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BLOCK-CAVING COPPER MINING METHODS AND COSTS AT THE MIAMI MINE, MIAMI COPPER COMPANY, GILA COUNTY, ARIZ.

By W. R. Hardwick

UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

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BLOCK-CAVING COPPER MINING METHODS AND COSTS AT THE MIAMI MINE, MIAMI COPPER COMPANY, GILA COUNTY, ARIZ.

by

W. R. Hardwick[1]

ABSTRACT

This Information Circular describes the development of block-caving methods and practices by the Miami Copper Co. at the Miami mine in Gila County, Arizona. It is one of a series published by the Bureau of Mines on mining methods, practices, and costs in various mining districts of the United States.

This report gives a history of the district and outlines early prospecting and exploration including methods of sampling and estimation of ore tonnage and value at the Miami mine. Early mining methods are described, particularly those that influenced developments in the caving method. Mine exploration, development, and operating methods for block caving are described with particular attention to those factors, physical, economic, engineering, and managerial that have improved efficiency in the mining operations over a long productive period. Extraction, ventilation, wage system, safety, water supply, power plant, and shop facilities are discussed. The last section gives a brief summary of such costs as are available for publication.

INTRODUCTION

A wealth of information about the underground mining operations at the Miami mine is scattered through technical literature, but a comprehensive account of the different mining methods used and the role of the earlier methods in developing block-caving techniques has not been published. The purpose of this paper is to describe the noteworthy Miami block-caving system and its development, including earlier mining methods. Many important modifications and improvements made during 60 years of operation before suspension of underground mining in 1959 are described.

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[1] Mining engineer, Bureau of Mines, Area V, Tucson, Ariz.

Work on manuscript completed December 1964.

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Acknowledgment is made to the executives and staff of Miami Copper Co. for permission to publish this paper. Special acknowledgment is due B. R. Coil, vice president and general manager, J. B. Fletcher, chief engineer, E. G. Williams, mine superintendent, and W. F. Sloan, assistant mine superintendent.

The mine was visited and information for this report was collected in early 1958. Annual reports of the Miami Copper Co. were available, and information from published articles has been used and is acknowledged by footnotes to the text. Maps, plans, and photographs for this paper were furnished by the company unless otherwise indicated.

LOCATION, PHYSICAL FEATURES AND CLIMATE

The Miami mine (fig. 1) is in the western part of the Globe-Miami mining district, T 1 N, R 14 E, Gila and Salt River base and meridian, Gila County, Arizona (fig. 2). The nearby town of Miami extends along Bloody Tanks Wash about 6 miles west of Globe, the county seat. U.S. Highway 60-70 forms the main street of the town. Many other copper mines have operated in the district (fig. 3).

The topography of the area near the Miami mine is characterized by a succession of high ridges with no apparent regularity of form or arrangement, sometimes exceedingly rugged in detail. Altitudes range from 3,300 feet in Miami Wash to 4,000 feet along Inspiration Ridge. Surface drainage is easterly into Russell Gulch, thence northerly via Miami Wash into Pinal Creek. There is no permanent surface water in the area, but enough water for domestic and industrial use has been developed at depth by mine workings east of Pinal Creek and deep wells along Miami Wash.

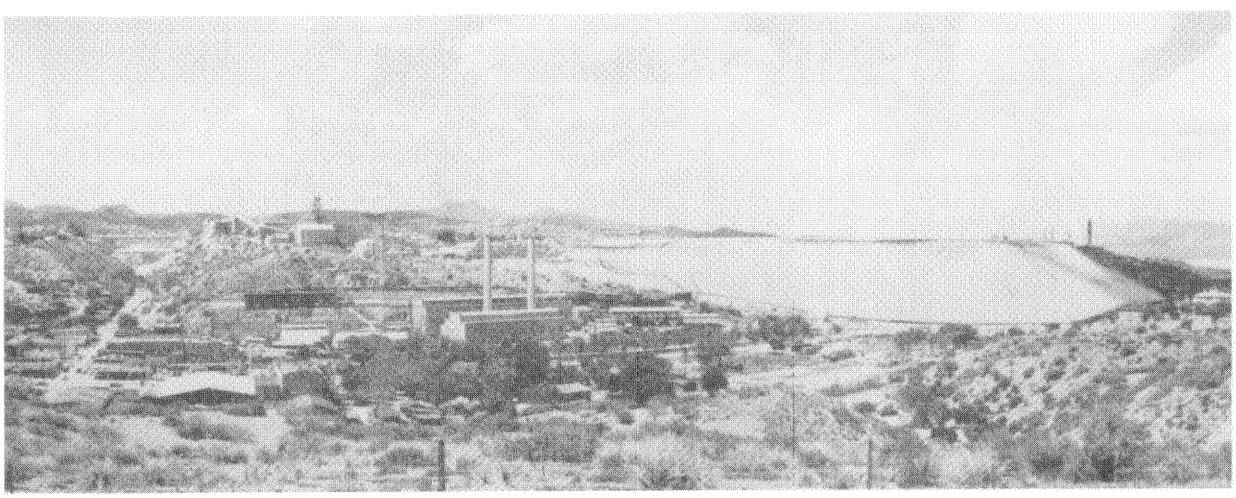

FIGURE 1. - The Miami Mine, Gila County, Ariz.

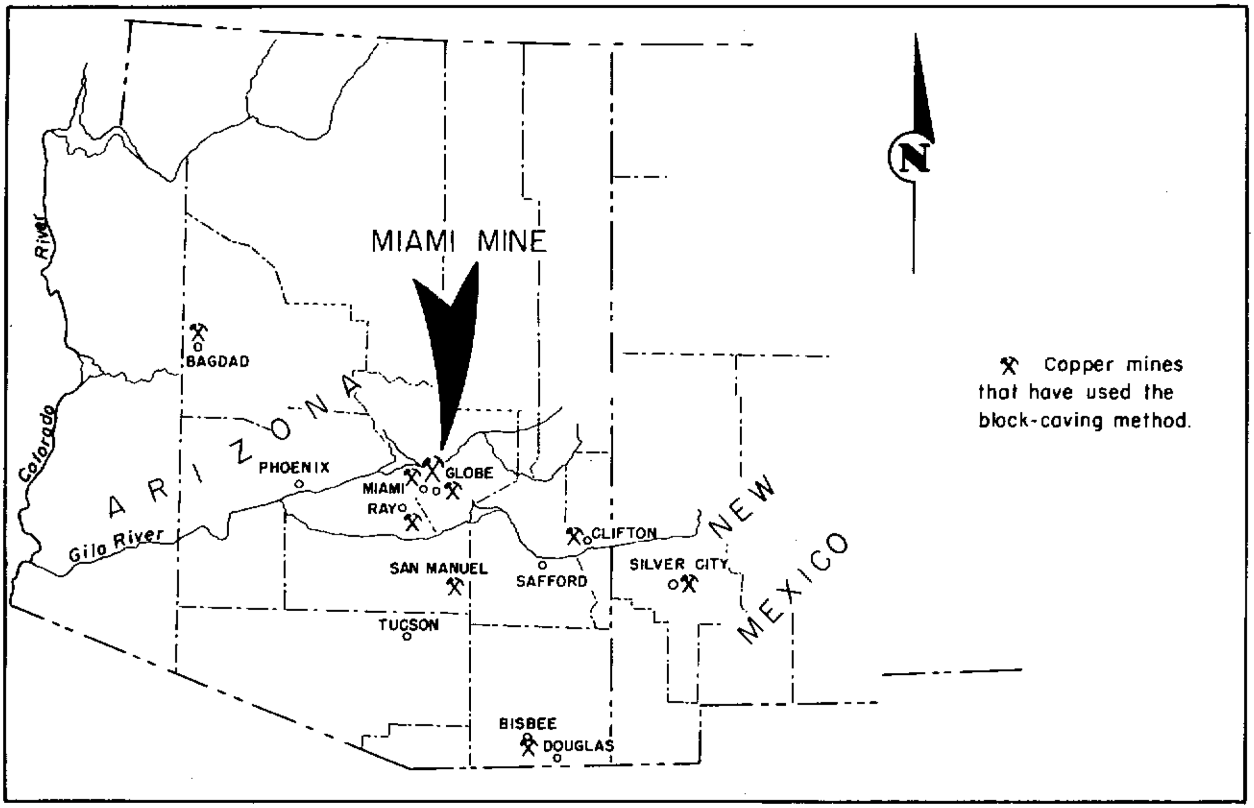

FIGURE 2. - Location of Miami Block-Caving Mine, Gila County, Ariz.

Near the Miami mine, annual precipitation of about 20.31 inches occurs chiefly as violent electrical storms in July and August. The mean annual snowfall, about 3.6 inches, occurs in December, January and February.[2] Temperatures range from 13° F in winter to 110° F in summer.

HISTORY AND PRODUCTION

The first mining claims in the Globe-Miami district were located in 1872 by a party of prospectors from Florence, Ariz., the nearest permanent settlement at that time. Copper lodes were found near Globe, but the hope of finding silver was the incentive for early exploration of the area. Eventually the large copper deposits were recognized and developed.

In 1873, rich silver veins were discovered at McMillanville, 12 miles north of Globe. Bamboz Camp was founded in 1875, and other settlements, Cottonwood Springs, Richmond Basin, and Watsonville, served mines that furnished high-grade silver ore from the oxidized portions of the veins in the Globe Hills and the nearby Apache Mountains. About 1878, Globe became the principal settlement. Early silver mining reached a peak in 1883, with 12 mills reported to be working in the area, then declined and finally ceased late in 1887, when the last operation, the Fame mine, was closed.

[2] Smith H. V. The Climate of Arizona. Univ. of Ariz. Agricultural Exp. Sta. Bull. 197, July 1945, p. 98.

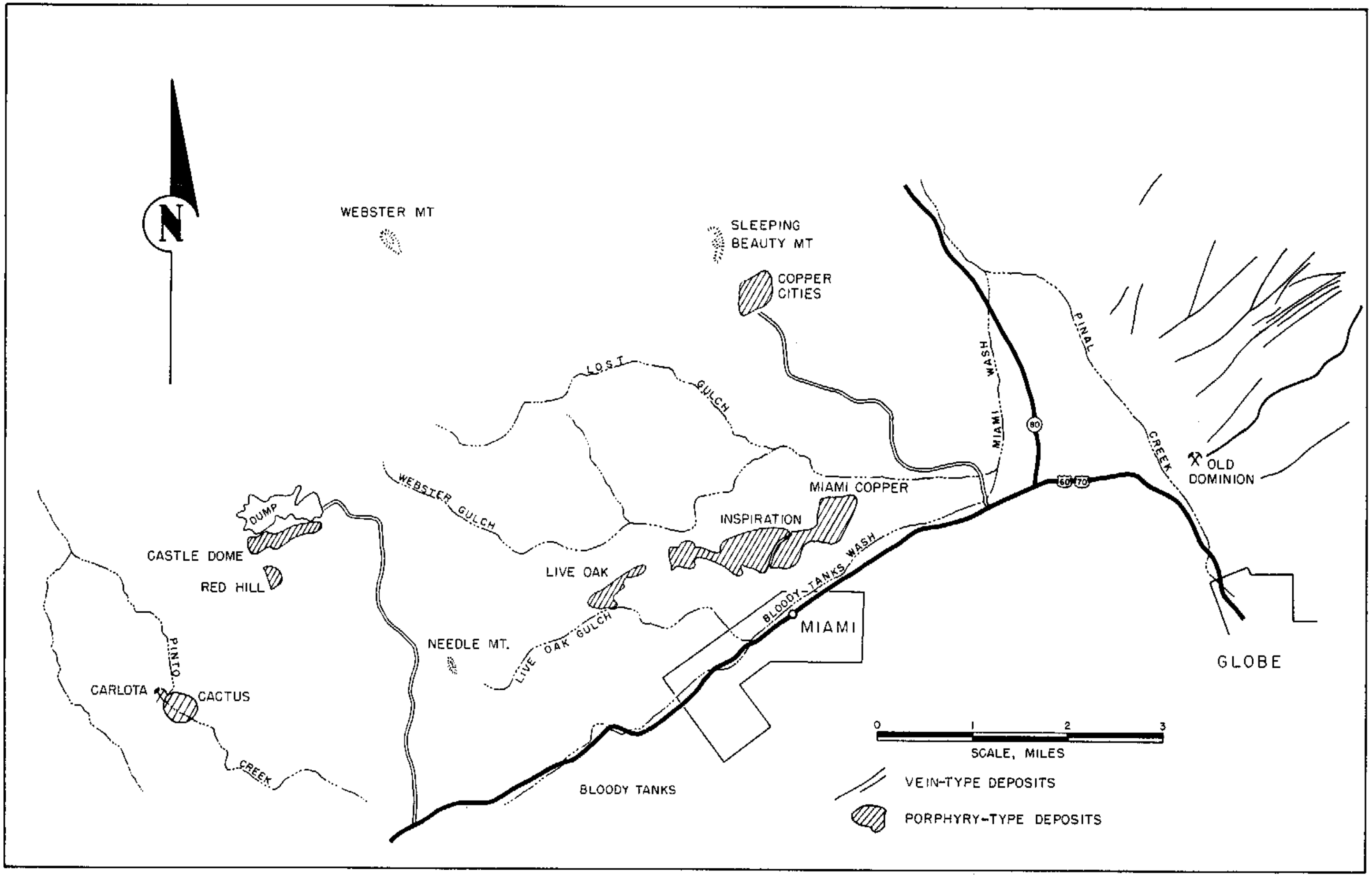

FIGURE 3. - Mines in the Miami Area, Gila County, Ariz.

After 1881, several smelters were built to treat the rich chalcocite and carbonate ore from the fissure-veins of the Globe area. By 1905 much of the high-grade, direct-smelting ore had been mined and the first copper concentrating plant began treating lower grade ore at the Old Dominion operation.[3] When this mine was closed in 1931 after some 49 years of operation, the district had produced metals valued at $166 million.[4] Since 1931, only minor production by lessees has been made on the vein deposits east of Pinal Creek.

The successful early exploration of the small silver mines near Globe had encouraged exploration of the mineralized outcrops in the Miami area 8 to 10 miles west of Globe. Copper carbonate and silicate minerals were recognized in Pinal schist near its contact with Schultze granite in the vicinity of Live Oak Gulch and on Pinto Creek near the mouth of Cottonwood Gulch. The first mining attempts were not too successful. In 1881, a pocket of ore mined in the schist west of the Miami mine was smelted in a small copper furnace built by the Old Dominion Co. 6 miles west of Globe and one-half of a mile northeast of Bloody Tanks. This plant was moved to Globe after the pocket was exhausted.

In 1901, the Keystone mine, west of the present Miami mine, produced chrysocolla ore from a fissure in porphyry above the Pinal schist. Mr. J. Parke Channing, an engineer for the General Development Co., investigated the area in 1904, and, in 1905 and 1906, acquired options on most of the ground later included in the Miami Copper Co.'s block-caving mine and started systematic exploration. A shaft on the Red Rock claim penetrated sulfide copper ore at a depth of 220 feet in 1906, and the Miami Copper Co. was incorporated in 1907 to develop and mine the ore. Mine development was started in 1910, followed by production in 1911.

The railroad was extended from Globe to the town of Miami in 1909, and by April 1911, the first three sections of a concentrating plant were in operation. In the first year 445,036 tons of ore with an average copper content of 2.48 percent was milled. The first concentrates were sent to a Globe smelter,[5] but in May 1915 a smelter was completed by the International Smelting Co. near the Miami mine to smelt concentrates from both Miami and Inspiration Consolidated Copper Properties.[6] After the Old Dominion smelter was closed in 1931, this was the only operating smelter in the district. It was acquired by the Inspiration Consolidated Copper Co. in 1960 and is now operated as an integrated unit by that company.

The capacity of the Miami concentrator, 2,000 tons per day at the start, eventually reached 18,000 tons. In 1956, the Miami mine produced 3,812,165 tons of ore that averaged 0.67 percent copper, about 12,000 tons per operating

[3] Forrester, D. J., and W. B. Cramer. Milling Methods and Costs at the Concentrator of the Old Dominion Co., Globe, Ariz. BuMines Inf. Circ. 6467, 1931, 31 pp.

[4] Peterson, Nels P. Geology of the Globe Quadrangle, Arizona. U.S. Geol. Survey, Geologic Map GQ41, 1954.

[5] Hunt, H. D. Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz. BuMines Inf. Circ. 6573, 1932, 26 pp.

[6] Weed, Walter Harvey. The Mines Handbook. V. 13, 1918, W. H. Weed, New York, p. 265.

day. On the basis of successful recovery of molybdenum sulfide in pilot tests in 1937, byproduct molybdenum was saved and marketed thereafter as molybdenum trioxide. Production figures for typical years, 1945 through 1950, are shown in table 1.

TABLE 1. - Production of copper and molybdenum, 1945-50

Year	Ore milled (short tons)	Copper content (percent)	Copper produced (pounds)		Molybdenum trioxide produced (pounds)
			Total	Per ton of ore	
1945.........	4,003,000	0.644	43,537,000	10.9	619,000
1946.........	4,223,000	.734	52,485,000	12.4	697,000
1947.........	4,557,000	.695	51,216,000	11.2	534,000
1948.........	4,198,000	.679	46,922,000	11.2	386,000
1949.........	3,844,000	.735	47,304,000	12.3	503,000
1950.........	4,003,000	.667	44,331,000	11.1	627,000

The total production of copper from the Miami mine from inception to 1925 was about 446,000 tons.[7] Block caving was started at that time, and, by 1951 1,050,000 tons of copper had been produced from 123 million tons of ore. By July 1, 1959, when underground mining was terminated, production had totaled 1,186,500 tons of copper from about 153 million tons of ore. The tonnages mined by the several methods used are as follows:

Mining method	Short tons
Square set and top slicing.................	4,500,000
Shrinkage stoping with sublevel caving.....	2,200,000
Undercut caving with hand tramming.........	15,400,000
Block caving...............................	130,900,000

Solution mining, the in-place leaching of mined areas by flooding with acidified water, was introduced in 1940 and has been continued since. It has accounted for more than 30,000 tons of copper during 1940-56, and the mine continues to produce about 9,000 tons of copper per year by this method.[8]

Byproduct metal production has totaled more than 10 million pounds of molybdic trioxide, 4,427 ounces of gold, and 636,492 ounces of silver. A trace of rhenium was contained in the ore and tended to concentrate in the flue dust from the molybdenum plant; a little was recovered for use in experimental work. The gross value of all minerals produced from the Miami mine has been estimated to exceed $450 million.[9]

The production from the underground mine, tabulated by period and ore type, is as follows:

[7] Parsons, A. B. The Porphyry Coppers in 1956. AIME, New York, 1957, p. 97.
[8] Larson, L. P. The Mineral Industry of Arizona. BuMines Minerals Yearbook, v. III, 1961, p. 105; 1962, p. 106.
[9] Tuck, Frank J. Stories of Arizona Copper Mines. Arizona Department of Mineral Resources, Phoeniz, Ariz. 1957, p. 15.

Period	Tons of ore	Type
1910-25....................	24,200,000	High grade
1926-54....................	101,000,000	Low grade
1936-43....................	9,800,000	Mixed ore
1954-59....................	18,000,000	Low grade
Total.................	153,000,000	

In addition to the Miami mine, three other disseminated porphyry copper type deposits in the area have been developed: Inspiration, Castle Dome, and Copper Cities.

Formed by merger of the Inspiration Copper Co. and the Live Oak Development Co. in 1912, Inspiration Consolidated Copper Co. has developed and mined large copper ore bodies adjoining the Miami mine. Production, starting in 1913, has totaled more than 3 billion pounds of copper from 165 million tons of ore through January 1, 1960.[10] The mine continues to produce and has reserves for many years.

The Castle Dome mine, a porphyry copper deposit about 5 miles west of Miami, was purchased by Miami Copper Co. in 1941 and operated from 1943 to 1953 when it was closed. A total of 48,484,188 tons of waste was moved to mine 41,442,617 tons of copper ore with a recovery of about 12 pounds of copper per ton.[11]

After the Castle Dome mine was closed, Miami Copper Co. began operating the Copper Cities mine at 12,000 tons per day. This disseminated porphyry ore body is about 4 miles north of the Miami mine. Between 1954 and 1957, the mine produced 12,566,600 tons of ore after removing 31,613,000 tons of waste. It continues to produce at about 3 million tons of copper ore per year.

DESCRIPTION OF THE DEPOSITS

The Globe-Miami area (see fig. 3) was divided into two parts by Pinal Creek, a small stream that flows northeast from the Pinal Mountains and joins the Salt River. East of Pinal Creek the lodes were generally fissure veins. Hundreds were found in the region; many were small, some contained high-grade silver ore and some, for example the Old Dominion, were extensive and contained copper ore.

The ore bodies west of Pinal Creek were generally disseminated or porphyry type associated with igneous intrusions. The host rocks for the ore bodies of the Miami mine were Pinal schist and a porphyritic border facies of the Schultze granite stock. The schist and granite cropped out in the mine area where younger overlying formations were removed by erosion. A substantial thickness of the host rocks also was removed.

[10] Hardwick, W. R. Mining Methods and Costs, Inspiration Consolidated Copper Co. Open-Pit Mine, Gila County, Ariz. BuMines Inf. Circ. 8154, 1963, 65 pp.

[11] Hardwick, W. R., and M. M. Stover. Open-Pit Copper Mining Methods and Practices, Copper Cities Division, Miami Copper Co., Gila County, Ariz. BuMines Inf. Circ. 7985, 1960, p. 4.

Remnants of dacite flows and diabase sills were exposed north of the mine, and the ore-bearing formations were cut off on the southeast by the northeast-trending Miami fault that placed them in contact with the outcropping Gila conglomerate.

Precambrian sediments were metamorphosed by intense folding and recyrstallization accompanying extensive igneous activity to form Pinal schist, the basement rock of the region. The Schultze granite stock was intruded into the schist as were quartz monzonite and other igneous rocks. These belong to the period of plutonic activity and mountain building during which mineralization at Miami as well as other Arizona base-metal districts was believed to have occurred.

Pinal schist, Schultze granite, and Gila conglomerate remained near the Miami mine, and additional geologic history of the region was obtained by study of the structural relationships of the remnants of sedimentary and igneous formations in the general area. In the long period between the metamorphism of the Pinal schist and the plutonic activity of later times, a thick sequence of sediments was laid down on the basement rock over wide areas and later removed by erosion in parts of the Miami district. Thick dacite flows and as much as 2,000 feet of Quaternary Gila conglomerate, a continental sediment derived from erosion of local intrusives, covered the area and were subsequently removed over the Miami and Inspiration ore bodies by erosion.

The geology of the Miami district was described in detail by Ransome.[12] The general relationships of the host rocks and the ore bodies of the Miami mine are shown in figures 4 and 5.

Pinal schist was a light gray to blue-gray rock, sometimes coarsely crystalline, containing abundant quartz and sericite. The sericite often gave it a lustrous, silvery appearance. Schultze granite was a light-colored rock that weathered to a pale yellow. It was distinctly porphyritic near the periphery of the stock and exhibited this characteristic in the Miami ore body. It approached quartz monzonite in composition.

Both Pinal schist and Schultze granite contained many clay-filled fissures and a network of small quartz stringers where mineralized, and the ore was soft and friable. The granite tended to be harder than the schist, but both rocks broke up more or less easily when disturbed. Eight percent of the Miami ore as mined passed through a 10-inch grizzly. Broken or caved ore tended to pack when damp, particularly the softer varieties. This was a disadvantage in mining because the ore sometimes hung up in the stopes or ore passes.

The capping above the ore was leached schist or granite porphyry. In addition to undergoing hydrothermal decomposition by the primary mineralizing solutions, the leached protore was subjected to the action of strong sulfuric

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[12] Ransome, F. L. The Copper Deposits of Ray and Miami, Arizona. U.S. Geol. Survey Prof. Paper 115, 1919, 192 pp.

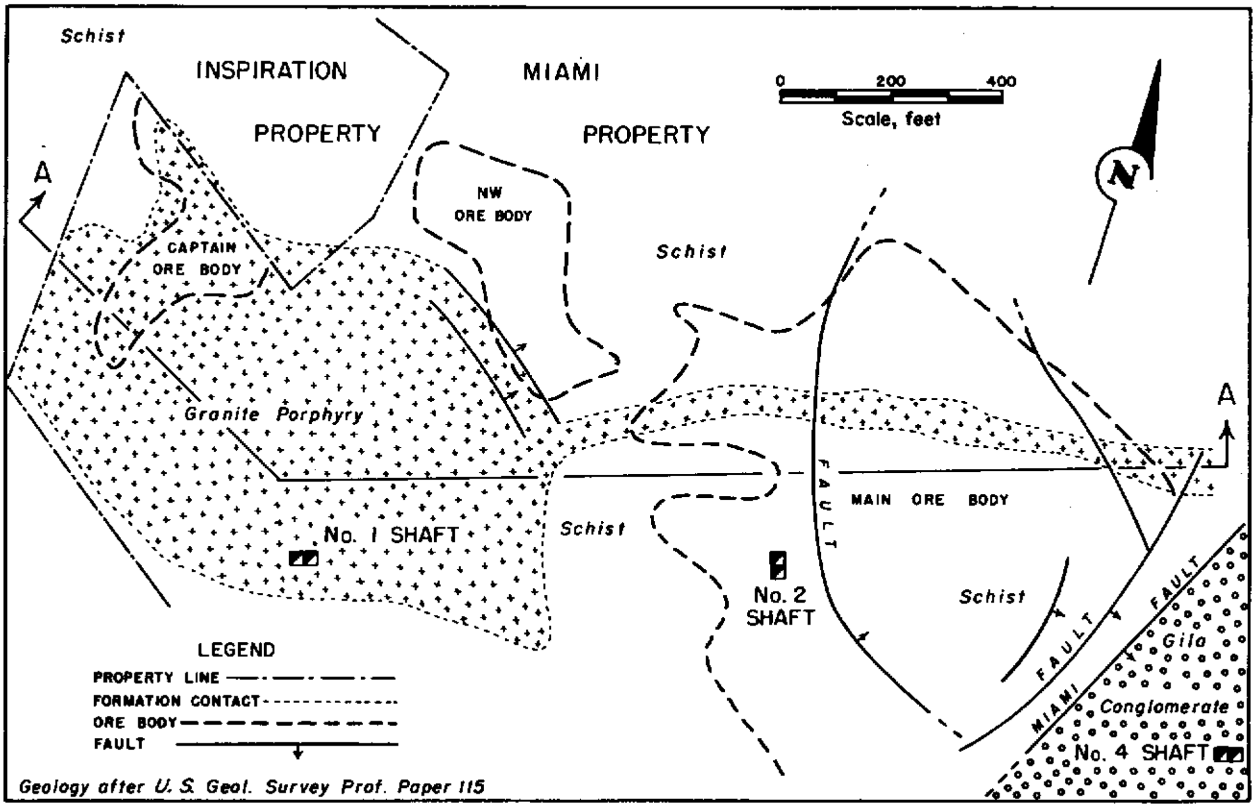

FIGURE 4. - Geologic Plan at the 420 Level, Miami Mine.

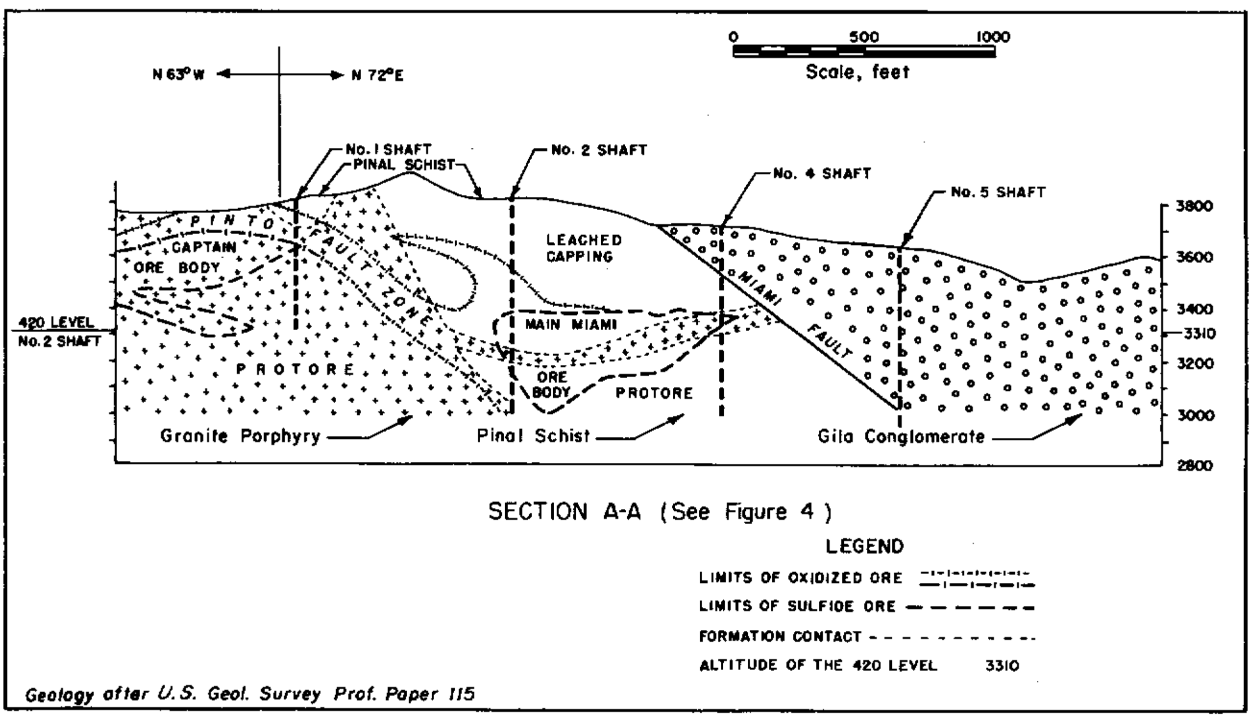

FIGURE 5. - Generalized Section of the Miami Mine.

acid solutions generated by oxidizing sulfide minerals during copper removal and redeposition in the zone of secondary enrichment below. The capping was converted into a rusty-brown, very friable, and unstable material that broke into fine particles under stress. This cap rock flowed more easily than ore and created a control problem in mining the ore below.

The mineralizing solutions that emplaced the metal sulfides in the Miami-Inspiration area were believed to have risen through channels in a structure identified with the contact between Pinal schist and the Schultze granite intrusion. The sulfides were deposited in permeable ground in both rocks. The primary metallic sulfides of the Miami protore are pyrite, chalcopyrite, and molybdenite deposited with quartz in small stringers and disseminated through the rock. Minable ore occurs only where the secondary enrichment had deposited additional copper minerals. The chief secondary sulfide was chalcocite, deposited in thin veinlets or as a coating on pyrite and chalcopyrite. The ore also contained small amounts of bornite and covellite.

Minor pyrite was disseminated in the ground mass of the schist protore, and pyrite with very small amounts of chalcopyrite and molybdenite was disseminated in granite porphyry ore.

The secondary sulfide ore had much the same appearance as the protore with some darkening caused by specks and veinlets of chalcocite introduced into the sulfide zone by downward-percolating copper-bearing solutions from the leached overlying protore. The upper portions of the larger Miami ore bodies and the mineralized Pinto fault zone contained various proportions of azurite, malachite, chyscolla, cuprite and native copper as well as sulfides. In the Pinto fault zone, the carbonates and silicates were believed to have resulted from partial reoxidation of secondary sulfides.

For all practical purposes, the Miami deposit consisted of three separate ore bodies (figs. 4 and 5). The Main Miami ore body was in Pinal schist. The Captain ore body in Schultze granite porphyry extended westward into the adjoining Inspiration mine. The Pinto ore body in schist lay between the other two and connected with both.

Post-mineral displacement along a series of northwesterly trending fissures in a fault zone roughly identified with the contact between schist and granite porphyry dropped the Pinto and Main Miami ore bodies in relation to the Captain ore body.

The ore bodies were irregular in shape. The outlines in figures 4, 5, and 6 illustrate the irregularity both laterally and vertically. The ore was cut off at some points by fault planes, but along other boundaries it blended gradually into the protore. Neither the upper nor lower limits of the sulfide zone were flat, and ore extended both in peaks into the cap rock above the main bodies and in pendants and troughs into the protore below.

In plan, the Miami ore body in fissile quartz-mica schist (fig. 6) was roughly triangular with a base of 3,700 feet and an altitude of 2,500 feet, maximum dimensions, and apex to the south. The average thickness was about

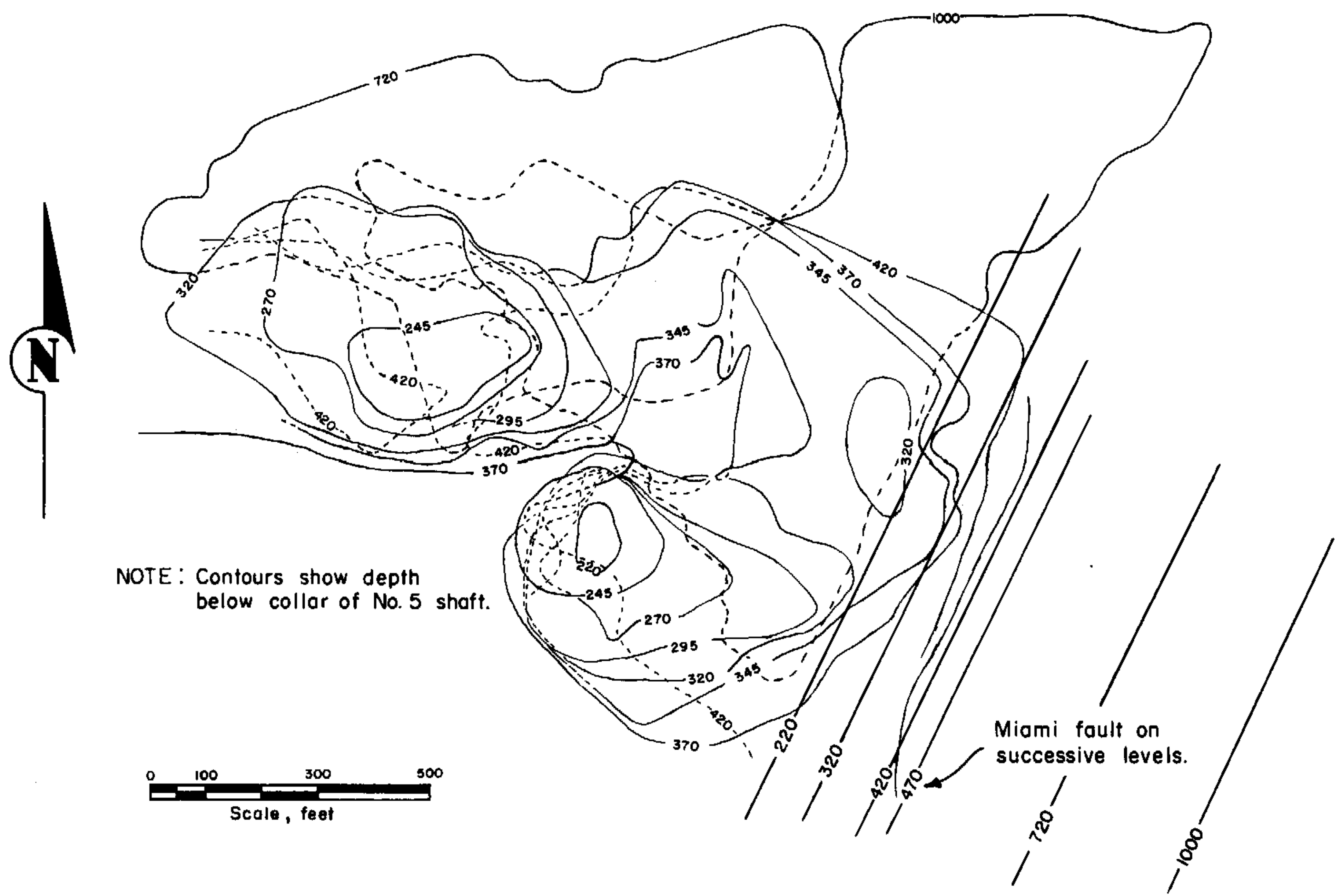

FIGURE 6. - Contours on Mine Levels Showing Approximate Shape of the Miami Ore Body.

350 feet. The top portion, sometimes called the northwest ore body (fig. 4), projected beyond the main body and was mined separately. Three peaks of good ore projected above the Main Miami ore body into the barren cap rock, as indicated by the contours in figure 6.

The Captain granite porphyry ore body was about 500 by 500 feet in horizontal dimensions and averaged 350 feet in thickness.

The Pinto schist ore body lying within the Pinto fault zone was a mineralized connecting link between the other two (fig. 5). It contained mixed (partially reoxidized) sulfide ore with good copper content that was mined separately.

The capping over the ore bodies ranged from 245 to 500 feet in thickness. In sections where it approached the maximum thickness, the higher ground pressures developed in the underlying mining operations increased support problems.

PROSPECTING AND EXPLORATION

Copper sulfide ore first was discovered by the General Development Co. in 1906 at a depth of 220 feet. Other discoveries followed rapidly, and in 1909 the irregular ore bodies were explored by churn drilling. Drill holes were put down at the corners of 200-foot squares to an average depth of 640 feet for the first 117 holes. By July 1912, a reserve of 20,800,000 tons ore averaging 2.48 percent copper has been outlined.[13]

After this first ore (called the high-grade ore body) was essentially mined out, the downward extension below the 720-foot level was explored. The original churn drill holes could not be deepened because the surface had caved. However, they provided worthwhile information because many had bottomed in 1.00 percent copper ore, considered the cutoff grade at the time they were drilled (the cutoff point was later reduced to 0.6 percent copper). Development planning was based in part on information obtained from this earlier drilling. A new, deeper level, called the 1000-foot level, was opened, and ore was explored by raising, drifting, and diamond drilling from both the 1000- and the 850-foot levels.

SAMPLING AND ESTIMATION OF TONNAGE AND VALUE

Ore reserves were divided into assumed ore and proven ore. Assumed ore potential was inferred from geological interpretations and from scattered workings or drill holes in which sampling results were not conclusive enough to confirm extent and grade of ore. Proven ore was estimated from the dimensions and grade of ore bodies as determined by information from all types of exploration work, including drifting, raising, diamond drilling, churn drilling, and percussion drilling.

Stope boundaries were defined by the results of sampling exploration openings driven to find the limits of ore zones. The boundary drifts used to outline stope blocks in the first use of block caving were an excellent source

[13] Page 97 of work cited in footnote 7.

of data for reserve calculations. After development by boundary drifts was abandoned, data from undercutting operations, although not as complete, was used for the same purpose.

Drifts were sampled by cutting continuous, waist-high, horizontal channels 1 to 2 inches deep and 4 to 5 inches wide in both walls. Material cut from opposite walls in 5-foot intervals was combined to form a sample representing that interval.

Sludge from churn-drill holes was sampled in 5-foot intervals. In most cases, only the sludge from diamond-drill holes was sampled. Core recovery was only about 15 percent, and the core was generally ground and added to the corresponding sludge.

Sludge from percussion-drill holes across major seams in the walls of development workings was sampled in 2½-foot intervals. Assay results were recorded on the drill logs made for each hole.

Occasional samples of some 6 to 8 tons were mined in 25-foot cutouts in the backs of development workings, and in one case, a 1,500-ton sample was mined from a narrow shrinkage stope and put through an automatic sampler. Samples of broken ore were taken as cars were loaded in development headings.

In grade calculations, car-sample assays were used as reported, as were churn- and percussion-drill results. Channel-sample assays were considered 13 percent high and diamond-drill sample results 10 percent high. Reported assays were corrected on this basis for grade estimation purposes.

All samples were plotted on plan maps and sections through the ore bodies drawn on a scale large enough, generally 50 feet per inch, to provide space for entering each individual assay. Data concerning geologic structure and location of inclusions of oxidized ore or waste as well as assay values were recorded on maps.

The vertical map sections were parallel and usually 50 feet apart, divided into blocks for convenience in calculation. Each section represented a 50-foot slice through the ore body, 25 feet on each side of the plane of the section. In drifts that crossed a section, the average assays for 25 feet on each side of the section were used as representative of the corresponding slice. Recorded assays were grouped in blocks to facilitate reserve calculations. The average grade of copper in a block was the summation of all the assays, including all adjusted results, divided by the number of samples. Ore tonnages were calculated by volume estimates based on the dimensions of ore bodies as established by development, using factor of 12½ cubic feet per ton for ore in place and 20 cubic feet per ton for broken ore. The indicated factor for the material packed in a mined-out stope was about 16 cubic feet per ton.

EARLY MINING METHODS

A number of different mining systems preceded block caving at the Miami mine. Some of the features of block caving as finally developed can be recognized as closely related to earlier methods. The management of the mine was outstanding in developing improved operating methods and adopting new equipment. More than 50 years of research and development by company engineering and operating personnel went into the design of the final mining method.

Square Set Stoping

The first mining system was square-set stoping and was used to mine the peaks of ore that extended above the Main Miami ore body into the cap rock. This costly method was feasible because the ore was relatively high in grade.

Wide Shrinkage Stope and Pillar Caving System

Mining by the wide shrinkage stope and pillar system was done in the upper part of the Captain ore body. The granite porphyry ore was relatively hard, and stopes 50 feet wide could be carried safely. A total of 2,300,000 tons of ore was mined by this method, as described by Scott.[14]

The following description of wide shrinkage stoping and pillar caving method is included in some detail because of the similarity in development with that of the later block-caving system. The tramming level, pocket raise, drawing off level, stope raise, and stope floor of the early system served purposes similar to the respective haulage level, transfer raise, grizzly control level, control raise and under cut level of the block-caving system. Furthermore, in the earlier system, ore was broken on the mining level in the 50-foot-wide stopes by drilling 15-foot horizontal blast holes from each edge toward the center and depending on the principal of caving to break the long rib left in the center of the stope.

Application of the system provided for alternating stopes and pillars across the short dimension of the ore body, each 50 feet wide and 200 to 500 feet in length, according to the width of the ore body (fig. 7). The height of stopes depended on the distance to the upper ore limit, a maximum of 125 feet. Necessary development consisted of a tramming level, drawing-off level, and sublevels with the required raise work to connect the several levels. Of these openings, the tramming and drawing-off levels and the pocket raises between were timbered. The level openings were supported with 10- by 10-inch sets with 2-inch lagging on 5- to 6-foot centers.

A new and distinctive feature of the method at the Miami mine was the drawing-off level, between the haulage level and the floor of the stopes used to handle all the ore broken in both stopes and pillars. Paired raises at 25-foot intervals were inclined up on both sides of the drawing-off drifts under the stopes and belled out to a diameter of 20 feet at the stope floor level. This left a pillar directly over the drawing-off drift. However, failure of the pillar often occurred during the life of the stope, and a drawing-off drift was protected more by timber support than by a pillar.

[14] Scott, David B. Stoping Methods of Miami Copper Co. Trans. AIME, v. 55, 1916, pp. 137-153.

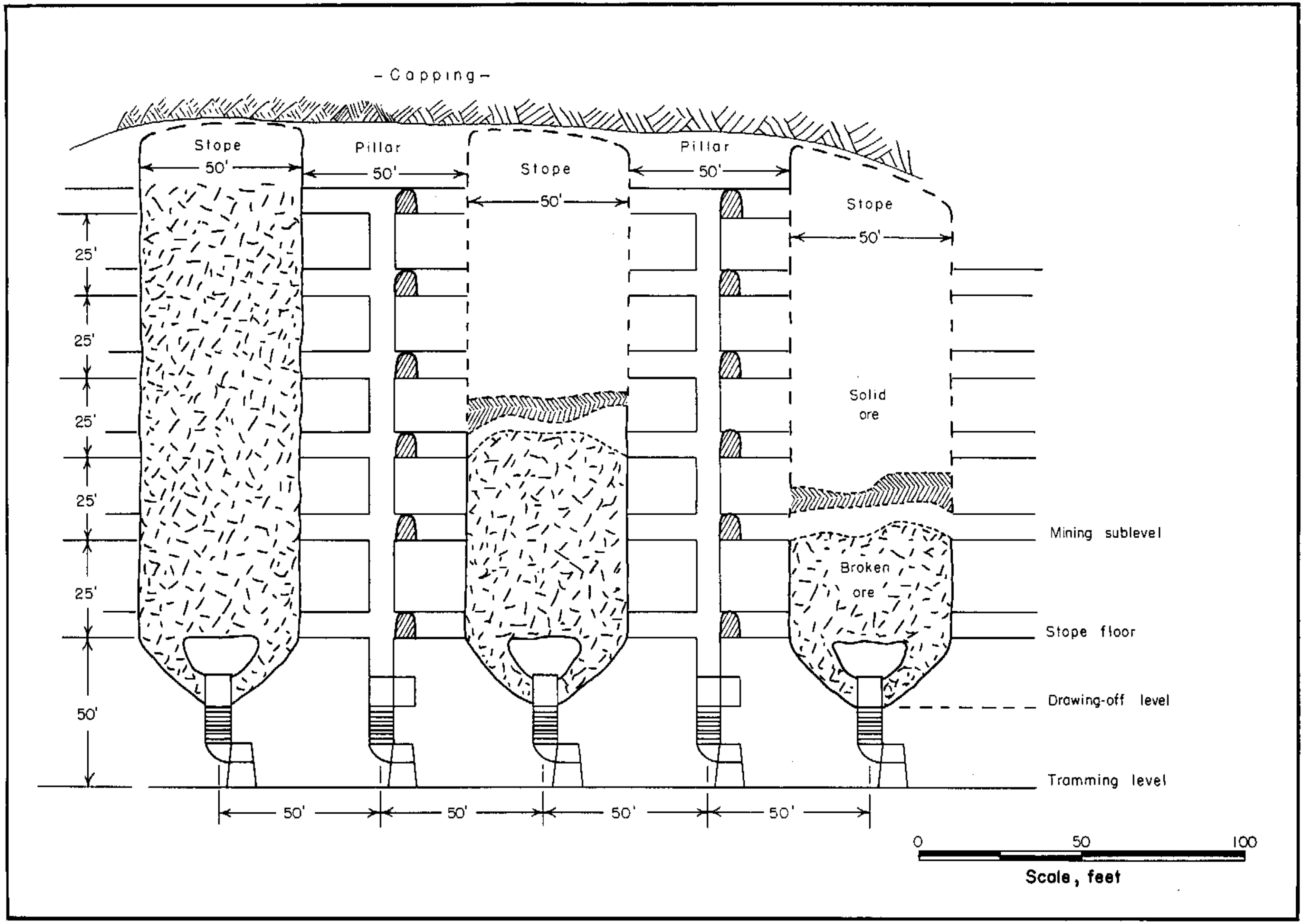

FIGURE 7. - General Cross Section, Wide Shrinkage Stope and Pillar Caving System.

Vertical, cribbed pocket raises, all placed at one side of the tramming drifts, were driven through to the drawing-off level, one raise for each pair of stope raises. Grizzlies with an 18-inch opening were installed at the top of the pocket raises in the floor of the drawing-off drifts directly under the control chutes of the stope raises.

Broken ore passed in turn through stope raises, control chutes at the drawing-off level, grizzlies in the floor of the drawing-off level, pocket raises, and finally the chutes at the haulage level into cars for transportation to the shaft.

Development in the pillars was started as quickly as possible. Vertical raises at 50-foot intervals were driven from the drawing-off drift to the upper limit of the ore body, and sublevels 25 feet apart were developed with central drifts driven to the ore limits and crosscuts at 50-foot intervals into the, as yet, unmined stope blocks. None of the sublevel openings were ordinarily timbered. The sublevel development was necessary, not only for breaking the pillars later but for access into the stopes as they were advanced, and was kept ahead of stoping.

When preparatory work had advanced sufficiently, stoping by the shrinkage system proceeded until the upper ore limit was reached. About 39 percent of the ore was drawn to provide working space. A shell of ore was usually left under the barren capping to form a cushion and minimize dilution when the stope was drawn. Sustained drawing was not started until pillars were broken.

Mining of pillars was undertaken after breaking in the stopes was finished. Breaking in the pillars proceeded from the top down by drilling and blasting from sublevels. Mining was started at one end of pillars, retreating toward the other end, and at stope edges, retreating toward the central drift. Using mounted drifters, successive rounds were drilled and blasted in the backs and walls of sublevels until the openings holed through to the next sublevel above or, in the uppermost sublevel, reached the capping. The last 5 feet of ground usually caved. The back in this type of stope operation was carried on a slope of about 45°. With proper scheduling, a large number of ore faces on several sublevels could be attacked at once. An important detail in the retreating system was temporary bulkheading of raises just under each sublevel as work on that level was completed to avoid any uncontrolled movement of ore or waste capping through open raises.

In the last stage of pillar mining, preparation for drawing the ore was made by driving and enlarging additional raises from the drawing-off drift into the base of the pillar to provide control facilities similar to those under the shrinkage stopes.

As much as one-third of the total tonnage in a block of ore mined by the wide shrinkage stope and pillar system was removed in preparatory work or by drawing during stoping and pillar mining to compensate for the additional volume (shrinkage) occupied by broken ore.

Systematic drawing of ore, other than that required to provide working space, was not started until about 70 percent of stope and pillar ore had been broken. Drawing was carefully controlled to bring down the overlying capping evenly and minimize the chances of the finely divided capping running into any cavities that might form through "arching" or "piping" in the broken ore column. Drawing was not permitted within 100 feet of a point where drilling was still in progress. When drawing was started, some difficulty was experienced in making ore run steadily. Hard ore could be drawn with comparative ease, but ore with a high content of soft clay minerals tended to pack after standing for more than a short period. This tendency of soft ore to pack under compression, particularly when damp, was shown by the fact that sublevel crosscuts occasionally driven from pillars into the broken ore in shrinkage stopes would stand open without timbering. Movement of ore sometimes had to be induced by driving intermediate raises from the drawing-off drifts to break up packed material.

Under certain conditions, failure of timber support on the drawing-off level under extreme weight was a problem. In the northern part of the ore body, the capping above the ore averaged 340 feet in thickness, which, with 125 feet of broken ore in a completed stope or pillar, gave a total column of about 465 feet of ground in a mobile condition. Crushing in this section necessitated complete retimbering in some stope lines. In the southern part, the capping was about 250 feet thick, and the broken ore in a completed stope 100 feet thick, making a total of 350 feet of moving ground. The weight in this section was negligible and only minor repairs were required during the life of the ore body. A difference of about 100 feet in total height of the ore-and-waste column was critical.

In a small area, when blocks of unbroken pillar ore were unavoidably left above the drawing-off drift, the weight of such blocks tended to concentrate on a few points with resultant failure of support underneath. After drawing had progressed on a large scale over a large area, the weight problem in heavy ground tended to decrease. Apparently the weight became more uniformly distributed as a state of equilibrium was approached.

Not all of the broken ore in a stope or pillar could be drawn without additional work. At the end of the regular drawing period, a wedge of broken ore always remained on top of the pillar forming the back of the drawing-off drift. Most of the ore was recovered by breaking this pillar (often partly or completely crushed) through to the broken ore above, and drawing the material into the pocket raises in a retreating procedure.

Blocks of solid ore, triangular in cross section, between adjacent lines of drawing-off drifts and below the junction of stopes and pillars also remained at the close of regular drawing operations. Most of this ore was recovered by driving timbered intermediate drifts between the original drawing-off drifts, installing a line of chutes, and driving small shrinkage stopes until broken ore was encountered, or the remaining ground caved. The ore was trammed along the drifts to pocket raises driven from crosscuts on the tramming level below.

Extraction by the wide shrinkage stope and pillar system was about 95 percent of the calculated tonnage. Data kept on each stope and pillar indicated that extraction in the stopes was about 10 percent better.

Top Slicing

When the upper part of the Captain ore body had been nearly mined out as far as shaft depth at the time would permit, preparations were made to continue mining the Main Miami ore body below the earlier square-set stopes. The schist ore was too soft to mine by the wide shrinkage stope and pillar system, and top slicing was adopted. The collapsed timbers in the abandoned square-set stopes above reinforced the heavy mat necessary in top slicing to isolate the working level from the overburden. In this case, the overburden was friable cap rock, which, when dry, would readily sift down and mix with the ore, unless controlled.

In the first attempt, slicing faces from 50 to several hundred feet in length were used, retreating along the ore body toward a service shaft outside the slicing area. The method was unsatisfactory. Working faces advanced so irregularly that control of operations was difficult, and the long connecting drifts required excessive maintenance. As a result mining costs were high.

Long slicing faces were abandoned in favor of a more successful block-slicing system.[15] In this system, haulage levels were opened at vertical intervals of 150 feet with two sublevels between to facilitate driving ore raises and distributing ventilating air. The haulage level drifts were spaced at 50 feet and ore raises were put up at 50-foot intervals along the drifts. Two-compartment supply raises were put up to the top of the ore body in the center of square blocks that were 250 feet on a side (fig. 8). At the top, four bulkheads were built in a square to protect the raise. Drifts were extended on the long axis of the supply raise to within 25 feet of the block limits. Crosscuts 125 feet long were driven each way from the ends of these drifts and later from the supply raise to the block limits. Ore broken in driving was disposed of through the raises with which connections were made as work progressed.

Mining was carried on simultaneously in all quadrants of a block. Starting at the ends of the outer crosscuts, 25-foot slices were taken between these crosscuts and the block limits. When the first slice had advanced a few feet, a second was started and so on. When work in the outer rows was well underway, slicing in interior rows was started at the outer ends of the interior crosscuts, always retreating toward the central raise. Slices were 12½ feet wide and 10 feet high. Except in the exterior rows, slices were 50 feet long.

Caps and posts were used for ground support. If the timbers showed signs of taking weight before all the ore was removed, bulkheads were built from old timbers in the mat. As soon as the ore had been removed the timbers were drilled and shot out and a new slice was started below.

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[15] Dean, E. G. The Block Method of Top Slicing of the Miami Copper Co. Trans. AIME, v. 55, 1917, p. 240.

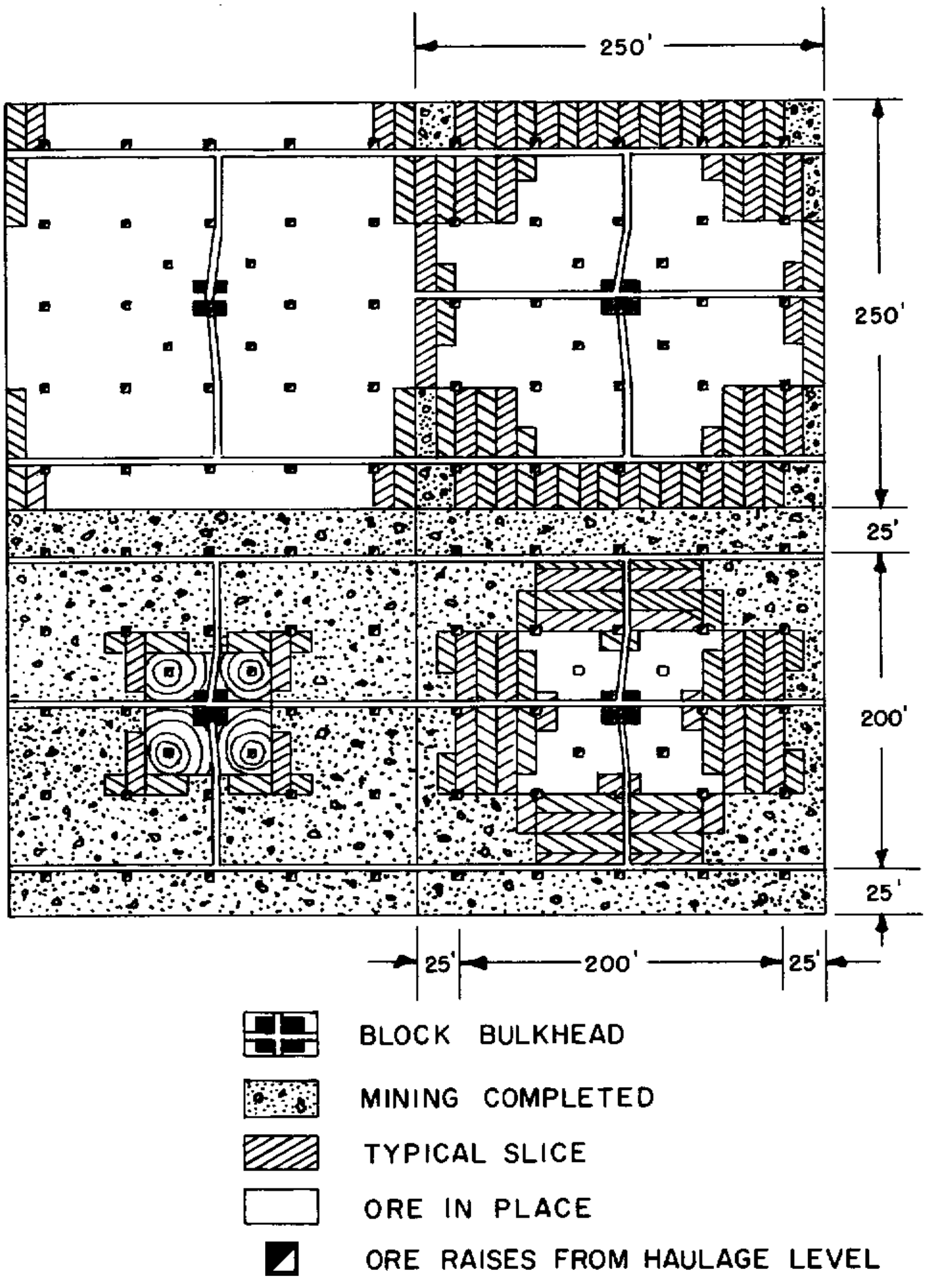


FIGURE 8. - Block Method of Top Slicing.

Mining was carried on as rapidly as possible to minimize support problems and maintain a high rate of production. Production ranged from 9 to 20 tons per mucker-shift and from 5 to 10 tons per manshift. The 250-foot-square slice block yielded an average of 877 tons per day for the life of the slice. This method had the advantages that 100 percent of the ore was recovered and dilution was avoided; costs, however, were high.

Narrow Shrinkage Stoping

A system of narrow shrinkage stope and pillar mining replaced the more costly block method of top-slicing in the Main Miami ore body. The system was an adaptation of the wide shrinkage stope and pillar method used in the Captain ore body but with stope width reduced to 8 feet and pillars to 25 feet in the soft schist ore. It had been found that some of the 50-foot pillars left between stopes in the hard Captain ore body would start to cave before they could be drilled and blasted. Caving of the 25-foot pillars in the soft schist ore of the Main Miami ore was easily induced by undercutting alone. This system could be classed as the first Miami caving operation, if the narrow shrinkage stopes are considered as part of the necessary preparation. Some reduction in mining costs was achieved by this system.

Panel Caving with Hand Tramming

Panel caving evolved out of an unsuccessful attempt to use a caving system to mine an entire ore body as a single stoping unit. Starting at one end of the ore body, the ore was under cut and caved for its full width, the undercut retreating toward the other end. This approach was unsatisfactory because it was impossible to maintain good draw control over a large mining area. Consequently, the waste capping broke through and prevented substantial tonnages of ore from reaching many draw points.

For optimum results, two conditions are necessary in drawing caved ore. First, the rate of drawing must be adjusted to the natural rate of caving. Ideally, the caved ore is drawn at a rate that permits the solid ore to "work" and cave freely, but not so rapidly that large cavities are formed above the caved material. Second, drawing should be continuous rather than intermittent so far as possible, because, in periods of rest, caved ore, particularly soft schist, tends to pack, and then hang up when drawing is resumed. The draw should be uniform across the entire stope width so that the waste capping will settle evenly and remain above the ore.

These objectives could not be attained at Miami in a large stope because the ore was not uniform in hardness and consequently in caving characteristics. Soft ore caved more easily than hard ore and more readily near the center of a stope than near the margins. The friable waste capping above the ore broke up into fine material that ran much more easily than caved ore. Consequently, irregularities in the draw permitted waste to descend more rapidly in one section, and the free-running waste was likely to break through the contact between capping and ore, and dilute adjoining sections. Under these conditions, waste, moving laterally as well as downward, would reach draw points ahead of the ore directly overhead. When waste appeared in a draw point, the ore drawn thereafter was diluted, if it could be recovered at all.

Another disadvantage of large caving units was the heavy maintenance of ground support necessary to keep extraction workings open during long periods of operation. The heavy ground pressures encountered at many points caused support failures before extraction openings could be abandoned, and timbering in haulage drifts had to be repaired or replaced several times during the life of a large stope. Maintenance was not only costly, but interference with drawing schedules aggravated the problem of draw control.

To solve the problems of poor draw control and high maintenance costs, the ore body was divided into panels 150 feet wide extending across the width of the ore. Alternate panels were mined, leaving intermediate panels as pillars. The pillars were mined a year or so later after the waste filling in the earlier stopes had consolidated enough to stand when the pillars were caved. Panel caving was first applied in the isolated Northwest schist ore body, but the system was applicable to both hard granite-porphyry and soft schist ores.[16]

[16] Hensley, J. H., Jr. Mining Methods of Miami Copper Co. Trans. AIME, v. 72, 1925, pp. 78-99.

Preparatory work for panel caving is illustrated in figure 9. To keep development at a minimum, the horizontal spacing of haulage drifts was set at 150 feet, and the 150-foot vertical interval between main haulage levels used with the block-slicing system was selected. Haulage drifts connected to main haulage laterals outside the ore body. A single haulage level was designed to serve, in turn, two caving lifts of 75 feet each. Each lift was served directly by parallel tramming drifts on 25-foot centers on a tramming or transfer level 25 feet below the undercut or mining level.

Paired, diverging transfer raises on 100-foot centers were driven from both sides of haulage drifts to connect with and serve, with branches, the six tramming level drifts for the upper caving lift of each panel. The design of the raise complexes, shown only in an end view in figure 9, was similar to the paired, branching raises described more fully in the section on block caving. Drifts, both on upper tramming levels and haulage levels, were parallel. The 150-foot spacing of the haulage level drifts was fixed by the minimum slope on which ore would run (53°) and the distance to the upper tramming level (100 feet).

The distance from the haulage level to the lower tramming level was only 25 feet, insufficient for an inclined transfer-raise system. To utilize the same haulage drifts, the lower tramming level drifts were oriented at 90° to the haulage drifts. Short vertical raises where tramming drifts crossed over haulage drifts then provided enough ore passes to limit the maximum hand-tramming distance to 75 feet. Crosscuts between tramming drifts were driven at 100-foot intervals. Crosscuts were a source of trouble later because they tended to collapse when drawing operations reached them, but they were necessary to furnish access for men and supplies. They had to be completely filled with waste timber as drawing progressed.

Next, paired inclined raises were driven on opposite sides of alternate tramming drifts and widened to 14 feet at the level of the under cut 25 feet above. All the raises on one side of a tramming drift were then connected at the top to form a drift at the undercut level parallel to and offset halfway between lines of tramming drifts. The undercut level drifts were enlarged to 8- by 8-foot sections, and the pillars between drift lines were drilled with fan-shaped rounds of holes 2 to 3 feet apart and blasted in 25-foot sections. At this point, only alternate tramming drifts had been connected to the undercut level drifts. The final stage in undercutting was opening 8- by 8-foot drifts immediately above the level sets in all tramming drifts and installing alternate drawing chutes at 6¼-foot intervals on both sides (chutes on one side of a drift were therefore 12½ feet apart). From these intermediate drifts, the stope-floor pillars between the undercut level and the tramming level were drilled and shot out, leaving unobstructed passageways for drawing the ore. The drift work on the undercut level was carried on as far in advance of mining operations as tramming level development permitted. The undercutting phase began at one end of the panel and was completed on a schedule to keep it 50 feet or more ahead of drawing. In advance of sustained drawing, enough of the broken ore was drawn to give the solid back of the undercut a chance to work and start caving.

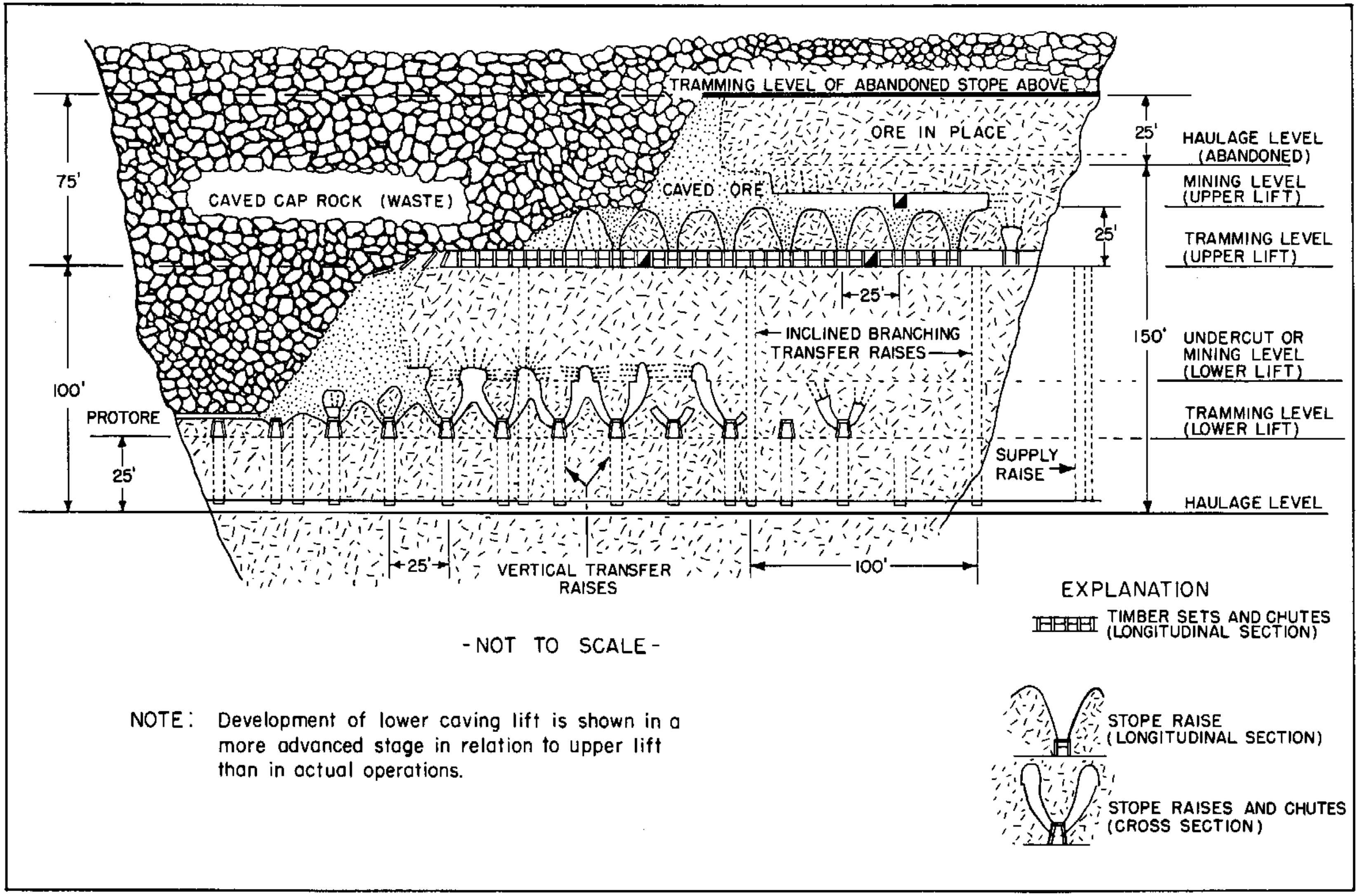


FIGURE 9. - Longitudinal Section of Panel-Caving Stope.

Sustained drawing followed undercutting as soon as ore had started to cave properly and the undercut had advanced far enough for the safety of the undercutting crews. The operation consisted of drawing the ore according to a definite schedule through the lines of chutes on the tramming level into cars, tramming by hand a maximum of 50 or 75 feet, and dumping the ore through an 8-inch grizzly at the top of the nearest transfer raise. Any boulders in the ore were blasted to pass through the drawing chutes, and practically no block-holing or sledging was necessary on the grizzlies.

All haulage and transfer level drifts were timbered. Conventional drift sets with 2-inch plank or peeled 3½- to 5-inch lagging were used. Ten- by ten-inch squared posts and caps were used on haulage drifts and 12- by 12-inch timber on tramming levels. The tramming drifts were driven without timber and less than finished size. They normally remained open until enlarged and timbered as needed. Long, inclined transfer raises to the upper tramming level were untimbered in hard ground, except for manway stulls and partitions, removed after completion. In soft ground, these raises were timbered with 4-post framed sets, lined with 2-inch plank. A 10-foot vertical section at the top of each transfer raise branch below the grizzly was cribbed with 6- by 8-inch or 8- by 8-inch cribbing. The short vertical transfer raises serving the lower tramming level were cribbed throughout.

Experience gained during the panel-caving operation brought about changes in procedure, improved stope performance, and knowledge of ground characteristics that were helpful later in designing block-caving stopes. For instance, it was found that soft ore would cave readily with stope widths of as little as 50 feet, that hard ore required at least 150 feet, and that 250 feet would have been better in very hard ground.

Small shrinkage stopes driven along block boundaries facilitated caving, especially in hard ore, and limited caving to the block in operation. However, they were not always necessary. In very soft ore, it was found that caving would start without enlarging the undercut level drifts beyond the original size of 8 by 8 feet in cross section, so this phase of undercutting was dispensed with at times.

The ore body mined by panel caving was about 500 by 1,500 feet in horizontal dimensions. The ore above had been mined to an even horizon by the top-slicing method before caving was started, so the tonnage of the cave block could be calculated with reasonable accuracy. The expectancy for the first 75-foot lift was 2,464,300 tons of ore assaying 2.19 percent copper. The tonnage extraction realized was 100.78 percent; grade extraction 82.70 percent; copper extraction 83.34 percent. The production of ore per man shift for all underground labor was 8 tons.[17] About 1,000 tons per day was produced from a stope area of 11,337 square feet.

BLOCK CAVING

The grade of Miami ore reserves steadily declined as the richer ores were depleted, and by 1930 was estimated at only 17.6 pounds of copper per ton

[17] MacLennan, F. W. Subsidence From Block Caving at Miami Mine, Arizona. Trans. AIME, 1929, pp. 167-168.

(0.88 percent copper) of which only 14 pounds or 0.70 percent was present in sulfide minerals. To continue profitable mining, a further reduction in cost was essential. This was accomplished by introducing a low-cost block-caving system, a logical improvement of panel caving. In this system, the ore body was mined in individual stopes or blocks originally 150 by 300 feet in cross section and 300 feet or more high. Alternate blocks were mined with intervening blocks or pillar stopes left in place. The pillar stopes were caved a year or so later after the waste filling in the first blocks had consolidated.

The term "pillar stope" was used at Miami to designate a stope that was bounded on one or more sides by subsided capping from previously mined stopes. The term "pillar" was used to designate the partition between stopes.

Block caving replaced other systems after 1925.[18] During 1925-29, a total of 16,556,296 tons of ore were mined by block caving at a cost of 39.94 cents per ton. In the same period, production per man shift underground averaged 28 tons compared to 8 tons for panel caving. With modifications dictated largely by ground conditions, including departures from the 150 by 300-foot stope-block size and eventual adoption of lateral transfer by conveyor and scraper to supplement raises in moving ore from stopes to main haulage levels, the block-caving system was used for the remaining life of the mine.

General Mine Development

The term "development" as used at the Miami mine was defined as preliminary preparatinn for mining. It included work common to all or a large part of the ore body. With respect to block caving, mine development included constructing and equipping of hoisting, service, and ventilation shafts, haulage levels, transfer raises, access workings from shafts to the stoping areas at the grizzly level and service raises between levels. Some of the facilities served earlier mine operations.

Stope preparation, often called development by the operating crew, included the work done on a specific stope block such as, control drifts at the grizzly level, raises above the grizzly level (control raises), the undercut level, fringe drifts, boundary drifts and corner raises.

Shafts

Nine major shafts were sunk at the Miami mine. Shafts Nos. 1, 2, and 3, sunk during the exploration period, were also used for early production. No. 3 shaft served in turn for exploration, ore hoisting, and finally as an escapeway and ventilation shaft. No. 4 shaft was the main operating shaft until 1921. The usefulness of the first four shafts had been destroyed by ground subsidence before block caving was introduced.

[18] MacLennan, F. W. Miami Copper Co. Method of Mining Low-Grade Ore Body. AIME, Tech. Pub. No. 314-A.34, March 1930, 44 pp.

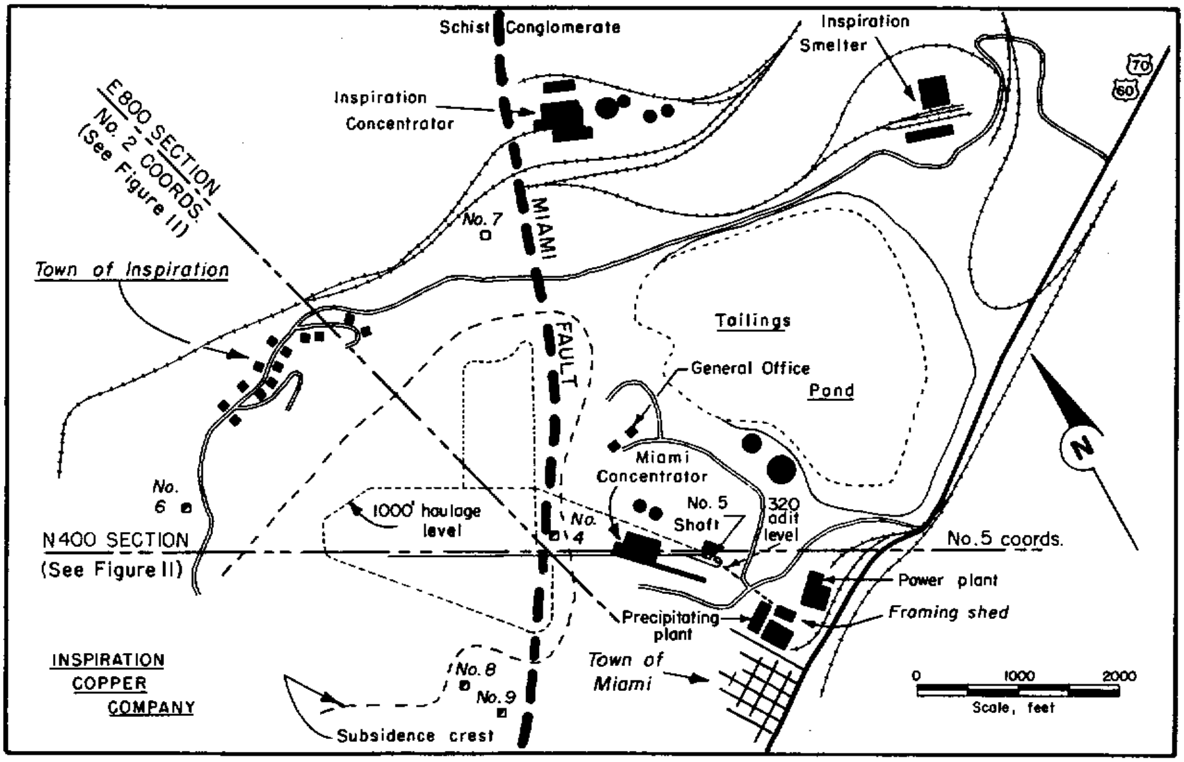


FIGURE 10. - Plan of Miami Mine Area.

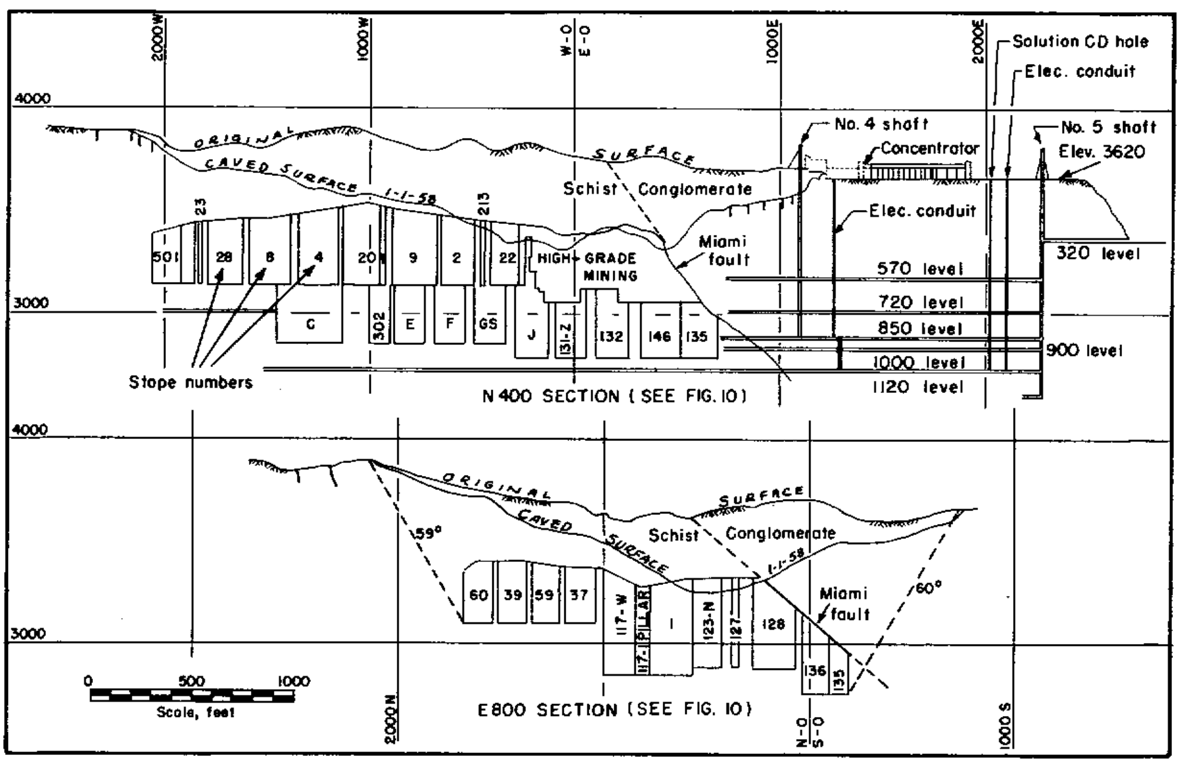


FIGURE 11. - Sections Showing Stoped Area, Subsidence, and Shafts.

The location of later shafts are shown in figure 10. No. 5 shaft was sunk about 1,500 feet east of No. 4 shaft and well outside the subsidence area to replace the latter as the main operating shaft (fig. 11). Four additional service shafts, all inclined, were sunk at various times to provide ventilation and entrance for supplies as the need arose. Of these, No. 7 shaft was sunk specifically to replace the abandoned No. 3 shaft.

No. 5 shaft, the main shaft in the block-caving operation, was a vertical, concrete-lined, four compartment installation sunk just east of the Miami concentrator on a ridge overlooking the town of Miami (fig. 1). The collar elevation was 3,620 feet. A service and supply adit driven from a portal in Bloody Tanks Wash at an altitude of 3,400 feet connected the shaft with a timber-framing plant on a railroad spur adjacent to the town of Miami. Measuring from the collar, stations were cut at the 320, 570, 720, 850, 900, 1000 and 1120 levels.

Ore passed through concrete-lined pockets constructed below the 720 and 1000 main haulage levels. For hoisting, ore was discharged into skips through measuring chutes. The flow was regulated by compressed-air-actuated, undercut, guillotine gates, under visual control of a skip tender. The arrangement of a haulage-level shaft station is shown in figure 12.

Because No. 5 shaft was sunk some distance away from the mining area to avoid the possibility of future damage from subsidence due to caving, travel between operating levels by way of the shaft was time consuming. To shorten travel for personnel and supplies, an interior service raise was driven between levels near the mining area and equipped with an automatic warehouse elevator large enough to accommodate a loaded supply car or 3-ton battery-powered locomotive.

Ore Hoisting Equipment

The double-drum hoist on No. 5 shaft was designed originally to raise 12,000 tons of ore per day a distance 796 feet from the 720-foot haulage level. Later, after the 1000-foot haulage level was developed, the hoisting distance was increased to 1,087 feet and the capacity to 18,000 tons per day. When block caving was started, the hoist was driven by a 1,400-hp direct-current motor connected to the main hoist shaft through a single-reduction gear drive. A second identical motor to double the horsepower was added when hoisting capacity was increased. The motors were connected to the hoist separately through pinions that engaged the main-shaft bull gear on opposite sides. To apply the increased horsepower effectively, hoisting speed, originally 1,500 feet per minute, was increased 50 percent to about 2,290 feet per minute by changing the gear ratio of the drive.[19]

Direct current for the main-hoist motors was supplied by motor-generator sets, one for each motor. Each set, rated at 1,000 kw, consisted of a direct-current generator direct-connected through a flywheel to a 1,100-hp induction motor designed for three-phase 25-cycle alternating current service at 6,700 volts.

[19] Grant, F. R. Miami's Automatic Hoist. Eng. and Min. J., v. 129, No. 7, April 7, 1930, p. 347.

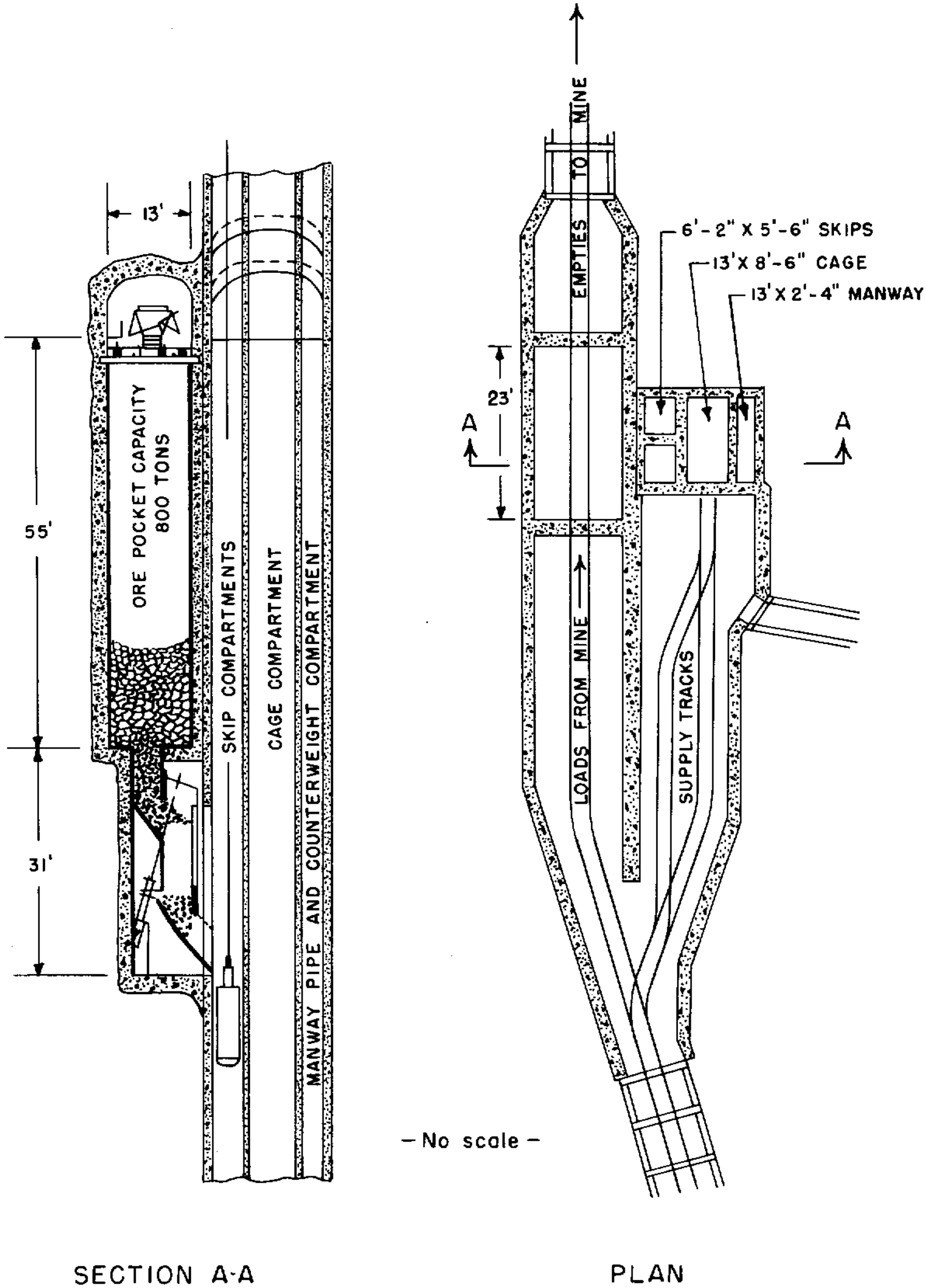


FIGURE 12. - Haulage Level Station, No. 5 Shaft.

The hoist was equipped with retardation, overspeed, and overwind safety devices, including electric limit switches in the skip roads in the headframe. A hoisting cycle was started when the skip tender pushed a control button at a loading point. Normally, accelerating, retarding, and dumping of the skips were automatic, but manual control from the engine room was possible.

Each drum was 10 feet in diameter with a 5-foot face grooved for 1-3/4 inch rope. Ore was hoisted in 10½-ton skips running in balance. Hoist ropes were discarded after an arbitrary service life of 1,600,000 tons of ore hoisted. Additional hoist data are given in table 2.

TABLE 2. - Performance and characteristics of No. 5 skip hoist

Distance hoisted....feet..	1,087
Drum speed....revolutions per minute..	73.0
Hoist motor speed....do...	338
Acceleration time....seconds..	11.4
Retardation time....do...	9.25
Running time....do...	42
Rest period....do...	3
Total time of trip....do...	45
Motor generator set speed, no load....revolutions per minute..	747
Motor generator maximum operating speed....do...	722
Motor generator minimum operating speed....do...	668
Generator voltage....volts...	535
DC line amperes, maximum....amperes...	3,900
DC line amperes, sustained....do...	1,300
Exciter voltage....volts...	247
AC line voltage....do...	6,700
AC frequency....	25
No. of skips hoisted per hour....	83
Average hoisting time per skip....seconds...	45
Average rope speed....feet per minute...	2,290
Power consumption per ton hoisted....kilowatt hours...	1.47

Personnel, supplies, and waste rock were handled by a single-deck cage operating in counter balance through a 6½- by 13-foot compartment of No. 5 shaft. Cage capacity was 45 men, and equipment as large as 6-ton locomotives could be handled without dismantling. The single-drum service hoist was driven through single-reduction gearing by a 250-hp, variable-speed alternating-current motor. Three-phase, 25-cycle service at 440 volts was required. The manually controlled hoist was equipped with safety devices similar to those on the ore hoist.

Grizzly or Control Level Access

The control or grizzly levels were connected through to the shaft by access drifts for passage to the stoping area for men, timber, and supplies without interfering with the movement of ore on the haulage levels. Existing workings above the 720 level were used as control levels when they were suitably located. Below the 720 level, new levels were established; several were necessary because of the uneven bottom of the ore body. Six grizzly (control) levels, connected to No. 5 shaft and were served by the man hoist. The haulage system on these levels was equipped with 3-ton battery locomotives running on 24-inch gage track.

Haulage Levels

The Miami block-caving operation was served by two main haulage levels through which all ore was moved to the shaft. Existing facilities on the 720 level were used for most of the ore remaining above that level. The new haulage level, established on the 1000 level, 100 feet below the bottom of all but a small section of the ore body, served mining operations below the 720 level.

Block caving below the 720 level was started in 1939 before mining above the level had been completed. By 1941, all the ore above the 720 level had been mined out, and production thereafter came through the 1000 level.

Track Haulage

The haulage system as finally developed for block caving provided for movement of 18,000 tons of ore per day to the shaft, first on two levels and later on the 1000 level only. One-way traffic was provided by a closed-loop system in which empty trains moved from the shaft to the stoping area through one lateral and returned through the loading drifts for filling and back to the shaft through a second lateral (fig. 13). The round-trip distance averaged about 10,000 feet.

Ore was hauled in trains of 30 saddle-back cars of 86 cubic feet capacity (3.73 tons) pulled by two 6-ton trolley locomotives equipped for tandem operation by one motorman. Trains were loaded at the transfer raise chutes through hand-operated arc gates and later air-actuated undercut guillotine gates at about 50 seconds per car or 30 minutes per train (fig. 14). The cars, equipped with hand-operated door latches, were unloaded as the trains passed slowly over the ore pockets without stopping (fig. 15). The time for a round trip between the mining area and the shaft was about one hour on the 1000-foot level.

Haulage track was 24-inch gage with 45-pound rail in loading drifts and 70-pound rail in the main haulage drifts and shaft approaches, where traffic was heaviest. Six- by eight-inch wood ties three feet six inches long placed on 30-inch centers supported the rails. Parallel-ground-throw switch stands, cast-manganese frogs, and split switch points with spring connecting rods were used. A standardized curve radius of 41 feet 3 inches was used for all turnouts. A grade of 0.4 percent favoring the load was established.

Auxiliary Conveyor Haulage

Conveyor belts were first used underground at Miami in 1947 to get haulage drifts out from under the stoping area because the ground was too heavy near the center of blocks. At this time, the mine began producing from a block of about 18 million tons of ore that bottomed directly over the 1000-foot haulage level, the grizzly level being on the 975-foot level. With the

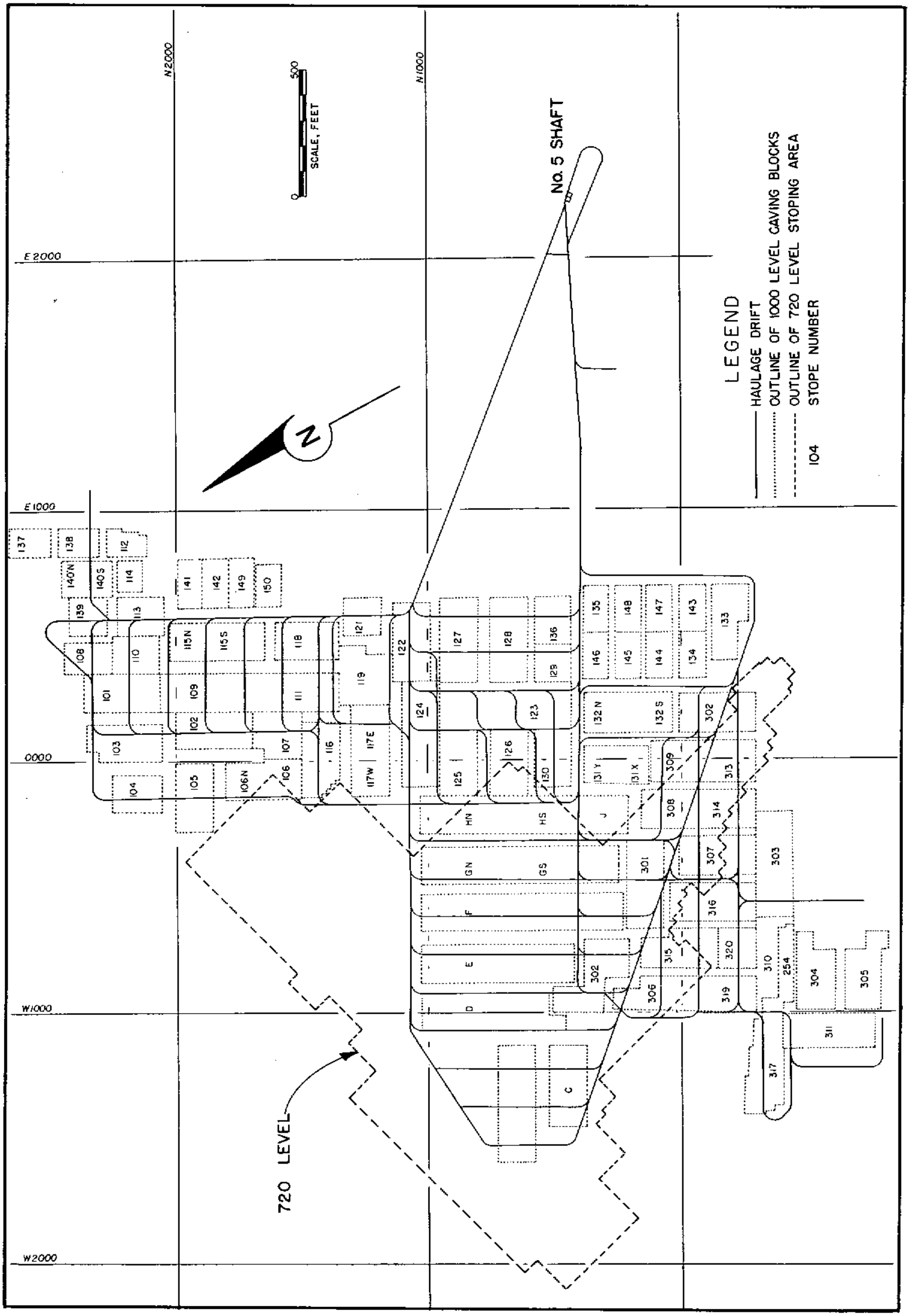


FIGURE 13. - Layout of 1000 Haulage Level.

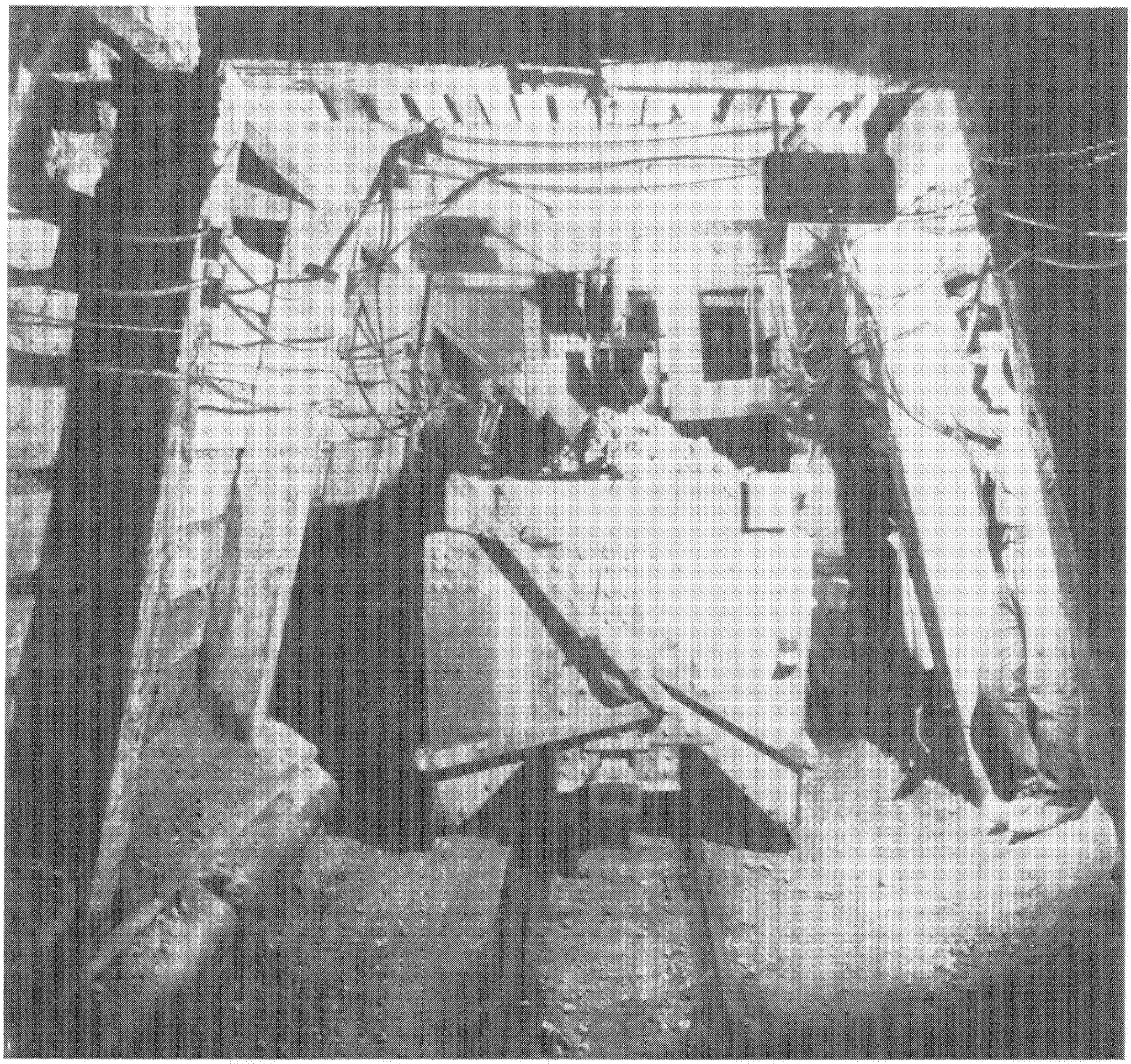


FIGURE 14. - Loading Ore From Transfer-Raise Chute.

low height above the haulage level, it was impossible to use a full gravity system. Scrapers were used feeding a short raise directly into haulage drifts, which were moved to the edge of blocks. Even then, weight developed in the weaker areas on the timber in the haulage levels, and the loading drifts were moved still farther from the active block, thus requiring additional lateral transfer. Slushers on the control level then fed a 200-foot-long, 30-inch belt conveyor on an intermediate level through a short raise. The belt, running at 400 feet per minute carried the ore to the haulage drift now at a location outside the weight area.

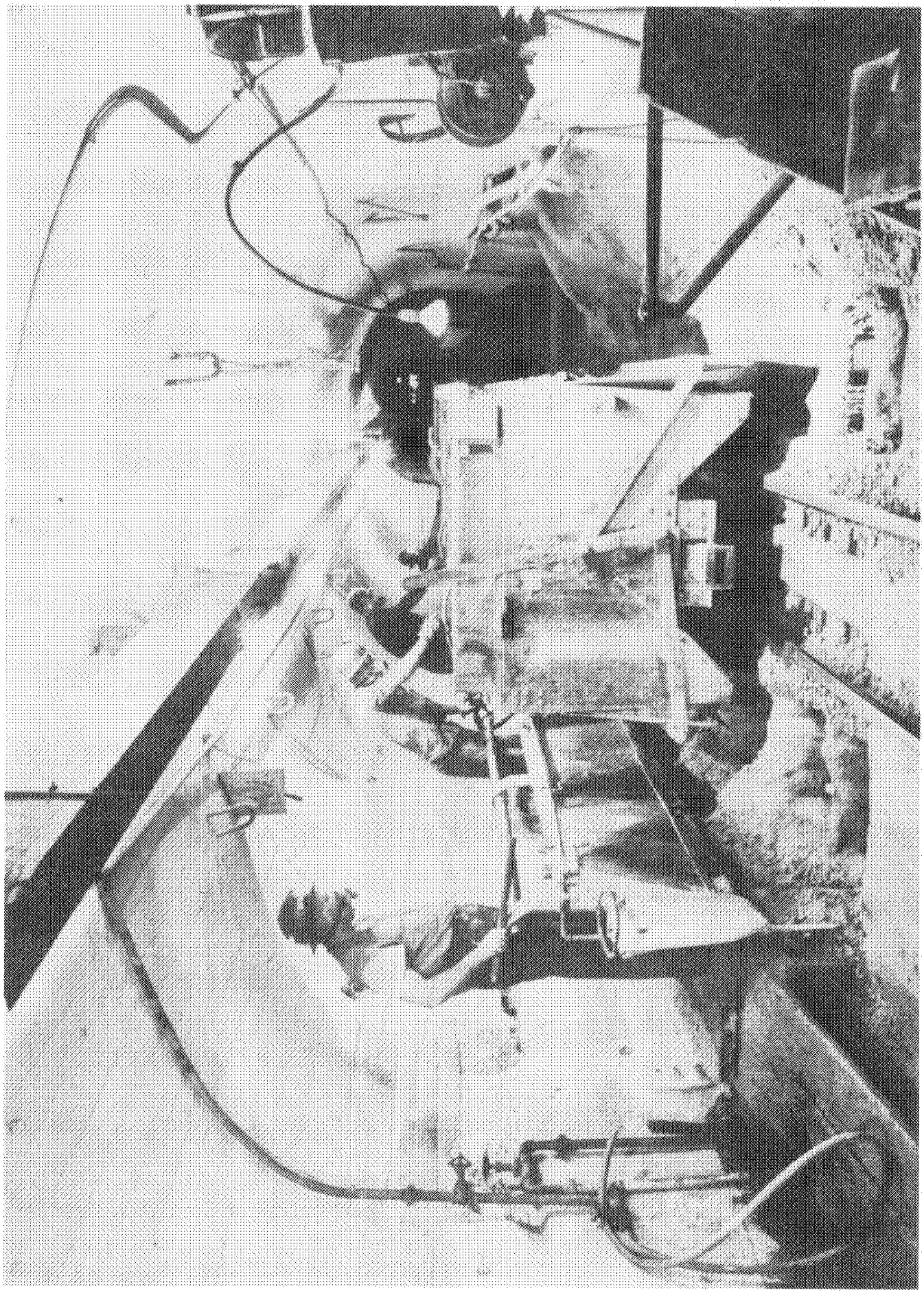


FIGURE 15. - Unloading Ore at No. 5 Shaft Pocket.

As mining operations approached an end, sampling and examination of the records indicated that in several locations, particularly on the upper levels, some low-grade ore remained below the old stopes and on the fringes of the ore body. This resulted from the fact that when caving first started the cutoff grade was 1.00 percent copper and for a long period it was much higher than the average grade of 0.63 percent sulfide copper in ore being mined in 1958.

This low-grade ore was in low blocks and much of it was on the 570- and 720-levels in areas that were not serviced by the 1,000-foot haulage level. Rather than extend costly haulage laterals under the upper levels, auxiliary conveyor haulage was installed (fig. 16). To handle the necessary tonnage of about 2,000 tons per shift, a 36-inch belt driven about 300 feet per minute by a 20-hp motor was installed.[20]

Ore was moved from draw control points by a 42-inch scraper pulled by a 25-hp motor and was discharged through a 10-inch grizzly into a raise. At the bottom of the raise on the 670 level the ore was fed on the belt by a locally designed auxiliary chute. The chute gave the ore a velocity and direction at the point of contact with the belt matching that of the belt. Sometimes belts on two intermediate levels were necessary to move the ore to a centrally located concrete ore pass to the 1000-foot haulage level for handling by conventional train haulage.

Underground Power System

Electric power for the haulage system was delivered to substations (fig. 17) on the haulage levels at 6,600 volts ac. Service for the 1000-foot level came through a lead armored cable suspended in a churn-drill hole. Power was converted to direct current in Ignitron[21] rectifiers and fed at 275 volts into a trolley system designed to minimize voltage drop and resulting power losses. The system was sectionalized in four independent blocks, each served by a 1,250,000 circular-mil feeder line. Trolley lines of size 0000 copper wire were connected at frequent intervals with paralleling 500,000 circular-mil feeder lines, and a similar current-return circuit was tied to the bonded rails of the track system at about 50-foot intervals.

Power for battery chargers, scraper-hoists, and other equipment using alternating-current motors was delivered into the mine through an auxiliary shaft at 2,400 volts and reduced to 440 volts in 225 kva transformers installed near the point of use.

Stope Preparation

The extraction openings at Miami for full-gravity block caving, listed in the direction of ore flow from stopes to haulage level, were undercut levels, control raises, grizzly-control drifts, transfer-raise complexes, loading and

[20] Williams, E. G. The Use of Belts and Concrete Ore Passes in Transporting Ore From the 570 Grizzly Level to the 1000-Train Level. Paper delivered at Arizona Section, AIME, May 1955, Miami, Ariz., 3 pp.

[21] Reference to specific makes or models of equipment is made only to facilitate understanding and does not imply endorsement by the Bureau of Mines.

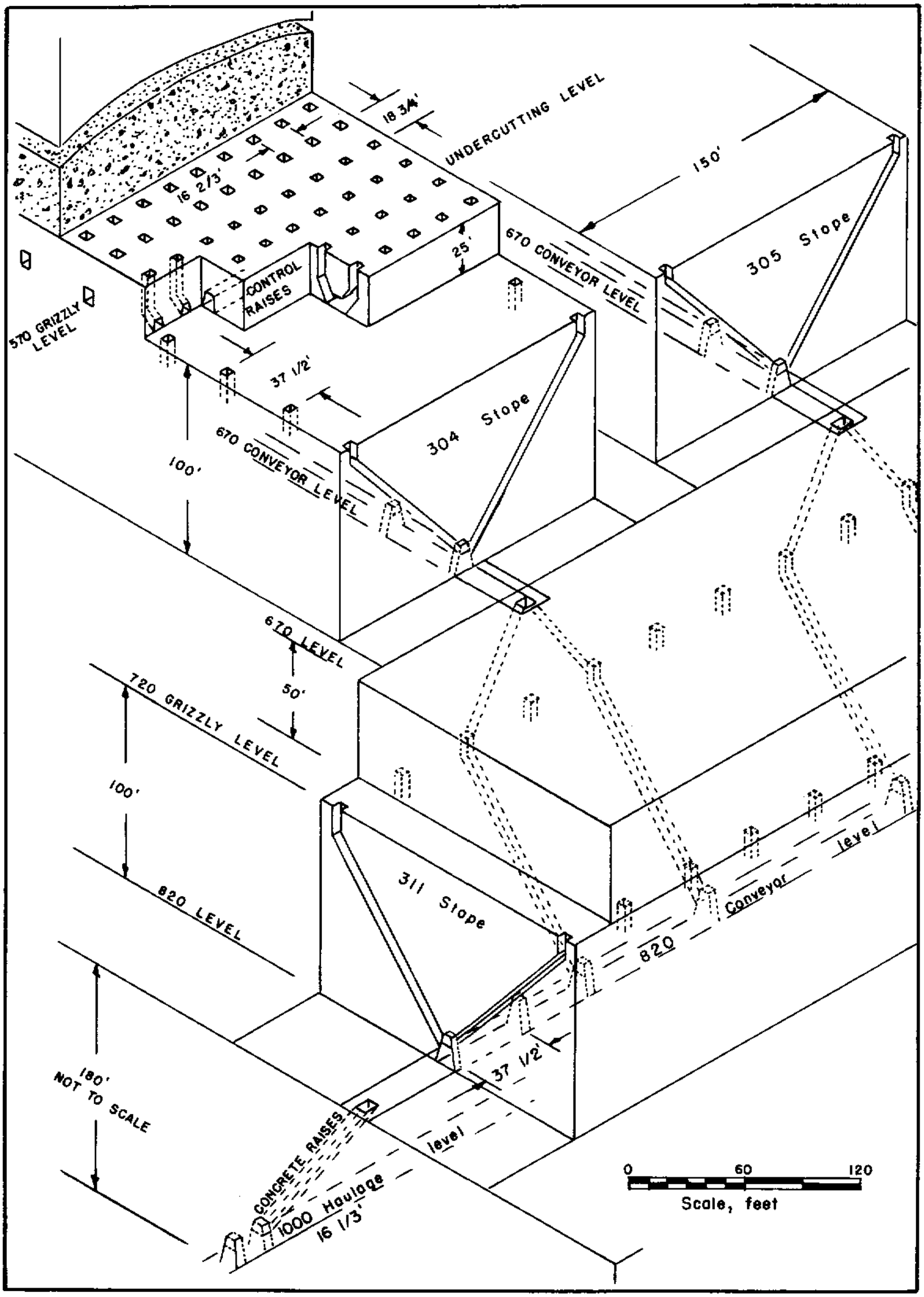


FIGURE 16. - Typical Scraper Caving Block With Auxiliary Lateral Conveyor and Transfer to Haulage Level.


FIGURE 17. - Underground Power Substation.

haulage drifts. In later operations, grizzly drifts were replaced by shaker-conveyor or scraper drifts at the control level, and transfer-raise complexes by simple ore-pass-type raises.

Planning and Design

The success of the block-caving operation was dependent on careful planning, adequate design of extractive openings, and scheduling of stope preparation to give efficient and prompt removal of the ore at the proper time. A continually revised plan set up a sequence and time schedule for both stope preparation and ore extraction based on 110 percent draw at 9 inches per 24 hours. Few standards relating to block size, methods of ground support, ore-drawing procedures, or ore transfer were universally applicable. Flexibility was required, and an outstanding feature of the operation was the successful adaptation of the mining system to meet changing ground conditions.

Stope preparation for block caving included work under the stopes at or above the grizzly control level. Loading drifts on the haulage level and transfer raise complexes for full-gravity stoping were classed as mine development. However, these items of development were correlated closely with stope preparation, and their construction was scheduled only as needed to serve the ore blocks selected for mining. Detailed stope preparation plans were made about two years in advance of actual mining to meet projected production schedules.

Although stope preparation was not carried on continuously, the number of men assigned to this work averaged out over a long period in accordance with planning. An attempt was made to stay well ahead of schedule on the slower part of the work such as haulage drifts, transfer raises and fringe drifts. Schedule for work on grizzly or scraper drifts, control raises, and undercuts was more flexible because many more working faces were available, and overall rate of advance could be controlled simply by the number of men assigned to the work.

The development rate was adjusted in accordance with a running study of the performance of operating stopes. Stopes were prepared for a rate of production 10 percent above the 12,000 tons per 24 hours desired. However, ore was not undercut far in advance of mining because adverse weight conditions or packing were likely to develop if ore was not drawn promptly and continuously after being undercut.

Block Dimensions

Because of the friable nature of the Miami ore, it was believed at the time panel caving was introduced that the maximum block height for successful caving was 75 feet, and the 150-foot block height between haulage levels established for panel caving was divided into two lifts (fig. 9). The development cost per ton of ore in lifts of this height was high, and after the high-grade ore was exhausted, it was imperative that costs be reduced. After panels were replaced by blocks, larger tonnages were made available without increasing the amount of development by increasing the height of blocks, and the costs per ton for stope preparation were reduced correspondingly. Blocks 300 feet high were tried, and it was found that when the ore was drawn down evenly, dilution was not excessive.

Most of the ore above the 720 level was mined from that level, but a few remaining blocks as high as 500 feet were successfully mines from the 1000-foot level. During the last years of the operation, blocks less than 100 feet high were mined using the scraper transfer system. Blocks ranged from 80 to 500 feet in height, and the average block height over the life of the block-caving operation was 300 feet.

Other factors being equal, dilution by waste increased as block height was increased. This was particularly true when the draw was not even over an entire stope, a condition that developed either through faulty draw control or unavoidable weight surge.

The area that was undercut for a stope was large enough so that the ground would cave freely at a uniform rate, yet not so large that caving action progressed faster than the draw, thus subjecting openings below to excessive weight. The extremes in stope size at Miami ranged from 37½ feet by 75 feet to 150 feet by 800 feet.[22] The size was primarily dependent on the strength of the rock and its resulting tendency to form a self-supporting arch across the back of a stope; the stronger the rock, the larger the stope area necessary to induce caving. In the later years, stopes were often limited by the size of remaining unmined segments of ore.

When the block-caving system was first started in 1925, a stope size of 150 by 300 feet was adopted. Stopes this size performed satisfactorily in strong ground but gave trouble in weak ground; here the size was reduced to an area 150 by 150 feet. The 150- by 300-foot size in strong ground was generally used as a standard, except in the east end of the ore body, where a size of 150 by 200 feet was used because stope sills were closer to the haulage level.

[22] Still, J. W. Block-Caving at Miami. Min. Eng., v. 41, No. 4, April 1955, p. 90.

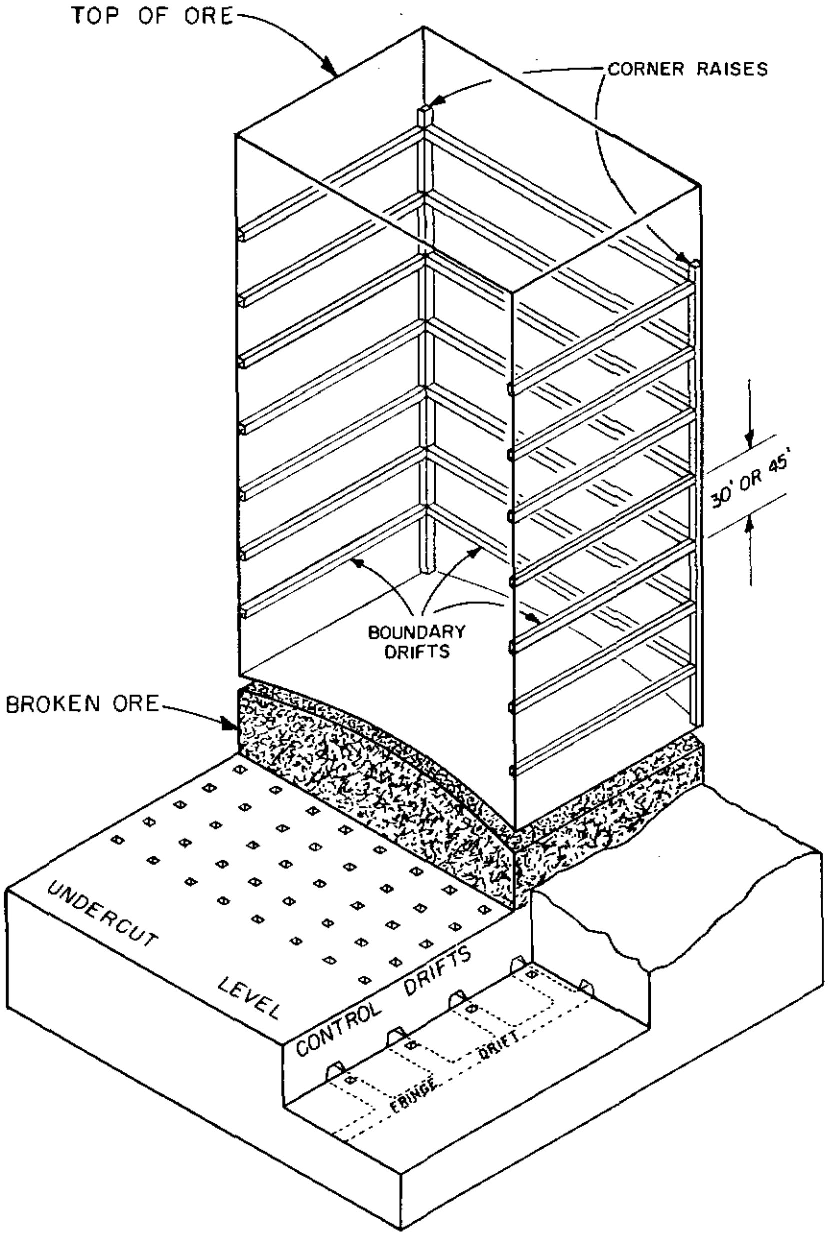


FIGURE 18. - Fringe Drift, Boundary Drift, and Corner Raise Locations With Respect to Undercut.

The final size of a block was not determined until the undercut was completed. The practice was to start the undercut the desired maximum size, then, if excessive weight developed, drop two or more lines, thus reducing the block in length. Some ore in abandoned lines was mined later along with that in adjoining partition pillars as small stopes.

Boundary Drifts and Corner Raises

In the original design of workings for caving, boundary drifts and corner raises were driven at block boundaries (fig. 18). The supposition was that, if the ore to be caved was not isolated from the surrounding ground by weakening the boundary plane, it either might arch to the center and stop caving or follow slips and planes of weakness in the formation and cave beyond the intended block boundaries. Raises were driven the height of the block at each corner and untimbered drifts 7½ feet high were driven at 30-foot vertical intervals completely around a block, thereby weakening the boundary plane by 25 percent. Later, the interval between boundary drifts was increased to 45 feet, and an 8-foot cutout round was drilled and blasted in the back of each drift, thus increasing the total weakening effect to 33½ percent.

Maps of the structure in these drifts showing slips, inclusions of capping, oxidized ore, and other data, were useful later on in control of ore drawing. After 1940, because of the shortage of skilled miners during World

War II, boundary drifts were discontinued with little or no change in the performance of the block. Corner raises were still driven in hard ore but were unnecessary in most of the last blocks mined.

Partition Pillars

To protect pillar stopes from dilution by waste from adjoining previously mined blocks, partition pillars were left against the mined-out blocks except in the mixed ore body along the Pinto fault.

The Pinto fault limits the mixed ore on the west and dips about 45 degrees with many local variations. No pillars were left in this area and, in some blocks, a deliberate effort was made to encourage movement along the fault. Measurement of resulting additional ore from lateral movement was not possible, but the highest overall extraction was from this area. The mixed ore body was in rock classed as hard to medium hard. Table 3 shows the relation between extraction and pillar thickness.

TABLE 3. - Extraction by pillar thickness

From	Expected		Drawn		Extraction, percent		
	Tons	Percent copper	Tons	Percent copper	Tons	Grade	Copper
	720 Level--mixed ore--no pillars						
Stopes........	7,358,000	1.788	9,791,000	1.460	133.06	81.66	108.58
Pillars.......	-	-	-	-	-	-	-
Total....	7,358,00	1.788	9,791,000	1.460	133.06	81.66	108.58
	720 Level--sulfide ore 7½- and 15-foot pillars						
Stopes........	40,577,000	0.878	40,522,000	0.763	99.87	86.90	86.79
Pillars.......	7,130,000	.859	-	-	-	-	-
Total....	47,707,000	.875	40,522,000	.763	84.93	87.20	74.06
	1000 Level--sulfide ore 30- and 50-foot pillars						
Stopes........	53,331,000	0.749	59,023,000	0.690	110.67	92.12	101.96
Pillars.......	13,866,000	.766	1,888,000	.704	13.61	91.91	12.51
Total....	67,197,000	.753	60,911,000	.691	90.65	91.77	83.18

The second best extraction was on the 1000 level where pillars were 30 to 50 feet thick. Ore on the 1000 level was generally lower in grade, and it was important that dilution be held as low as possible. The thicker pillars helped prevent dilution. Based on stope maintenance on this level, 21.5 percent of the ore was hard, 72.9 percent medium, and 5.6 percent weak.[23]

On the 720 level in the sulfide ore, the pillars were 7½ and 15 feet thick against broken ground, and the extraction was lower than with the other two methods. Rock in the stopes on the 720 level was classed as medium to soft.

[23] Fletcher, J. B. Ground Movement and Subsidence From Block Caving at Miami Mine. Paper presented at Annual Meeting, AIME, San Francisco, Calif., Feb. 15, 1959, p. 8.

No attempt was made to mine the 7½- and 15-foot pillars. An attempt was made to mine some of the 30- and 50-foot pillars on the 1000 level, but low extraction results indicated that some of the ore presumably left in the pillars had probably been recovered when adjacent stopes were mined. Extraction for some partition pillars on the 1000 level is shown in table 4.

TABLE 4. - Mining results in 1000-level pillars

Amount of ore expected....................tons..	11,164,000
Amount drawn....................do..	1,752,000
Grade expected....................percent copper..	0.747
Grade drawn....................do..	0.716
Extractions, tonnage....................percent..	15.69
Extractions, grade....................do..	95.85
Extractions, copper....................do..	15.04

Control-Raise Spacing

The spacing of draw points was designed to give good draw control and especially to counteract a tendency of Miami ore to draw vertically over control points in small pipes that caved through broken ore to the waste capping with very little spread. When piping developed, waste capping followed the ore downward as it was pulled, causing dilution and leaving columns of ore in place between draw points.

Piping was minimized by spacing draw points closely. In the first block-caving operation, control was provided by a four-way chute set at the top of the grizzly raise from which four finger raises reached the undercut level on a spacing of 12½ feet in a square pattern (fig. 19). The fingers were driven about 3½ feet in diameter and completed by funneling at the undercut level, leaving little, if any, flat-topped pillar between them. This spacing was satisfactory and prevented piping. However, in practice, operation was unsatisfactory because the draw through the grizzly raises from multiple draw points was difficult to supervise, and the cost of building and maintaining the intricate four-way chute sets was high. Consequently the system was replaced by a design that shifted draw control to the bottom of the raise on the grizzly drift level. The grizzly raise was called a control raise and finger raises were abandoned.

Elimination of finger raises increased the spacing of raise openings into the undercut to 25 feet, and this was found to be too wide to prevent piping. The spacing of draw points was changed so that raises reached the undercut level with a spacing of 16-2/3 feet parallel to the control drifts and 18-3/4 feet normal to them (fig. 20). Raises were belled out at the undercut level during the course of undercutting. A corresponding reduction was required in spacing of control drifts from 50 feet to 37½ feet, and of transfer raise branches and grizzlies from 25 feet to 16-2/3 feet. This control-raise design served throughout the remaining life of the operation.

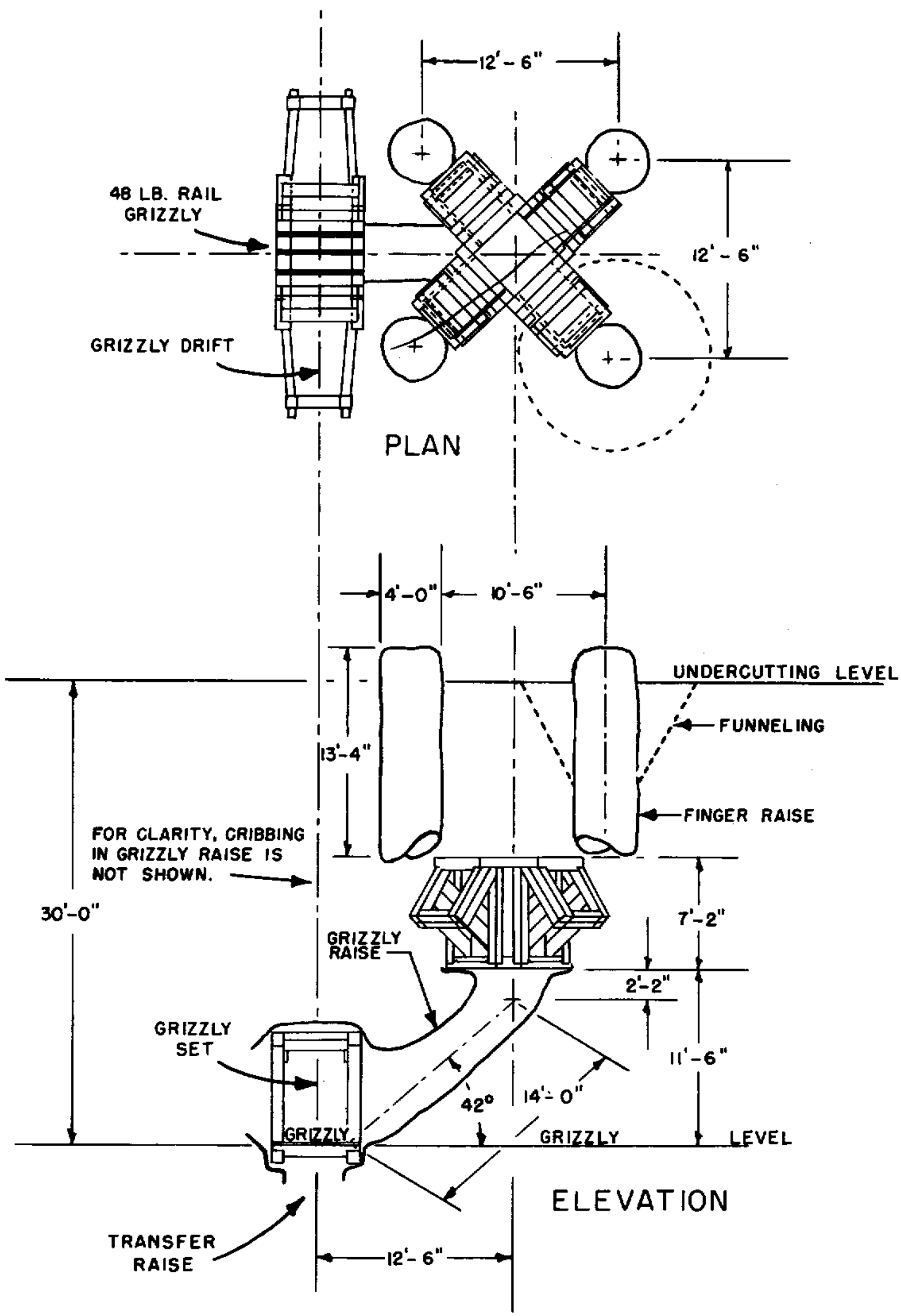


FIGURE 19. - Grizzly Raise and Finger Raises With Four-Way Chute Set for Control.

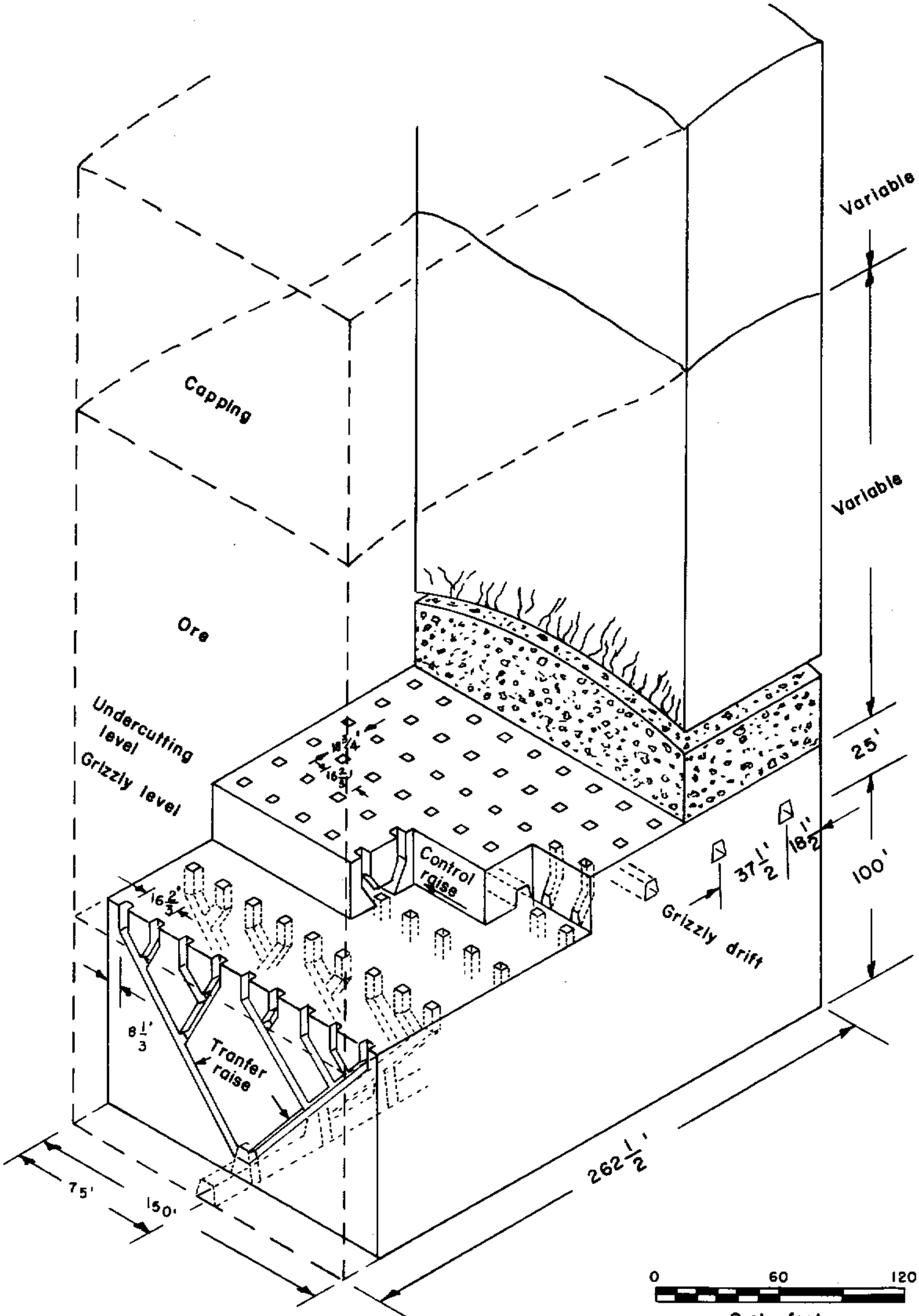


FIGURE 20. - Typical Gravity Caving Block.

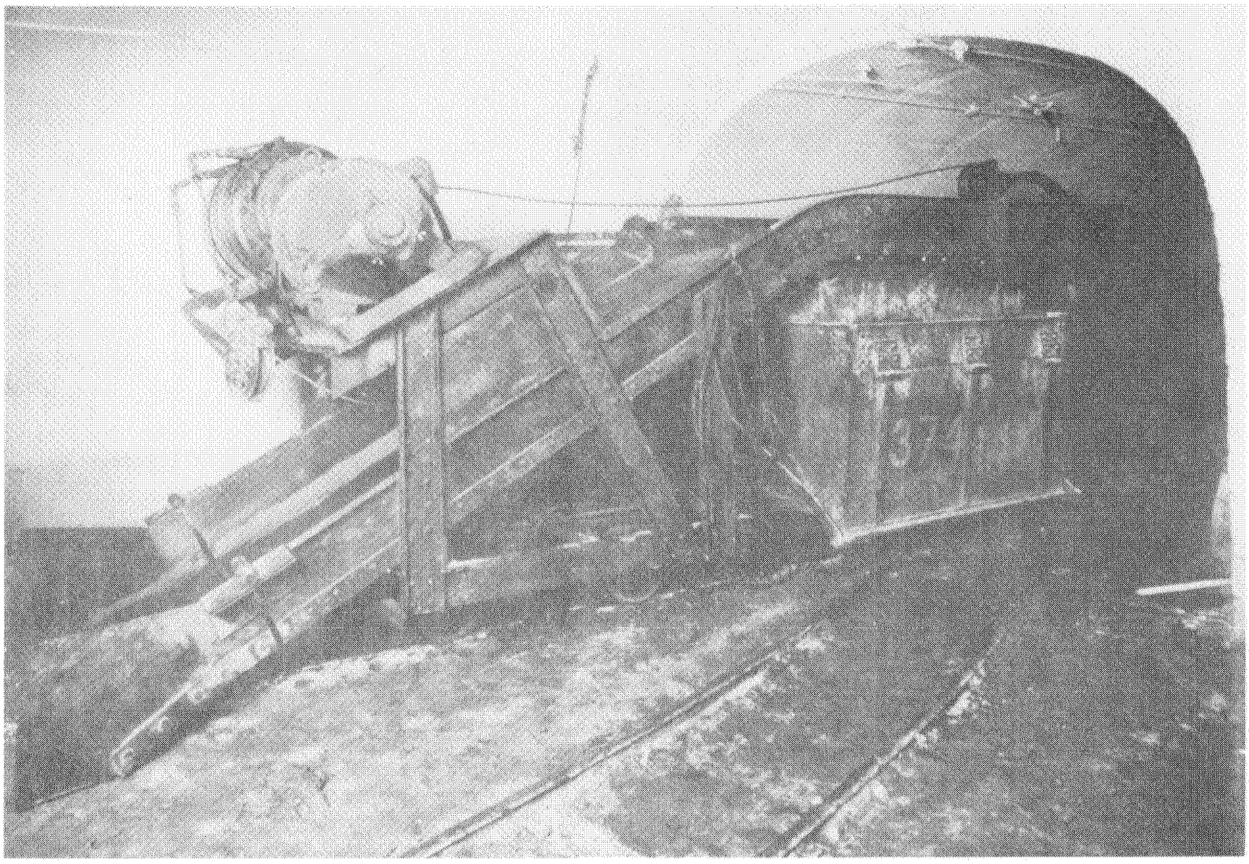


FIGURE 21. - Slusher Hoist and Scraper Ramp for Loading Cars.

Construction Methods

The methods used in constructing the extraction openings for the full-gravity system were generally serviceable throughout the block-caving period. Introduction of lateral ore transfer and unforeseen weight problems sometimes necessitated changes in design and function of some openings.

Mounted drifters were used in all drift headings until the introduction of air-leg-mounted (jack-leg or pusher) drills, which were adopted for medium and soft ground.

Rock-bit and drill-rod use followed changes in industry-developed design. Bits forged integrally with rods gave way to threaded detachable bits, and finally to throwaway-type detachable bits.

Originally, all ore broken in development headings on levels with track haulage was handled by scrapers. Broken rock was slushed to loading ramps and dumped into cars (fig. 21).[24] In 1932, air operated, track-mounted

[24]McDermid, A. J. Underground Scraping Practice. Eng. and Min. J., v. 130, No. 8, October 1930, p. 390.

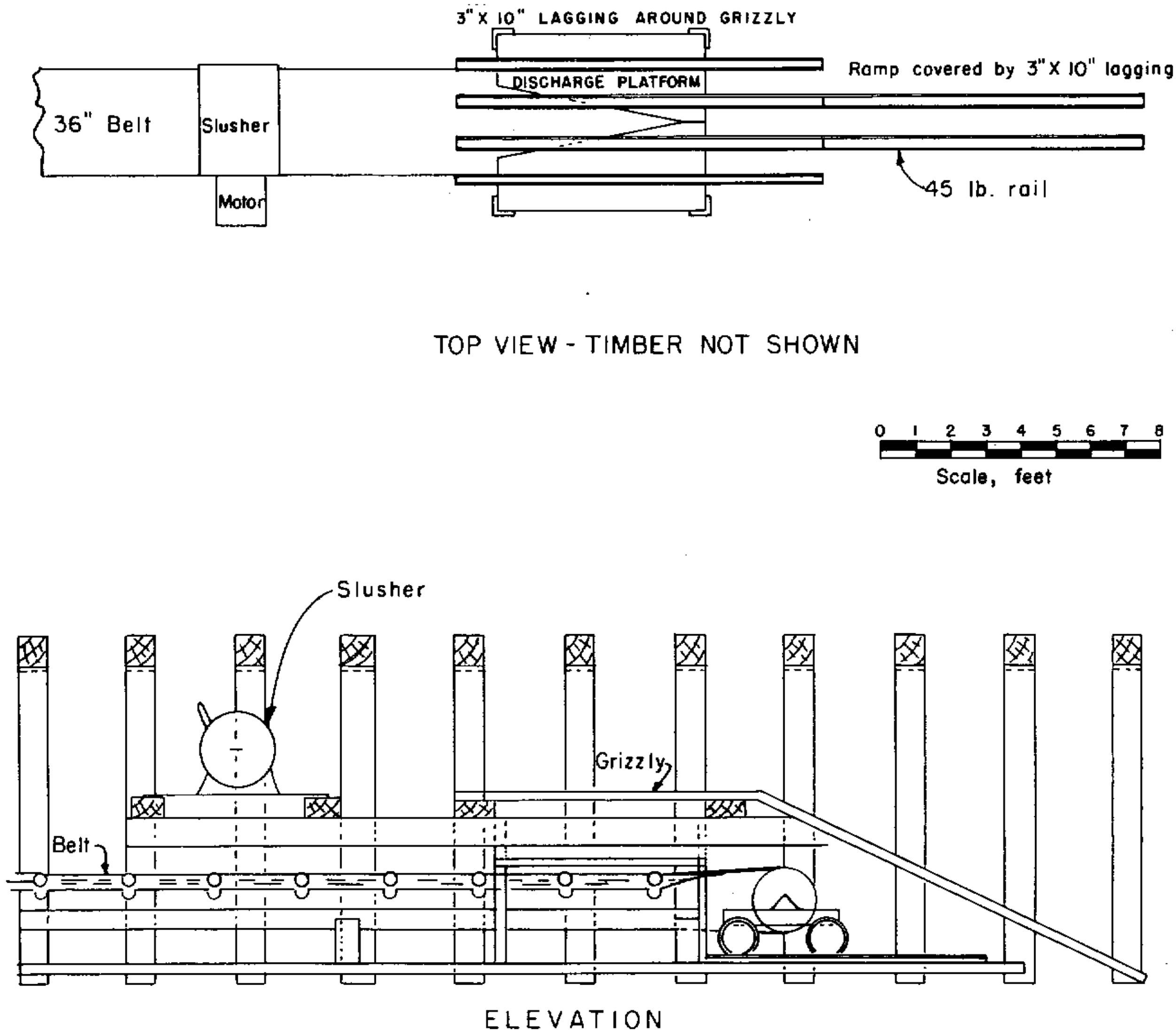


FIGURE 22. - Scraper Ramp for Loading Belt Conveyor During Development.

rocker-shovel loaders were adopted for haulage-level work and some control-level work. Scrapers continued to be standard equipment for moving ore that was to be discharged into nearby raises. Scrapers were used in construction work in conjunction with conveyors and temporary ramps when the equipment was to be used later for ore transfer (fig. 22).

Except for the use of concrete in shafts and loading pockets, ground support was originally accomplished with timber. Wood was replaced in whole or in part by steel in repair and new applications when timber was found to be unsatisfactory from an overall cost standpoint. Transfer chutes in loading drifts were normally constructed of timber with some steel parts in gates. Concrete linings and all-steel chutes were used in a few ore passes that handled large enough tonnages of ore to justify the higher initial cost, as in some of the ore passes serving the auxiliary lateral transfer system (fig. 16).

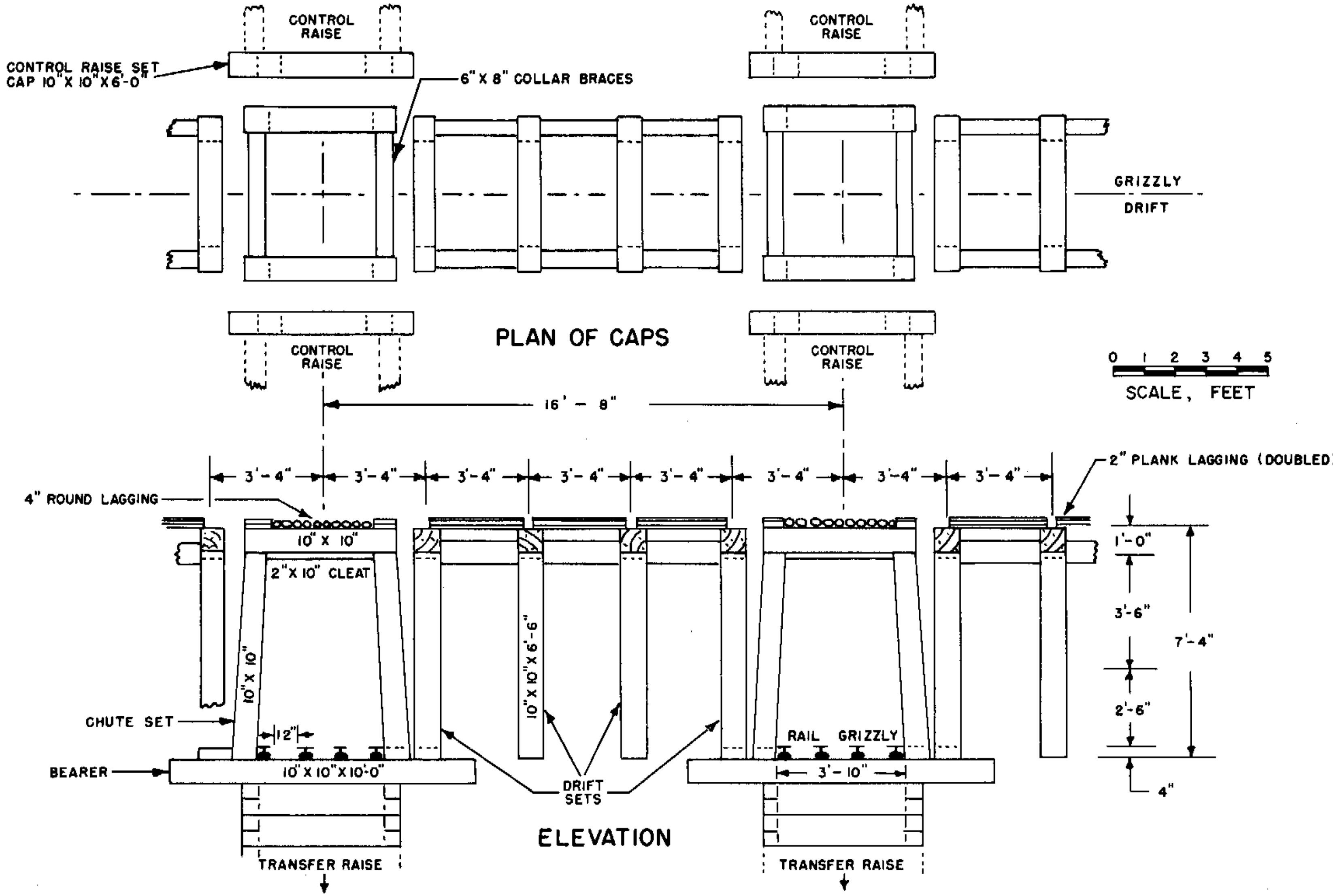


FIGURE 23. - Timber Support for Grizzly Control Drift.

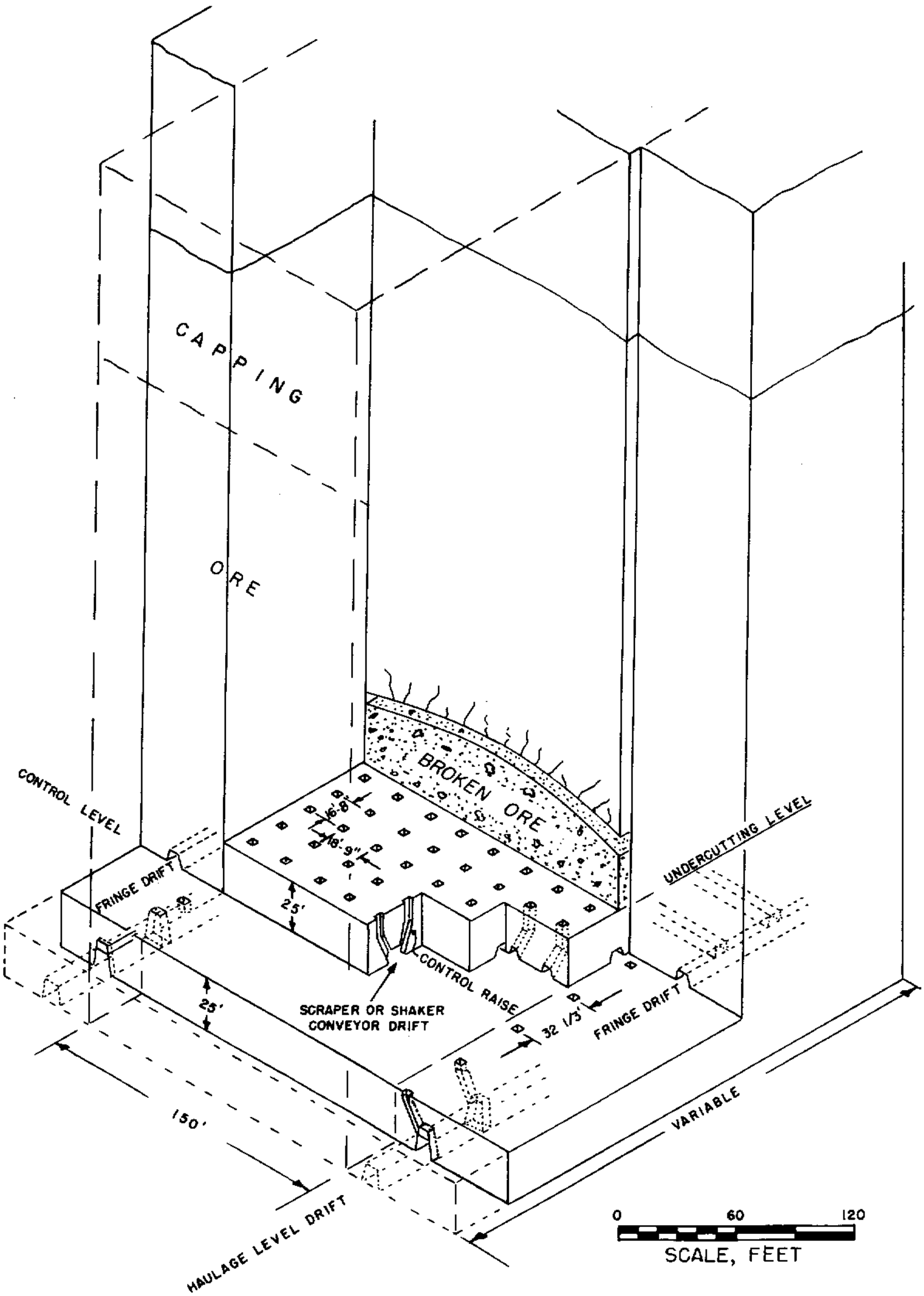


FIGURE 24. - Caving Block With Mechanical Lateral Transfer of Ore on Control Level.

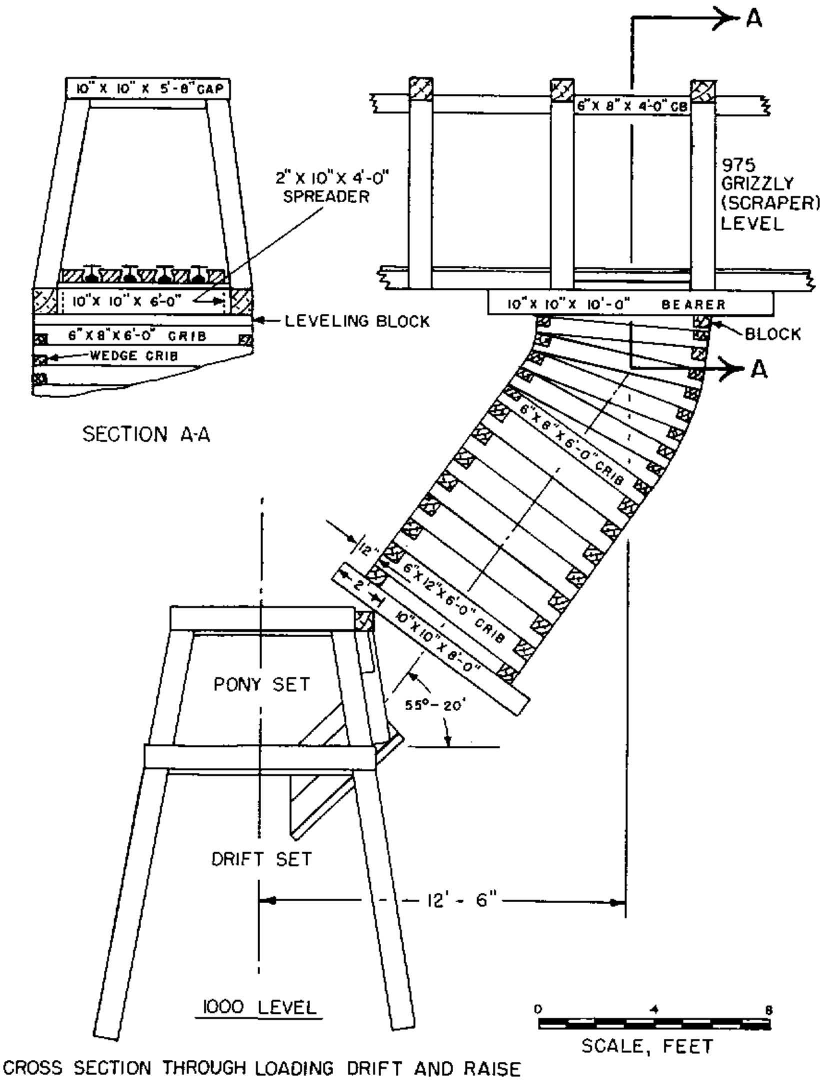


FIGURE 25. - Scraper Transfer Setup for Low Blocks.

Conventional three-piece drift sets were used in fringe and grizzly drifts with modifications at control raise locations (fig. 23). Reinforcing sets were placed inside of the regular sets when extra support was needed in control drifts used for shaker conveyors. Circular three-segment steel sets replaced timber in control drifts when scraper transfer was adopted, because maintenance of either conventional steel or timber sets having the 5-foot clear width needed was costly and time consuming. Support was not used in stub or boundary drifts except locally for safety reasons.

Ore Transfer Systems

The design of facilities for moving ore from stopes to haulage levels was dictated chiefly by two important considerations, the height of the block to mined and the vertical distance between stope sills and haulage levels. Other factors such as availability of material and labor, character of the ore or anticipated amount of maintenance also influenced the choice.

Two methods were used, a controlled gravity system and gravity combined with mechanical lateral transfer of the ore. The basic design at Miami was the full-gravity system by which the ore dropped from the grizzly control level to the haulage level solely by the use of inclined, branching transfer raises (fig. 20). A minimum distance of 125 feet between undercut and haulage levels was required to furnish sufficient lateral movement in transferring ore. It was the most economical from the standpoint of performance, extraction, and operating costs under conditions where it could be used to advantage. However, stope preparation costs were high, and a column of ore sufficiently high to amoritze the high development cost was necessary.

In parts of the ore body where these conditions could not be met, full-gravity transfer was replaced by lateral transfer at the control level (fig. 24). In this system, ore moved in shaker conveyors or scrapers from control points to short ore passes through which it dropped to the haulage level (fig. 25). About 4 million tons of ore were moved over shaker conveyors before they were replaced by scrapers, after it was proven in simultaneous tests that scrapers were superior for Miami ore. Table 5 compares the performance of the three systems in drawing similar ore blocks.

TABLE 5. - Production from standard blocks in tons per man-shift for three transfer systems

Month	Gravity, Block No. 111	Scraper plus gravity, Block No. 112	Shaker plus gravity, Block No. 114
January	251	192	132
February	169	109	118
March	214	155	114
Average	211	152	121

The gravity system was superior in operating efficiency (table 5) and more economical in direct labor cost. However, development costs were much higher for a stope of given lateral dimensions using the all-gravity system than for one of equal size using the combination method. The greater development cost offset the advantage of lower drawing costs unless spread over large tonnage. Table 6 is a comparison of the estimated development requirements for gravity and scraper-gravity transfer on blocks of similar lateral dimensions.

TABLE 6. - Comparison of estimated development requirements

	Gravity block	Scraper block
Transfer raise.............feet..	3,520	700
Fringe drift................do...	1,725	2,170
Total...................do...	5,245	2,870

Development required for all-gravity transfer in the example given exceeded that for scraper-gravity transfer by 2,375 feet or (at an assumed cost of $20.00 per foot) $47,500.00. The area of a stope 150 feet by 267.5 feet is 40,125 square feet and, at 12.5 cubic feet per ton, represents 3,120 tons per vertical foot of stope. If we assume there was a saving of 10 cents per ton in operating cost on the gravity stope, it required a column of ore 148 feet high to reach the break-even point. Other factors being equal, a block more than 150 feet high was mined as a gravity stope, while one less than 150 feet high was mined as a scraper stope. The actual development requirements for typical stopes are shown in table 7.

The choice between full-gravity transfer and gravity combined with lateral transfer of ore was further influenced by the distance between stope sills and haulage levels. A vertical distance of 125 feet between stope sills and haulage levels (100 feet between control and haulage levels) was the practical minimum for full-gravity transfer. With less separation, not enough lateral movement of ore could be effected in an inclined, branching raise system without providing an excessive number of loading drifts. Thus, full-gravity transfer was not applicable to parts of the ore body that bottomed less than 100 feet above the haulage level.

TABLE 7. - Development record for typical stopes

	Feet	Manshifts
	Gravity stope--310	
Haulage drift	291	379.75
9 chute stations (3 pony sets each)	-	47.5
18 chutes and gates	-	47.0
18 transfer raises	3,566	3,532.0
Fringe and grizzly drifts and 78 grizzly sets	2,418	1,908.0
156 control raises	4,368	1,097.75
Undercut drift	3,076	960.625
Undercut pillars, 36,446 sq ft	--	824.125
Corner raise	151	22.875
Miscellaneous	-	14.0
Total shifts	-	8,833.625
Stope area...sq ft	-	48,750.0
Square feet of stope developed per manshift	-	5.52
	Scraper stope--309	
Haulage drift	196.67	249.0
4 chute stations (3 pony sets each)	-	43.125
4 chutes and gates	-	12.5
4 transfer raises	688	629.625
Fringe and scraper drifts and 10 grizzly sets	1,185	895.625
100 control raises	2,800	860.625
Undercut drift	2,072	734.125
Undercut pillars, 22,659 sq ft	-	538.875
Total shifts		3,970.5
Stope area...sq ft	-	31,250.0
Square feet of stope developed per manshift	-	7.87

Further, it was found in the course of operations that a rapid weight transfer from active stopes to the haulage-level openings underneath took place when the distance between was less than 125 feet. This was attributed to a tendency of soft ore to develop a semi-plastic condition that readily transmitted ground pressure. In some instances, extreme pressures at the haulage level developed even with the full 125-foot vertical interval. Extreme weight around haulage level openings caused support failures, downtime for repairs, and delays in drawing operations. Such delays in turn caused further build-up in weight, more repairs, increased mining costs, poor stope performance, and lower ore extraction. Lateral transfer of ore was adopted to improve performance under these conditions even though, in some cases, sufficient head room for full-gravity transfer was available.

Full-Gravity Ore Transfer

The design of the full-gravity ore transfer raise system for block caving was based on experience gained in earlier mining systems. The minimum practicable angle for raises was 53 degrees. In practice, haulage levels were driven 125 feet below the undercut level and inclined branching raise systems were driven at a minimum angle of 53 degrees to serve a stope width of 150 feet.

Haulage Level Laterals and Loading Drifts.--The basic design of all haulage level workings was the same. Loading drifts under the blocks were driven in the same manner as the laterals from the shaft and modified later by installing pony sets and chutes to accommodate transfer raises. The positioning of loading drifts in relation to the cave blocks served underwent changes during the course of the block-caving operation, but, for full-gravity caving, they were centrally located under block lines (fig. 13). Their 150-foot spacing was correlated with a minimum vertical spacing of 100 feet between grizzly-control and haulage levels to provide the minimum raise slope on which ore would run (53° above the horizontal). Sufficient lateral movement was effected in the descent of the ore stream through the transfer raises to serve a standard 150-foot-wide block line with one loading drift.

Changes in the location and orientation of loading drifts were necessary in later operations for the following reasons: (1) block sizes changed; (2) parts of the ore body were less than the minimum 100 feet above the haulage level needed for full-gravity ore transfer; (3) and unforeseen, severe ground pressure developed around haulage-level openings underneath active stope blocks. In the later phases of block caving, loading drifts were constructed along block boundaries and finally in solid ground entirely outside stoping areas. This minimized the delay and cost of timber repair on the haulage level.

Haulage-level openings were driven 10 feet high and 9 feet wide to accomodate standard timber support (fig. 26). A 6-inch air line, a 2-inch water line, and a 6- by 12-inch wood-lined drainage ditch were installed at one side of the ballasted track.

Two column-mounted automatic-feed drifters, later supplanted by air-leg-mounted drills in medium and soft ground, were used to drill rounds averaging 17 holes 6½ feet deep. Rounds were blasted with about 50 pounds of 40-percent ammonium gelatin dynamite.

After a blast and before mucking was started, a boom-supported timber cap was placed close to the new face and lagged to support the back. After the round was mucked out, the new set was completed by setting posts and blocking.

Drifts were advanced on a 2-shift cycle: 5 hours setting up, drilling, and blasting; 2 hours advancing booms and setting cap; 4 hours mucking; and 4 hours timbering. Blasting was done between shifts with about 1 hour required to ventilate headings.

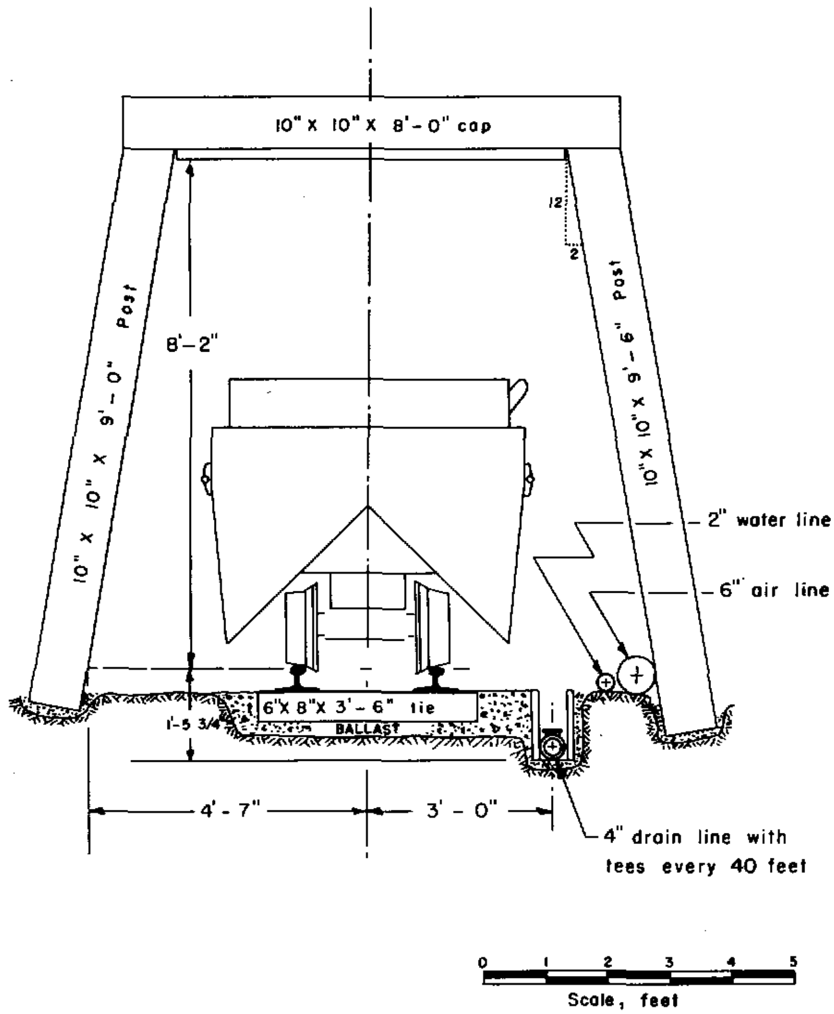


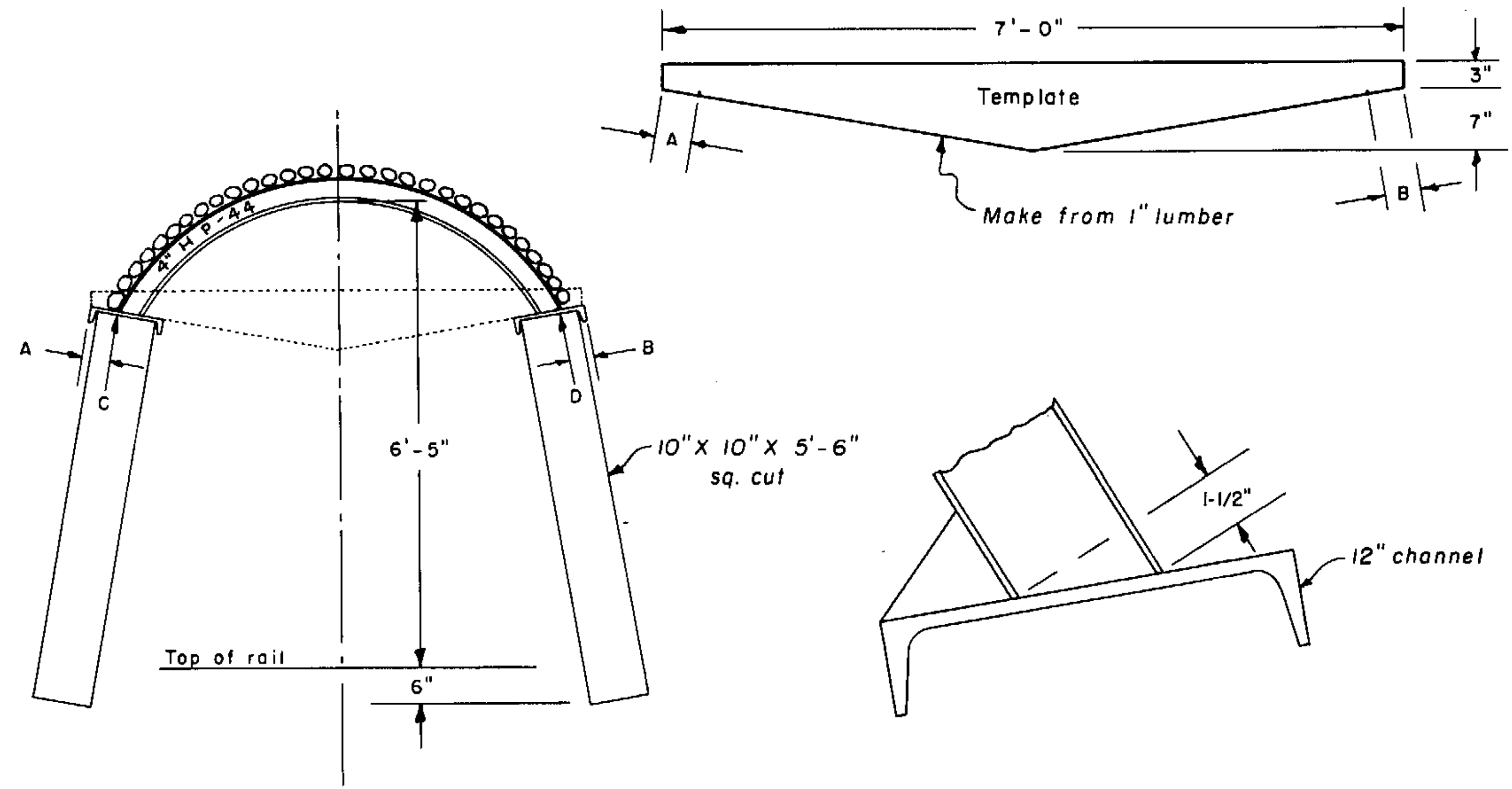
FIGURE 26. - Standard Haulage Drift.

The standard design of support for haulage level openings is shown in figure 26. Standard spacing for these 10- by 10-inch treated timber sets was 6 feet 3 inches. In heavy ground, this spacing was halved. Sets were close-lagged on top with 2-inch planks or with 3- or 4-inch round timber. Side lagging with 2-inch spaces between pieces was used in sloughing ground. H-beams were used for caps in turnouts where the span was too long for timber. Treated timber deteriorated eventually, and cap failures occurred at points that were in service a long time. In heavy ground, failing sets were replaced satisfactorily with sets having arcuate steel caps fabricated out of 4-inch H-beams (fig. 27).

Loading drifts were enlarged at 50-foot (later 37½-foot) intervals to accommodate transfer chutes. Pony sets were installed above three sets of timber at the location of each pair of transfer raises (fig. 28).

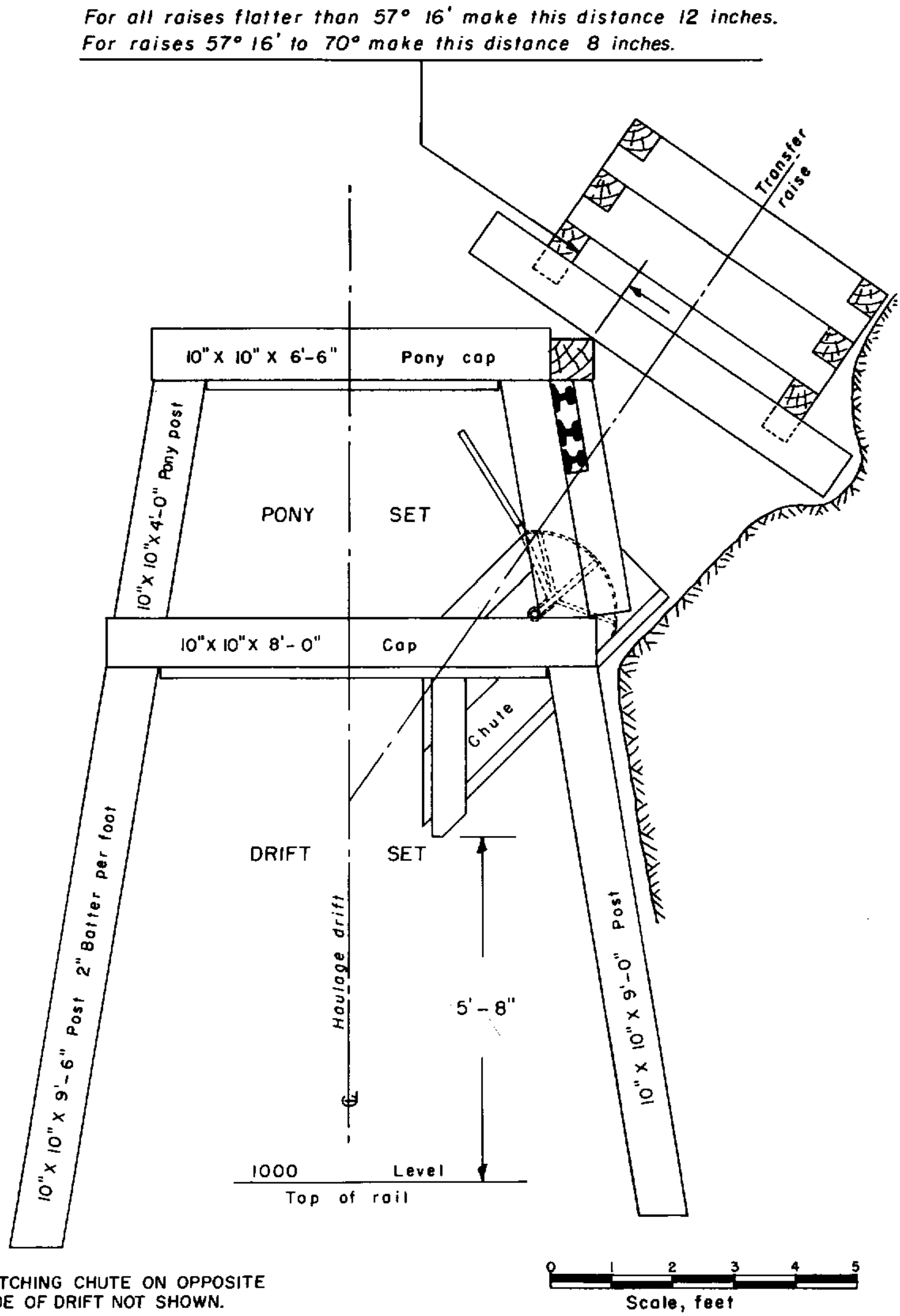
Haulage level construction was done by company employees under a bonus system. Work in the headings, including drilling, blasting, loading cars, and timbering was charged as direct labor, while supervision and transportation of broken ore and supplies was charged as indirect (other) labor.

The amount of haulage level development in the stoping area varied from year to year according to production schedules. Loading drifts, in particular, were not driven in advance of actual need in order to minimize maintenance requirements. In 1955, about 2,200 feet of drift were driven at an average rate of advance of 1.27 feet per man-shift for direct labor or 0.79 feet per man-shift including all labor.



Make measurement "A" equal measurement "B"
Make template pass over steel segment at "C" and "D"

FIGURE 27. - Replacement Drift Set With Arcuate Steel Cap.



MATCHING CHUTE ON OPPOSITE
SIDE OF DRIFT NOT SHOWN.

FIGURE 28. - Loading Drift Chute Set.

Control-Level Laterals.--Access laterals from the shaft to the stoping areas on control levels were similar to haulage laterals in design and method of construction. However, the control-level-lateral posts were shorter, 7 feet high inside timber.

Fringe, Stub, and Boundary Drifts.--Fringe and stub drifts in any block were driven on the grizzly level before the transfer-raise system for the block was ready for use. If service raises were not available, broken ore was disposed of in raises from the haulage level specifically driven for this purpose. As soon as possible, a stub was driven from the fringe drift to the first grizzly position in a control drift and the main branch raise for that position completed to provide for disposal of development rock. Loading was done at different times with track-mounted rocker-shovel loaders, with scrapers, and by hand.

Boundary drifts were driven above the grizzly level from the corner raises, using scrapers to move broken ore to a raise. The air-leg drill was particularly useful for intermediate-level work.

The ground in the corner raises and boundary drifts generally stood without support during construction, and no support was used except for light timbering in the raises.

Inclined Branching Transfer Raises.--The design of the transfer raise complex for block caving, while adopted from earlier panel-caving practice, differen in two respects: At draw-control points on the control level, ore passed through grizzlies in the floor of grizzly drifts directly into transfer-raise branches instead of being trammed along the control level first, as in the tramming drifts of the panel-caving system. Further, each raise complex was in line with and served a single grizzly drift, instead of being at right angles to and connected with all of the tramming drifts in a stope block, as in panel caving.

Transfer raises accomplished the functions of dropping ore from grizzly to haulage levels, providing as much as 75 feet of lateral movement, and providing intermediate storage. Each pair of inclined, branching transfer raises delivered ore from the grizzlies of a control drift 150 feet long in a stope block to a centrally located loading drift under the center line of the 150-foot-wide stope blocks (fig. 20). The paired raise complexes, spaced 37½ feet apart along loading drifts, were in planes at right angles to the loading drift and in line with corresponding grizzly drifts above. Raise branches connected with the grizzly level and ended at grizzlies installed at draw points. In early block-caving operations, grizzlies were spaced at 25-foot intervals in grizzly drifts 50 feet apart, corresponding to the spacing of control raises at that time. Later, when closer spacing was adopted for control raises, grizzly spacing was changed to conform, 16-2/3-foot intervals along grizzly drifts spaced 37½ feet apart. As shown in figure 20, each raise complex and associated grizzly drift served nine draw points under a stope section 150 feet across and 37½ feet thick.

Loading drifts were not always centered with relation to stopes and the width of stopes was not always 150 feet. Transfer raise design, shown in

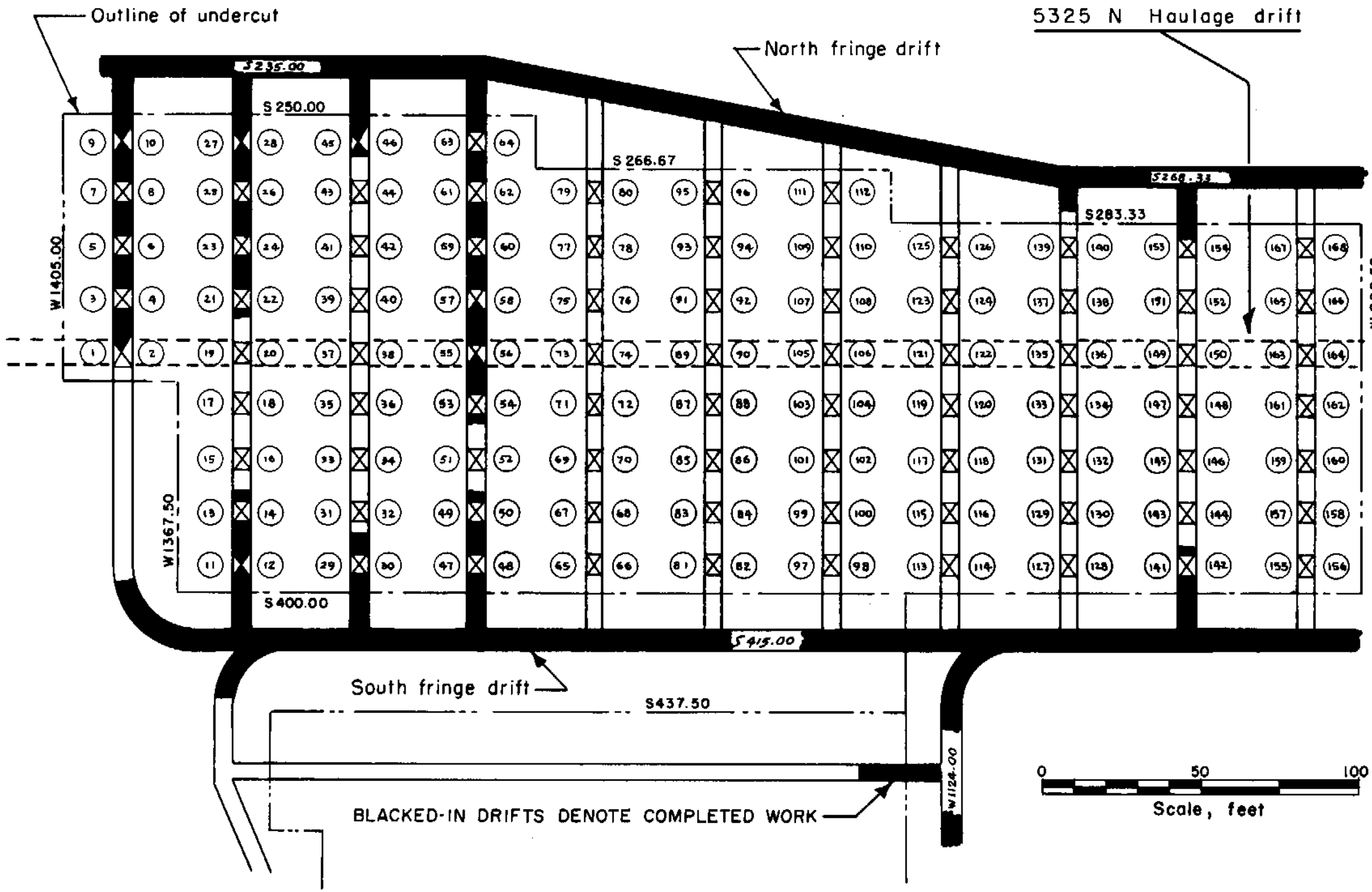


FIGURE 29. - Progress Map of a Typical Gravity-Stope Grizzly Level.

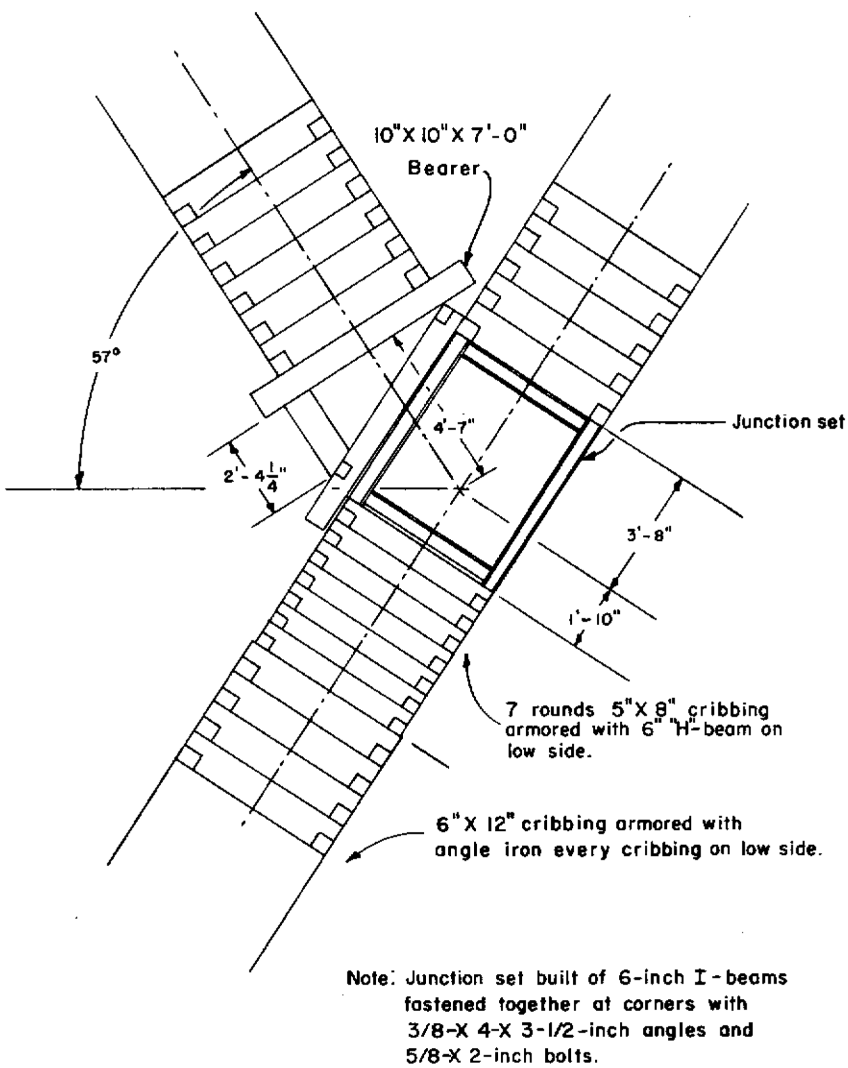


FIGURE 30. - Junction Set in Transfer Raise.

in figure 20, was changed to conform to local conditions. Changes were made to fit each case. After lateral transfer by conveyor or scraper replaced the full-gravity system, transfer raises were simple ore passes (fig. 25).

Transfer raise chutes were constructed in pairs in the enlarged sections of loading drifts prepared for them (fig. 28). The main branch of each transfer raise was driven first. At the same time, fringe drifts at the grizzly level above were being advanced, and, from them, stub drifts to the points where the main transfer raise branches would intersect the grizzly level (fig. 29).

The raises were driven in pairs by two miners, each man working in a raise with assistance from his partner when necessary. All raises were driven 4 by 4 feet inside timber on a slope exceeding the minimum 53° angle above the horizontal on which ore would run. Six by twelve-inch cribbing, resting on a bearing set above the chute sets, was installed as raising progressed (fig. 28). Junction sets fabricated out of 6-inch H-beams were installed where branch raises joined (fig. 30). Three sections of 90-pound rail were installed above each chute to protect the caps of the haulage drift pony sets from abrasion. Cribbing was subject to greater wear at certain points, particularly where a raise flattened. The wear points in main transfer raise branches were protected by placing wear plates of 3- by 3- by 5/16-inch angle iron over the upper inside edges of cribbing pieces. While being driven, raises were divided in two compartments with stulls placed 10 feet apart. The manway compartments carried a ladder and pipes but were not completely lagged off. Entrance to both manway and ore compartments was through one temporary chute installed at the mouth of each raise.

From 7 to 10 holes 4 or 5 feet deep were drilled with light, self-rotating stopers and loaded with 40-percent gelatine dynamite. Rounds were blasted with electric blasting caps, using the regular mine blasting circuit. Raises were drilled on day shift, and blasting was done toward the end of that shift. Broken ore was pulled on night shift, giving powder fumes a chance to clear before the next day.

Raise branches, including the main branch, were not holed through until the corresponding grizzly above it on the control level was in place. Installation of the grizzly included sinking a 4-foot cribbed winze below the drift. The actual connection was made below the bottom of this winze to protect the grizzly. A 6-foot drill rod was left in a hole drilled with a jackhammer in the center of the bottom of the winze, and, when the raise reached the end of this steel, the distance through to the bottom of the winze was then known. Also, if the raise was off line, it could be brought back before the connection was made by using shims to align the cribbing. It was important to avoid overbreak or misalignment in making the connection in order to leave solid ground to support the 10- by 10-inch stringers on which the grizzly rested.

As soon as each main raise branch was completed and connected with the grizzly-level drift, it was cleaned out, and a permanent chute with a manually operated, overcut, art gate was installed at the haulage level (fig. 28). Later, an air-operated undercut guillotine gate (fig. 31) was used on some chutes. Next, the subordinate raise branches were driven without manways by crews who entered from the grizzly level. Timber and supplies were stored on the grizzly level, lowered down the main branch to the junction, then hoisted with rope and snatch block up the branches.

Grizzly Drifts.--In the full-gravity block caving system, the function of grizzly drifts was simply that of providing access to draw control points where ore was metered from stopes to transfer raises. Grizzly drifts were in line with the short dimension of stope blocks and with corresponding transfer-raise complexes. Large boulders were broken up and sized through 12-inch rail grizzlies in the floor of control levels at draw points.

A grizzly drift was constructed by advancing two headings toward each other from the main raise branches in each transfer-raise complex. Broken ore was disposed by slushing into the main raise branches. Next, grizzlies were installed at future control points, and connections were holed through from the corresponding subordinate raise branches below.

Grizzly drifts were supported with conventional timber sets modified at control points to accommodate chute sets and grizzlies (fig. 23).

Control Raises.--The function of a control raise was to provide a metered passage for the ore from stope to transfer raise. Control was necessary to regulate caving rate and weight increase on extraction openings at each draw point in accordance with practices aimed at maximum extraction and minimum maintenance costs. Two control raises, one in each wall of a grizzly drift above a single grizzly in the floor, constituted a draw point.

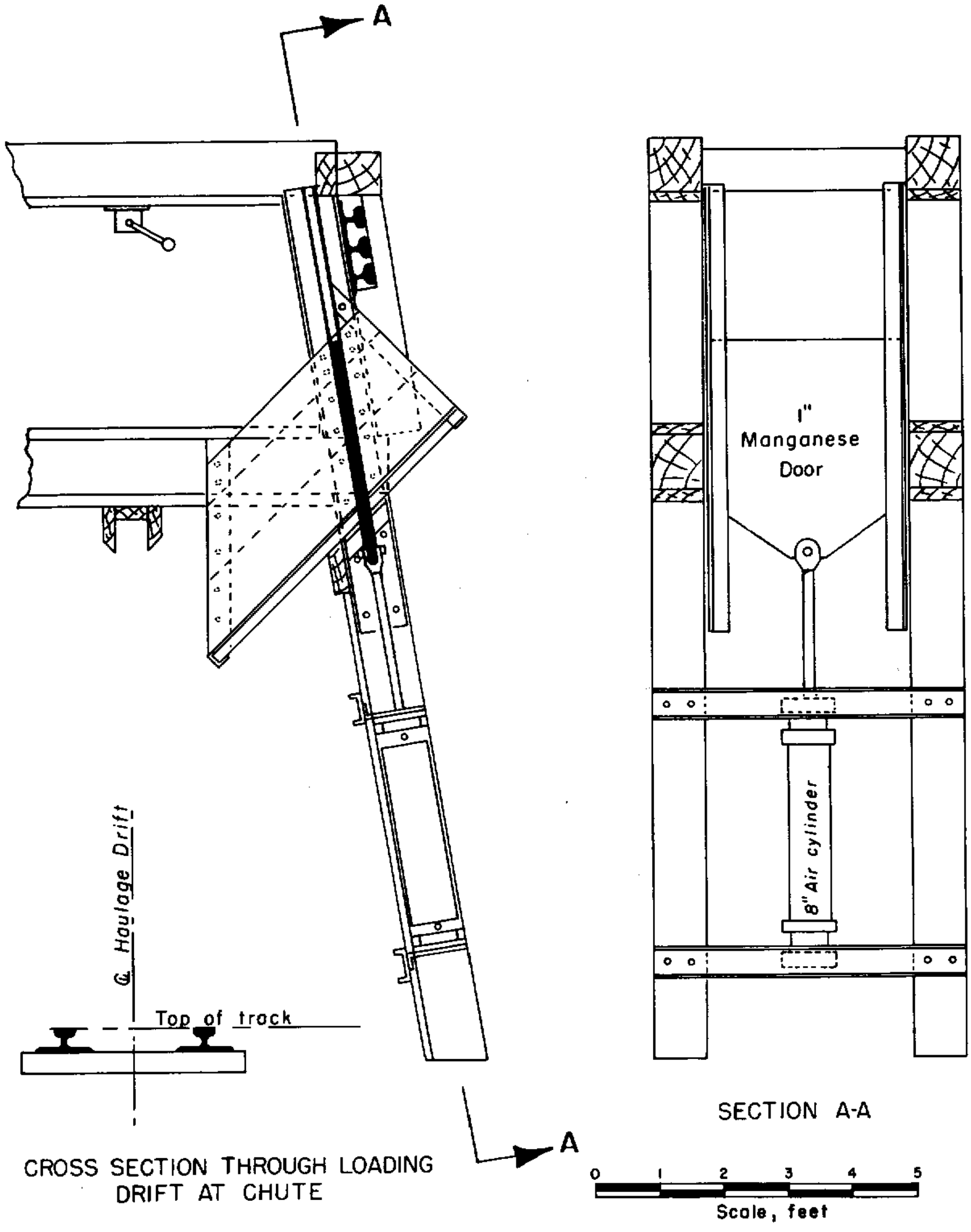


FIGURE 31. - Undercut Guillotine Chute Gate.

In the original design, four draw points fed ore into chute sets at the top of each grizzly raise and these were spaced at 25-foot intervals along grizzly drifts. It was found that the theoretical flexibility in draw control was not achieved with 12 or 16 control chutes feeding into one transfer raise. Moreover, the pillars around some finger raises broke up as drawing proceeded, the chute set was lost, and draw control had to be shifted to the grizzly level.

Raises were driven in pairs from opposite sides of each grizzly in control drifts. After the finger raise and chute set design was discarded the name was changed from grizzly to control raise. Control raises were driven on an incline 63° above the horizontal for 14½ feet and then vertically to a point about 2 feet above the floor of the future undercut (fig. 32). Raises were belled out at the undercut level during the course of undercutting. This control raise design resulted in a spacing of 16-2/3 by 18-3/4-feet at the undercut level and served throughout the remaining life of the operation.

Drilling in the raises was accomplished with light self-rotating stopers. Funneling of a raise at the undercut was started by enlarging the cross section in the last round drilled in the raise. The funneling process was completed along with undercutting.

Cribbed support was used in all control raises, 6 by 12 inches in the lower inclined sections, and 2 by 12 inches in the vertical sections (fig. 32). Any timbering in the vertical sections of raises was removed when the raises were funneled out to prepare them for ore drawing.

A 3-inch plank thrust through chain loops secured to the posts of draw sets was used to control the flow of ore to grizzlies (fig. 33). The chains also provided safe handholds for personnel crossing the grizzlies.

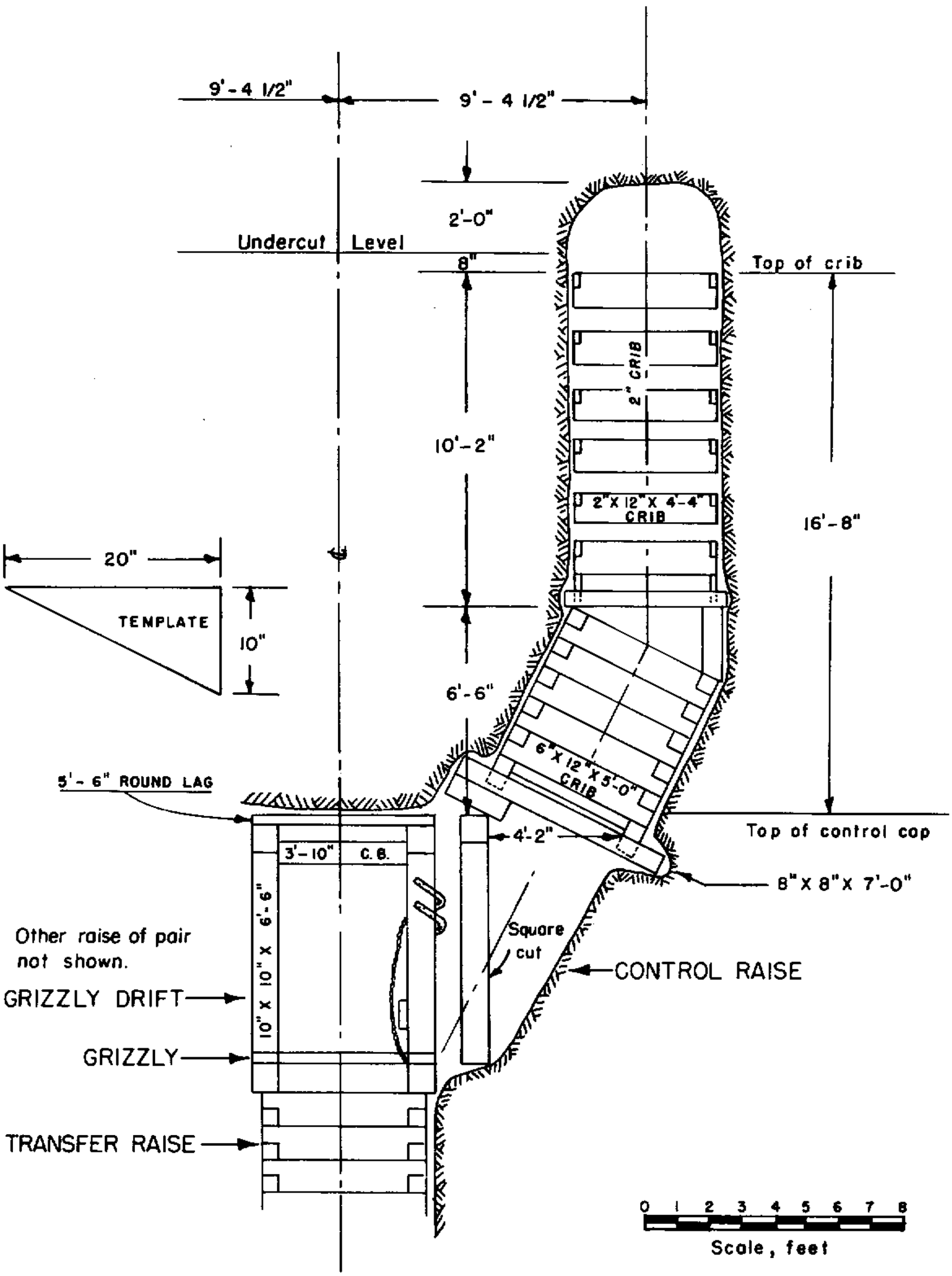
Ore broken in raise construction was disposed of by whichever method was adopted for ore drawing--discharge into transfer raises, lateral transfer by shaking conveyor, or scraper.

Lateral Ore Transfer

A substantial part of the Miami ore body bottomed within 50 feet of the 1000 haulage level; some 18 million tons was less than 125 feet above the haulage level, too close for full-gravity transfer. Construction of a new haulage complex at a lower level would have required a capital expenditure not economically justified by the amount of ore involved.

In 1938, a system of lateral transfer at the control level was introduced utilizing pan (shaker) conveyors or scrapers to move ore along control drifts to the edges of stope blocks and thence through ore passes to the haulage level (figs. 24 and 25). These drifts, sometimes called scraper or conveyor drifts, replaced the grizzly drifts of the gravity system and performed a function similar to that of the tramming drifts of the earlier panel-caving operation.

The practical operating limit for either pan conveyors or scrapers in production ore transfer was about 75 feet, and each control drift across a block 150 feet wide required two opposed units moving ore toward opposite block boundaries. The number and spacing of conveyor or scraper drifts under



VERTICAL CROSS SECTION THROUGH CONTROL POINT

FIGURE 32. - Grizzly Control Raise Design.

FIGURE 33. - Tapping a Control Raise.

each stope block remained the same as for the grizzly drifts, six or seven drifts driven parallel to the short dimension of blocks on 37½ foot centers. However, each control drift required only one ore pass at each end instead of the more costly branching-raise complex of full-gravity transfer. The ore passes were above and connected with loading drifts along block boundaries parallel to the long dimension of blocks. Thus, both ore passes and loading drifts were where ground pressures tended to be less.

Shaker Conveyors.--Two reciprocating (shaker) pan conveyors in an opposed arrangement served a line of control points in each control drift. The conveyor mechanism was suspended on chains from the caps of timber sets. At the draw points, aprons placed over the edge of the conveyors guided the ore from the control raises into the pans (fig. 34). The ore stream was carried along by a reciprocating motion imparted by drive mechanisms under the discharge

FIGURE 34. - Drawing Ore From Control Raise on Shaker Conveyor.

FIGURE 35. - Discharge End of Shaker Conveyor.

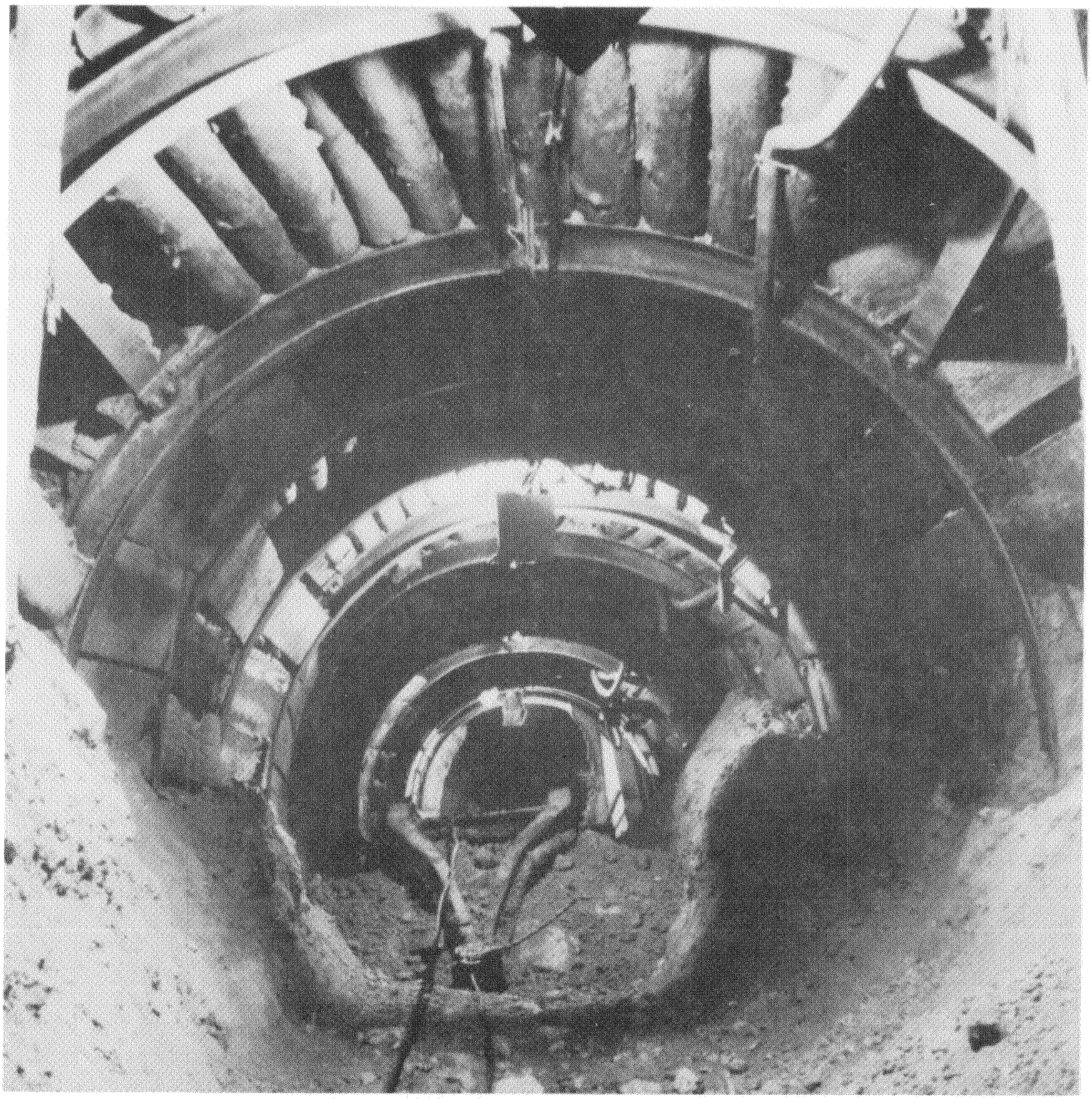


FIGURE 36. - Scraper Drift With Circular Steel Support.

ends of conveyors and was discharged through 10-inch grizzlies into ore passes (fig. 35).

The design of shaker conveyor drifts, including ground support, was the same as for grizzly drifts, and driving methods were similar. Rock broken in heading advance was handled by rocker-shovel loading into tramcars or by shaker conveyor fed by hand. The conveyor, installed in sections as the heading was advanced, became the permanent lateral-transfer setup.

An apparent advantage of pan conveyors was the 3-foot operating space required. Extra reinforcing sets could be placed inside the regular timber sets in bad ground and still leave operating clearance. In practice, failure of the reinforced sets was common. Installation, repair, and operating costs proved to be high, and repair of both timber and equipment was difficult in the restricted working space. Resultant delays in ore-drawing schedules affected stope performance.

Scraper Transfer.--Scrapers were adopted in place of shaker conveyors to improve lateral-transfer performance (fig. 36). Comparison tests proved that scraper transfer was superior at Miami even though ground support in the larger openings was more costly, and power consumption was one third higher than for shaker conveyors. Lower repair and labor costs for scraper operation (one scraper hoist operator against two conveyor attendants for each equipment unit) more than offset the disadvantages. Shaker conveyors were replaced by scrapers in all operations.

Two opposed scraper installations with tail pulleys at the center of a block served each control drift, pulling ore to ore passes at block boundaries.

Twenty-five hp, double-drum, electric hoists were used to pull a 42-inch scraper. A five-eighths, 6 by 19, regular-lay rope was used to pull the scraper, and a one-half inch 3 by 19 rope was used for a tail rope. About 480 feet of rope was required per scraper setup. Some information on rope performance is shown in table 8.

TABLE 8. - Scraper rope performance

Tons pulled..	1,785,888
Feet of rope required..............................	41,217
Cost of rope (1945)................................	$6,566.00
Cost per ton.......................................	$0.0037
Average tons per foot of rope......................	43.3
Feet per set of ropes..............................	480
Tons per set of rope...............................	20,784
Life per set of ropes......................months..	2.12

Scraper drifts were driven in the same manner as grizzly drifts, starting from ore passes at block boundaries. The permanent scraper installation for ore drawing was used to remove muck broken in heading advance.

Conventional support using timber or steel was tried in scraper drifts, which required 5-foot clear width. Because the heavy ground pressures encountered caused rapid distortion and failure of conventional support in wide drifts, circular sets were adopted (figs. 36 and 37). A circular set, assembled in place from three 4-inch, 13-pound, H-beam segments flanged and bolted together, proved to be satisfactory. Clear diameter inside the sets was 6½ feet, ample for scraper operation.

The circular sets had a longer service life than conventional steel sets and were easy to repair or replace. The first sets were installed with one

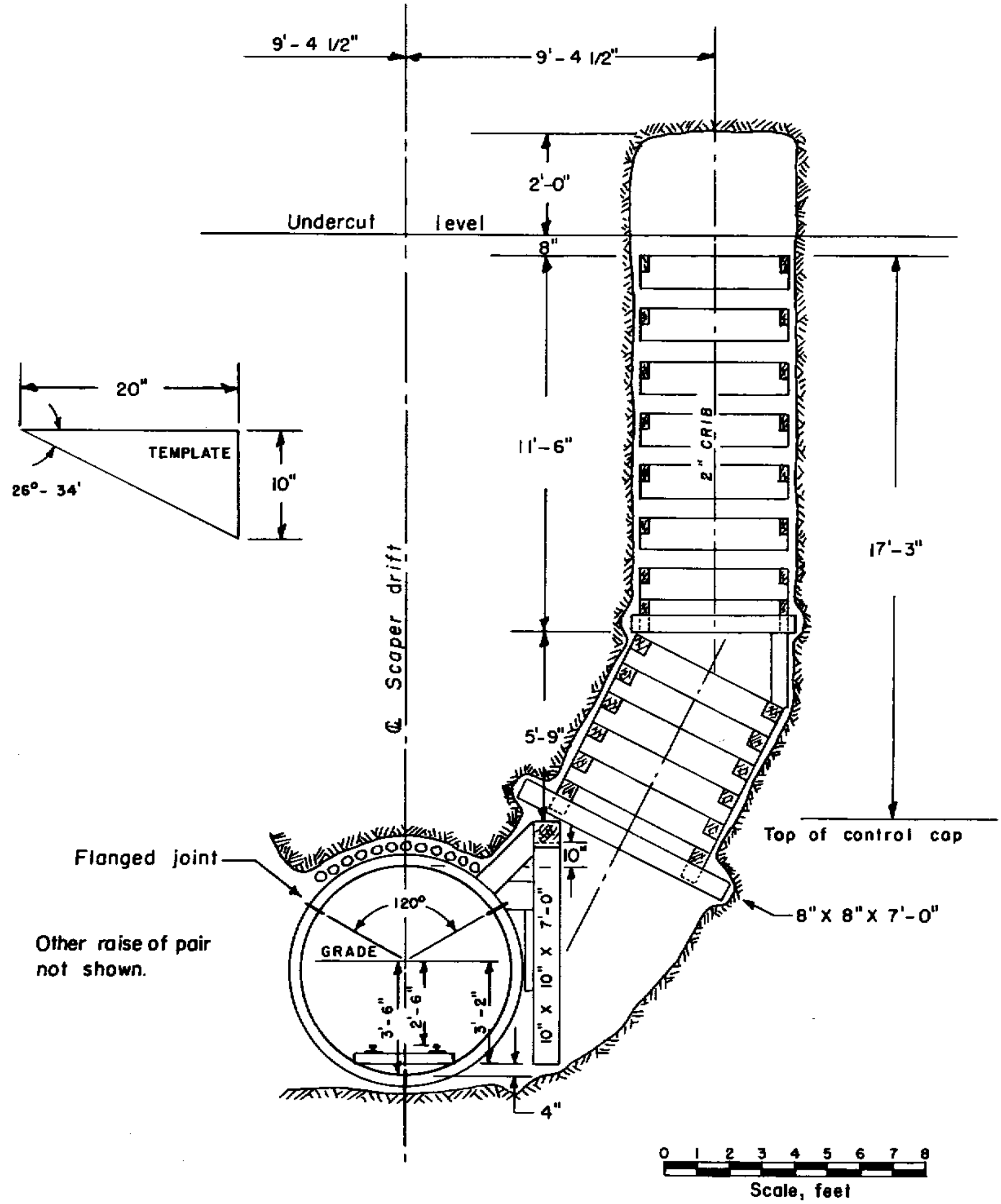


FIGURE 37. - Cross Section of Scraper Drift and Control Raise.

segment across the bottom so that joints were on each lower rib and in the center of the back. Failure usually occurred in the back, and it was found that installation of sets with one segment across the back permitted repair by replacing the top segment (fig. 37).

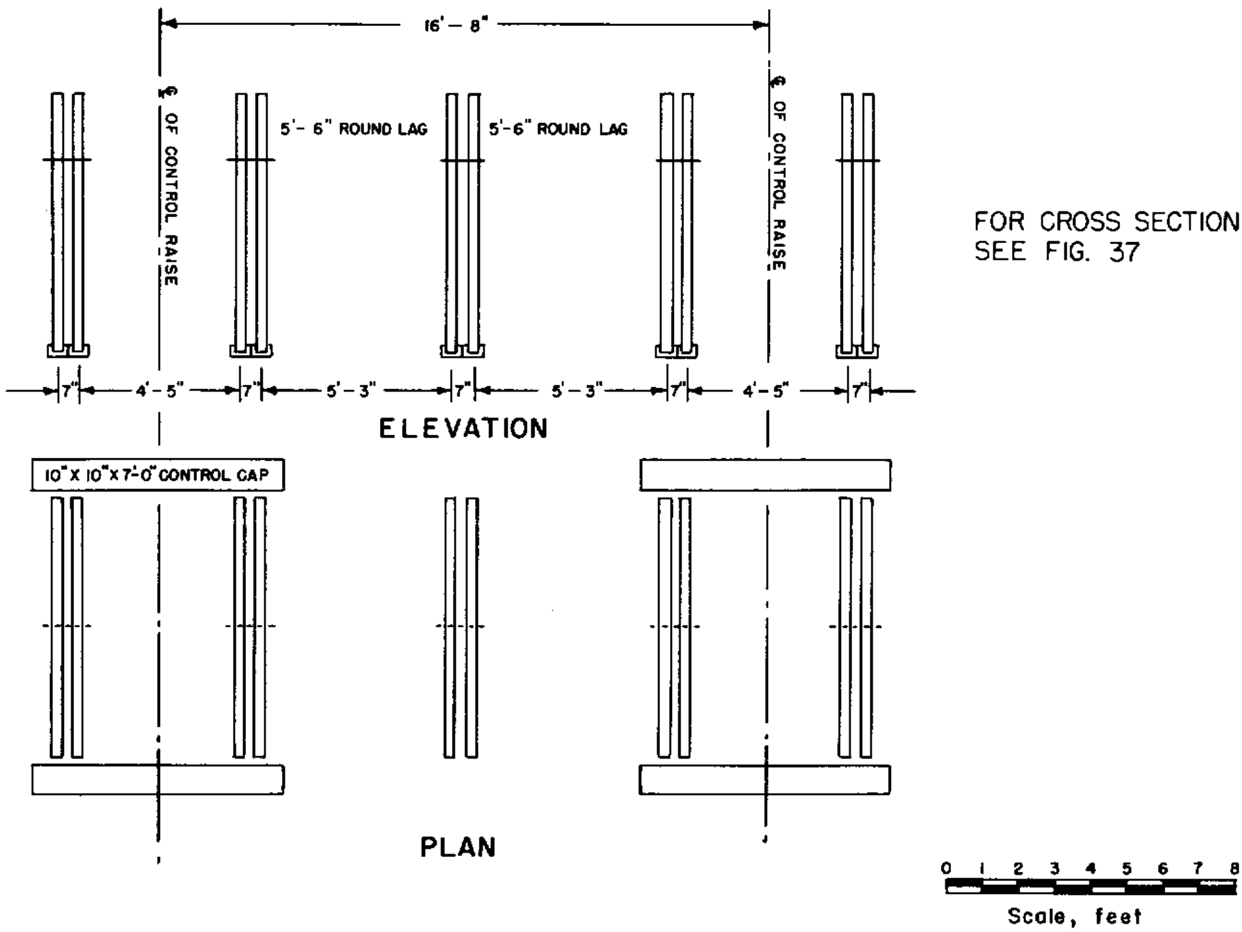


FIGURE 38. - Circular Steel Set Arrangment.

Three- to four-inch round timber was used for back lagging and 2-inch plank for side lagging. Lagging failed rapidly and had to be replaced often during the life of a stope. Rock pressures tended to shear the lagging along beam edges leaving the sets intact. Where single sets were used, a set failure destroyed two spans of lagging. By using circular sets in pairs, so that each span of lagging was independently supported, a set failure normally disturbed only one set of lagging. The arrangement and spacing of the double sets is shown in figure 38. The initial cost of the circular sets was 50 percent greater than for timber, but lower repair costs and salvage of the sets for reuse tended to equalize overall costs. However, the chief advantage of steel sets was reduction of downtime in drawing operations because of lower repair requirements. Steel sets, assembled with flange and bolt, proved more economical than those incorporating a yieldable joint.

Conventional steel sets fabricated out of 6-inch H-beams with 4-inch angles for collar braces were used at the slusher hoist installation in the ends of scraper drifts or in fringe drifts where more space was required. At those points this type of support stood up well, even though the openings were larger, because ground pressures near or outside the edges of active blocks normally were moderate.

Intermediate Lateral Ore Transfer.--Lateral ore transfer, other than on control levels, was first adopted to alleviate a ground support problem at the haulage level underneath active caving blocks. It was found that excessive ground pressures developed around openings, particularly in soft ore and where the pillar thickness between undercut and haulage levels was no more than 50 feet. This was true even when loading drifts were driven along block boundaries where ground pressures tended to be less. Support failures necessitated frequent repairs that interfered greatly with drawing schedules and resulted in poor stope performance. To correct this, haulage drifts were relocated outside the area taking weight and reoriented parallel to the short side of blocks.

With this change, haulage drifts were no longer in a position to serve ore passes from the control level directly. Intermediate transfer drifts just above the haulage level were driven length-wise of blocks along boundaries to form a link between ore passes and loading drifts, and equipped with belt conveyors. Two conveyor drifts, each connected with six or seven ore passes from the control level above and on opposite sides of the block, served a standard block setup of six or seven scraper drifts. Collecting belt conveyors in these drifts terminated just above the relocated loading drifts and discharged ore into short raises at these points (fig. 39). A conveyor drift often served more than one block along the same panel.

Conveyor sections about 210 feet long carrying a 30-inch-wide belt were used. Power supplied by a 15-hp induction motor was applied through V-belts and a variable speed reducer for smooth acceleration and deceleration during the frequent starts and stopes. Direction of belt travel could be reversed to move timber and supplies along the intermediate levels.

Ore, sized through 10-inch grizzlies at the top of ore passes, was distributed on collecting belts through a feeder chute sloping in the direction of belt travel (fig. 40). Chutes made of three-eighths-inch plate with a slot two feet seven inches long in the inclined bottom tapering from one foot eight inches at the discharge end to four inches at the top. Fine ore passed through the slot and formed a protecting cushion on the belt for the larger pieces, which slid down the V-slot and landed on the finer material. Counterweighted chutes rode on hinges so that they could be easily pushed up to give clearance for large ore fragments from chutes further down the line and for timber and supplies.

Intermediate belt-conveyor transfer was very successful. Belt operation was economical, and timber repairs in the small drifts were infrequent, relatively inexpensive, and could be made without interrupting the draw schedule.

Intermediate belt-conveyor drifts were driven with mounted drifters or air-leg drills, using scrapers to move muck from the headings. If the drift exceeded the length of a conveyor section (200 feet), a section was installed, and the slushing setup was moved ahead to continue the work. The scraper delivered ore from the heading to a loading ramp at the end of a conveyor and discharged it on the belt (fig. 22). When a belt-conveyor drift was extended under several blocks in the same panel, additional conveyor sections were installed as work proceeded.

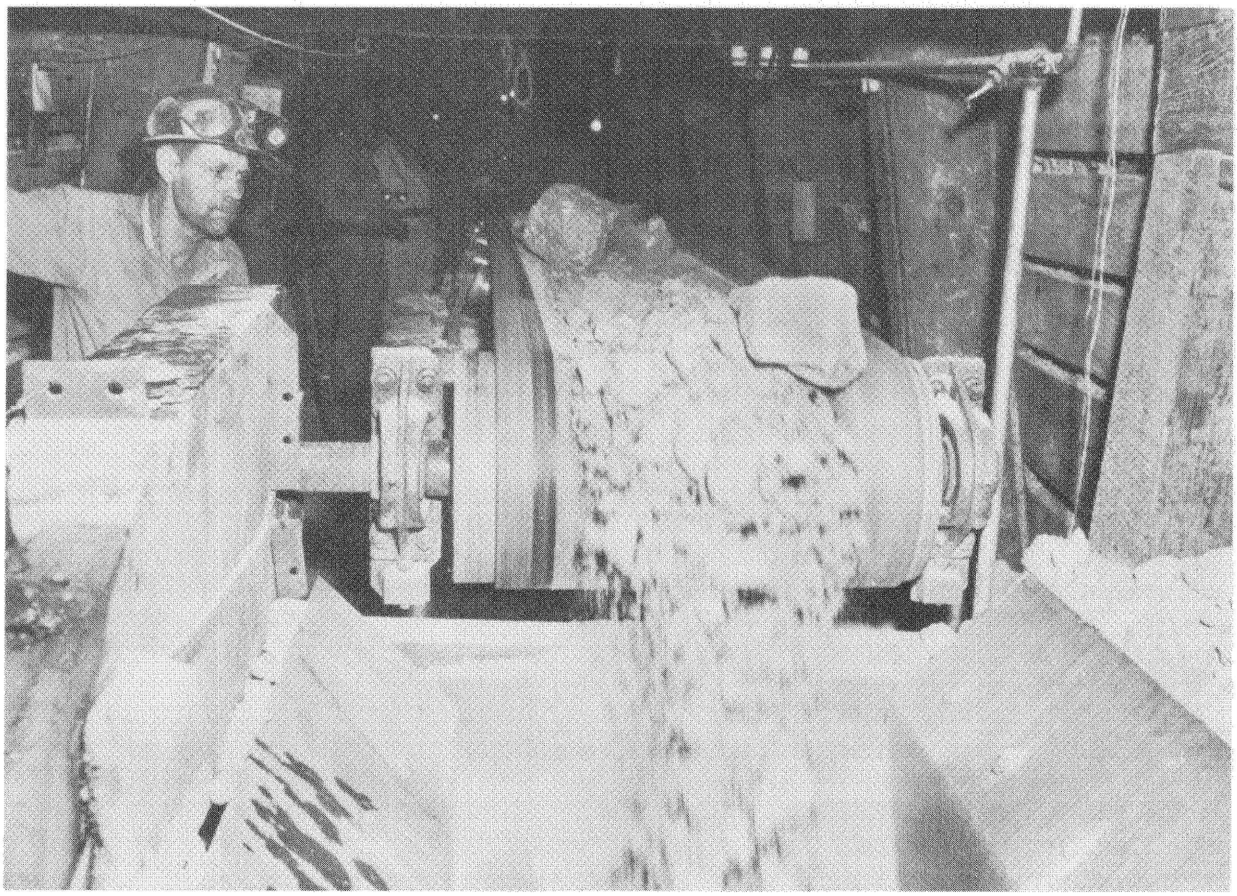


FIGURE 39. - Discharge End of Conveyor Belt.

Design of support in belt conveyor drifts was originally patterned after haulage-drift timbering. In later operations, if drifts were to be in service long enough to justify the initial expenditure, steel sets were often used instead of timber because the extra cost of steel was more than offset by lower repair costs.

In other applications of intermediate-level transfer, belt conveyors functioned as an auxiliary haulage system to serve ore blocks in areas above the 720 level and not reached by 1000 level workings. These blocks were mined after the haulage facilities on the 720 level had been abandoned. Two intermediate levels equipped with belt conveyors and three sets of ore passes provided a means of transferring ore to the existing 1000 level haulage system without the necessity of extending 1000 level workings.

When existing workings could not be used, laterals in these setups were driven in the same manner as intermediate conveyor drifts. Conveyors 200 to 400 feet long, equipped with 36-inch-wide belts, provided lateral transfer on intermediate levels. Each conveyor, powered by a 20-hp motor, had a rated belt speed of 400 feet per minute that provided a capacity of 2,000 tons of ore per shift. On the upper of the two intermediate levels and serving from

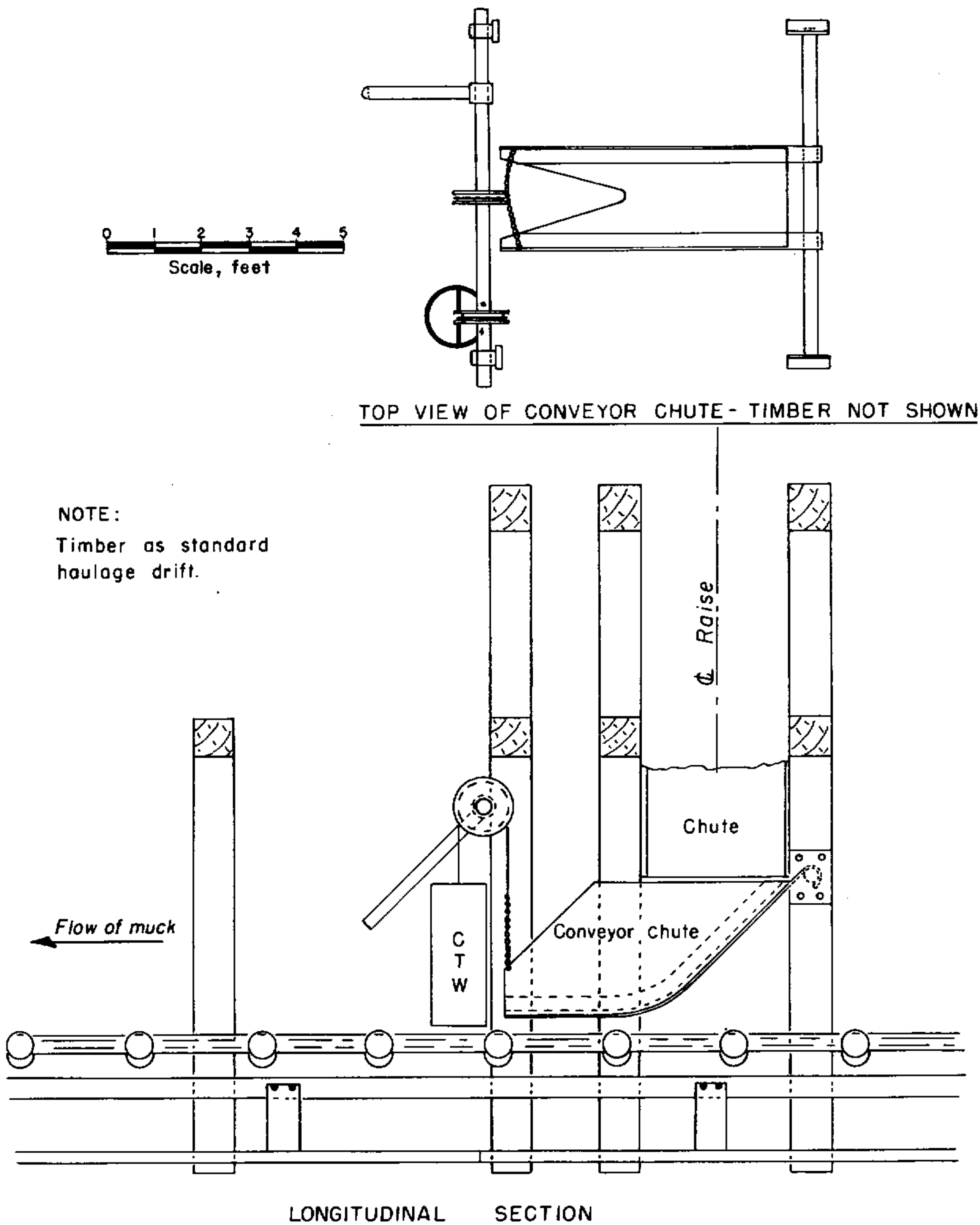


FIGURE 40. - Chute for Loading Belt.

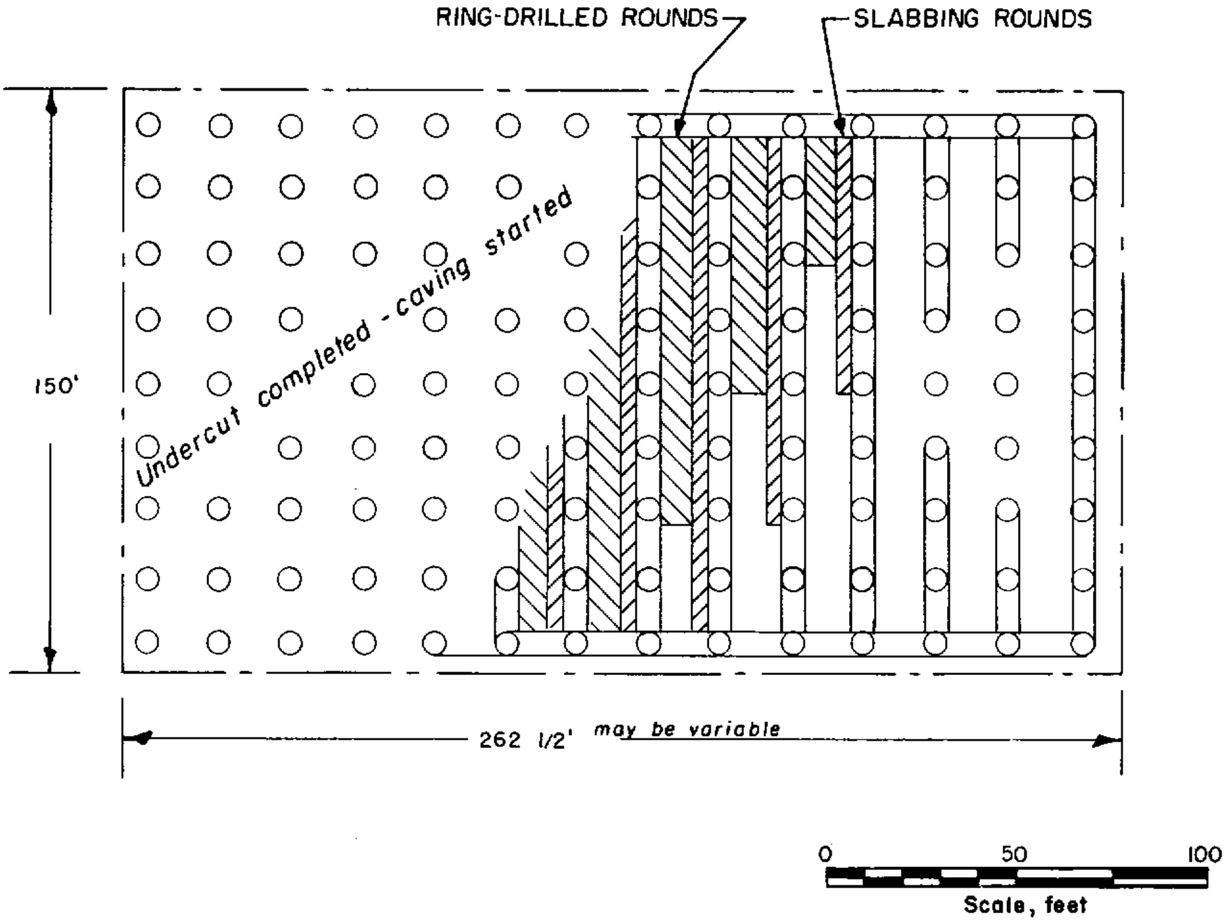


FIGURE 41. - Stope Undercut Sequence.

8 to 30 scraper drifts each, four collecting conveyors in drifts spaced 187.5 feet apart drew from scraper-fed raises and discharged into paired intermediate ore passes arranged so that one ore pass could be closed off for repairs while the other remained in service. One conveyor on the lower intermediate level handled the entire ore flow, discharging into a double ore pass leading to the 1000 haulage level. To handle the anticipated large tonnage of ore converging on them, these lower ore passes were lined with concrete and equipped with steel chutes of three-eighths-inch plate and undercut guillotine gates made of one-inch manganese steel and controlled by heavy duty 30-inch-stroke air cylinders (figs. 24 and 31).

Undercut Method

Undercut procedure was standardized regardless of the method of ore transfer below the stope. The objective was to remove all support under a column of ore and induce caving. Good stope performance resulted when the undercut was completed rapidly, caving started and drawing continued at a uniform rate. In the initial period of caving, the ground at the undercut level was cut into individual pillars by driving two sets of drifts at right angles to each other. Then, starting in one corner and advancing along the

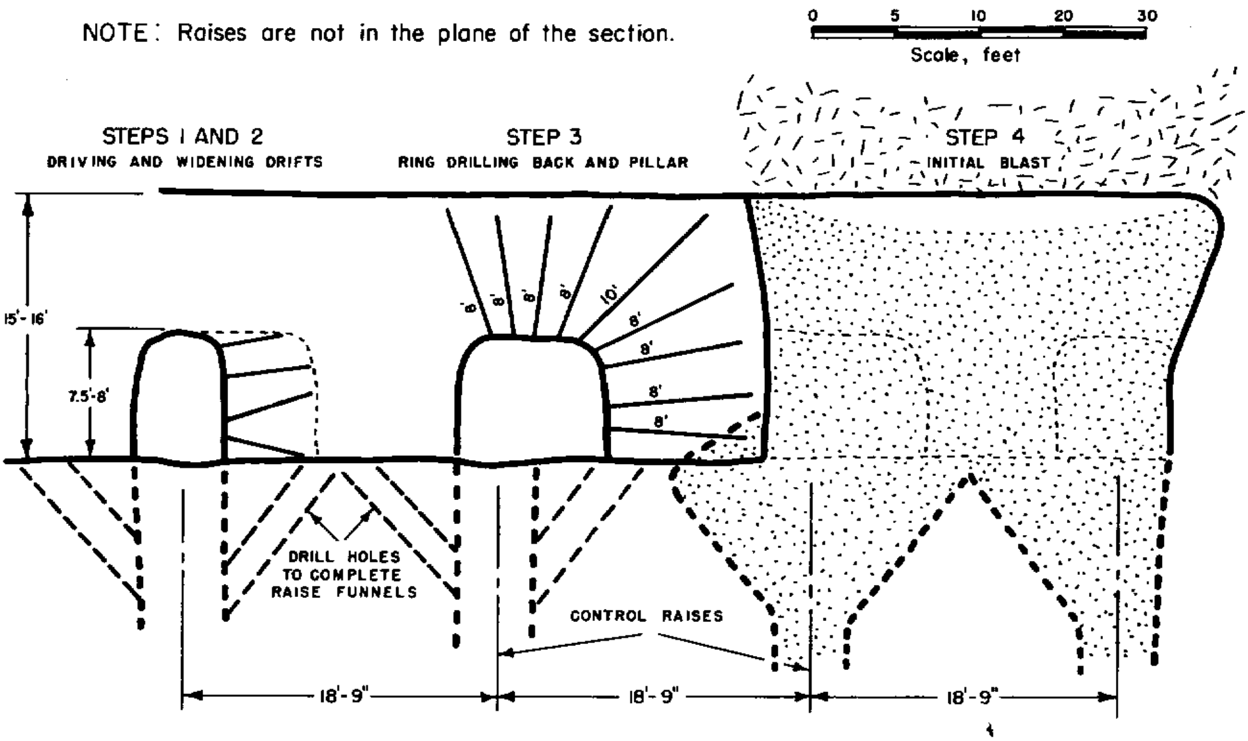


FIGURE 42. - Section Through Undercut.

block as rapidly as possible, the pillars were shot out by blasting the walls and backs of drifts.

Access to the undercut level was provided through the control raises from the control-level drifts below the stope. Ore was shot directly into the raises, and progress of the undercut was dependent on rate of raise completion.

The method was modified, control raises were completed to the undercut level, and small untimbered drifts were driven lengthwise at opposite sides of a block, connecting the tops of raises in the lines nearest the block boundaries (fig. 41). From these boundary drifts, untimbered cross drifts were driven on 16-2/3-foot centers to connect each line of raises across the block, leaving long pillars extending the full width of the block. Several cross drifts in different stages of advance were under construction concurrently. Beginning at the corner of the block where caving was to start, the cross drifts were first slabbed to about 8 feet wide; then the remaining pillar sections and the back of the drifts were ring-drilled and blasted in sections above two control raises at a time. This procedure removed all support from the ore column above. Each section of the undercut was started as soon as the preceding section was completed (figs. 41 and 42).

As soon as the undercut was completed over a draw point, enough ore was drawn through the chute below to remove any support of the ore column furnished by the broken ore and induce caving. Regularly scheduled drawing followed the initial caving action as soon as safety requirements permitted.

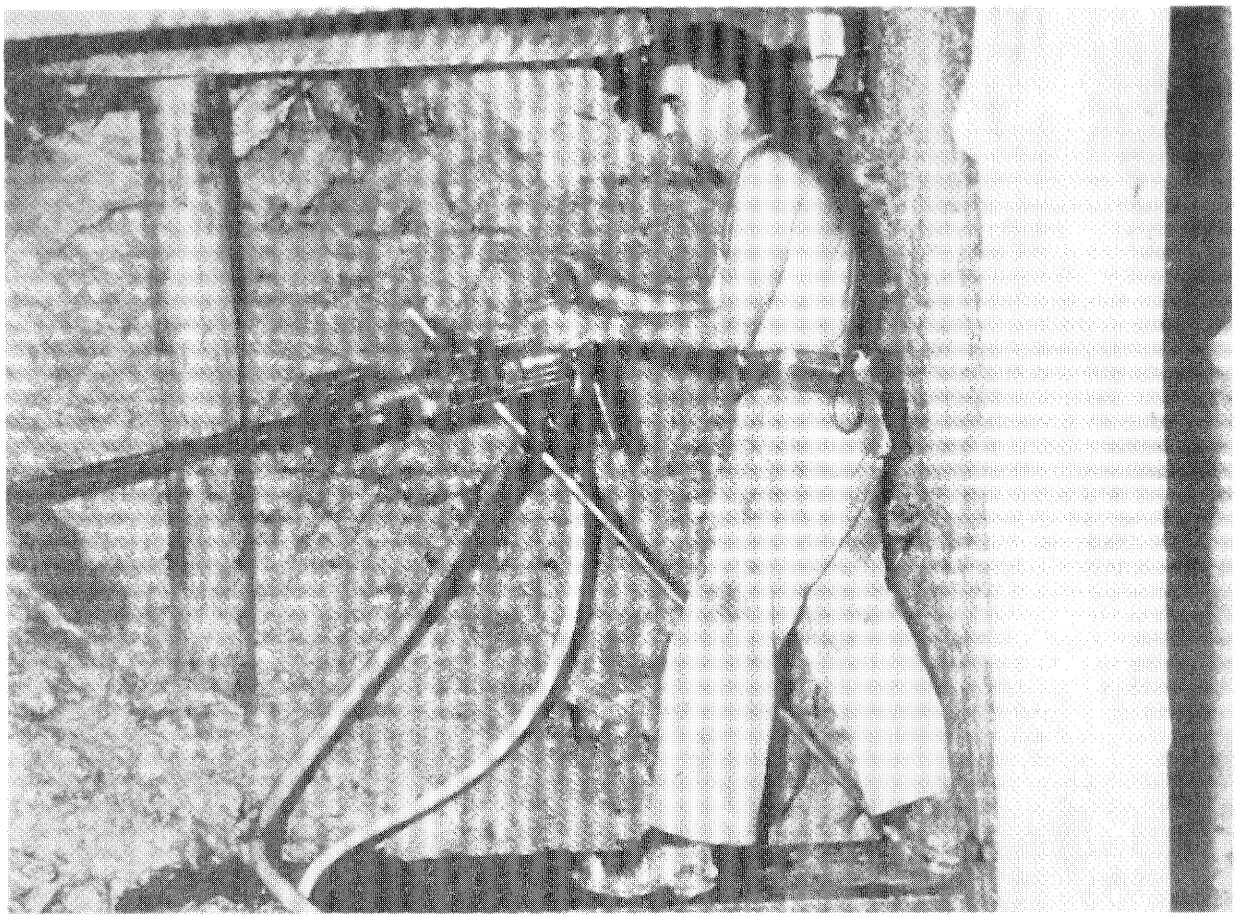


FIGURE 43. - Drilling Flat Holes With Mexican Setup Developed at Miami Mine.

Normally, drawing was not started closer than two chutes away from the point at which undercutting was in progress. Several raises were always in use as manways for the undercutting crews.

Mounted drifters were used to drill headings on the undercut level originally, but were replaced by air-leg-mounted drills. Before the air-leg drill was introduced, a system that might be called a forerunner was devised by contract miners of Mexican descent and used for flat drill holes. This system, called the Mexican setup, utilized a 45-pound jackhammer drill secured by a looped chain to the upper (shank) end of an upright drill rod of suitable length. The rod was anchored by a pointed detachable bit driven into a piece of lagging on the floor (fig. 43). The weight of the drill being thus supported, the operator leaned against the drill to keep the drill bit against the ground. The setup could be realigned as drilling progressed by adjusting the chain or the length of the drill rod. The Mexican setup and air-leg drills were used to widen drifts and to drill lateral holes in the final ring-drilling procedure. Light stopers were used for up-holes. The pattern of drilling is shown in figure 42.

Funneling of control raises below the undercut level had been started by enlarging the last round shot in control-raise work. The process was continued by cutting slots in the undercut floor on both sides of each raise as part of drift advance. The slots assisted in the movement of broken rock into the raises. Drilling to complete enlargement of raises was done from the raises, using inclined stoper holes, or from the level, using jackhammers. The funneling rounds and the ring-drilled rounds at the same draw point were shot together. Generally timbering was not used on the undercut level except over raises used to enter the undercut. Occasionally points of weakness required light support locally.

Ore Production

Three objectives of primary importance in drawing ore were maintenance of a scheduled tonnage, minimum dilution, and prevention of weight build up on extraction openings. In practice, the drawing schedule resulted in a compromise between these objectives. Difficulties resulted when the drawing rate was not in step with caving rate.

Immediately after a section of an undercut was blasted, the broken ore was drawn to induce caving. Sometimes an appreciable time interval elapsed before the back caved but, once caving started, every effort was made to draw ore at a uniform rate in sufficient quantity to keep the broken ore in the stope and control raises moving and loose.

The first ore that caved usually broke coarsely and arrived at the draw points in large pieces. These were broken up by sledging and, if necessary, by blasting at the grizzly. Also, any timber from the undercut or raises was removed at this point. Hangups of boulders sometimes occurred in control raises during this phase.

While good stope performance from the standpoint of both cost and extraction required an even rate of drawing throughout the life of a stope, special emphasis was placed on maintaining a uniform draw schedule until about 15 percent of the tonnage expectancy had been drawn. This procedure prevented ore from packing in the control raises. It was also important at this stage because the pillars and the timber support for extraction openings under the stope were not protected by the cushion of broken ore that formed as drawing progressed.

After 15 percent of the tonnage expectancy in a stope had been drawn, moving strain set up by the caving process and crushing action within the broken ore column as it moved downward produced more finely broken material. This formed a blanket of ore above the control-level workings that tended to distribute weight uniformly over the pillars and timber support and thus minimize concentrated loading at any point.

Failure to draw ore as rapidly as caving took place caused excessive buildup and concentration of weight, timber failure, and delays during repairs which caused further weight buildup and more timber failure. On the other hand, drawing at a rate faster than caving action caused formation of cavities below the unsupported arch in the back of a stope. This caused surges of weight on extraction openings due to mass block movements or air blasts when the arch collapsed. Uneven draw caused piping and increased dilution.

Many factors inherent in the block-caving method prevented an ideal schedule in day-to-day operation. Caving rate was a function of both hardness of the ore and size of the stope. Softer ore and larger stopes caved more rapidly, but it was not always possible to anticipate areas of soft ore or of fault gouge and adjust the draw schedule to correspond. Such material affected caving rate and also tended to pack in control raises and interfere with drawing. This led to weight buildup and support failure before the blocked raises could be opened. Raise hangups due to boulders also occurred and, if not removed promptly, could result in packing above. Packing of soft ore in stopes or control raises after a suspension of drawing often occurred.

Collapse of the ground around extraction openings developed from structural failure of the pillars between stope floors and control-level workings. This sometimes occurred without excessive loading from the broken ore column above. Pillars that broke up into fairly fine fragments did not ordinarily cause trouble because the weight of the crushed material was well distributed and therefore not excessive at any point; on the other hand, the weight of a failing pillar that broke loose as an unshattered rock mass often caused timber failure in the openings below.

Sometimes weight problems could be abated only by abandoning entire sections of a stope and concentrating on other sections to maintain mine production at the required level.

Delays in drawing often developed from causes not directly involved in the extraction phase. Disruption of the flow of ore anywhere in the ore transportation system affected stope performance adversely. Common causes were blocked transfer raises, support failures in raises or haulage-level workings, or any circumstances that involved more than brief suspension of drawing operations. Even interruption of operations over a long weekend sometimes delayed the draw schedule long enough to cause a surge of weight and timber failure.

Stopes that were drawn at the highest rate gave the best copper extraction on the basis of the records of 55 stopes shown in table 9. In general, high drawing rates accompanied heavy ground.

TABLE 9. - Comparison of drawing rates and copper extraction

Range in tons per man-shift	Number of stopes	Tons drawn	Extraction, percent		
			Tons	Grade	Copper
Over 300..............	9	9,764,000	122.6	94.8	116.2
200 to 300............	16	16,719,000	117.8	97.4	114.7
100 to 200............	23	19,924,000	106.9	94.5	101.0
Under 100.............	7	2,454,000	84.4	88.1	74.3
Total or average..	55	48,861,000	111.8	95.4	106.7

Pillar stopes were structurally weaker than original stopes. In weak ground, a large amount of repair with consequent delay in drawing schedules was required, and generally a poor draw resulted. Pillar stopes in firm ground required little repair, permitted good draw control, and gave good extraction. Dilution of pillar ore by waste entering along the sides exposed to mined-out, waste-filled blocks was not a problem in firm ground.

In general, there was a small tendency for a pillar block to draw outside the vertical limits of the undercut on the sides bounded by broken waste. This was countered by delaying the draw along stope boundaries until 100 percent of the calculated extraction was approached. Dilution was also reduced by increasing the original boundary-pillar thicknesses of 7½ or 15 feet to 30 or 50 feet. A comparison of the extraction as related to number of sides exposed to waste and pillar thickness is made in table 10.

TABLE 10. - Extraction by stopes in relation to pillar thickness and exposure to mined-out ground

No. of stopes	No. of sides of stope exposed to mined-out ground	Percent of total tonnage drawn	Extraction, percent		
			Tonnage	Grade	Copper
720 level, 7½- and 15-foot boundary pillars					
16	0	33.3	115.48	92.56	106.86
17	1	24.4	101.52	87.72	89.05
13	2	15.2	97.47	84.55	82.41
15	3	16.3	82.28	78.54	64.62
8	4	10.8	90.84	79.54	72.65
Total or average 69	[1] 43	100.00	[2] 99.83	[2] 86.79	[2] 86.64
1000 level, 30- and 50-foot boundary pillars					
37	0	65.5	116.67	94.43	110.17
12	1	14.5	105.02	96.0	100.82
14	2	17.0	103.15	90.24	93.08
2	3	3.0	96.02	86.98	83.52
Total or average 65	[1] 13.4	100.0	[2] 111.65	[2] 93.39	[2] 104.27

[1] Percent; average stope perimeter exposed to mined-out ground.
[2] Weighted average.

Ore extraction schedules were dependent in large measure on physical characteristics of the ore in each block. The actual draw rate ranged from 18 inches in 24 hours in a heavy stope that caved easily to less in a stope without a weight problem. The average was about 9 inches in 24 hours. Ore drawing and repairs were scheduled to permit maximum production from the stope. Detailed schedules were made each day; they could not be set up in advance of actual mining, and, during the life of any stope, many adjustments were necessary.

The tapper drew from each control raise in the line for a set time period. The objective was to pull the same amount of ore from each draw point.

Various situations sometimes interfered, such as the appearance of capping, necessity for repairs, or development of weight on timbers at some point. For example, it might be necessary to draw at a point to relieve the weight whether the position was listed for drawing or not. Variation in grade of ore and changes in order to draw to maintain proper distribution to haulage level for efficient operation of trains were also interfering factors.

The scheduling of production and repairs to realize maximum production both day-by-day and for the life of the stope required close cooperation of draw bosses and stope engineers. Foremen and engineers met each shift to discuss work progress and make necessary changes in schedules.

All rock broken in development work was sampled and handled with the ore. The average grade of such material was low, but the material generally contained enough copper to pay other than mining costs, and operation of a separate waste-disposal system was avoided.

Blasting

Dynamite was used to free hangups in transfer and control raises. All charges were detonated by electric blasting caps, and 95 percent of the blasting was done with permanent electric blasting circuits. In a few cases, caps were exploded with a blasting machine, and when a blasting circuit was not available, single isolated charges were exploded with an electric miner's lamp equipped with a special shot firing attachment.

All blasting in the stopes was done during lunch. Drawing was stopped in time to load before lunch. When ore was hungup in a control raise, a bomb made of 3- by 4½-inch cartridges of 40-percent semigelatine dynamite tied to the end of a blasting pole was placed against the lodged boulders. The blasting pole was wedged to hold the charge in place. The blasting poles, supplied in 9 foot lengths with a 1-inch cross section, were spliced or cut to the required length. Special safety precautions were taken, and all blasting was supervised by bosses.

When a transfer raise branch was obstructed at a point that could not be reached from the haulage level, the other branches were pulled clean, and a workman climbed down on a rope ladder from the control level to the intersecting plugged branch and placed the bomb against the packed ore from below. During 1956 about 0.024 pounds of dynamite were used to open raises and draw points per ton of ore drawn. Total consumption of dynamite including development, stope preparation, repair, and ore production was 0.23 pounds per ton of ore.

Repair

Of the total labor required to develop and prepare a stope for production and to repair and maintain the extraction openings during production, 37 to 48 percent was expended on development while the balance of from 52 to 63 percent was required for repair of ground support. Repair was one of the inherent problems of caving and was responsible for a substantial part of mining costs.

FORM 528

MIAMI COPPER COMPANY

DAILY ORE REPORT

	PLACE	Tons	% Total Cu.	% Oxide Cu.		PLACE	Tons	% Total Cu.	% Oxide Cu.
1					51				
2					52				
3					53				
4					54				
5					55				
43					93				
44					94				
45					95				
46					96				
47					97				
48					98				
49					99				
50					100				

SUMMARY

Total Tonnage__________

Total Skips __________

Shift	CRUSHER PLANT ASSAYS A	B	C	FLOTATION FEED ASSAY
Total Copper				%
Oxide "				%
Sulphide "				%

Date__________

FIGURE 44. - Daily Ore Report.

This was particularly true of Miami where no practical method was found to support permanently the accumulation of weight that would develop under a moving column of broken ore. Although ground movement at Miami was never sudden or instantaneous, increase of weight became serious if production from part of the block was stopped for any reason. Weight was influenced by rock strength and rate of draw, and the only sure method of combating weight was to pull ore from the stope as fast as caving characteristics permitted. To do this, repair work was planned and followed through so that drawing ore was interrupted as little as possible.

With production at 12,000 tons per 24 hours, ore was drawn on two 8-hour shifts. Repair crews were able to work on the third shift without interference from the operating crews, and many repairs were completed in one shift or less without disrupting drawing schedules. Urgently needed repairs were advanced 24 hours per day until completed. Even with this schedule, at times it was advisable to abandon sections of a stope, let the ore pack for a time, and devote all effort in another section to keep up production.

Stope Performance

Drawing was under the supervision of stope engineers who inspected the stopes daily and issued written draw orders that showed the tonnage desired from each control raise. All control raises in the stope were numbered, starting in the southwest corner, and each transfer raise was numbered. In the gravity stopes, the draw boss estimated the draw from each point and each raise on each trip. The engineering department received these reports and, using the figures from the draw boss report, prorated the actual car count back to the draw points and prepared a daily ore report that recorded the calculated tonnage drawn from each point (fig. 44). The skips were counted by an automatic counting device, and the actual weight of the ore was recorded by a weightometer at the head of the concentrating plant. Once a month, the actual tons were prorated back to determine the skip and car factors, which averaged 10.3 and 3.73 tons of ore respectively.

For grade control, a grab sample was taken from each car at the transfer raise chute. Check grab samples were also taken as cars were dumped in the shaft pocket. A weighted average of the haulage level samples gave the daily mine grade and this was corrected to the mill grade. Mine grade and the flotation feed for a typical period are compared in table 11. All development workings were sampled, and ore broken in driving them was included at the indicated grade. Special samples were taken at any control raise when dilution was indicated. However, because of the difference in the color of ore and waste, visual control was quite effective.

TABLE 11. - Comparison of mine and mill sampling results, January 1956

Date	Tons	Percent copper					
		Total		Sulfide		Oxide	
		Mine	Mill	Mine	Mill	Mine	Mill
2....................	11,080	0.603	0.617	0.438	0.457	0.165	0.160
3....................	10,689	.572	.621	.397	.452	.175	.169
4....................	11,546	.627	.607	.445	.424	.182	.183
5....................	11,378	.616	.614	.441	.429	.175	.185
6....................	12,137	.595	.580	.430	.395	.165	.185
7....................	11,188	.623	.595	.456	.427	.168	.168
9....................	11,681	.857	.642	.618	.460	.240	.182
10....................	11,912	.635	.635	.484	.474	.151	.161
11....................	11,818	.567	.631	.406	.468	.161	.163
12....................	11,651	.597	.621	.465	.477	.133	.144
13....................	11,834	.702	.601	.532	.437	.170	.164
14....................	11,500	.672	.638	.506	.437	.166	.201
16....................	11,423	.670	.576	.481	.415	.190	.161
17....................	11,318	.706	.636	.533	.456	.173	.170
18....................	11,128	.677	.633	.512	.472	.165	.161
19....................	11,570	.652	.618	.485	.476	.167	.142
20....................	11,251	.635	.621	.494	.476	.140	.145
21....................	11,230	.679	.581	.536	.456	.143	.125
23....................	11,777	.643	.629	.489	.479	.154	.150
24....................	10,930	.681	.582	.532	.433	.149	.149
25....................	11,084	.617	.616	.486	.487	.131	.129
26....................	11,514	.658	.663	.492	.494	.166	.164
27....................	9,967	.639	.653	.479	.503	.160	.150
28....................	10,842	.437	.630	.470	.458	.167	.172
30....................	11,170	.697	.634	.536	.457	.161	.153
31....................	11,673	.659	.651	.530	.474	.429	.150

Dilution from capping was an integral part of mining at Miami and was compensated for by an overdraw. Generally waste began to appear in a few drawpoints when a stope was about 30 percent drawn and in an increasing number of drawpoints as the draw approached completion. When waste showed in a drawpoint, the control raise was sealed for a limited time, 5 to 10 days, and then reopened; usually the waste packed and ore again could be drawn. As the stope approached 100 percent extraction, the percentage of drawpoints that could be cleaned up by this method decreased. A comparison between the percentage of tons drawn from a stope block and the percentage of draw points at which waste appeared for the first time is shown in table 12.

TABLE 12. - Appearance of waste in drawpoints related to stope completion

Percent of tons drawn	Percent of drawpoints showing waste	
	Minimum	Maximum
0	0	0
10	0	3
20	0	12
30	7	25
40	18	35
50	28	47
60	38	59
70	48	70
80	58	82
90	68	92
100	78	100

Ore was drawn from each chute until the grade of the ore dropped to a point that was not profitable. This usually did not occur until after 100 percent of the calculated ore was drawn. This cut-off point was fixed for each stope by the minimum profit required. Profit depended on the cost of operation in the particular stope, market price of metal, capacity of the treatment plant, and ratio of oxide to sulfide copper minerals.

Extraction

The extraction obtained in mining more than 108 million tons of ore by block-caving method is shown in table 13. The performances of typical stopes using full gravity, shaker conveyor, and scraper transfer are compared in table 14. The items of tonnage, grade and copper extraction show the actual content of ore recovered expressed as a percentage of the calculated content of the reserve.

The influence of caving rates on copper and tonnage extraction was discussed in the section of this report on ore production and illustrated by the data on representative stopes in table 9.

As shown in table 13, about 81 percent of the calculated copper content of the ore body was contained in the ore mined, and the mining loss was 19 percent. Estimates indicate that 15 percent of the total copper remaining (81 percent of 19 percent) was in pillars, and 4 percent, diluted with waste, was left in the stopes. Much of this copper will be eventually recovered by solution mining along with some copper from the capping.

TABLE 13. - Extraction, block-caving method

	Expected		Drawn		Extraction, percent		
	Tons	Grade, percent copper	Tons	Grade, percent copper	Tonnage	Grade	Copper
720 stopes.............	40,577,000	0.878	40,522,000	0.763	99.87	86.90	86.79
720 pillars............	7,130,000	.859	-	-	-	-	-
Total, 720 level....	47,707,000	.875	40,522,000	.763	84.93	87.20	74.06
1000-level stopes......	49,204,000	.741	54,937,000	.692	111.65	93.39	104.27
1000-level pillars.....	11,164,000	.747	1,752,000	.716	15.69	95.85	15.04
Total 1000 level....	60,368,000	.742	56,689,000	.693	93.90	93.93	87.69
Total or average both levels........	108,075,000	.801	97,211,000	.722	89.94	90.14	81.07

TABLE 14. - Performance of typical stopes

Block	Length of draw, months	Expected percent copper		Extraction, percent		
		Tons	Grade	Tons	Grade	Copper
Gravity stopes						
127..............	30	1,432,888	0.785	114.86	69.60	79.94
128..............	18	1,487,850	.788	131.53	75.13	117.01
1..............	31	1,646,855	.750	111.73	97.87	109.37
113..............	23	443,367	.787	106.04	83.10	88.12
118..............	36	737,524	.729	114.34	96.02	109.79
129..............	24	660,682	.821	113.80	91.14	103.72
139..............	20	580,376	.787	106.01	108.51	115.03
G-3..............	19	640,410	.724	114.46	106.63	122.04
H-3..............	22	924,936	.593	98.43	99.33	97.77
Shaker conveyor stopes						
J................	31	1,032,159	0.880	110.70	99.45	112.81
114-N............	9	183,651	.832	90.02	86.18	77.58
132-S............	17	425,531	.914	110.50	84.57	93.46
117-W............	27	1,077,637	.702	106.35	92.31	98.17
117-E............	27	752,425	.716	86.97	96.79	84.18
115-W............	20	559,237	.721	99.35	119.56	118.77
115-S............	27	535,071	.719	100.83	108.90	109.81
123-S............	26	451,105	.884	113.72	74.89	85.16
Scraper stopes						
134..............	15	380,265	0.720	113.87	78.61	89.51
142..............	12	327,900	.840	87.57	80.48	70.48
143..............	13	304,836	.838	91.63	70.17	64.30
141..............	12	314,850	.831	110.05	88.09	96.94
133..............	22	756,635	.779	131.36	86.01	112.98
132-N............	19	741,538	.784	117.90	84.06	99.10
120..............	26	837,612	.724	98.92	84.81	83.89
119..............	24	826,776	.702	105.70	85.33	90.19
121..............	29	383,658	.631	123.04	108.87	133.96
138..............	19	187,694	.910	38.00	95.27	36.20

SOLUTION MINING

Some minerals and metals can be dissolved with suitable solvent solutions. Many minerals of copper are soluble in a dilute solution of sulfuric acid. Some copper ore bodies are in porphyry, granite, or other crystalline rocks that are not acid consuming. After ore bodies of this type are well broken, sulfuric acid solutions applied at the surface percolate through the ore dissolving the copper and are collected at the bottom. The copper is recovered from the solution. This method of extracting valuable minerals from the earth has been called "in situ or in place leaching and solution mining."

The Miami ore body was broken and fractured by the removal of 153 million tons of ore by the caving method. In-place leaching of copper was started in 1940 in some underground areas where workings had been abandoned. By 1956, production of copper by this method had reached an annual rate of 6,653,000 pounds, or 17 percent of the mine total. The precipitating plant was enlarged during the last years of the block-caving operation. After 1959 the mine produced at the rate of about 18 million pounds of copper per year by solution mining and by 1962 had produced 140 million pounds of copper by this method.

The leach cycle started with preparation of leaching solution. Leach liquor from the precipitation plant, about 2,000 gallons per minute, was mixed with fresh wash water, pumped to a tank above the mine area, and sulfuric acid was added in a lead-lined box at the discharge. Acid strength of about 10 pounds per ton of water was required because Miami ore did not contain enough sulfide mineral to oxidize and furnish the acid necessary to leach copper minerals.

The acidified leach solution flowed by gravity through 3-inch polyethylene pipes to the mining area and was distributed over the subsided surface through a system of flexible plastic pipe with branches that were shifted at will to reach different points above the ore body. Final distribution was through sprays at the point of application. Water loss by evaporation was about 10 percent. Approximately this amount is added to the circuit when copper precipitates are washed.

Leach solution passed freely through the barren capping of the ore body without significant acid loss. On the southeast side of the mining area, the ore body was covered with as much as 150 feet of Gila conglomerate, which consumed an excessive amount of acid. To eliminate the loss, the distribution pipes were extended to the underlying schist-granite complex through 6-inch rotary drill holes through the conglomerate.

After filtering through the caved stopes, the pregnant leach solution was gathered in ditches on the 1000 level of the mine, conducted to a sump, and pumped through No. 5 shaft to surface. Four vertical, 1,000-gallons-per-minute, stainless-steel pumping units, powered with 300-hp motors and operating against a 700-foot head, returned the solution to the surface through a 12-inch diameter stainless-steel pipe.

The copper-bearing solution contained about 4.28 pounds of copper per ton of solution and a trace of acid as it reached the precipitation plant. About

99 percent of the copper was recovered in a system of launders by precipitation from solution on shredded iron. The launders are arranged in sections; one section was bypassed while the precipitated copper sludge (cement copper) in it was flushed with a stream of high-pressure water through a wood screen in the bottom. The cement copper was conducted to an open cement settling pad where it was allowed to dry. Dried copper sludge was loaded into railroad cars with a front end loader and shipped to a smelter for further treatment. The dried precipitates contained about 13.5 percent moisture and 79 percent copper.

Iron for precipitation was in the form of detinned and shredded scrap cans purchased in carload quantities from large cities. The shredded scrap iron was charged with a magnet-equipped crane from stockpiles into a hopper which fed a belt conveyor traveling over the center of the precipitation launders. At Miami, about 1.3 pounds of prepared scrap and 2.4 pounds of sulfuric acid was required per pound of copper produced.

CAVING MECHANICS

The ground movement and subsidence at the Miami mine has been described by MacLennan[25] and Fletcher.[26]

After a block was undercut, the back of the stope, acting as a loaded beam, began to fail because the span of the undercut was too great for the strength of the rock. The back or roof of the stope formed a dome (commonly called arch) which continued to fail. For the average block at Miami this dome progressed upward at 5 or 6 feet per day when ore was drawn at 9 inches per day. Smaller blocks, or those in harder ground, caved more slowly. The cave boundary remained inside of the vertical stope boundary.

The crown of the dome reached the capping before the haunches; thus, for a period, the waste from the crown fell into and diluted the ore from the haunches. When the crown of the dome reached the surface, the first visible sign was a breakthrough 20 or 30 feet in diameter.

The cave reached the surface from stopes on the 1000 level after 11 to 25 percent of the ore was drawn. The rock surrounding the stope was under stress, and tension cracks developed on the surface around a caved area (fig. 45). The line from the outside surface crack to the edge of the undercut was about 50 degrees from the horizontal for a large area but was steeper for a single isolated block. Experiments with the flow of in-place-leaching solution indicated that these cracks were surface phenomena and did not extend in depth (fig. 46). No cracks were observed underground alongside the caved block.

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[25] MacLennan, F. W. Subsidence From Block Caving at Miami Mine, Ariz. Trans. AIME, Yearbook 1929, pp. 167-178.

[26] Fletcher, J. B. Ground Movement and Subsidence From Block Caving at Miami Mine. Presented at Annual Meeting AIME, San Francisco, Calif., Feb. 15, 1959, 28 pp.

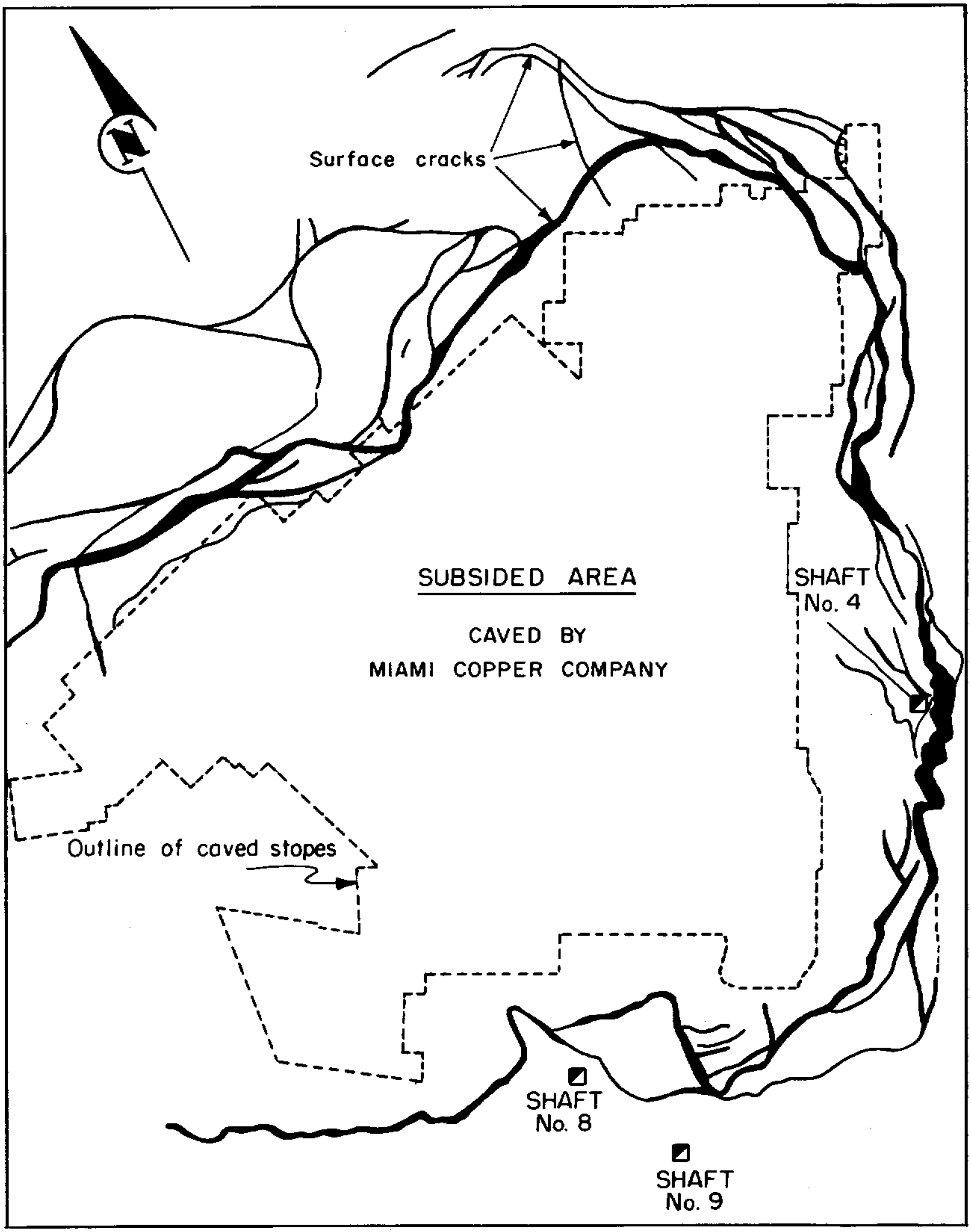
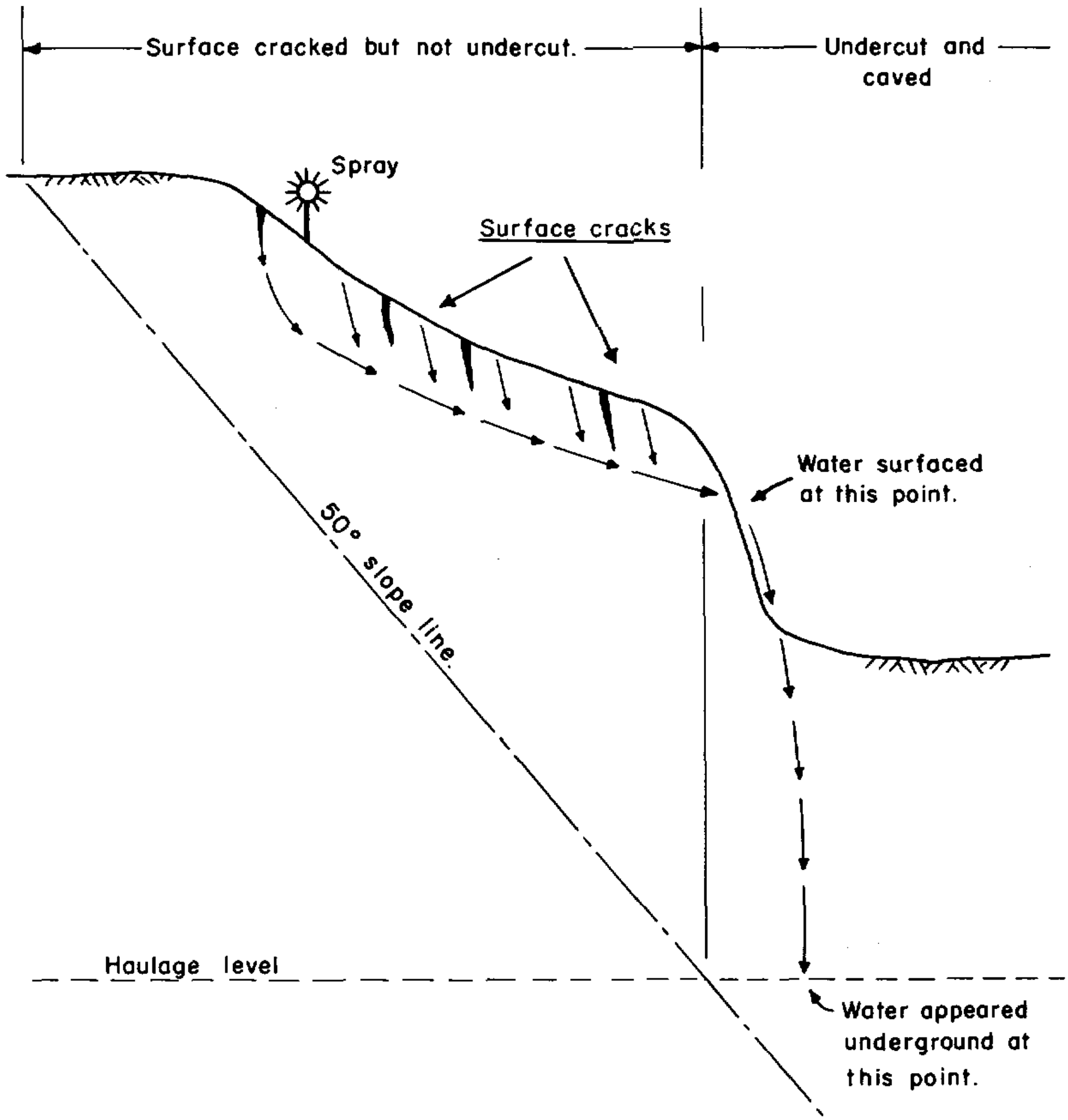


FIGURE 45. - Surface Cracks Around Caved Stopes.



VERTICAL SECTION ACROSS STOPE BOUNDARY

FIGURE 46. - Route of Leach Water After Entering Surface Cracks.

A dome (arch) structure in broken ore over a single drawpoint at Miami would not support itself over a wide span, and consequently pipes had very small diameters. Broken ore descending in a pipe occupied the same volume in the pipe as in the stope and there was no resulting volume increase or buildup of material in the pipe to eventually support the dome and stop the piping action. Consequently, piping, once started, would progress rapidly

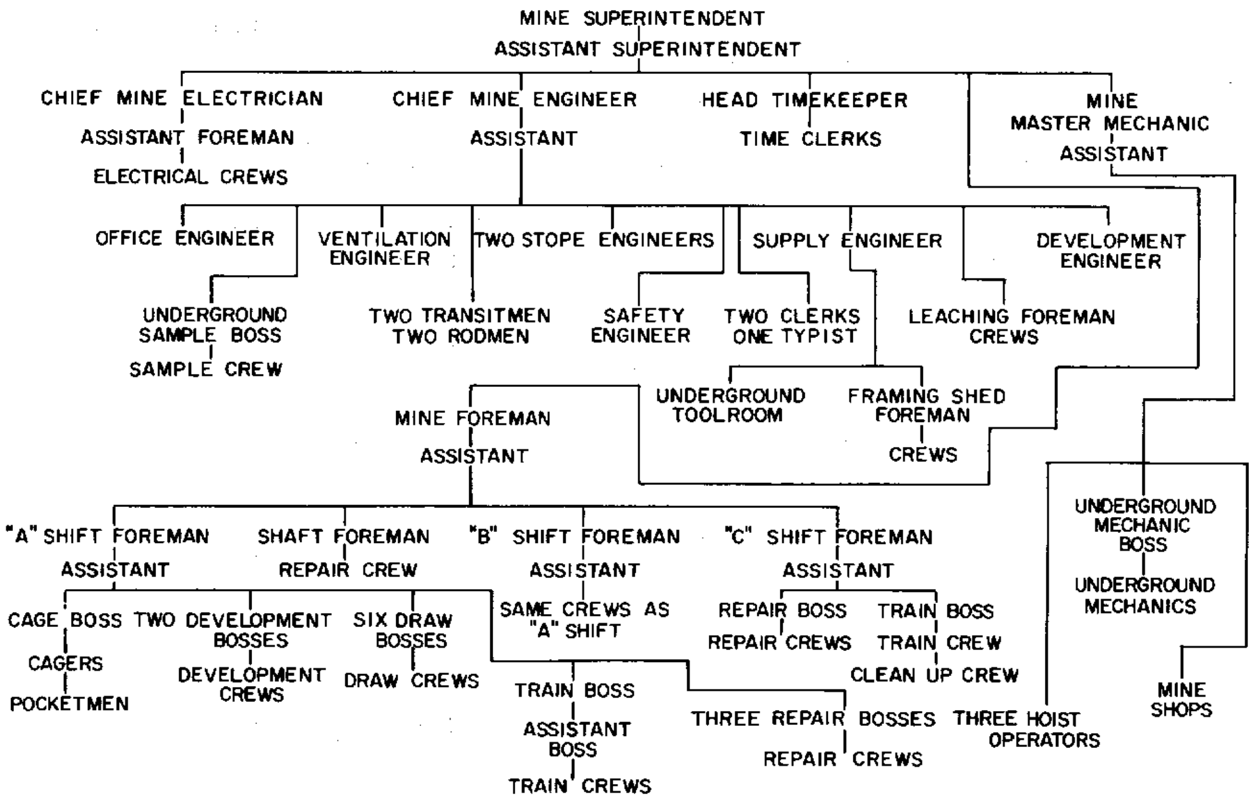


FIGURE 47. - Mine Organization Chart.

upwards through the waste capping to surface and, in the process, bring down a certain amount of waste to mix with and dilute the ore. Piping was prevented by uniform draw from all drawpoints and by spacing drawpoints closely.

MINE ORGANIZATION

During the period of underground mining, Miami Copper was one of three mining divisions of Miami Copper Co. The other two were Copper Cities and Castle Dome. The Miami Copper division operated the block-caving mine at Miami, Ariz. Copper concentrates from the Miami mine were smelted and refined under contract on a toll basis. The copper and molybdic trioxide was sold by a subsidiary company, the Adolph Lewisohn Selling Corp. Another subsidiary company, the Chester Cable Corp., manufactured a wide variety of products.

The organization of the mining department of Miami Copper division is shown on figure 47. The organization and supervision by engineers and bosses was one of the important factors in the success of the operation. About 45 bosses plus 12 engineers supervised 475 day's pay men. The engineers' and mine bosses' duties were overlapping and cooperative. The engineers prepared stope-draw orders, planned development and repair sequence, drew up work specifications, and supervised bonus contract rates and payments. The bosses and foremen assisted and followed through with these plans and supervised the men.

Mine labor requirements for 1954-56, in terms of production per stope employee, production per mine employee, and average daily man-shifts are given in tables 15, 16, and 17.

TABLE 15. - Production per stope employee, 1954-56

Stope	Tons per man-shift		
	1954	1955	1956
124	246	217	-
125	194	167	-
302	-	-	166
303	189	204	-
304	-	197	198
305	-	188	205
308	-	223	171
309	-	-	184
312	304	219	-
310	-	-	244
316	-	-	217
All stopes	178	164	186
Tons mined per day	11,937	11,970	12,417

TABLE 16. - Production per mine employee, 1954-56

Operation	Tons per man-shift		
	1954	1955	1956
Mining	51	59	72
Haulage	66	75	83
Miscellaneous services	95	103	104
All mine	22	25	28
Tons mined per day	11,937	11,970	1,2147

TABLE 17. - Average daily man-shifts, 1954-56

Average daily man-shifts	1954	1955	1956
Mining	234	205	173
Haulage	179	161	149
Miscellaneous services	126	116	119
Total	539	482	441
Tons mined per day	11,937	11,970	12,417

WAGE, CONTRACT, AND BONUS SYSTEMS

Engineers, foremen, and supervisors were on a monthly salary. Labor and skilled employees were on a day's-pay or hourly rate. All employees received paid vacations and participated in a retirement plan. Day's-pay employees were members of a union. Labor contracts were negotiated between union and company.

Generally, development, stope preparation, and some repair work was done on incentive-bonus contract. A rate for units of work was set by the engineer, and an agreement was made with the employee, who then completed the job and was paid for the number of units completed. The employee was guaranteed base pay at his regular rate, which was $18.86 per shift for miners during 1957. The contract system gave good results; development work was completed more rapidly, and mining costs were reduced. In general, miners who worked on contract earned more than wages. Results for one stope are illustrated in table 18.

TABLE 18. - Typical contract-labor earnings

	Feet completed	Feet per man-shift of contract labor	Contract earnings in percent of base pay
Haulage drift	2,203	1.27	209
Transfer raise	7,879	2.02	168
Control raise	19,004	6.23	206
Undercut drift	11,662	4.01	222

The crew on a given incentive-bonus contract performed all direct labor such as drilling, blasting, loading, and placing timber, but services such as moving-in supplies and haulage were classified as indirect labor and done by day's-pay employees. The labor performance for some typical operations is shown in table 19.

TABLE 19. - Performance data for development work

	Labor, man-shifts			Units of work in feet	Feet per man-shift	
	Direct	Indirect	Total		Direct labor	Total labor
Fringe, scraper, and grizzly drift	4,929.375	3,143.125	8,072.5	9,999	2.00	1.24
Haulage drift	1,730.875	1,045.625	2,776.5	2,203	1.27	0.79
Transfer raise	3,901.250	4,065.375	7,966.625	7,879	2.02	0.99
Control raise	3,052.375	3,005.125	6,057.5	19,004	6.23	3.14
Undercut drilling	2,910.500	1,428,375	4,338.875	11,662	4.01	2.69
Undercut pillars	2,363.500	901.250	3,264.75	[1] 139,964	[1] 59.22	[1] 42.87

[1] Square feet.

SAFETY, FIRE PREVENTION, AND FIRST AID

Safety was the primary responsibility of the supervisor, foreman, or engineer who was in charge of a job or activity. A safety engineer was employed, but his duties were advisory only. Any violation of safe practice received prompt attention and correction by the foreman. The excellent safety record at Miami, which merited award of the Joseph A. Holmes Safety Association's Certificate of Honor, has been attributed to careful safety planning and cooperation by employees, foremen, engineers, and management.

After 1919, mine-rescue work and first-aid training at the Miami mine was provided by a cooperative association organized by the various mine operators of the Globe-Miami district. A central mine-rescue station was built and equipped by the member companies, and mine-rescue and first-aid training in accordance with Bureau of Mines standards was given by full-time employees of the Association. A mine-rescue team of 15 men trained regularly and first-aid training was offered and recommended to all employees. Bureau of Mines mine-rescue and first-aid certificates were awarded qualified trainees. The operation of the Association has been described by Look and Van Fleet.[27]

In addition to the equipment available at the central station, a special rescue car was kept in readiness at the surface plant of the Miami Copper Co.'s No. 5 shaft (fig. 48). It was equipped with 10 McCaa 2-hour oxygen-breathing apparatus, a power saw, drill, air hose, and adapter fittings for all sizes of pipe in the mine.

All men trained in first-aid classes were paid their regular hourly rate and on completing the training were awarded certificates. All first-aid supplies were purchased by the Association and distributed to the member companies. Miami Copper Co. maintained first-aid cabinets at various strategic locations in mine and surface plant, and certain designated employees were charged with the responsibility of keeping the cabinets stocked.

Fire fighting equipment was provided near the principal buildings on the surface, and portable extinguishers of a suitable type were placed near operating machinery and equipment.

Miami Copper Co. and Inspiration Consolidated Copper Co. jointly operate a modern fifty-bed hospital fully approved by the American College of Surgeons, and a modern clinic on the Globe-Miami Highway. These facilities not only serve industrial cases but provide general medical care for employees, their families, and the community.

Particular attention was given to storage, handling, and use of explosives. Two explosives-storage magazines on surface, constructed according to Bureau of Mines standards,[28] were stocked with 30-ton lots at a time. Dynamite was issued from a single magazine until it was emptied. A magazine was normally emptied in about 3 months and when empty was thoroughly cleaned and inspected. Newly purchased stock was always placed in a completely emptied and cleaned magazine. This procedure insured using the oldest dynamite first. Detonators and fuses were stored in an auxiliary magazine.

Dynamite was received in railroad cars and transported to the storage magazines in plainly marked trucks grounded for protection from static electricity. It was hauled to the headframe daily and transferred to a wood-lined car that was lowered without any person on the cage. A 24-hour supply was stored in locked underground magazines. Dynamite was issued from the magazine only on written orders signed by the bosses and carried from the magazine to the place of use in sacks provided especially for that purpose.

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[27] Look, Allan D., and L. A. Van Fleet. Central Mine Rescue Station, Globe-Miami District Mine Rescue and First Aid Association, Globe, Ariz. BuMines Inf. Circ. 7577, 1950, 20 pp.

[28] Harrington, D., and J. H. East, Jr. Safe Storage, Handling, and Use of Commercial Explosives in Metal Mines, Non-Metallic Mines, and Quarries. BuMines Inf. Circ. 7674, 1954, 28 pp.

FIGURE 48. - Mine Rescue Car.

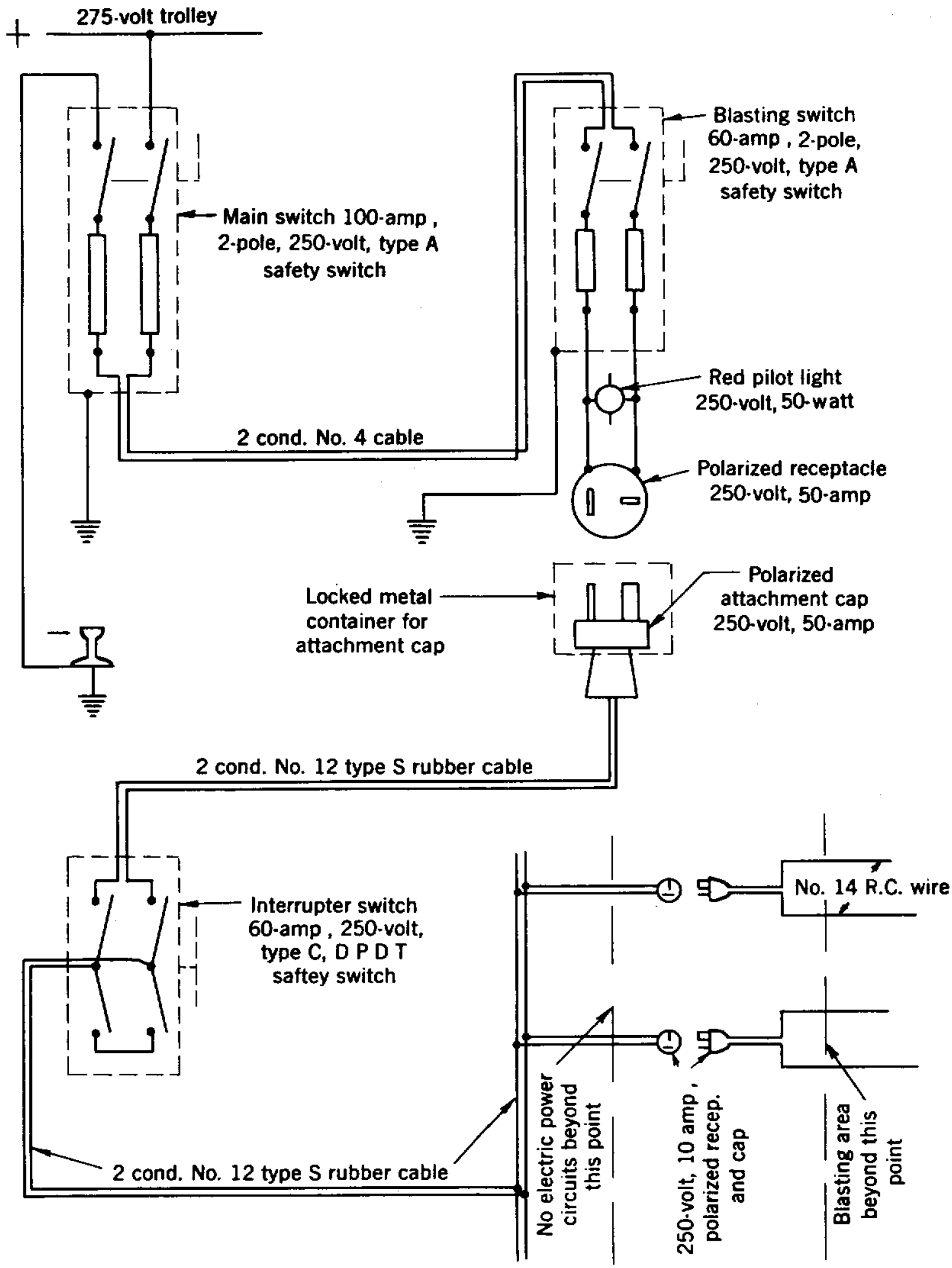

FIGURE 49. - Switch and Wiring Diagram Showing Permanent Blasting Circuit for Stopes.

Electric blasting was used almost exclusively and a separate blasting circuit was provided. Figure 49 shows a permanent circuit for blasting in stopes. All underground electric wiring and equipment was checked periodically, and detection of electrical leaks and faults was insured by a rigid maintenance patrol.

An effort was made to eliminate all hazards and to provide safe working conditions. Each employee was given a set of instructions and rules covering his job. The foreman assisted the employee by explaining the rules, then questioned and observed the actions of the employee to make certain that he understood and applied these rules in his work. Failure to observe the rules was valid cause for disciplinary action, including discharge.

MINE VENTILATION

The temperature in mine workings ranged from 70° to 80° F and humidity from 80 to 90 percent. Rock temperature was 70°, and temperature outside the mine ranged from 13° to 110° F. No attempt was made to control the humidity or temperature of the air; ventilating air delivery was planned to provide 300 cfm per man. Air pressure was maintained at about 10 inches water gauge, and air velocity in the grizzly drifts was about 40 feet per minute.

The most critical areas to ventilate were the draw points, belt dumps, and haulage-level chutes. Air passed from fringe drifts into grizzly drifts and down raises. Water sprays were used at critical points intermittently to settle dust. Fresh air entered No. 9 and No. 5 shafts and was directed through the mine by brattices and doors. Small blowers and tubing were used in development headings, and compressed air injectors were used to improve temporary local conditions. Air was exhausted from the mine through the No. 7 shaft by a 150-hp ventilating fan turning at 325 rpm. Excess compressed air was available and was used freely to supplement other ventilation.

POWER

Most of the power used in the Miami block-caving operation was generated in a company plant. In the first plant, completed in 1912, five oil-fired boilers rated at 600 hp each furnished steam for engine-driven generators and air compressors. As the demand for electrical energy increased, the plant was expanded and modernized. In 1959, it consisted of modern steam-turbine-driven generators connected to four superheated gas-fired boilers delivering steam at 425 psi. Electric power was generated and distributed at 6,600 volts in a 3-phase, 25-cycle system. Lines were connected to the Salt River Valley Water Users system from Roosevelt Dam for emergency service. The production and distribution of electric power for 1956 is shown in table 20; in that year, 98.27 percent of the power consumed was generated by the company, and the remainder was purchased.

In addition to direct use for generation of electric power, steam and gas were used in various parts of the mine plant for heating and processing. The amount of steam and gas used in this way in a typical month is shown in table 21.

TABLE 20. - Electric power distribution, 1956

	Kwhr
Mine:	
Drawing ore	832,874
Electric haulage	1,260,174
Handling men and supplies	240,000
Ventilation and sanitation	1,160,101
Underground lighting	120,081
Hoisting expense	5,527,014
Engineering and sampling	14,473
Mine surface	177,920
Mine machine shop	196,724
Total mine	9,529,361
Mine and mill:	
Total mine	9,529,361
Sales and sanitation	115,421
Total mill and crusher	55,747,795
Total leaching	4,575,108
General	425,710
Pumping and water	18,116,280
Total molybdenum plant	1,561,772
Total power used	90,073,275

TABLE 21. - Steam and gas distribution for a typical month

	Steam, pounds	Gas, million Btu
Mine change room	451,840	-
Sample drying	169,440	-
Y.M.C.A.	720,120	-
Assay office	70,600	-
Molybdenum plant	3,197,700	423
Concentrator shops	-	40
Mine shops	-	135
Concentrator	-	46
Hospital	-	30
Crusher	-	14

Air was compressed at the powerhouse and piped to the mine and plant. Capacity was developed to support a mine production of 18,000 tons per day, and at the actual rate of 12,000 tons per day, exceeded demand. The compressed air distribution for a typical year in the plant is shown in table 22.

TABLE 22. - Compressed air distribution, 1956

	Cubic feet
Mine ventilation	188,911,500
Air drills	44,625,000
Concentrating plant	44,007,000
Crusher	5,811,000
Shops	577,500
Molybdenum conditioning	3,744,000
Total	288,676,000

WATER SUPPLY

Milling required large quantities of water, and the Miami mine produced very little. Enough to supply early operations was obtained from wells in gravel along Miami Wash. When plans were made for plant expansion, additional water was required, and in 1918 an agreement was made to purchase water from the Old Dominion mine in Globe from which an average of 3,730,000 gallons of water was pumped daily during operation. While the Old Dominion mine was active, enough water was obtained from the lower workings of this mine to supply the Miami mine.

After the Old Dominion mine was closed in 1931, Miami again was short of water. Additional wells were drilled, and a 180-foot diameter dewatering tank was built to recover water from the tailings. In the spring of 1940 the water shortage became so acute that production was seriously threatened, and the Old Dominion mine was purchased. This source provided an ample supply for capacity operation at the Miami mine and also a part of the water for the Castle Dome and Copper Cities operations.

SERVICE FACILITIES

The mine service facilities consisted of blacksmith, welding, machine shops and a timber-framing shed. Separate areas were provided for car repair, scraper hoist, and mucking machine repair, air drill repair, and parts storage. The mine shops were supervised by a master mechanic who also supervised the underground mechanics. Many minor repairs were made in service shops underground.

After throwaway bits were adopted, blacksmith work consisted mainly of drill-rod conversion, sharpening pick bars, and miscellaneous work.

The mine machine shop was equipped with 14- and 20-inch lathes, shaper, hydraulic press, radial drill, and pipe and bolt machines. Mine work that could not be handled with this equipment was taken to the general machine shop.

Scraper hoists were brought to the surface for cleanup and service after completing a block. With the increase in the number of these machines, this became an important part of the mechanical repair work.

Ore cars were inspected twice a month underground, and bearings were greased once every 60 days. All car repairs were made in the mine shops on surface, which were equipped to change and turn wheels and rebuild car bodies.

Timber for mine use was prepared in a timber-framing shed adjacent to a railroad spur where lumber from the Northwestern States could be unloaded directly from freight cars and stored in a convenient area. Timber was prepared in accordance with daily orders given by the supply engineer to the framing-shed foreman. Orders were loaded on timber cars at the framing shed, hauled to No. 5 shaft through the 320-level adit and delivered to designated locations underground.

Timber consumption at Miami had a substantial range in quantity depending on the rate of new development. In 1956, consumption was 1.47 board feet per ton of ore, or nearly double the 0.79 board feet per ton of ore in 1957. The quantity of timber used for ground support at Miami during 1956 is shown in table 23.

TABLE 23. - Timber used underground, 1956

	Quantity, board feet	Percent of total
New development	3,122,080	56.6
Repair	2,188,266	39.9
Other	197,243	3.5
Total	5,507,589	100.00
Board feet per ton	1.47	

All timber used in permanent openings were pressure treated with a preservative solution of salts composed of 35.6 percent chromate, 23.8 percent sodium fluoride, 23.8 percent disodium hydrogen arsenate, 11.8 percent dinitrophenol, and 5.0 percent inert matter. The arsenic content, all in water soluble form, was equivalent to 9.5 percent metallic arsenic.

The preservative solutions, made up in the ratio of 17.5 pounds of salts to 100 gallons of water, was mixed and heated to 80° C in a vertical pressure tank. Timber, loaded on cars, was charged into a horizontal pressure tank, the tank was sealed, and the preservative solution was then charged into the tank at 150 psi. Treatment required about 6 hours. About 2,800 to 3,000 board feet were treated per shift, and about 500 gallons of solution was consumed.

CONCENTRATING PLANT

Although development and operation of the mine by advanced engineering and operating techniques were important in reducing costs at Miami, the development of a suitable metallurgical process and concentrator operating technique were equally important. When copper was selling for 10 cents per pound, Miami Copper Co. treated ore giving a net yield of 10.55 pounds per ton. Throughout the long life of the operation, continued improvements in treatment practices were made to treat ore with decreasing copper content profitably. In 1956, when all but a fraction of the ore came from the low-grade ore bodies, the mine produced 3,812,165 tons of ore from which 31,911,120 pounds of copper, or 8.37 pounds per ton, was recovered in the concentrator (fig. 50).

Ore dressing techniques followed the mineral industries' advances in milling practices. Modifications of processes were necessitated by the changing character of ore from relatively high-grade chalcocite to chalcocite deposited as a thin film on pyrite disseminated in low-grade ore and from straight sulfide ores to mixed sulfide-oxide ores.

A molybdenum content in the sulfide ore estimated at an average of 0.01 percent was first recovered in 1937 by selective flotation of copper

FIGURE 50. - Miami Concentrator.

concentrates to yield a molybdenite (MoS_2) concentrate. The concentrate was roasted to an oxide in which form it was sold.

The flue dust from the molybdenum roasting plant was found to contain up to 1 percent rhenium, and this byproduct was recovered experimentally in 1942 to furnish the raw material for production of the first domestic rhenium at the University of Tennessee.[29]

COSTS

Development was capitalized and charged as prepaid development against the estimated ore tonnage to be mined. Stope preparation was charged as an operating cost against the individual blocks.

Performance in terms of man-hours per unit have been listed in the report when available for publication. Direct costs are not available for publication; however, a percentage breakdown of mining, concentrating and total costs is shown in tables 24, 25 and 26.

[29] Mineral Facts and Problems. BuMines Bull. 585, 1960, p. 692.

TABLE 24. - Mining costs

	Percent
Amortization of preliminary development	-
Remaining development (stope preparation)	16.5
Operating stoping	44.0
Electric haulage	16.9
General underground	5.3
Mine ventilation	6.5
Underground lighting	0.4
Hoisting	4.4
Engineering and sampling	2.0
Mine surface expense	4.0
Total mining	100.0

TABLE 25. - Concentrating costs

	Percent
Coarse crushing and conveying	20.7
Grinding	33.3
Flotation	15.4
Water	8.8
Concentrate disposal	4.4
Tailings disposal	5.2
Sampling	1.5
Experimental	0.5
General mill	10.2
Total concentrating	100.0

TABLE 26. - Total cost of producing copper

	Percent
Mining	38.5
Concentrating	23.8
Smelting, refining, etc	23.2
General	11.6
Legal and administrative	2.9
Total	100.0

Total cost in table 26 does not include credit to the copper account for molybdenum sold, which amounted to about 2.7 percent of the total cost. Also, it should be noted that no allowance has been made for amortization of preliminary development for mine and plant, as these charges had been completely written off. The listed costs are based on a period during which the mine produced at a daily rate of about 12,000 tons of ore that contained 11 to 12 pounds of recoverable copper.