BLOCK-CAVING MINING METHODS AND COSTS
BAGDAD MINE, BAGDAD COPPER CORP.
YAVAPAI COUNTY, ARIZ.

BY W. R. HARDWICK

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BLOCK-CAVING MINING METHODS AND COSTS, BAGDAD MINE, BAGDAD COPPER CORP., YAVAPAI COUNTY, ARIZ.[1]

by

W. R. Hardwick[2]

SUMMARY AND INTRODUCTION

This paper describes block-caving methods and practices of the Bagdad Copper Corp. in Yavapai County, Ariz. It is one of a series being prepared by the Federal Bureau of Mines on mining methods, practices, and costs in various mining districts of the United States.

The story of the Bagdad caving operation is of particular interest because the ore body was not as thick as at other mines where the method had been successful. At Bagdad the caving method worked very well as a small-scale operation of 300 tons per day but was abandoned when production was increased to 2,000 tons per day.

The Bagdad mine is in western Yavapai County, Ariz., about 22 miles northwest of Hillside, a station on the Phoenix-Ash Fork branch of the Atchison, Topeka & Santa Fe Railway system (fig. 1). Copper ore now is mined by open-pit methods. The ore is concentrated, and the concentrates are trucked to Hillside and shipped by rail to a custom smelter. Some molybdenum sulfide is produced as a byproduct.

This report gives a brief history of the district, describes the ore deposit, and outlines the methods of prospecting, exploration, and estimation of ore. Development and former underground mining, including stope preparation, stoping, and transportation, are described. Extraction of ore, production rates, miscellaneous operations related to mining, wage and contract systems, and method of concentration are described. The final section discusses mining costs.

The underground mine was visited by the author several times between 1937 and 1942, and some information has been drawn from personal notes. The property again was visited in June 1957, and maps and plans for this paper were furnished by the company at that time; they have been edited to conform with Bureau of Mines format. The underground mine could not be visited then because it had been closed in 1948 when open-pit mining was begun.

1/ Work on manuscript completed March 1958.
2/ Mining engineer, Bureau of Mines, Region III, Tucson, Ariz.

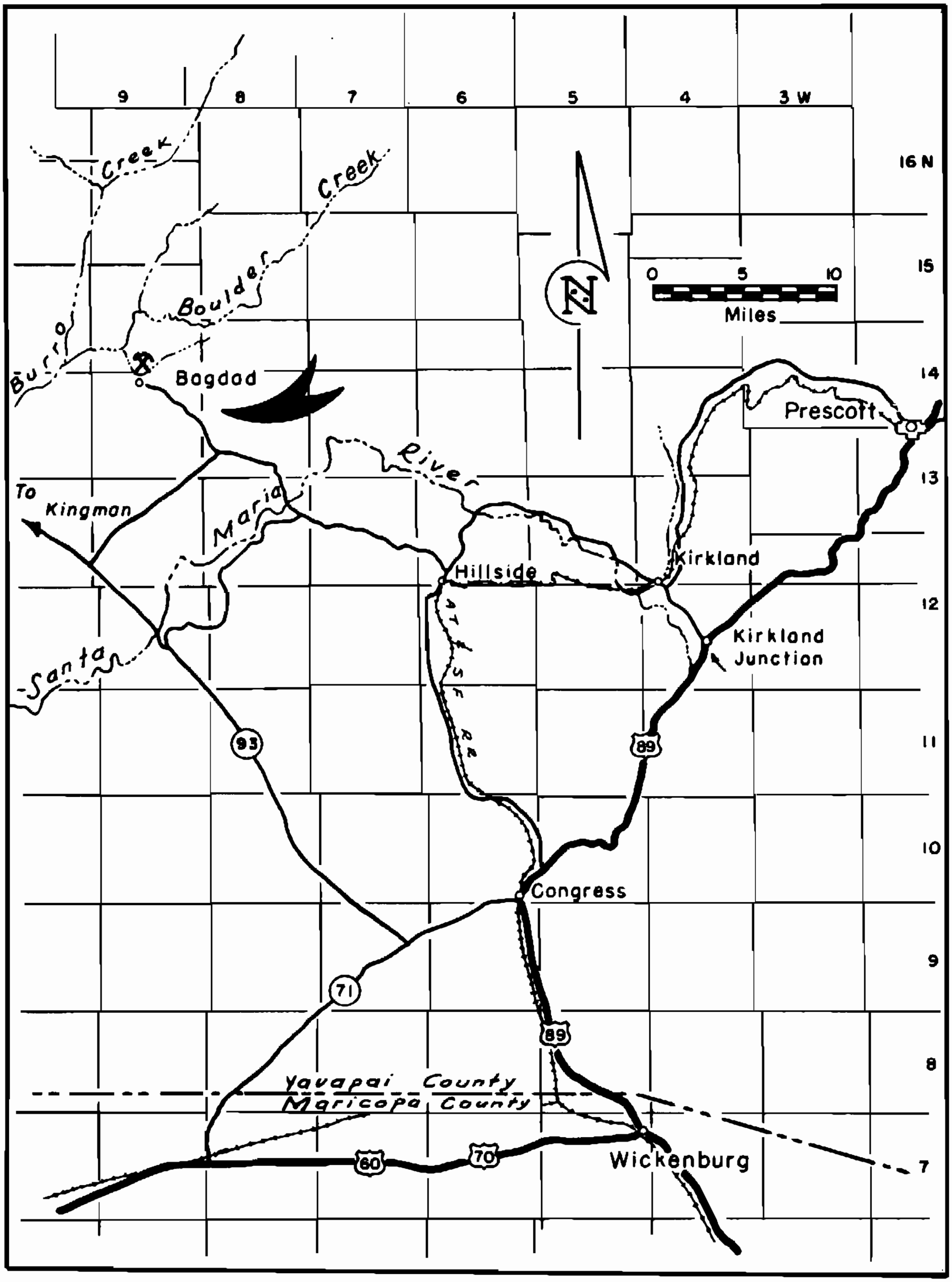

FIGURE 1. - Location Map, Bagdad Mine, Yavapai County, Ariz.

ACKNOWLEDGMENTS

Acknowledgment is made to the executives and staff of the Bagdad Copper Corp. for permission to publish this paper. Special acknowledgment is due Edward L. Jones, III, chief mine engineer of the Bagdad mine.

The geology of the deposit has been described by Anderson, Scholz, and Strobell.3/ Information also was obtained from the annual reports to the stockholders of the Bagdad Copper Corp. Maps and plans were furnished by the company unless otherwise stated. Much valuable information on the details of the operation was provided in a lengthy discussion with J. W. Still, consulting mining engineer, who was general manager of Bagdad Copper Corp. during the planning and initial stages of the block-caving mining operation.

HISTORY OF DISTRICT4/

The Bagdad mine is on Copper Creek, a few miles from its confluence with Burro Creek in sec. 4, T. 14 N., R. 9 W., Gila and Salt River meridian, at an altitude of about 3,200 feet, in the Eureka mining district. The mine is reached by 27 miles of paved road from Hillside, a station on the Ash Fork-Phoenix branch of the Santa Fe Railway.

The first claims in the district were located by John Lawler in 1880. Early production was confined to gold-silver-lead fissure veins. The Bagdad claims were discovered in 1886, and the ore body was developed by several adits. The first production was in 1906 by the Giroux Syndicate. In 1919 the Arizona-Bagdad Copper Co. acquired the claims, did some churn-drill sampling, then attempted to leach the ore in place. The leaching was not successful, and in 1927 the present Bagdad Copper Corp. succeeded the Arizona-Bagdad Copper Co.

The Bagdad Copper Corp. drilled 130 churn-drill sample holes, outlined a large part of the ore body, sank 2 shafts, and built a 50-ton pilot plant to test the ore. The pilot plant was operated in 1929, and the results were favorable. Plans then were made to mine 3,000 tons of sulfide ore per day by the block-caving method and to mill this ore by flotation. The financial collapse late in 1929 prevented completion of these plans, but a 200-ton mill was built in 1930, and mine production was increased to 150 tons per day. The plant was idle from 1931 to 1934. In 1936 the first ore was mined by the block-caving method, and in 1940 and 1941, at a rate of 250 tons per day, the company made a small operating profit for the first time. The ore was being successfully mined by the caving method, and mill recovery was satisfactory; however, the results indicated that greater profits would result if plant capacity were increased. Under the stimulus of premium prices for copper and with the help of a loan of $2,500,000 from the Reconstruction Finance Corporation (RFC), a 2,500-ton-per-day flotation mill was built, and the mine was expanded for larger production by the block-caving method.

3/ Anderson, C. A., Scholz, E. A., and Strobell, J. D., Jr., Geology and Ore Deposits of the Bagdad Area, Yavapai County, Aris.: Geol. Survey Prof. Paper 278, 1956, 103 pp.

4/ Tuck, Frank J., Stories of Arizona Copper Mines: Arizona Dept. of Mineral Resources, 1956, p. 34.

The ore body was thin, and the rock was hard for block caving.[5] Successful caving depended on small blocks; boundary drifts and corner raises were necessary. During World War II labor was scarce, and stope development lagged. Existing blocks were drawn rapidly, with the result that dilution was high and copper recovery was low. To increase production ore was mined by converting some caved stopes to glory holes where the capping was thin. Finally the mining method was changed, and the open-pit method with truck haulage was adopted. The underground operation was discontinued in 1948 after about 4 million tons of ore had been mined.

DESCRIPTION OF DEPOSIT

The copper minerals in the Bagdad mine occur in minor fractures and as disseminated grains in a quartz monzonite, which form a typical "porphyry" copper deposit. The ore occurrence has been described by Anderson, Scholz, and Strobell[6] as follows:

> In detail, the hypogene sulfides chalcopyrite and pyrite, replaced locally by supergene chalcocite, occur in minute veinlets, and some sulfide grains are disseminated between the veinlets, but from a broader viewpoint the copper minerals are "disseminated" rather than concentrated in larger veins. Some narrow quartz-pyrite-chalcopyrite veins are present, but the bulk of the copper occurs in the fractured quartz monzonite. Actually most of the so-called porphyry, or disseminated copper ore, is concentrated in fractures rather than truly disseminated in the host rock. The chalcocite enrichment in large part is related to the pre-Gila(?) erosion surface, and the highest grade of ore is found in a typical "blanket" beneath a leached zone and above the primary-sulfide zone, or protore. Some chalcocite enrichment in rock along Copper Creek northwest of the Bagdad mine appears to be related to the present erosion surface.

Molybdenite is common, and a molybdenum sulfide byproduct is separated from the copper concentrate and sold.

Supergene enrichment of the copper ore is of major importance in the Bagdad mine. After the primary ore minerals were deposited, the surface was eroded and the upper part of the mineralized zone oxidized and leached. The copper was carried down and deposited to form an enriched blanket of chalcocite (fig. 2). Afterward the Gila conglomerate covered the area and was in turn covered by tuff and basalt. Copper Creek eroded through the basalt and Gila conglomerate across the center of the ore body and removed these rocks and some of the oxidized capping.

Quartz monzonite crops out as an irregular stock that contains the ore body. Along Copper Creek sulfide ore occurs almost to the surface. Generally it is capped by 150 to 200 feet of leached rock or carbonate ore, 200 feet of

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5/ Mining World, This is What Streamlined Mining and Improved Haulage Did For Bagdad: Vol. 16, No. 9, August 1954, pp. 40-43.

6/ Work cited in footnote 3 (p. 3), p. 52.

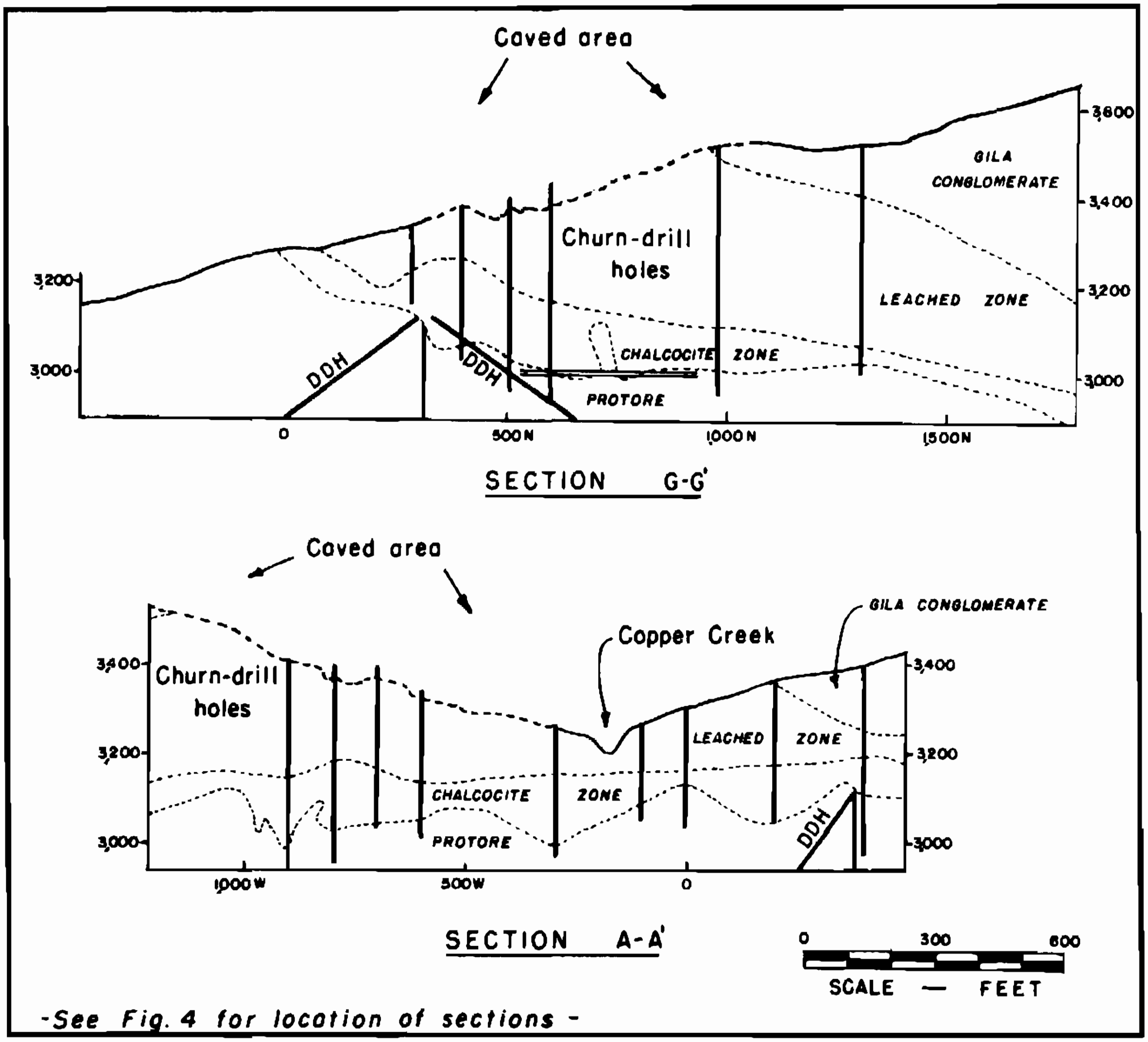

FIGURE 2. - Sections Through Mine.

Gila conglomerate, and 50 feet of basalt. The west ore body, which the company proposed to mine by caving, was roughly tabular and approximately 1,400 by 800 feet in area. Minable thickness averaged 125 feet. The deposit dipped 10° to 15° to the northeast. The quartz-monzonite stock is roughly 2 miles long, east and west (fig. 3), and 1 mile wide, north and south. It joins a schist complex on the north and is enclosed by a granitic complex on the other sides. A low-grade ore body covers a much larger area than the high-grade west ore body, which was mined by caving before 1948 (fig. 4).

PROSPECTING AND EXPLORATION

Ore first was discovered by underground work at Bagdad when adits were driven into the banks of Copper Creek. The Arizona-Bagdad Copper Co. began a churn-drill sampling program that was completed before 1930 by its successor, the Bagdad Copper Corp. About 130 sample holes, which averaged 300 feet in depth had been completed, outlining the ore essentially as it is known today. An engineer's report7/ in June 1937 listed the estimated reserves at 6 million tons of high-grade ore suitable for mining by underground methods.

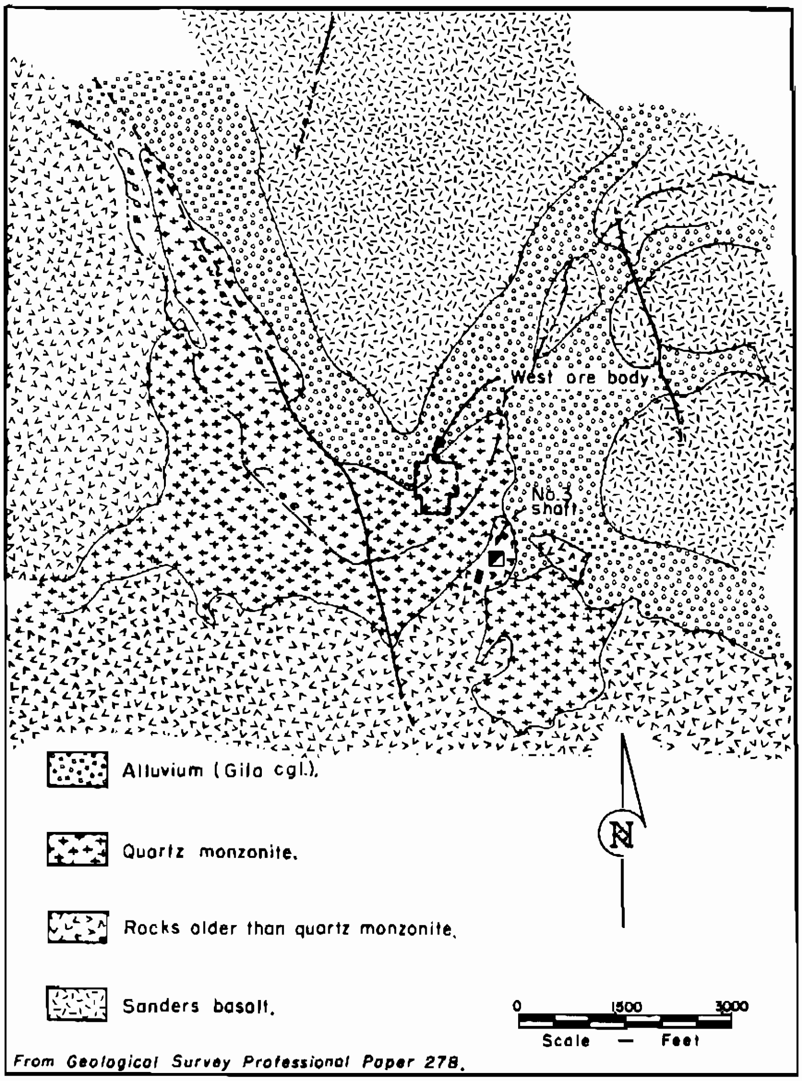

FIGURE 3. - Surface Geology.

SAMPLING AND ESTIMATION OF ORE

In the more important parts of the ore body churn-drill sample holes were spaced at 100-foot intervals and in the remainder of the ore body, at 200-foot intervals. Samples were split from the sludge for each 5-foot vertical interval in the drill holes. Cuttings were examined for type of rock and assayed for copper content. The results were recorded in drill-hole logs and plotted on sections.

Volume for various grades of ore was calculated and converted to tons by using a factor of 1 ton for each 12-1/2 cu. ft. of ore in place. In 1936 various estimates had placed ore reserves at 1 to 6 million tons of plus 1.5 percent copper sulfide ore and 20 to 40 million tons of 1.25 percent copper ore. The last information published places sulfide reserves at 30 million tons, with an average grade of 0.754 percent copper, and oxide-carbonate reserves at 300 million, with an average grade of 0.435 percent copper.8/

7/ Bagdad Copper Corp., Annual Report to Stockholders: Feb. 9, 1942.

8/ Parsons, A. B., The Porphyry Coppers in 1956: AIME, New York, N. Y., 1957, p. 221.

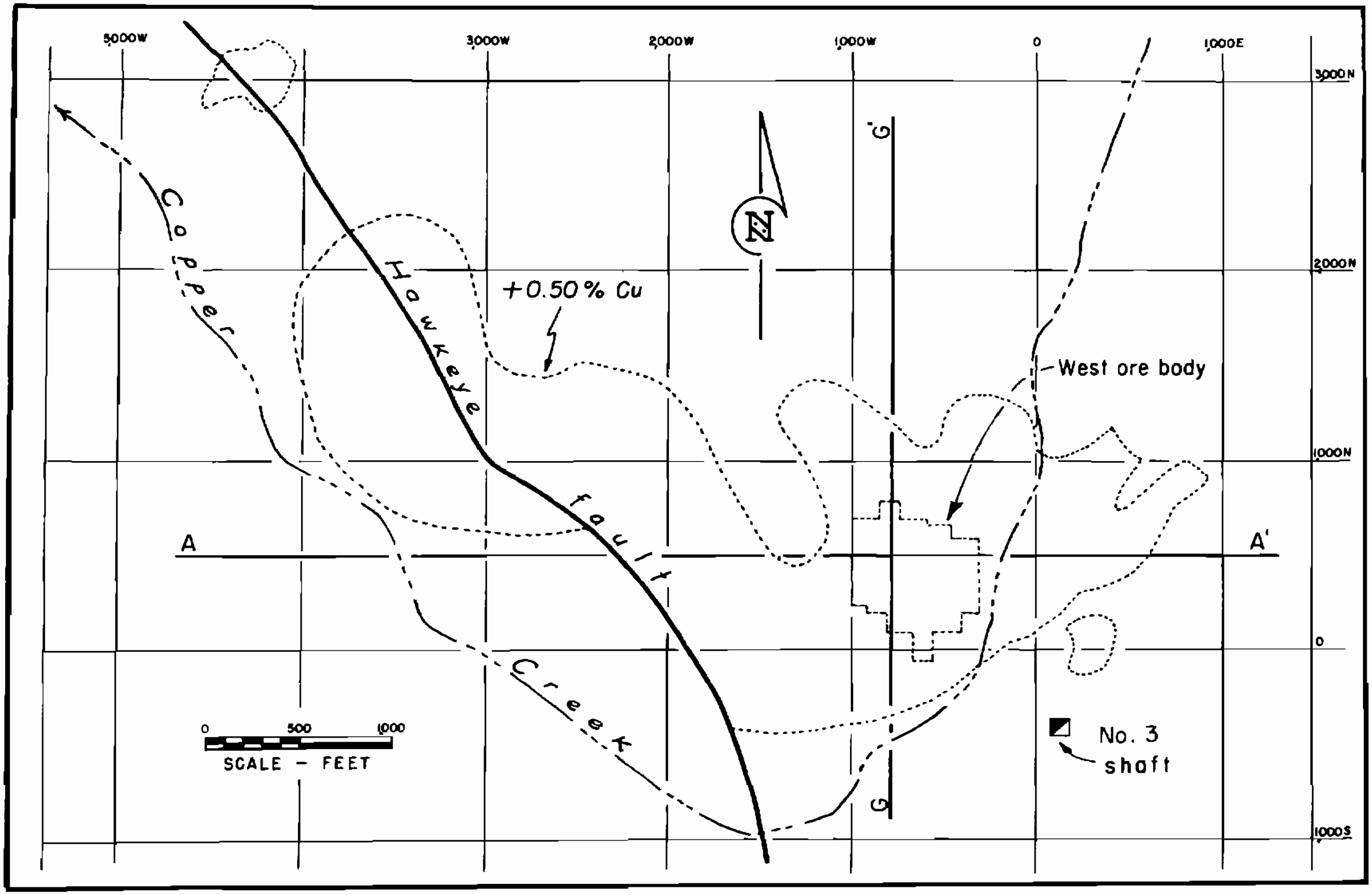

FIGURE 4. - Outline of Sulfide Ore Body.

DEVELOPMENT

By 1930 the mine had been developed through two shafts. No. 1 shaft, 100 feet deep, was equipped with a 50-foot timber headframe and a single-drum hoist powered by a 40-hp. motor. No. 2 shaft (depth unknown) was equipped with a 45-foot steel headframe. Cars were caged and hoisted with a double-drum hoist operated by a 100-hp. electric motor. No. 2 shaft was used to hoist ore until No. 3 shaft was completed.

The new No. 3 shaft, 465 feet deep, was completed and placed in operation in March 1943, and after that date all ore was hoisted through it (fig. 5). The main haulage level was at 340 feet in the new shaft, equivalent to an altitude of 2,960 feet. A level serving the upper workings was at 220 feet.9/ The skip-loading station at 400 feet was served by a 500-ton concrete storage pocket. Ore entering the pocket passed through a grizzly with steel rails spaced 12 inches apart. The shaft had two hoisting compartments, 6 by 7 feet inside the timber, and a manway, 5 by 7 feet (fig. 6). The pump station was at 440 feet. The shaft was covered by a steel headframe 125 feet high. Two 3-ton rockover-type skips were raised and lowered by a 300-hp. double-drum hoist. Ore was released from the pocket into a measuring chute, then into the skip through two air-operated doors. At the surface mine-run ore was dumped into a 1,000-ton storage bin.

Drifts on the main haulage level were driven 8 by 8 feet in cross section. Drifts in the upper levels were 5 by 7 feet and timbered only where necessary. Draw raises and corner raises were 4 by 4 feet and untimbered. Jackhammers were used in drifts, and light self-rotating stopers in raises; both machines used threaded steel and jackbits. The ground was highly fractured, and a 5- by 7-foot drift was broken to a depth of 4 feet by a round of 9 to 11 holes. The round was loaded with 40 to 50 sticks of 40 percent dynamite.

In the haulage drifts muck was loaded into 2-ton gable-bottom cars with rocker-type loaders. On the upper levels slusher hoists with scrapers were used; the muck was pulled to a raise, then loaded through chutes into cars on the haulage level. Timber was used only where necessary.

MINING

From 1936 to March 1943 the No. 2 shaft was used in hoisting ore. Stopes were developed as described later. One-ton cars of ore were caged and hoisted to the surface, where they were dumped over a 6-inch grizzly into a 25-ton-capacity storage bin. Large boulders were broken by hand with hammers. Ore from the storage bin passed through a jaw crusher in closed circuit with a 2-inch grizzly. The product was carried by a belt over a weightometer to a sampler, then to a second jaw crusher in closed circuit with a 1/2-inch Hummer screen. The undersize was carried by a two-bucket wire-rope tram to the fine-ore bin at the mill, a distance of 480 feet.

9/ Huttl, J. B., Bagdad-Arizona's Latest Porphyry Copper: Eng. Min. Jour., vol. 144, No. 6., January 1943, p. 62.

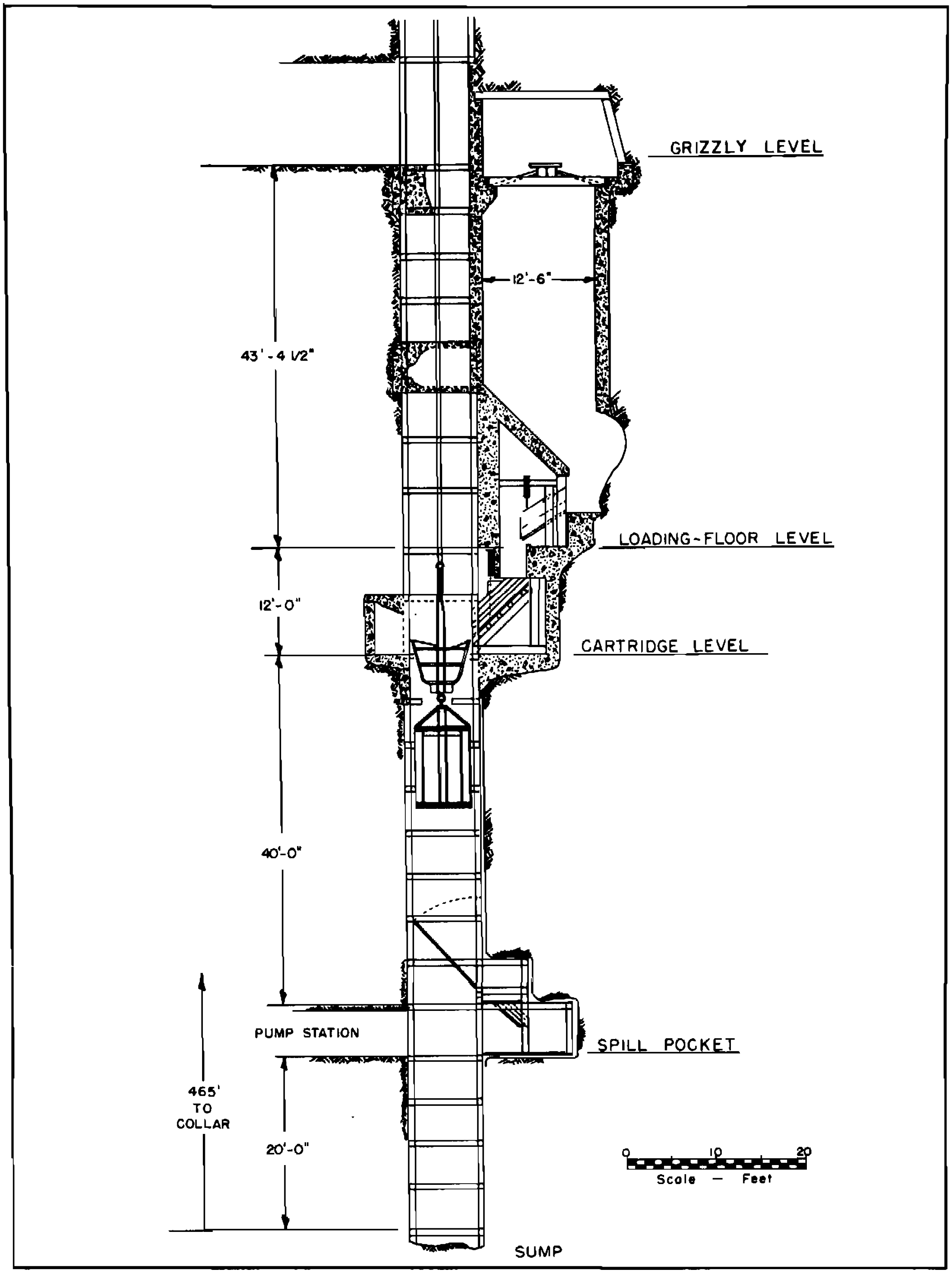

FIGURE 5. - Ideal Section, No. 3 Shaft.

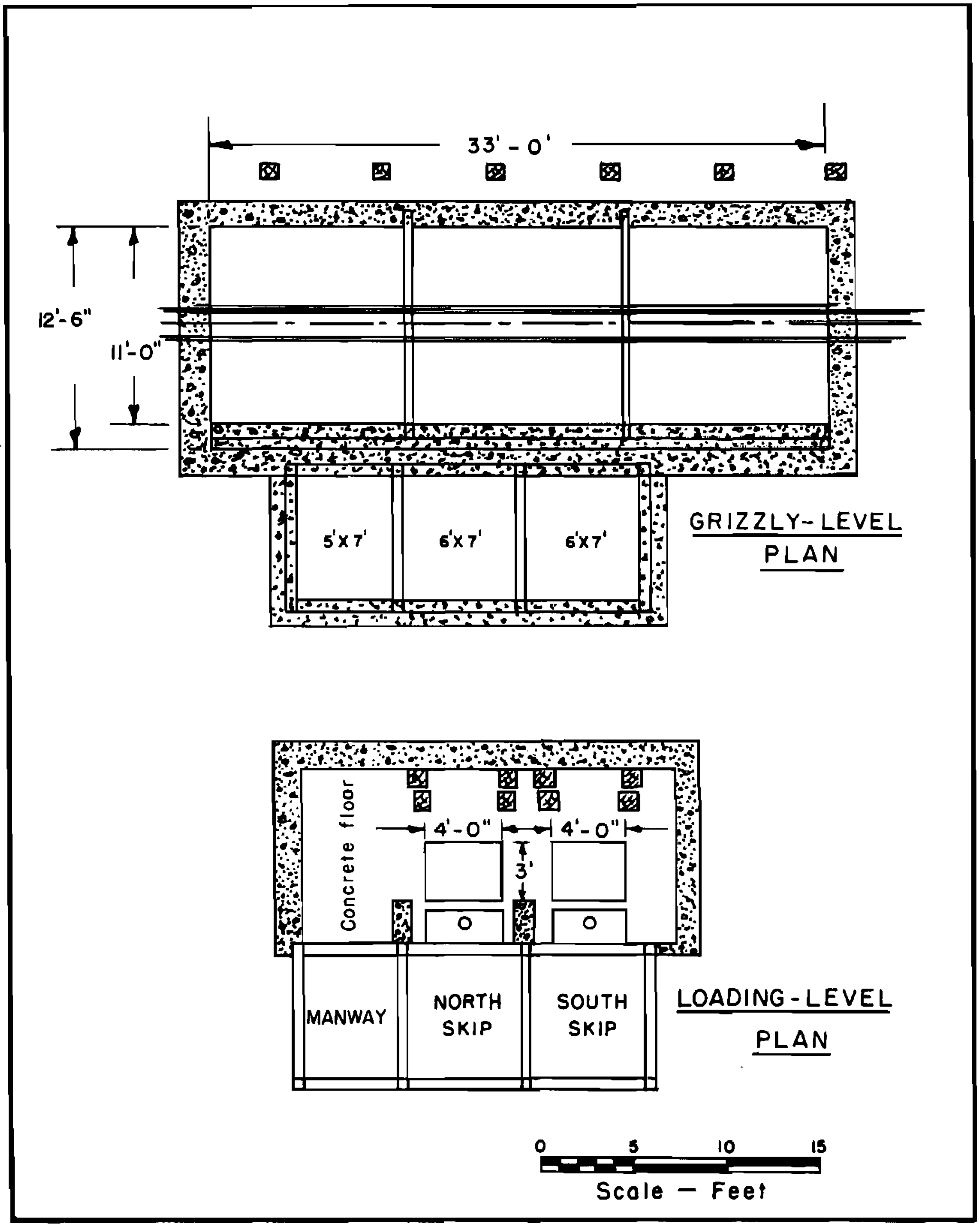

FIGURE 6. - Ideal Plan, No. 3 Shaft.

After March 1943 all ore was hoisted in 3-ton skips through No. 3 shaft and dumped into a 1,000-ton-capacity steel bin at the headframe. The crusher at the mill was fed by a pan feeder from this bin.

Block-Caving Method

Mining was divided into stope development, stoping, and transportation. The west or main ore body was developed first. Main haulage drifts were driven to this area (fig. 7), which was divided into panels 100 feet wide and extending across the ore body. Blocks with a base 100 feet square and extending the height of the ore body were developed as necessary, and the ore was drawn and trammed to the shaft pocket in 2-ton cars pulled by battery-type locomotives.

Stope Development

After a block was selected for mining, two haulage drifts spaced 50 feet apart and centered with the block were driven on the haulage level, which was 30 feet below the undercut level. Haulage drifts were timbered under the block (fig. 8). Grizzly or control drifts were not used; the draw was controlled at the chute set in the haulage drift. Draw or chute sets (fig 9) were spaced 25 feet apart along the drift. Intermediate sets filled in between the combination draw and chute sets (fig. 10). From one corner or draw-chute set, either in the stope being developed or in an adjacent stope, a raise was driven to the corner of the block at the undercut level, then driven vertically. The raise was used for a working entrance and ore pass for boundary-slice level drifts (fig. 11), established at 30-foot intervals.

In the second stage of development, draw raises (fig. 12) were driven at each chute set. These raises, 4 by 4 feet without timber, were driven inclined at 50° for 14 or 15 feet then vertically for 13 feet to the undercut level. Permanent steel arc-type chute gates were installed as the raises were completed. Corner raises were completed at all corners, and boundary drifts around the block at 30-foot vertical intervals. Stoper holes, 8 feet deep, were drilled every 3 feet and staggered along the backs of the boundary drifts (fig. 13). As a third stage, track and pipe were removed from the boundary-slice drifts, and the backs were blasted down. Undercut drifts (fig. 14), driven diagonally across the block, connected the tops of the draw raises. Finally, the pillars between the undercut drifts were drilled with 5- and 6-foot holes (three holes to the row and rows spaced 3 feet along the drift) and blasted. The tops of the draw raises were belled out with three rings of holes. Blasting was begun in one corner of the stope, and "bell out" in the draw raises was blasted with the undercut as it advanced across the stope.

When development work on a stope was begun, an outline of the block was sketched in light lines. As the development was completed, heavy lines were used. Figure 15 is such a sketch, copied from company records.

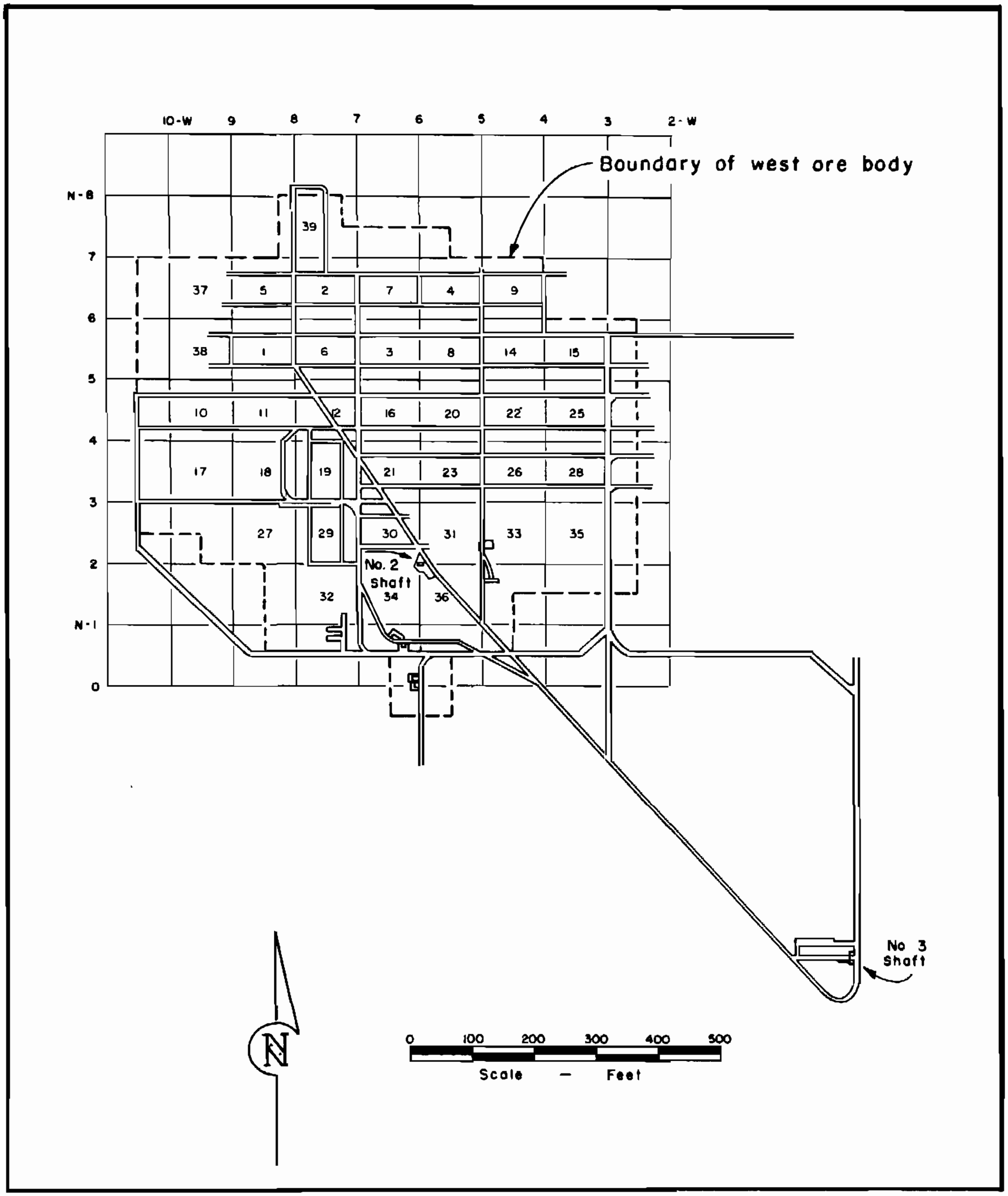

FIGURE 7. - The 2960 Haulage Level.

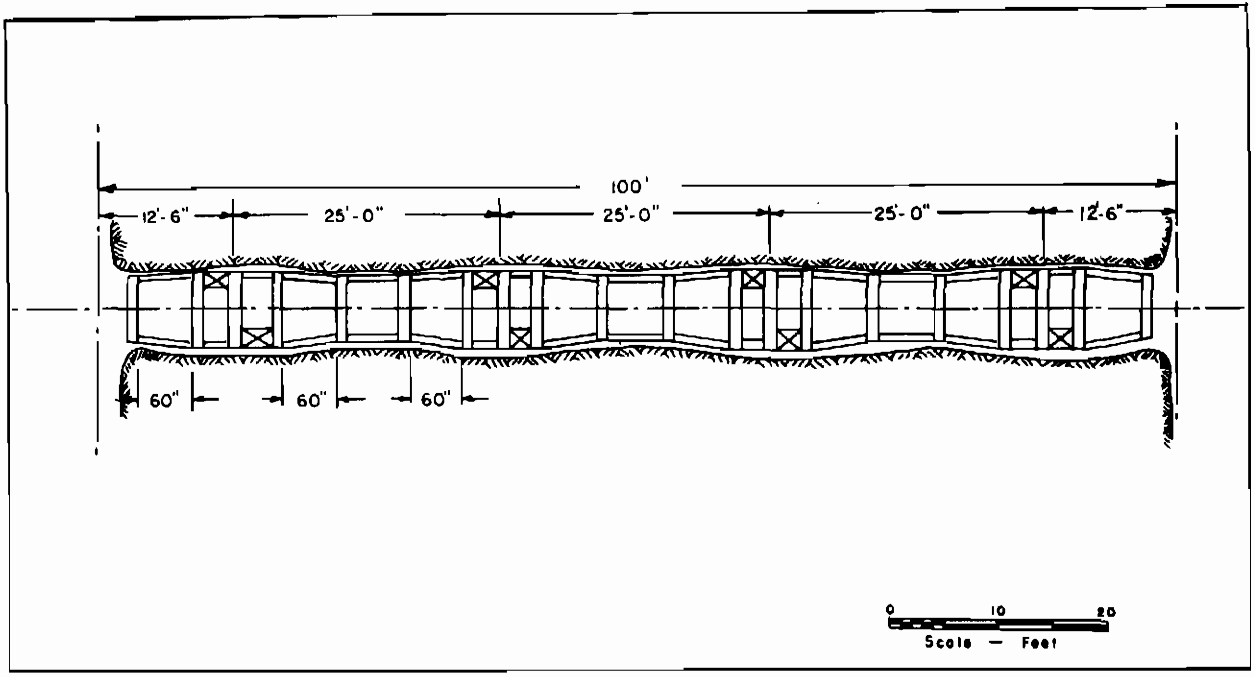

FIGURE 8. - Plan of Chute and Intermediate Sets on Haulage Level.

Stoping

Stopes were 100 feet square. No pillars were left between the stopes. The rock was hard and broke coarse and blocky. This coarse material frequently clogged the throat of the draw raise and made it necessary to blast the larger boulders. A bomb made by tying sticks of dynamite to the end of a 1- by 1-inch blasting stick was placed against the boulder, and the bottom end of the blasting stick was wedged to hold the charge in place. The dynamite was fired with an electric blasting cap.

Transportation

Ore was drawn from the stope through a 20- by 30-inch opening into 2-ton gable-bottom cars. The rate of draw was controlled by hand-operated, steel arc gates. Three 1-1/2-ton battery locomotives were used to position the cars under the loading chutes and assemble them in trains. Two 3-ton battery locomotives pulled the loaded trains 1,200 feet to the shaft pocket, where the ore was dumped through a grizzly of steel rails spaced 12 inches apart.

Ore from the 500-ton concrete-lined pocket was drawn into a measuring cartridge, then dropped into the skip. Flow was controlled by air-operated gates. Spillage was collected in a pocket and returned to the haulage level with an auxiliary hoist.

Glory-Hole Method

The caving method proved successful at 200 to 300 tons per day; however, when the capacity of the plant was expanded to 2,500 tons daily in 1943, labor

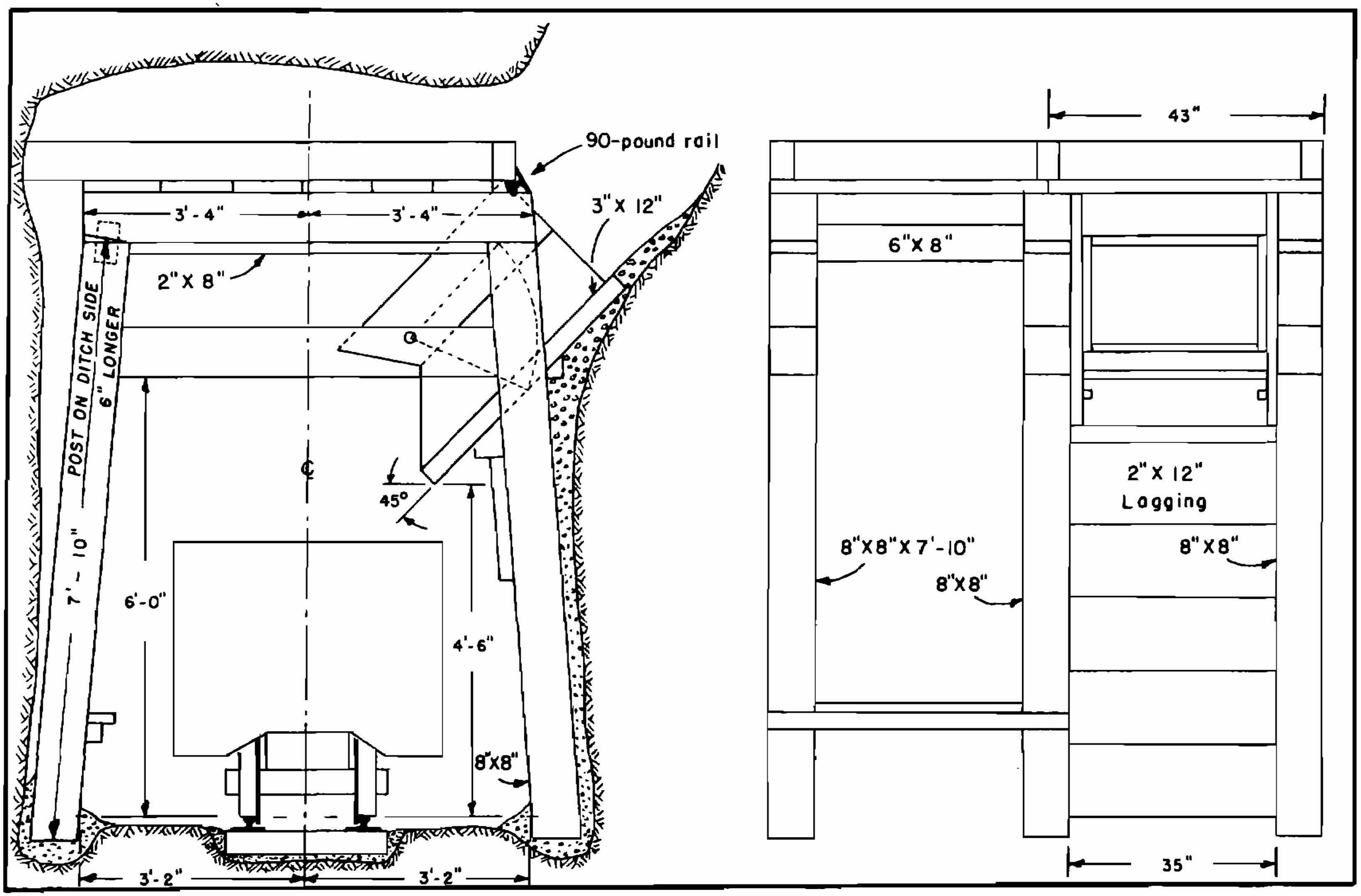

FIGURE 9. - Standard Chute Set.

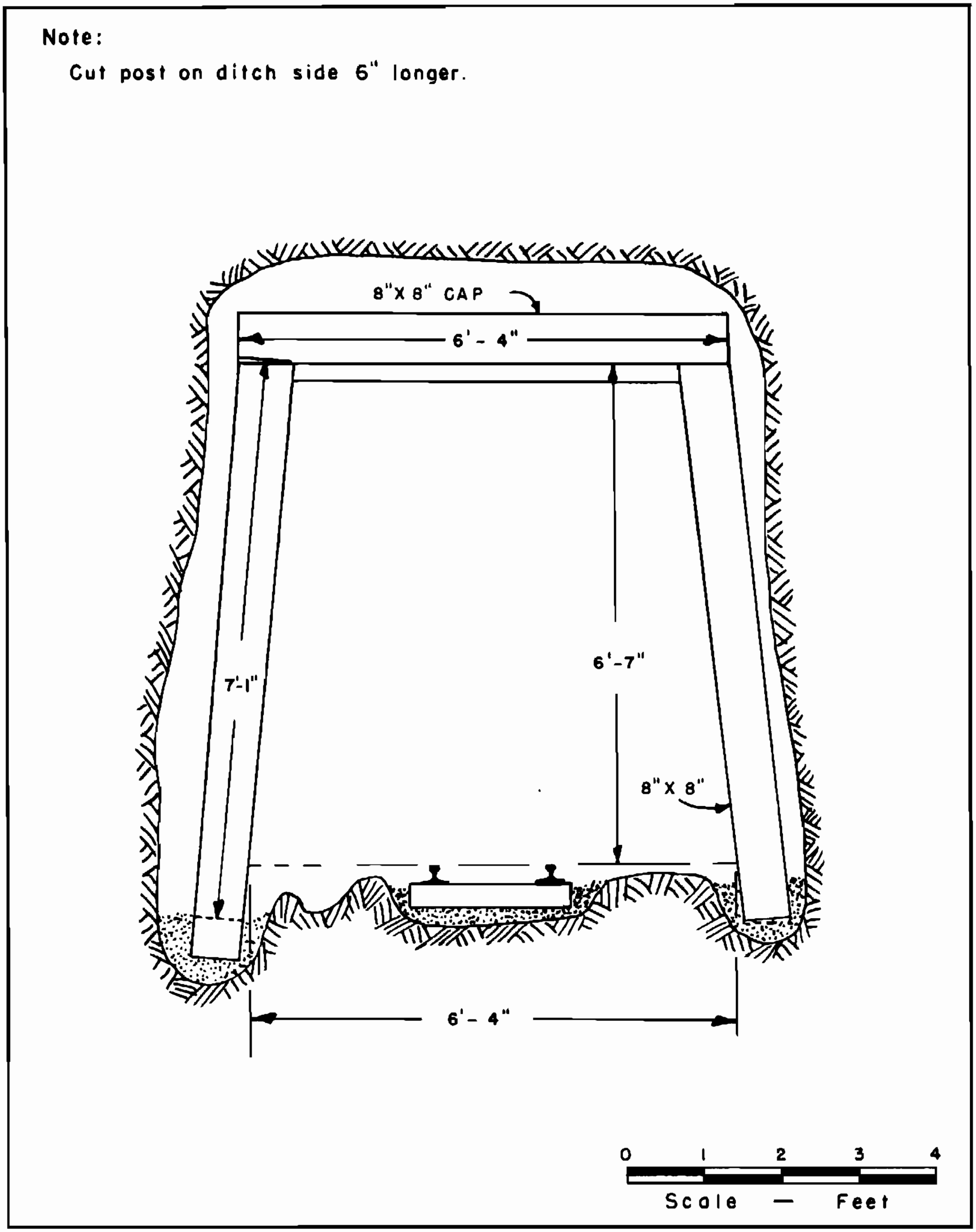

FIGURE 10. - Standard Intermediate Set.

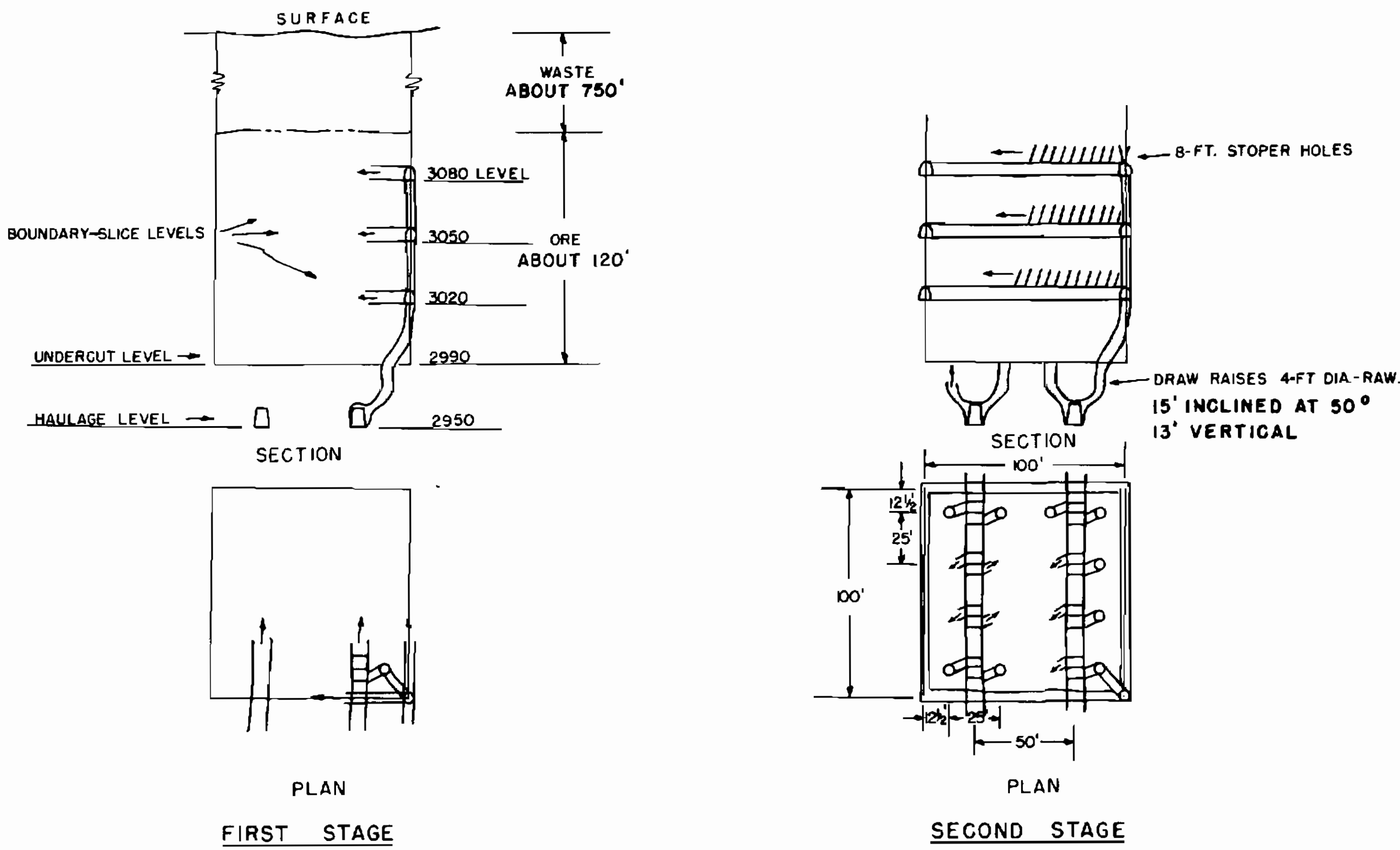

FIGURE 11. - Procedure in Stope Development, Stages 1 and 2.

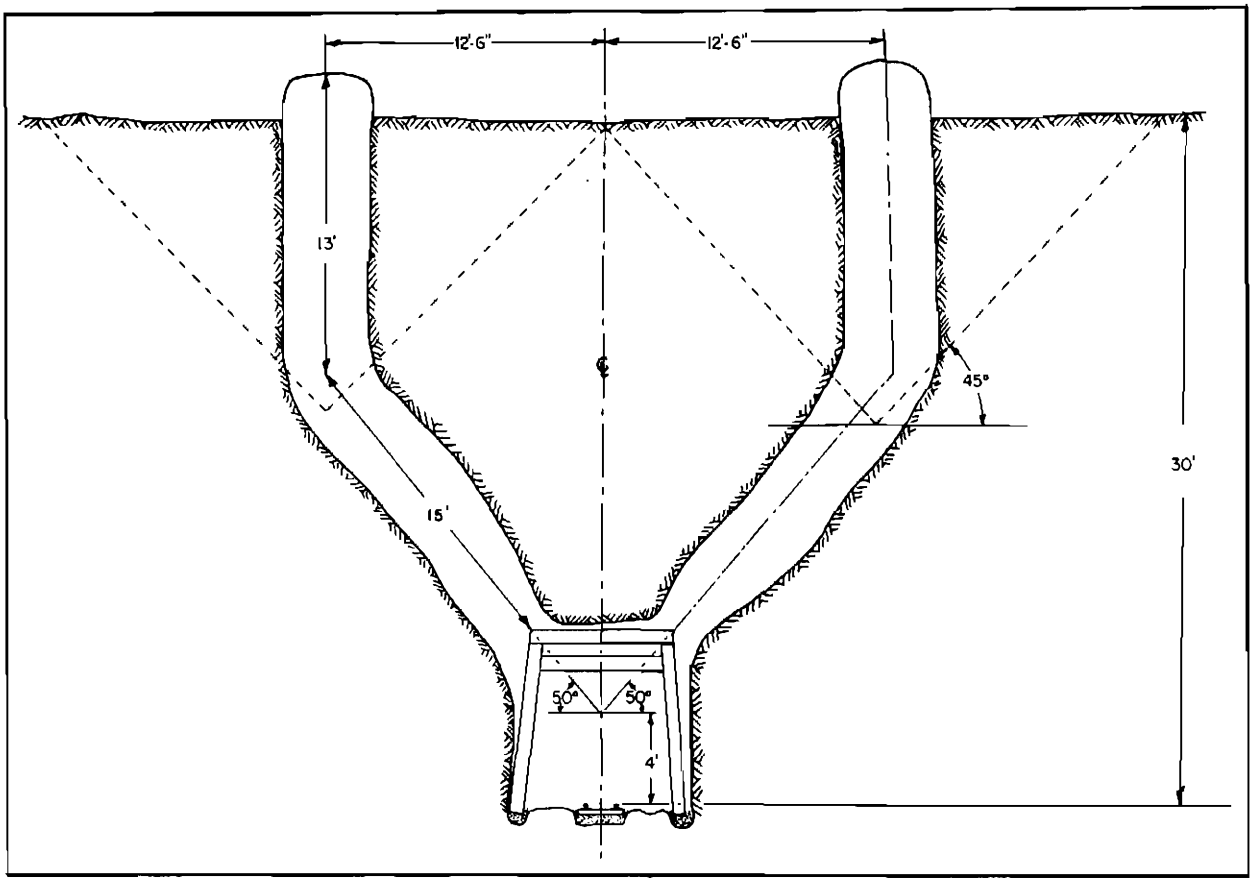

FIGURE 12. - Section Through Chute Raise Showing Development.

was scarce as a result of World War II, and development lagged. Furthermore, the average thickness and the hardness of the chalcocite ore were such that the tonnage recovered per foot of development was low. This resulted in rapid draw on the developed blocks, which increased the dilution from the oxide capping. Part of the ore body came close to the surface, and caved stopes broke through it (fig. 16). Some of the ore was covered with relatively thin overburden along Copper Creek.

An area was selected where the overburden was thinnest, then the overburden was stripped of waste by contract. A raise was started from the underground workings below the area and driven to the surface. After the raise reached the surface, the ore at the top was blasted and bulldozed directly into the raise. On the haulage level the ore was drawn into cars and trammed to the shaft pocket. This method was more economical than block caving and was expanded until four glory holes were in operation. The block caving then gradually was discontinued.

Mining Efficiencies

In 1941 the mine operated 336 days and yielded 88,209 tons of ore for an average of 262.5 tons per operating day. Operating efficiencies were as follows:

Department	Average man-shifts per day	Tons per man-shift
Mine	18.3	14.48
Mill	16.72	15.70
Powerhouse	4.25	58.03
Total plant	39.27	6.67

In January 1944 the draw crew produced 41.3 cars per man-shift. When chute blasters and car spotters were included, this was reduced to 27.2 cars per man-shift. The car factor was 1.446 tons. The stope efficiency of the muck and repair crews (in 2-week periods) is shown in table 1. During 1945 and 1946, both caving and glory-hole methods were used; stope efficiencies of the two methods (in 2-week periods) are compared in table 2. The figures in this table include stoping, tramming, and repair work, but not hoisting and development labor.

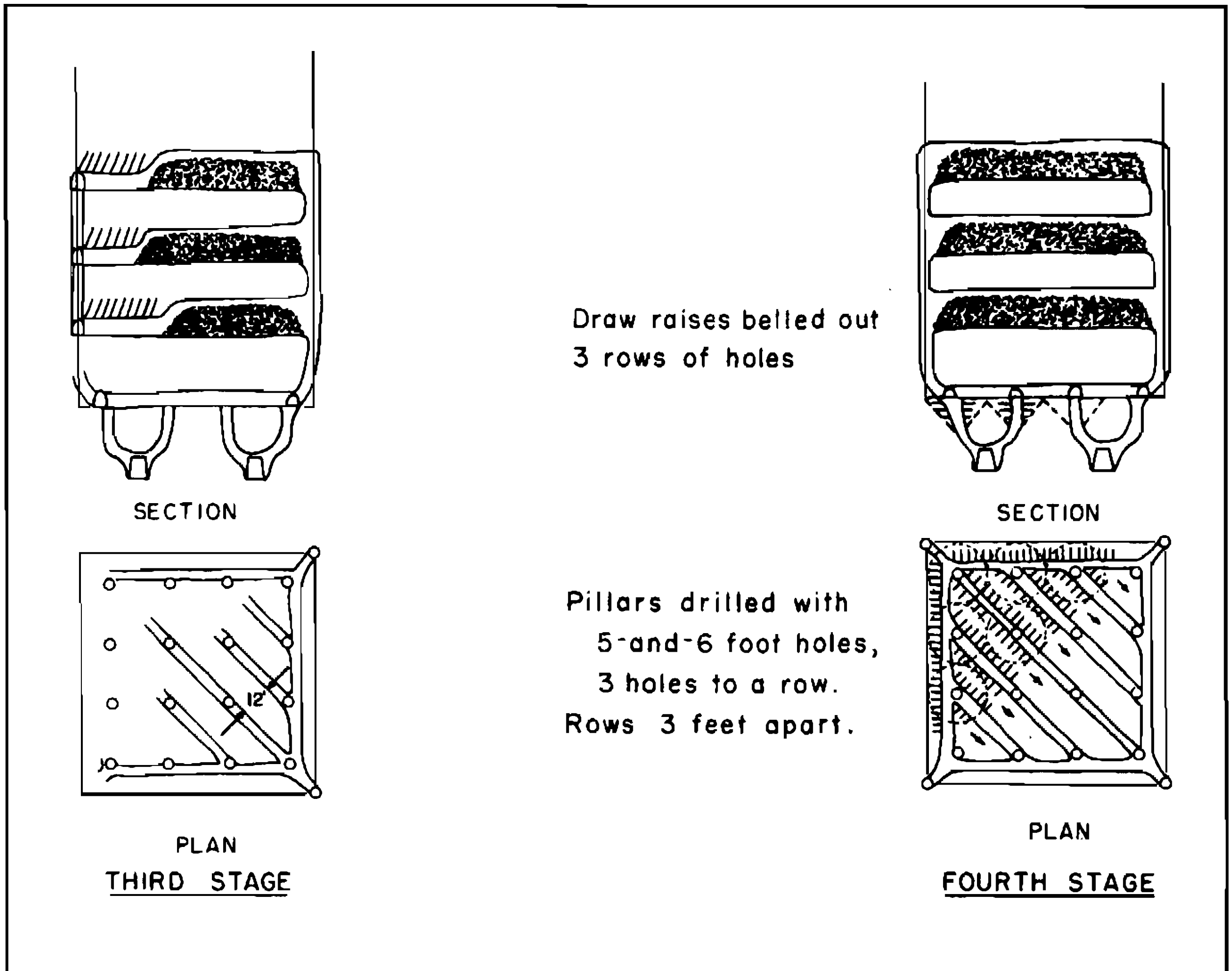

FIGURE 13. - Procedure in Stope Development, Stages 3 and 4.

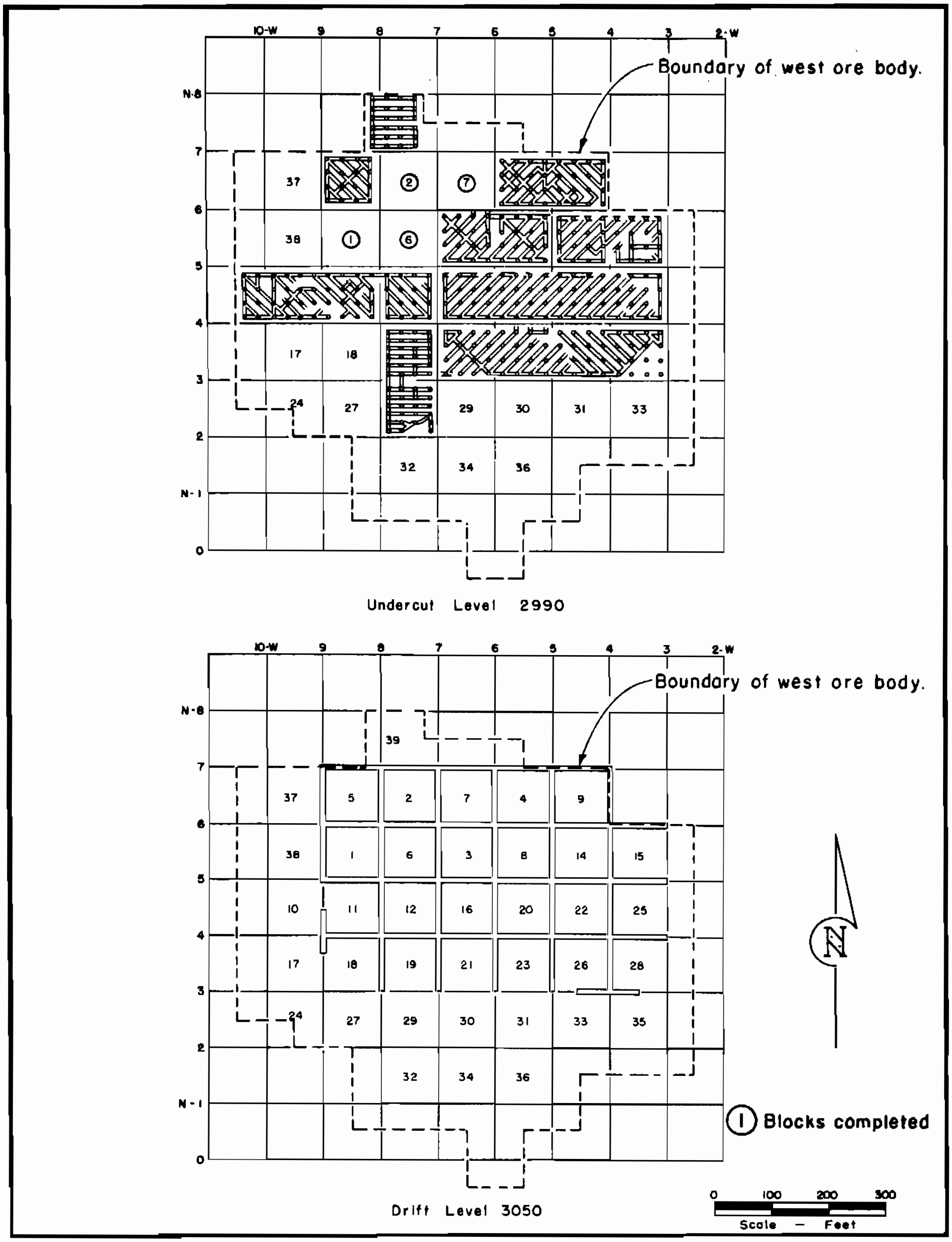

FIGURE 14. - Stope Development Levels.

EXTRACTION AND PRODUCTION

Development of stope 1 was completed in February 1937; 10,107 tons of ore was removed. Stoping was begun immediately and was completed in 1941. Stopes were numbered consecutively as development was begun (see fig. 7). The record for the first 15 stopes is shown in table 3, and the overall extraction for the caving method in the Bagdad ore body is indicated by the totals for these stopes.

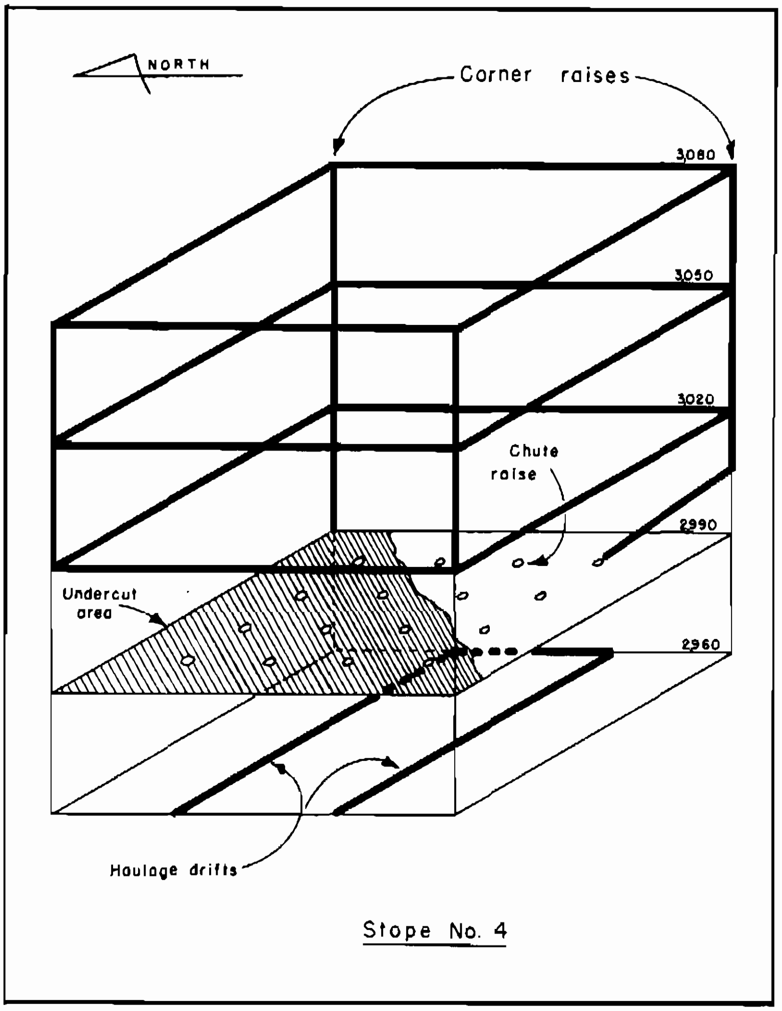

FIGURE 15. - Typical Sketch Showing Stope Progress.

Before 1928 some high-grade ore had been mined and shipped to custom smelters. Production before 1933 was 600,000 pounds of copper, according to Elsing and Heineman.10/ Some 277,501 pounds of copper was produced from the 50-ton test mill built in 1928 and operated through 1930. This test plant was rebuilt in 1930, and capacity was increased to 200 tons per day; but, because of low market prices for copper, the plant was not operated from 1931 to 1934. Beginning in 1935, the mine was prepared for mining by the block-caving method, and the capacity of the plant was increased to 250 tons per day by improving the flowsheet.

Production from 1935 to 1941 was primarily for testing application of the caving method to the Bagdad ore body. Stopes 1 and 2 were undercut and treated during this period under carefully supervised conditions. Although the development per ton of ore was high because of the thinness of the ore body, the caving method operated successfully; therefore, it was decided to expand the operation to 2,500 tons per day. Because of a shortage of copper in the United States during World War II, the smaller plant continued to operate until the new plant, with a capacity of 2,500 tons per day, was completed in March 1943.

10/ Elsing, Morris J., and Heineman, Robt. E. D., Arizona Metal Production: Arizona Bureau of Mines, Bull. 140, 1936, p. 102.

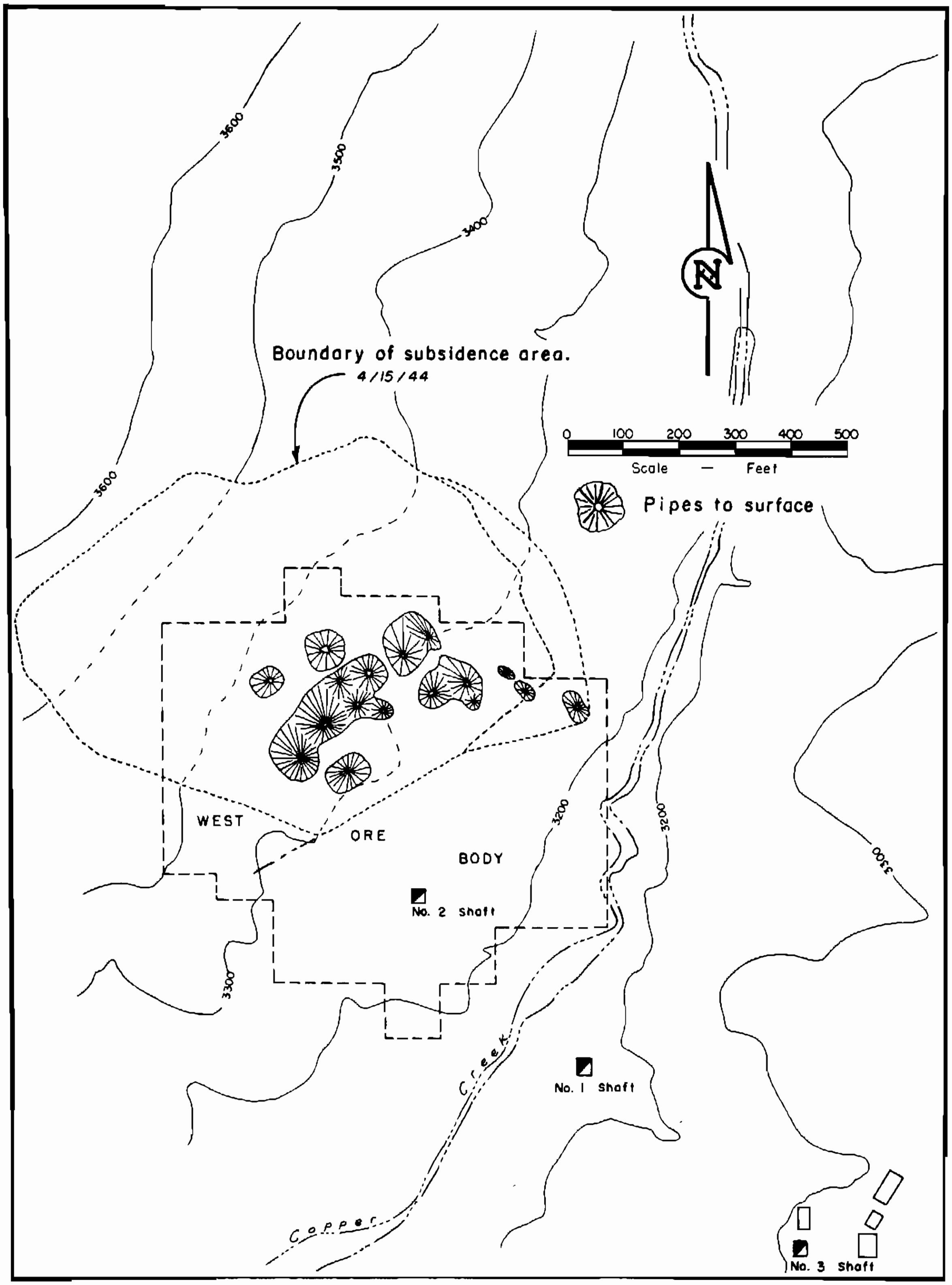

FIGURE 16. - Surface Subsidence.

TABLE 1. - Stope efficiency (1944)

Tons hauled in 2-week periods	Muck crew		Repair crew		Total	
	Shifts	Tons per man-shift	Shifts	Tons per man-shift	Shifts	Tons per man-shift
29,968	972	30.83	72	416.2	1,044	28.7
28,272	887	31.87	53	533.4	940	30.4
24,175	779	31.03	44	549.4	823	29.4
25,795	855	30.17	63	409.4	918	28.1
31,216	1,009.75	34.87	86	363.0	1,095.75	28.5
32,054	984	32.58	71	451.5	1,055	30.4
28,539	769	37.11	62	460.3	831	34.3
29,805	879	33.91	99	301.0	978	30.5
35,108	959.50	36.59	72	487.6	1,031.50	34.0
29,798	874	34.09	67	444.7	941	31.7
32,408	870	37.25	76	426.4	946	34.3
25,358	738	34.36	72	352.2	810	31.3
21,980	643.50	34.16	58	379.0	701.50	31.3
23,105	650	35.54	69	334.9	719	32.2
27,164	841.50	32.28	66	411.6	907.50	29.9
25,463	754.50	33.75	57	446.7	811.50	31.4
30,928	873.75	35.40	53	583.5	926.75	33.3
28,570	753	37.94	50	565.4	803	35.6
29,910	875	34.18	43	695.6	918	32.6
29,601	805	36.77	75	394.7	880	33.6
34,204	878	38.96	115	297.4	993	34.4
29,703	749	39.65	78	380.8	827	35.9
29,346	723	40.58	81	362.3	804	36.5
29,201	730.50	39.97	35	834.3	765.50	38.1
691,671	19,853.0	34.8	1,617	427.7	21,470	32.2

TABLE 2. - Comparison of stope and glory-hole efficiencies (methods used during 1945 and 1946)

Stope			Glory hole		
Tons in 2-week periods	Shifts	Tons/shift	Tons in 2-week periods	Shifts	Tons/shift
2,714	296	9.17	35,800	379	94.46
1,609	254	6.33	33,417	361	92.57
3,596	325	11.06	36,787	418	88.01
2,368	259	9.14	35,507	410	86.60
1,830	216	8.47	33,310	343	97.11
2,629	270	9.74	35,249	370	95.27
6,670	326	20.46	35,875	384	93.42
4,318	308	14.02	34,662	350	99.03
5,561	191	29.12	29,900	316	94.62
9,504	441	21.55	31,799	355	89.50
11,013	645	17.07	34,500	376	91.76
11,649	615	18.94	26,356	275	95.84
7,777	458	16.98	25,427	283	89.85
10,414	545	19.11	23,945	310	77.24
13,939	616	22.63	14,469	193	74.97
12,251	547	22.40	13,579	190	71.47
107,842	6,312	17.09	480,582	5,313	90.45

TABLE 3. - Stope record

Stope No.	Estimated			Drawn			Extraction, percent		
	Tons	Copper, percent	Copper, pounds	Tons	Copper, percent	Copper, pounds	Tons	Grade	Copper
1	102,300	1.495	3,058,770	104,808	1.396	2,926,239	102.4	93.4	95.7
2	98,800	1.697	3,353,272	103,571	1.525	3,262,486	104.8	89.9	97.3
3	102,470	1.890	3,874,000	118,533	1.370	3,237,672	115.6	72.5	83.5
4	99,800	1.530	3,084,442	66,249	.860	1,142,100	66.4	56.2	37.0
5	95,950	1.150	2,229,850	37,390	.910	679,700	38.5	79.1	30.5
6	110,000	1.975	4,345,000	113,807	1.380	3,142,364	103.5	69.9	72.4
7	98,800	1.851	3,657,576	111,981	1.490	3,342,170	113.3	80.5	91.4
8	97,200	1.602	3,114,290	104,465	1.230	2,574,172	107.5	76.8	82.7
9	96,100	1.370	2,634,000	36,546	.920	676,100	38.0	67.2	25.7
10	161,800	1.200	3,883,200	134,359	.740	2,000,700	83.0	61.7	51.5
11	138,760	1.330	3,699,342	123,500	1.100	2,722,650	89.0	82.7	73.7
12	132,720	1.805	4,777,920	166,150	1.290	4,301,287	125.1	71.5	90.0
14	94,650	1.270	2,404,110	77,639	.770	1,195,961	82.0	60.6	49.7
15	93,470	1.360	2,542,380	77,174	.930	1,432,427	82.6	68.4	56.3
16	100,050	1.569	3,139,570	65,880	.890	1,169,420	65.8	56.7	37.6
Total	1,662,760	1.534	49,797,722	1,442,052	1.172	33,805,448	88.9	76.4	67.9

Although the new concentrating plant was completed and placed in production in March 1943, the shortage of skilled miners and the lag in stope development prevented daily mine production from reaching the 2,500-ton rate until production from caving was augmented by some ore from glory-hole operation. Production by years during the life of the underground operation is shown in table 4.

TABLE 4. - Production by years

Year	Mill capacity, tons per day	Tons milled	Copper produced, pounds
1937	250	75,512	1,537,396
1938 1/	250	29,200	2/ 681,000
1939 1/	275	14,196	2/ 297,000
1940	275	73,600	1,540,000
1941	275	88,200	2,298,000
1942	275	67,900	1,090,000
1943	3/ 2,000	377,271	7,370,934
1944	2,000	682,484	9,818,181
1945	3,000	618,711	8,229,049
1946	3,000	862,535	12,226,848
1947 4/	3,000	957,302	13,569,945
Total	-	3,846,911	58,658,353

1/ Operations terminated May 1, 1938, and resumed Nov. 1, 1939.
2/ Estimated.
3/ New mill opened March 1943.
4/ Partly by open-pit method.

AUXILIARY OPERATIONS

The underground mine produced very little water - about 150 gallons per minute. Before 1942 water for milling the ore was obtained from wells in the gravel of Copper Creek and from springs. After the RFC approved a loan to equip the property for production at the rate of 2,500 tons per day, a 10-inch pipeline was constructed to wells along Burro Creek, and provision also was made to recover water from the tailings dam.

During the period before 1943, power was generated in a diesel plant at the mine. As part of the expansion program in 1942 a powerline was constructed from Parker Dam on the Colorado River to the mine, and after February 1943 power was obtained from that source.

Because the mine was isolated, houses for employees were constructed by the company. All materials and supplies needed for the operation were trucked from Hillside, a station on the Atchison, Topeka & Santa Fe branch line between Ash Fork and Phoenix. This highway from Hillside to the mining camp was improved by the State highway department.

WAGE, CONTRACT, LEASING, AND BONUS SYSTEMS

In 1941 wages for miners and mill operators were $6.08 per day and for muckers and mill helpers, $5.07 per day. Development work was completed by contract, and the miner received a bonus for all footage over standard. Later the bonus system was expanded to include the muck crew for both stope and glory hole.

CONCENTRATING PLANT

A complete description of the concentrating plant is beyond the scope of this paper. The size of the mine and the contemplated rate of production have been less than required for economical operation of a smelter on the property; however, the isolated location made a long truck haul necessary for supplies and concentrates. The long truck and rail hauls, plus custom smelting charges, necessitated the elimination of all possible weight from the concentrates. These factors also prompted experimental work with a leaching process.

With the objective of producing refined copper at the mine, milling tests were started. The first mill built for experimental testing was a 50-ton flotation mill. According to Young,[11] recoveries in this mill were 87.7 percent of the total copper and 91 percent of the sulfide copper with a concentration ratio of 32.8 : 1.0 and a concentrate grade of 37.7 percent copper. The flotation concentrate from this test mill was treated in an experimental electrolytic plant. Concentrates were roasted in three stages at 500°, 980°, and 1,100° F. The calcine was agitated with electrolyte, and the copper-bearing solution, which contained 60 grams of copper per liter, was passed through electrolytic cells, then through a trough filled with detinned scrap. During

11/ Young, Geo. J., Another Porphyry Copper in the Making: Eng. Min. Jour., vol. 130, No. 10, November 1930, p. 523.

a 9-month run 97 percent of the copper was precipitated in the electrolytic cells and 3 percent as cement copper. Metallurgically this experimental plant indicated a successful process.

However, more than 40,000 feet of sample-hole drilling had indicated a sulfide ore body containing 1.5 percent copper, which could be treated by flotation. Because of the high capital cost of a plant to roast and leach the ore and precipitate the copper by the electrolytic method, a decision was made to defer treatment of the oxide ore and mine the sulfide ore, which could be treated by the flotation process with smaller capital investment. Accordingly the 50-ton plant was enlarged to a 200-ton-per-day flotation plant in 1930. Because of the low prevailing price of copper, this plant was not operated from 1931 to 1934. Beginning in 1935 the plant operated, with minor improvements, until March 1943, when a new concentrator with a capacity of 2,500 tons per day was completed.

During 1941 the concentrator processed 88,209 tons of ore in 336 operating days, or 265.5 tons per day. Mill recovery was 82.77 percent. Approximately 2,590 tons of concentrate containing 44.33 percent copper was recovered from ore that averaged 1.57 percent copper. In 1945 the new plant processed 618,711 tons of ore.

COSTS

Mine costs are divided into two periods: First, 1937 to 1943, when the plant was operated at a rate of approximately 250 tons per day; and second, the period after expansion to 2,500 tons per day.

The year 1941 represents the first period. A total of 88,209 tons of ore was milled. Operating costs for this period are shown in table 5, and operating costs for a typical month after plant expansion in late 1943, when 38,245 tons of ore was milled, are shown in table 6.

Some additional information on consumption of power and steel and reagent data for a typical month representing this last period are shown in tables 7 and 8.

TABLE 5. - Operating costs, 1941

Stope development costs (typical month - 491 feet of advance):

	Cost per foot
Labor:	
Bonus	$1.60
Company	4.56
Total	6.16
Supplies:	
Powder	.55
Caps and fuse	.15
Timber	.46
Miscellaneous	1.11
Total	2.27
Compressor	1.01
Hoist	.27
Proportion of general expense	3.15
Total	12.86

Total operating costs (88,209 tons of ore milled):

	Cost per ton	Cost per pound of copper, cents
Mining	$0.588	2.257
Milling	1.071	4.111
General	.163	.626
Smelting, refining, marketing	.724	2.779
Total	2.546	9.773

TABLE 6. - Operating costs, 1943 (typical month, 38,245 tons milled)

Mining	Labor	Supplies	Power	Miscellaneous	Total	Cost per ton
Stope preparation	-	-	-	$9,070.89	$9,070.89	$0.237
Drawing and tramming ..	$12,576.10	$2,805.60	$84.37	30.00	15,496.07	.405
Maintenance	1,241.13	60.00	-	-	1,301.80	.034
Hoisting	1,740.98	207.90	162.50	-	2,061.38	.054
Pumping	6.82	64.55	30.80	-	102.17	.003
Ventilation	13.64	52.86	42.64	-	159.14	.004
Proportion of general..	-	-	-	8,187.76	8,187.76	.214
Total	15,578.67	3,191.58	320.31	17,288.65	36,379.21	.951

Milling	Labor	Supplies	Parts	Power	Miscellaneous	Total	Cost per ton
Crushing	$1,134.51	$493.48	$489.91	$211.60	-	$2,329.50	$0.061
Ball mills	774.26	5,311.05	661.44	1,468.05	-	8,214.80	.215
Flotation, copper	720.03	1,512.00	618.16	432.72	-	3,282.91	.086
Flotation, moly........	1,095.56	558.52	464.02	55.28	-	2,173.38	.057
Filtering	917.73	22.17	145.72	78.63	-	1,144.25	.029
Tailings disposal	1,380.68	19.19	236.34	-	-	2,136.21	.056
Water	921.44	304.86	24.56	500.39	-	1,751.25	.045
Proportion of general..	-	-	-	-	$4,371.19	4,371.19	.114
Total	6,944.21	8,201.27	3,190.15	2,746.67	4,371.19	25,403.49	.662
Cost per ton	$0.118	$0.214	$0.081	$0.072	$0.114	$0.662	

TABLE 7. - Power consumption (typical month)

	Kw.-hr./ton
Mine:	
Hoisting	0.656
Underground haulage	.306
Mine pumps	.222
Ventilation	.399
Compressors	1.249
Total	2.832
Concentrator:	
Ball mills	8.800
Flotation, copper	2.590
Flotation, moly	.330
Filtering	.470
Water	3.000
Total	16.460
Total mine and mill	19.292

TABLE 8. - Balls, liners, and reagent data (typical month)

Item	Pounds per ton	Cost per ton
Balls	2.04	$0.095
Liners	.25	.032
Lime	3.28	.279
Zanthate	.027	.004
Pine oil	.045	.005
Moly reagents	-	.009
Total		.424