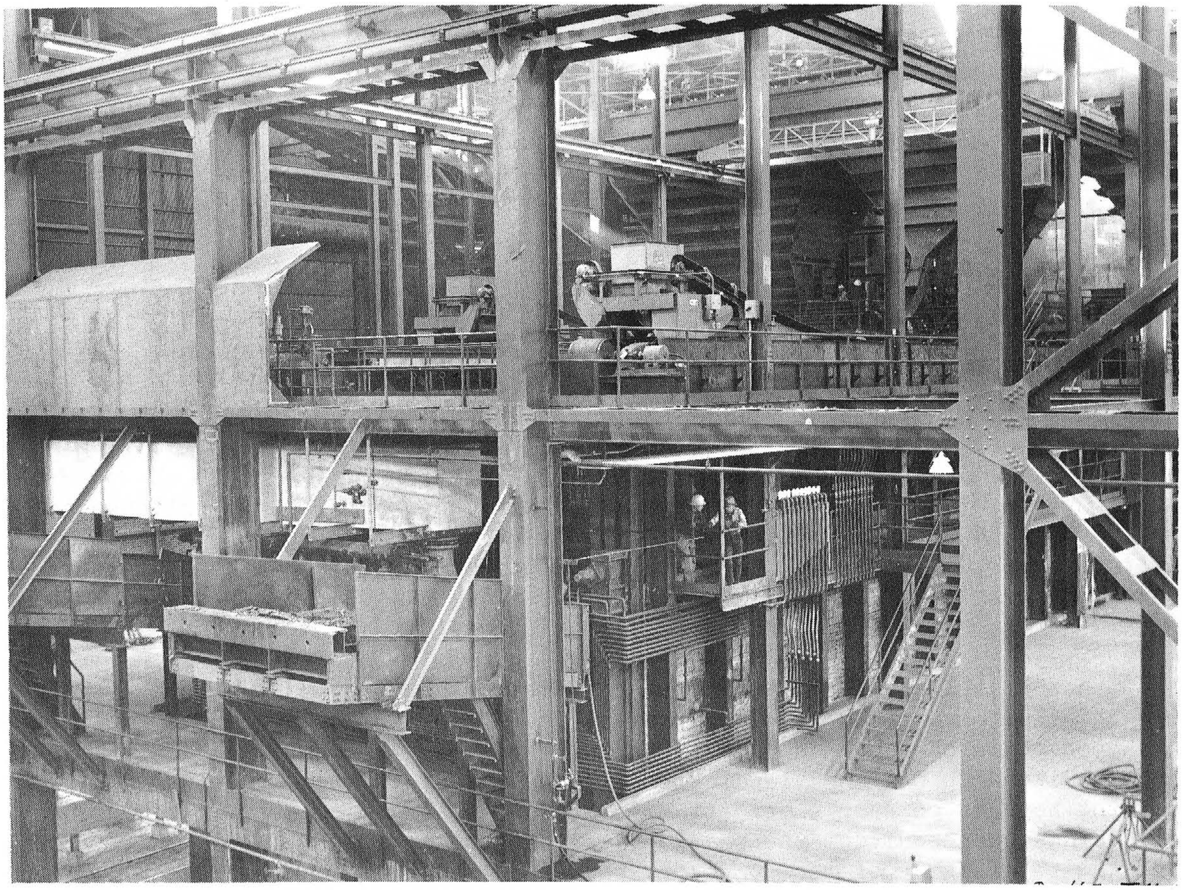

FIGURE 98. - Reverberatory Furnace.

aisle on electrically operated cars that move on standard-gage track along each side of the furnace. Slag can be skimmed from either side of the furnace. It is transported to the slag pile in four slag cars, with 225 cubic-foot capacities, that are pulled by a 25-ton diesel-electric locomotive. Converter slag is returned to the reverberatory furnace through two launders in the burner wall.

The furnace is fired by natural gas through eight multiple-jet burners. Each burner has 84 ports and is operated at $7\frac{1}{2}$-p.s.i. gas pressure. Combustion air is admitted to the burners at $2\frac{1}{2}$-inch water pressure. The combustion air is preheated to 700° F. in a separately fired preheater.

Waste heat from the furnace is recovered as steam in two boilers. The boilers have no baffles, and the exhaust gases have a nearly straight passage from furnace burners to the boiler output flue. Steam is produced at 475 p.s.i.g. and 725° F. At the maximum smelting rate, the boilers will produce more than 100,000 pounds of steam an hour. The steam is used to generate electrical power. The boilers can be operated simultaneously, or all the waste gases can be diverted to either boiler without curtailment of reverberatory operations materially below the rated capacity.

A control center in the smelter regulates combustion conditions and draft on the reverberatory furnace and air preheater. Boiler operations, including automatic soot blowing, are controlled at this point.

The reverberatory furnace is supported on a foundation that rises 10 feet 5 inches and extends 5 feet beneath the converter aisle gradeline. The foundation consists of side walls, end walls, and a bottom, which form a tank of reinforced concrete with outside dimensions of 39 and 119 feet. The interior of the tank is filled with tamped clay. The refractory walls of the furnace rest on the concrete walls of the foundation. Magnetite sand was tamped between the refractory walls and above the clay base to a depth of 5 feet. This formed the subhearth of the furnace.

Cooling tubes pass horizontally through the foundation from sidewall to sidewall. The tubes are 6 inches in diameter and are placed on 18-inch centers, 6 inches below the top of the clay fill. They are open at both ends to allow heat transfer from the bottom by natural convection. Bottom temperatures are recorded in the control room by six thermocouples spaced along the centerline of the furnace. If any of the thermocouples indicate bottom failure by showing a sudden rise in temperature, cooling air can be blown through the tubes in the danger zone.

The formation of the operating hearth level required 18 days. The air preheater was started, and the temperature was gradually increased for 5 days until 700° F. was reached. This temperature was maintained for 5 days to allow the foundation and brickwork to dry. Then, two main burners were ignited and operated at the minimum firing rate. The firing rate was gradually increased until, at the end of the third day, all eight burners were operating. The exit-gas temperature had risen to 2,500° F., and the temperature of the arch at the middle of the furnace was 2,650° F. At this time, 250 tons of converter slag, blown to contain a maximum amount of magnetite, was introduced into the furnace through the charge holes. The slag melted rapidly and flowed over the magnetite sand, forming a pool about 6 inches deep. The firing rate then was reduced gradually until the temperature at the uptake and wall was 1,800° F. This temperature was maintained for 8 days to allow the slag to cool slowly and anneal with a dense texture. A few contraction cracks appeared during the annealing period. The cracks closed and healed themselves when the temperature was raised again prior to the introduction of the first charge of concentrates and flux. Active smelting began on January 1, 1956, and the first copper was cast on January 8, 1956.

Deep-bath smelting is practiced; the bath is maintained at depths between 37 and 40 inches. Soundings have indicated that there has been no bottom buildup above the normal hearth line at the uptake end of the furnace. Experience has shown that bottom buildup at the firing end can be melted out by regulating the temperature of the preheated air; however, excessive temperature then occurs in the fettling chutes.

Clay for stopping reverberatory skim bays, ladle lips, and converter-lip repairs is obtained locally.

Converters

Three converters are situated on one side of the converter aisle, which is apanned by two cranes 60 feet long. Each crane has a main hook with a hoisting capacity of 60 tons and two auxiliary hooks rated at 25 tons. The cabs of the cranes are air conditioned. Matte and slag ladles used in the aisle have 200-cubic-foot capacities (fig. 99).

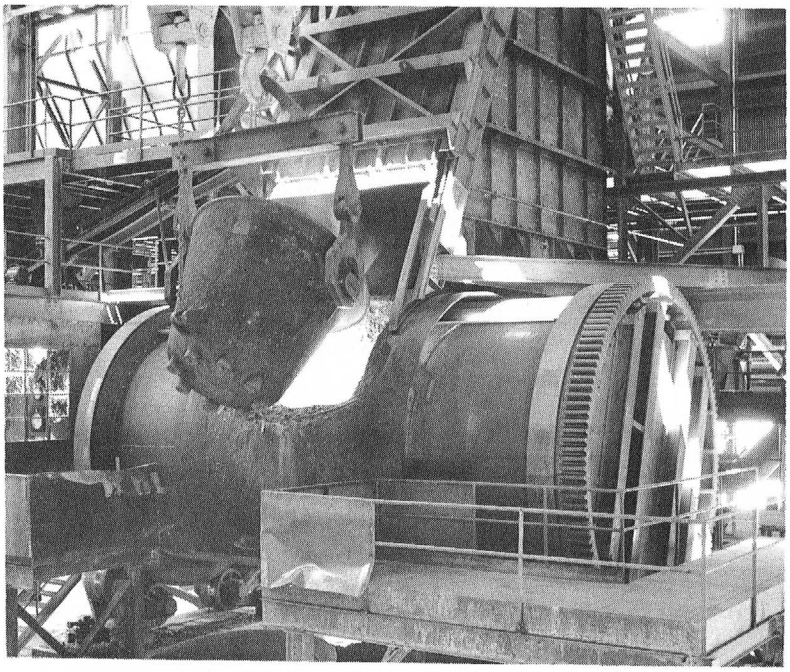

FIGURE 99. - Matte Being Ladled Into Converter.

The converters are 13 feet in diameter and 30 feet long. Each converter is equipped with 48 tuyeres made of 2-inch tubing with an inside diameter of 1.72 inches. Blast air is supplied by two blowers, each of which can produce 30,000 c.f.m. at 15 pounds pressure. Blast air can be delivered to any of the three converters by either blower. During normal maximum operations, when two converters are used, each converter has a separate blower. The blowers are located in the powerhouse and are controlled from the converter platforms. Each converter platform is equipped with a panel for the control of air pressure; idle and blow; the blast gate valve; and the uptake damper, hood door, and flux conveyors. The panels also contain an air flow indicator, an air failure alarm, and a converter rotation control. An air pressure gage and temperature recorder are mounted beside the panel. The temperature gage is operated from an optical pyrometer that sights through the uptake onto the bath. A high-temperature alarm from the optical recorder is located in the foreman's office. An airflow indicator for the convenience of the punchers is mounted in back of the converter. Airflow also is recorded in the foreman's office.

Converter flux is introduced through the side of the uptake by a conveyor belt and chute. Platforms on which punchers work can be adjusted to the desired height and are raised or lowered by air cylinders.

Flue Dust

Gases from the converter are exhausted through an insulated balloon flue that joins the flue from the reverberatory. The flue from the reverberatory is behind the waste heat boilers. The mixed gases are treated in a nine-unit electrical precipitator before being discharged to the atmosphere through a 500-foot-high stack.

Converter dust that settles in the balloon flue immediately behind the converters is moved by screw conveyors to three storage hoppers. The dust is drawn into trucks and sent to the converter aisle, where it is smelted, or it can be sent to the concentrate bins for smelting in the reverberatory.

Other dusts from the balloon flues are conveyed to a central point, where they are joined by dusts from the boiler and Cottrell precipitator. The dusts are moistened with water and treated in a pugmill. After treatment, the dust is conveyed to the concentrate storage bins above the reverberatory. All movements of dust ahead of the pugmill are made by screw conveyors. About 700 pounds of dust are removed daily, by hand, from the boiler ashpits.

Anode Casting

Blister copper from the converters is received in a holding furnace 13 feet in diameter and 30 feet long. This furnace is a rotating, or converter-like furnace, fired by natural gas burners at each end. The receiving spout is located in the center and serves as a stack for gases that are exhausted to the atmosphere. Oxygen in the blister copper is reduced from 0.80 to 0.09 percent by oak poles fed through one end of the furnace. The poles are shipped from Texas. They are 28 feet long, and 42 and 10 inches in diameter at the butt and top ends, respectively. The poles are fed to the furnace by a small air-operated hoist. Six or seven poles are consumed when treating a batch of blister copper; one pole treats approximately 45 tons of copper. Visual inspection of cold specimens determines when the charge is ready for casting anodes.

Anodes, weighing 700 pounds, are made on a hydraulically driven casting wheel (fig. 100) which is 34 feet in diameter and carries 22 molds. Pulverized silica is used for the mold wash. The casting wheel is operated from an enclosed air-conditioned booth, where the wheelman controls the tilting of the furnace, the pouring spoon, and the movement of the wheel. The wheel is positioned for each pour by an automatic switch arrangement in the hub.

The anodes are removed from the wheel by air-operated tongs and are deposited in bosh tanks for cooling. An overhead crane removes the anodes from the bosh tank and deposits them on a rack holding 45 bars. When their capacity is reached, the racks are unloaded on storage docks. After inspection and trimming, the anodes are moved by forklift trucks to railroad cars for shipment to a refinery (fig. 101). A scale for determining the weight of shipments is located on the track adjacent to the loading dock.

Summary of Operations

The smelter operates 7 days a week and has smelted more than 1,000 tons of concentrate a day, when the material has been available. The labor force consists of 49 men on the A shift and 25 men each for the B and C shifts.

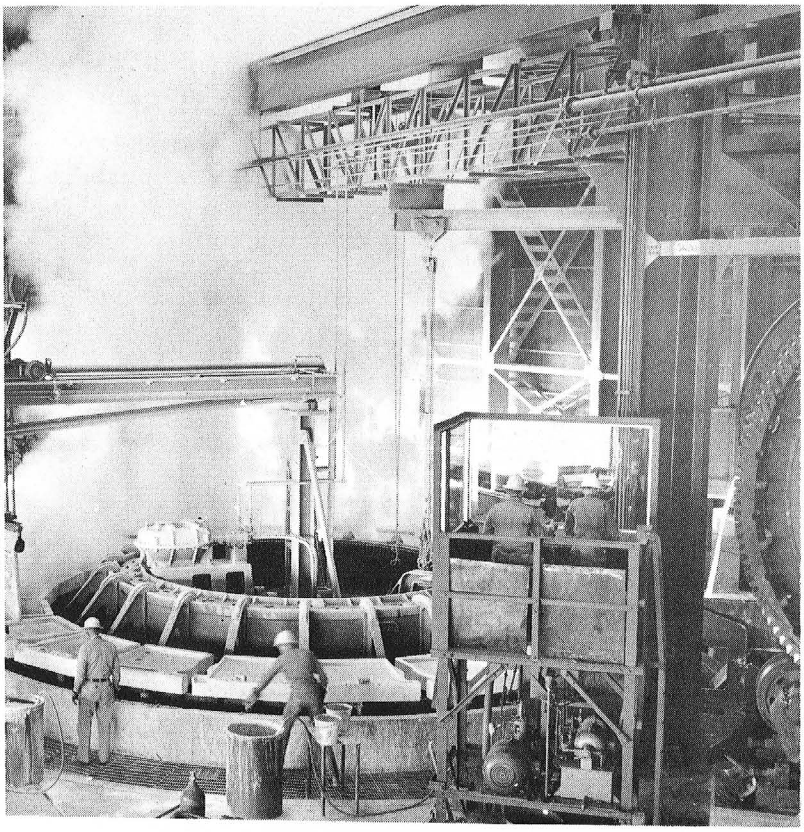

FIGURE 100. - Anode Casting Wheel and Furnace.

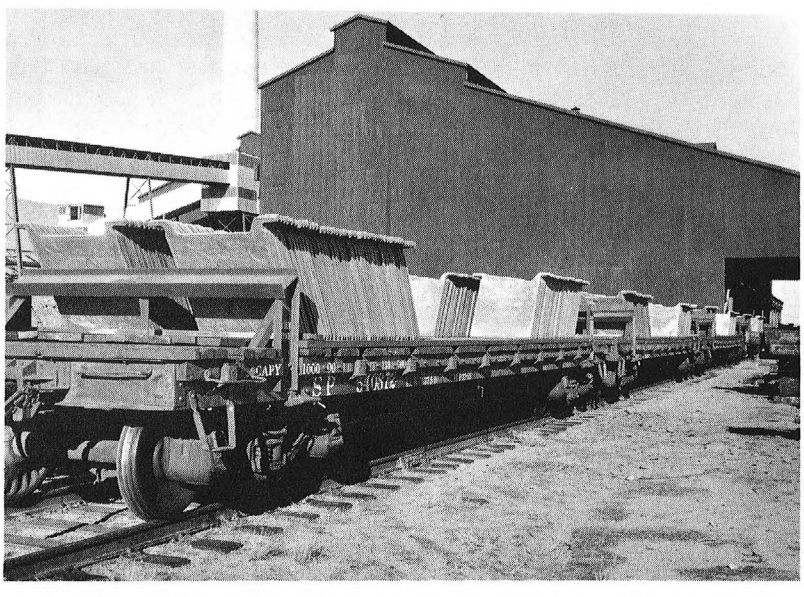

FIGURE 101. - Anodes Loaded for Shipment to Refinery.

Mechanical and electrical repairs are made by repair crews at the smelter. Repairs to large equipment are made in the machine shop. Brick masons, who are part of the operating crew, repair refractory linings of the converters. The maximum life of a converter lining is 24,000 tons of blister copper, and the average life is about 17,000 tons.

Tables 23, 24, and 25 give statistics about the reverberatory furnace, converter, and anode casting departments.

Machine Shop

A well-equipped machine shop is situated near the concentrator and smelter. The building is 90 by 300 feet and houses equipment for making major and minor repairs for the reduction plant. Some major repair work, such as turning down worn or flat wheels from locomotives and ore cars, is done for the railroad and the mine.

The layout of the equipment is shown in figure 102. The accompanying legend lists the equipment.

TABLE 23. - Reverberatory furnace department (statistics) (25 operating days in June 1959)

Concentrates smelted	tons	24,765
Limerock	do.	2,411
Silica (burner wall fettling)	do.	290
Flue dust	do.	580
Total solid charge	do.	28,046
B.t.u. per ton of concentrate (x 1,000,000)		5.7
B.t.u. per ton of solid charge (x 1,000,000)		5.1

Assay, percent

	Cu	SiO_2	Fe	CaO	Al_2O_3	S
Concentrate	28.2	6.2	28.8	0.6	2.0	30.0
Reverberatory slag	.40	36.4	33.4	5.5	7.1	--
Matte	32.0	--	37.9	--	--	30.0

TABLE 24. - Converter department (statistics) (50.7 operating days in June 1959)

Matte converted	tons	19,916
Flux	do.	7,021
Blister produced	do.	6,827
Blister per converter day	do.	135

Assay, percent

	Cu	SiO_2	Fe	CaO	Al_2O_3
Converter slag	1.6	27.5	44.9	--	--
Sulfide ore	.74	64.8	3.2	1.2	15.0
Quartzite	--	94.8	--	--	1.4

TABLE 25. - Anode department (statistics) (29 pours during June 1959)

Anodes cast	tons	6,709
No. of anodes cast		19,014
Average weight of anode	pounds	705.7
Rejected anodes	percent	.20
Average weight per hour	tons	231.3
Pouring rate per hour	do.	39.1
Copper refined per pole	do.	44
Copper content of anodes	percent	99.80
Anode per man-shift[1]	tons	1.97

[1] Total smelter operating crew plus repair men and electricians.

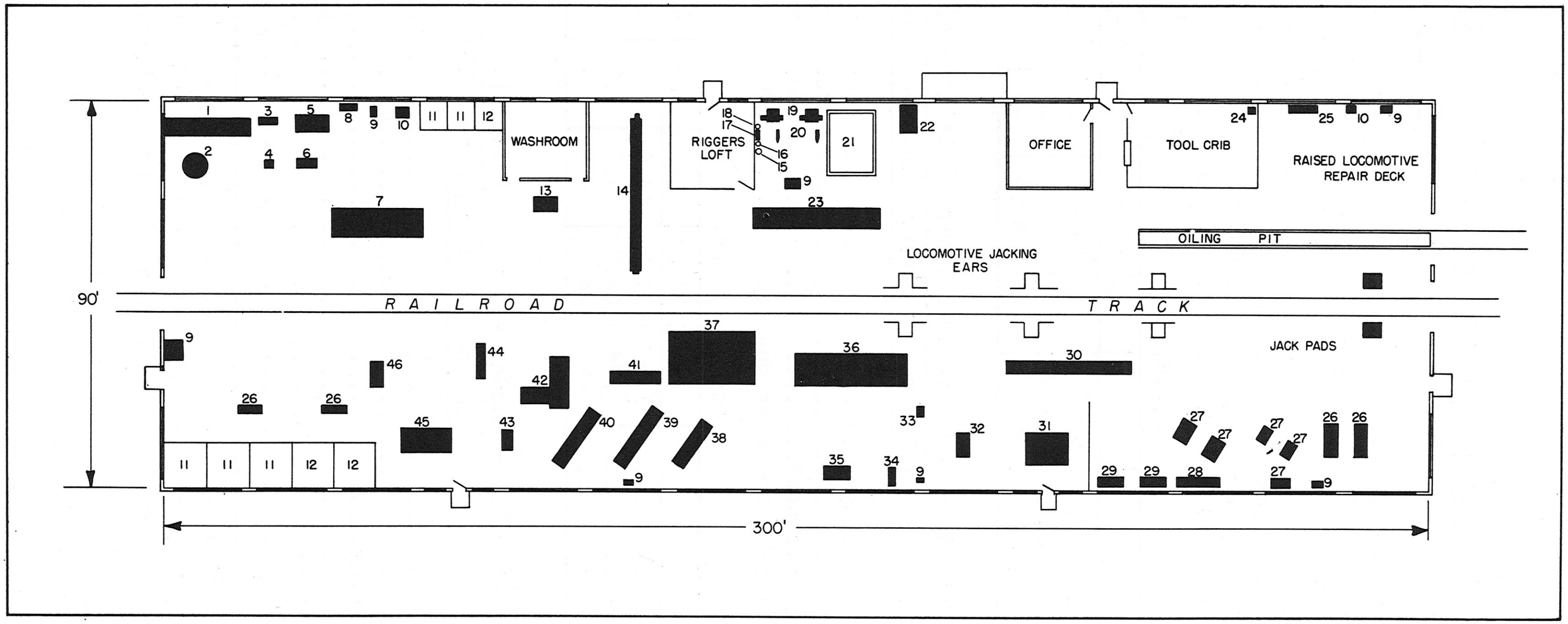

FIGURE 102. - Machine Shop Area.

Legend - Machine shop area

1. 4- by 20-foot workbench.
2. 24-inch punch.
3. Roll.
4. Shear.
5. Bend brake.
6. Squaring shear.
7. Pyramid bending rolls.
8. Gas furnace.
9. Grinder.
10. Drill press.
11. Welding.
12. Brazing.
13. Bolt-threading machine.
14. 32-inch by 16-foot shaper planer.
15. Tempering oil.
16. Salt brine.
17. Tool furnace.
18. Cyanide bath - portable.
19. Forge.
20. Anvil.
21. Forging hammer.
22. Furnace.
23. 600-ton press.
24. Drill grinder.
25. 13-inch by 4-foot lathe.
26. Workbenches.
27. Pipe machines.
28. Lockers for tools.
29. Welding tables.
30. 42-inch lathe.
31. No. 5 milling machine.
32. 24-inch shaper.
33. Bandsaw.
34. Keyseater.
35. 60-ton press.
36. 28/50-inch by 18-foot bed gap lathe.
37. 250- by 150-inch boring-mill area.
38. 16-inch by 8-foot lathe.
39. 16-inch by 12-foot lathe.
40. 28½-inch by 10-foot lathe.
41. Boring bar.
42. 7-foot radial drill.
43. Bandsaw.
44. Metal saw.
45. Punch and shear.

BIBLIOGRAPHY

PETERSON, N. P. Geology and Ore Deposits of the Mammoth Mining Camp Area, Pinal County, Arizona. Arizona Bureau of Mines Bull. 144, 1938, 63 pp.

_____. Some Arizona Ore Deposits. Part 2, Mining Districts, Mammoth Mining Camp Area, Pinal County, Arizona. Arizona Bureau of Mines Bull. 145, 1938, pp. 124-127.

SCHWARTZ, G. M. Geology of the San Manuel Area, Pinal County, Arizona. Geol. Survey Strategic Minerals Investigation, Preliminary Map 3-180, 1945.

CHAPMAN, T. L. San Manuel Copper D posits, Pinal County, Ariz. Bureau of Mines Rept. of Investigations 4108, 1947, 93 pp.

STEELE, H. J., AND RUBLY, G. R. San Manuel Prospect. AIME Tech. Pub. 2255, 1947, 12 pp.

LOVERING, T. S. Geothermal Gradients, Recent Climatic Changes, and Rate of Sulfide Oxidation in the San Manuel District, Arizona. Econ. Geol., vol. 43, 1948, pp. 1-20.

GOSS, W. P. San Manuel Copper Corporation. Eng. and Min. Jour., vol. 150, No. 7, 1949, pp. 92-95.

SCHWARTZ, G. M. Oxidation and Enrichment in the San Manuel Copper Deposit, Arizona. Econ. Geol., vol. 44, 1949, pp. 253-277.

CREASEY, S. C. Geology of the St. Anthony (Mammoth) Area, Pinal County, Ariz. Arizona Bureau of Mines Bull. 156, 1950, pp. 63-84.

LOVERING, T. S., HUFF, L. C., AND ALMOND, H. Dispersion of Copper From San Manuel Copper Deposit, Pinal County, Ariz. Econ. Geol., vol. 45, No. 6, 1950, pp. 493-514.

SCHWARTZ, G. M. Geology of the San Manuel Copper Deposit, Arizona. Geol. Survey Prof. Paper 256, 1953, 65 pp.

PILLAR, C. L. Progress on Three Big Shafts Reveals Up-to Date Sinking Practice. Min. Eng., vol. 6, No. 7, July 1954, pp. 688-695.

KNOERR, A. W. San Manuel -- America's Newest Large Copper Producer. Eng. and Min. Jour., vol. 157, April 1956, p. 77.

WILSON, E. D. Geologic Factors Related to Block Caving at San Manuel Copper Mine, Pinal County, Ariz. Progress Report, April 1954-March 1956. Bureau of Mines Rept. of Investigations 5336, 1957, 78 pp.

GRISWOLD, G. B. A Study of Subsidence at the San Manuel Mine, Tiger, Arizona. Univ. of Arizona M.S. Thesis, 1957, 86 pp.

PELLETIER, J. D. Geology of the San Manuel Mine. Min. Eng., vol. 9, No. 7, 1957, pp. 760-762.

WILSON, R. C. The San Manuel Smelter. Proc. Arizona Section Meeting, AIME, November 1957, 13 pp.

PARSONS, A. B. The Porphyry Coppers in 1956: AIME, 1957, pp. 244-256.

McLEHANEY, J. D., JR. A Study of Subsidence Due to Block Caving, San Manuel Mine, Pinal County, Ariz. Univ. of Arizona M.S. Thesis, 1958, 56 pp.

CIGLIANA, C. F. Ore Transportation at San Manuel. Min. Eng., vol. 10, No. 5, May 1958, pp. 573-576.

GIVEN, E. V. Milling Methods at the Concentrator of the San Manuel Corp. Proc. Arizona Section Meeting, AIME, San Manuel, Ariz., May 1958, 45 pp.

WARD, M. H. Underground Concreting at San Manuel Mine, San Manuel, Arizona. Proc. Arizona Section Meeting, AIME, Tucson, Ariz., December 1958, 11 pp.

GIVEN, E. V. Design Requirements for Tailing Disposal in the Southwest. Proc. Ann. Meeting AIME, San Francisco, Calif., February 1959, 14 pp.

PILLAR, C. L. Mine Communication System at San Manuel. Proc. Ann. Metting AIME, San Francisco, Calif., February 1959, 9 pp.

______. The Placement and Use of Concrete Underground at the San Manuel Mine, San Manuel, Ariz. Proc. Am. Min. Cong., Denver, Colo., September 1959, 15 pp.

WILSON, E. D. Geologic Factors Related to Block Caving at San Manuel Copper Mine, Pinal County, Ariz. Progress Report, April 1956-March 1958. Bureau of Mines Rept. of Investigations 5561, 1960, 43 pp.

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