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BY

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MINING METHOD AND COSTS AT THE UTAH COPPER CO., BINGHAM CANYON, UTAH[1]

By A. Soderberg[2]

INTRODUCTION

This paper describing the mining practice at the Utah Copper Co., Bingham Canyon, Utah, is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The Utah Copper Co. operates the low-grade open-cut copper mine at Bingham Canyon, Utah, which is 30 miles in a southwesterly direction from Salt Lake City. In addition to the mine the company operates two flotation concentrators, the Magma plant and the Arthur plant, which have a combined capacity at maximum metallurgical efficiency of 60,000 tons daily. The concentrators are at Magna, about 18 miles from the mine. The company operates its own standard-gage railway, known as the Bingham and Garfield Railway, between the mine and the mills (fig. 1). Concentrates are smelted at the Garfield smelter of the American Smelting and Refining Co., situated 4 miles from the mills.

ACKNOWLEDGMENTS

Acknowledgment for material and assistance received in the preparation of this paper is made to E. E. Barker, engineer of mines, H. C. Goodrich, chief engineer, Louis Buchman, assistant superintendent of mines, J. A. Marsh, mine geologist, and H. L. Garrity, safety engineer, of the Utah Copper Co.

HISTORY

The early history of the Utah Copper Co. has been published so many times that a brief review of its early beginnings will suffice for the purpose of this discussion. Long before the organization of the Utah Copper Co. it was known that large tonnages of low-grade sulphide ore existed at Bingham Canyon. As early as 1887 Colonel Enos A. Wall realized that the property had a potential value. In 1895 Captain Joseph R. DeLamar heard about the deposit and had it examined. A mill test was made and it was determined that a 60 per cent

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U S. Bureau of Mines Information Circular 6234."

2 - One of the consulting engineers, U. S. Bureau of Mines.

recovery could be effected. In 1898 Robert C. Gemmell examined the property and made a very encouraging report to his principals. Efforts to promote the property, failed, however, and it was not until 1903 that D. C. Jackling was able to obtain the necessary capital to purchase options from Colonel Wall. Among Mr. Jackling's backers were Charles M. MacNeil, Spencer Penrose, and R. A. F. Penrose. At this time the property was examined by F. H. Minard, who made another favorable report, and shortly afterwards the Copperton mill of 300 tons daily capacity was designed and constructed.

The Utah Copper Co. was first organized on June 4, 1903, under the laws of Colorado, with a capitalization of $500,000 and shares at $1 each; D. C. Jackling was given the position of general manager. Later, in April, 1904, the company was reorganized under the laws of New Jersey, and the capitalization was made $4,500,000 with shares having a par value of $10. In 1904 the Copperton mill began its operations with G. G. Janney, mill superintendent, in charge. It was on the basis of results obtained in this plant that the Magna mill was later designed.

At first, arrangements were made with the Denver and Rio Grande Railroad Co. to transport the ore from the mines to the mills, but in 1911 it was found necessary for the company to lay its own rails in order to assure a sufficient tonnage at all times for the mills.

The company acquired the Arthur mill in 1910 when the adjoining Boston Consolidated Co. mine was purchased. Since that time additions and improvements have taken place until the present daily capacity of 60,000 tons has been established.

The first shovel was placed in operation in 1906, with J. D. Shilling, sr., as mine superintendent.

GEOLOGY

The geology of the Bingham deposit is simple. Pennsylvania quartzite beds thousands of feet thick are intercalated with a few narrow limestone beds, and intruded by a mass of monzonite porphyry (fig. 2). The genesis of the commercial ore deposit is a typical example of secondary enrichment. The mineralization within the porphyry is the result of a period of intense action following the solidification of the mass; hot hydrothermal solutions caused the comparatively uniform dissemination of chalcopyrite, bornite, and pyrite. Subsequent secondary enrichment has increased the values; meteoric waters dissolved the copper in the upper portions, leaving a red capping, and redeposited it as chalcocite, covellite, and bornite. This mass of porphyry ore, when present reserves are added to the tonnage already mined, measures approximately 800,000,000 tons. The grade of the present reserves averages 1.066 per cent copper.

PHYSICAL CHARACTERISTICS

The Utah Copper ore body has its long axis in a northeasterly and southwesterly direction with an over-all length of about 6,000 feet, a maximum width of 4,000 feet, and a vertical depth of about 2,000 feet. Approximately an average of 115 feet of capping or completely leached porphyry covered the ore; in places, however, the sulphide zone was not

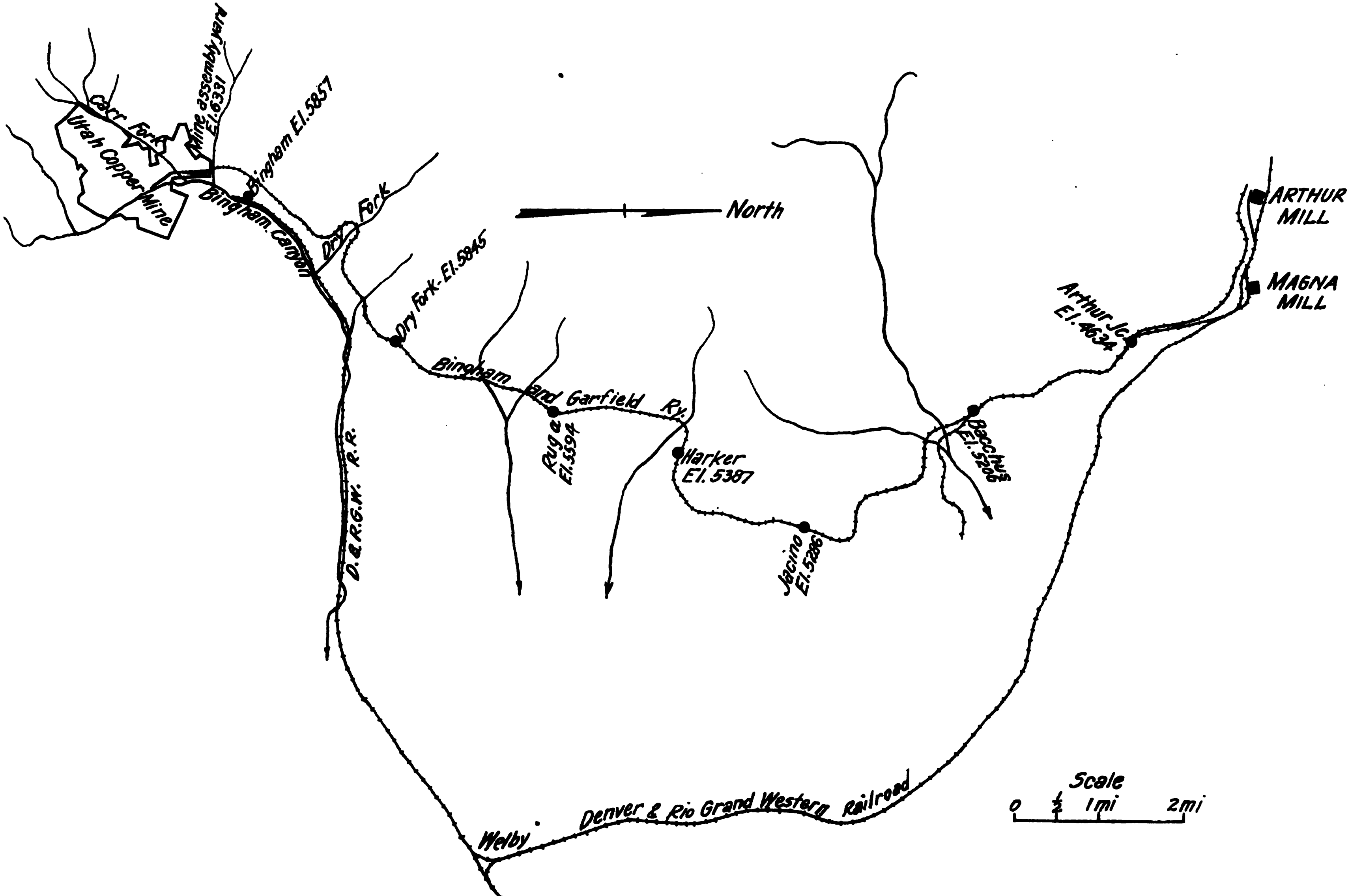

Figure 1.- Map of Bingham and Garfield Railway showing Utah Copper Company Mine and Mills

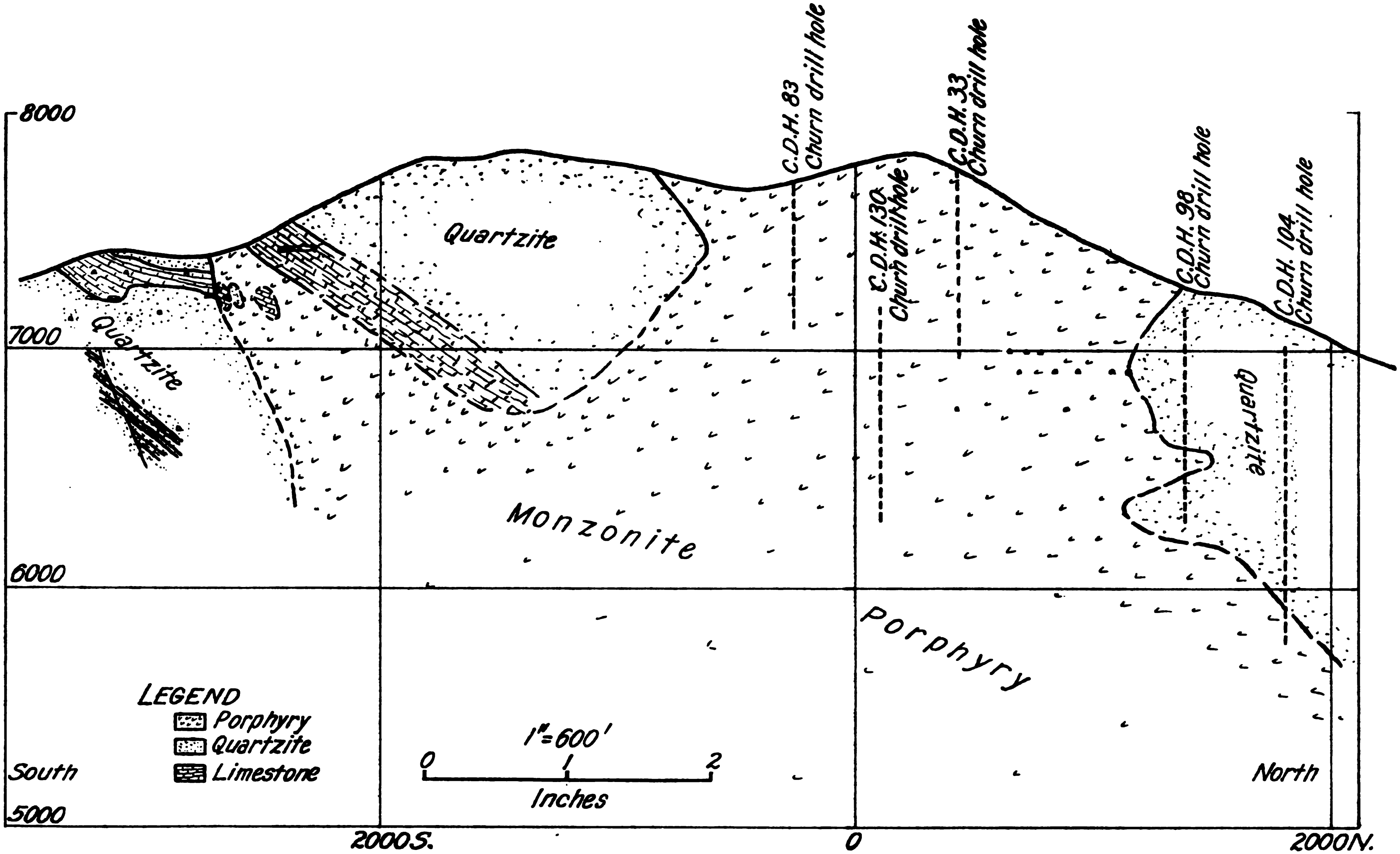

Figure 2.- Typical Geological Section

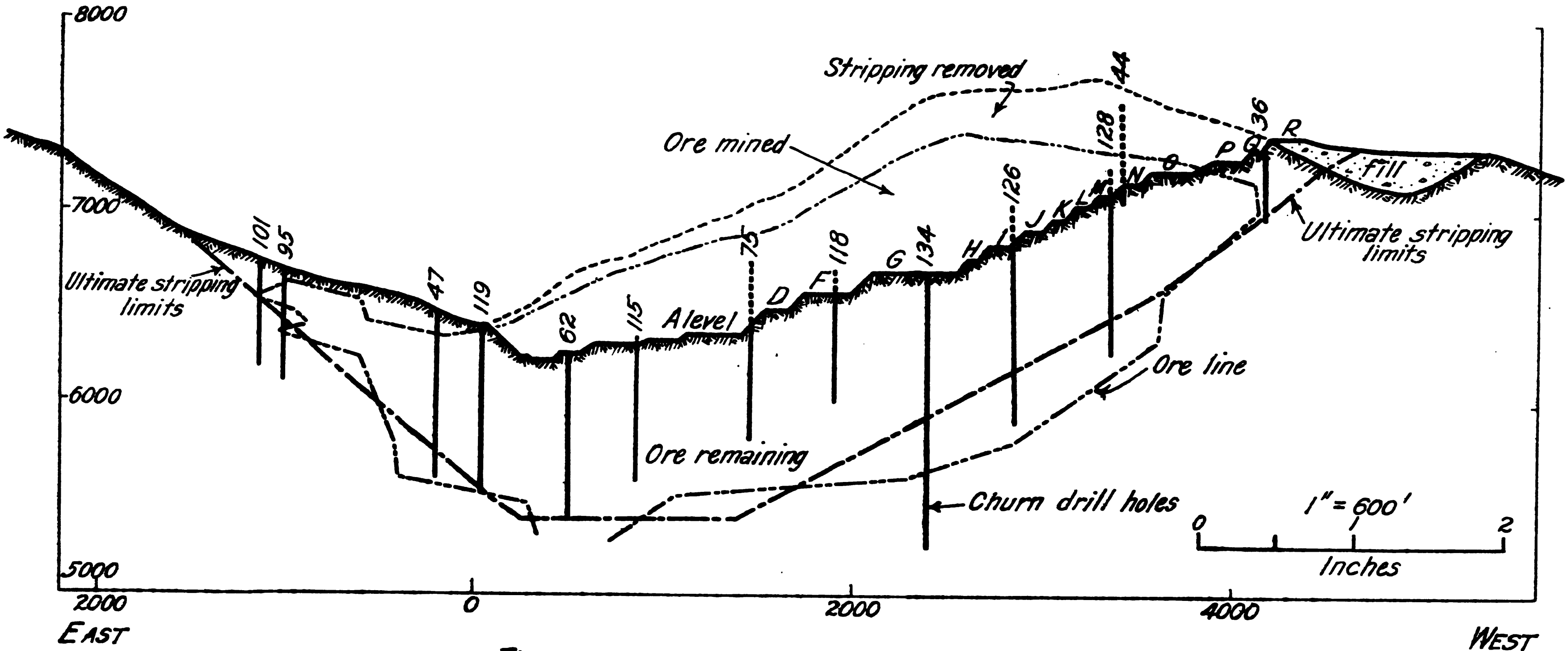

Figure 3. - Typical east-west section

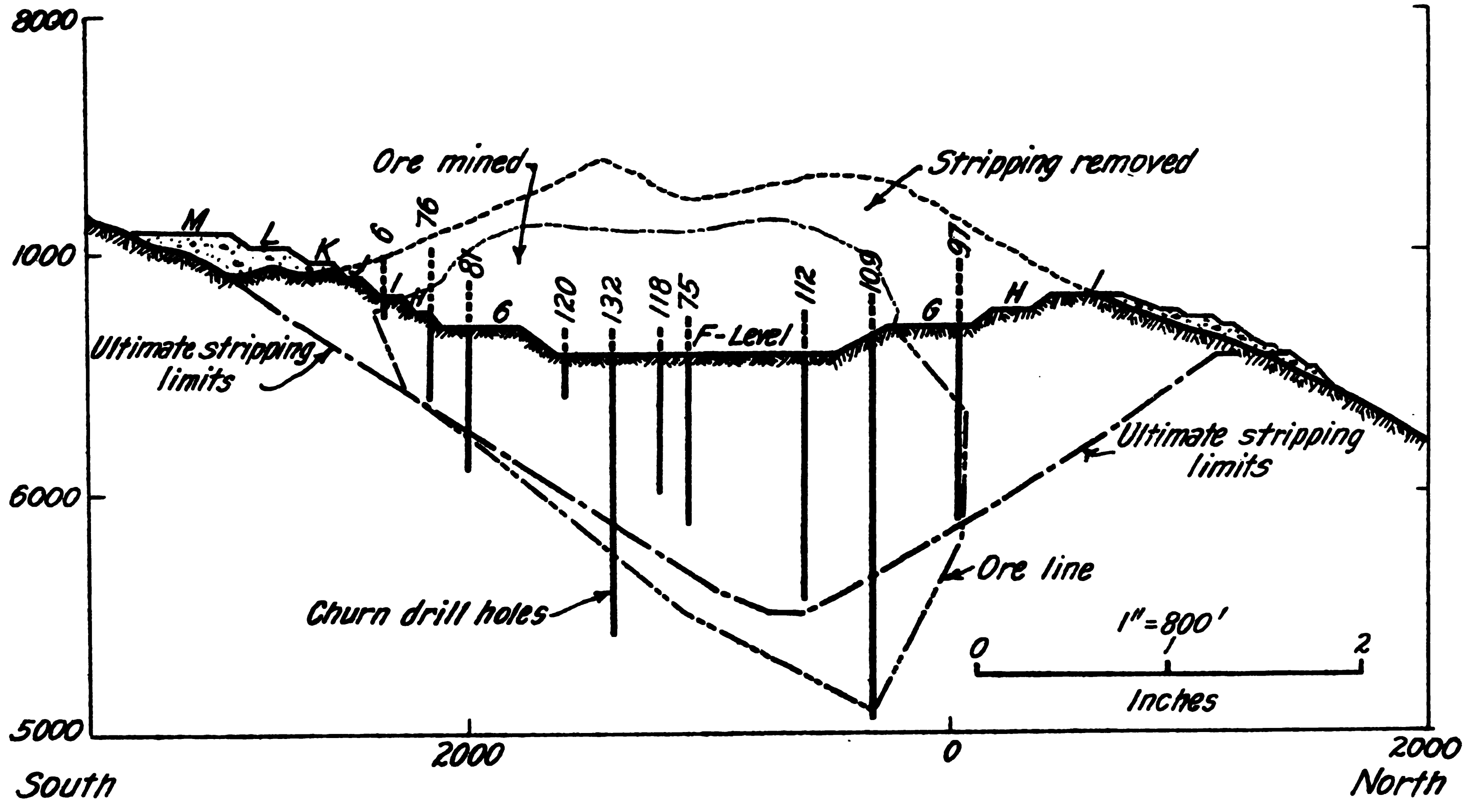

Figure 4. - Typical North-South Section

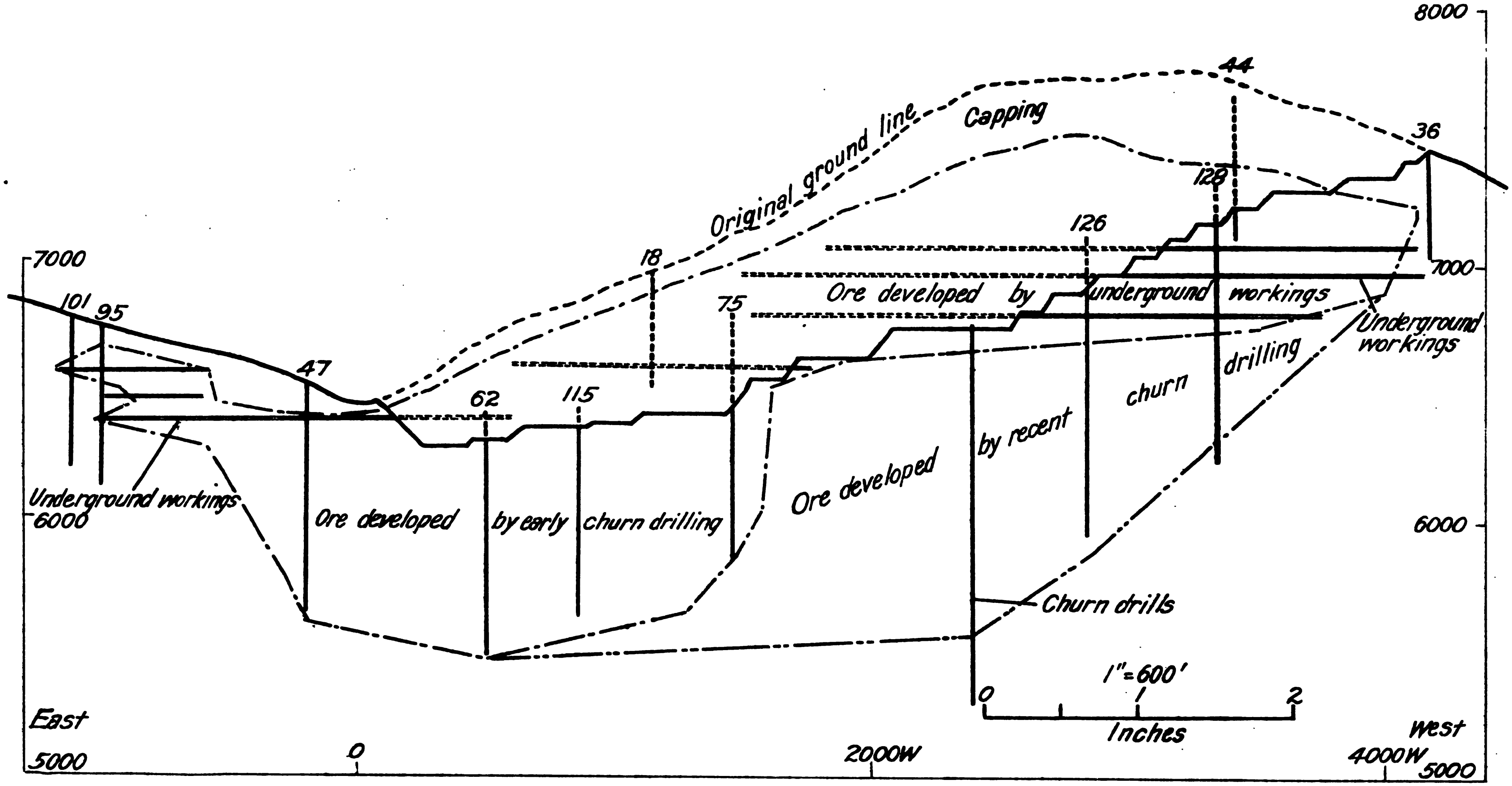

Figure 5-. Illustrative section showing comparative ore development by underground workings and churn drills

more than 20 to 50 feet from the surface. About 100,000,000 cubic yards of this capping has been removed. The ore is relatively soft and breaks easily along fracture planes into sizes that can readily be loaded by power shovels with a minimum of secondary blasting. Copper occurs in the ore as chalcopyrite (80 per cent), chalcocite (9 per cent), covellite (7 per cent), and bornite (4 per cent) uniformly disseminated throughout the intrusive mass.

It was found early that the ore body was one of magnitude and that the ratio of stripping to ore would not be more than one of stripping to two of ore; with these facts in mind it was decided from the outset to strip the overburden with steam shovels and then load the ore by the same method into cars for transportation to the mills. The angle of repose of the finished slope of a steam shovel mine is of prime importance, and it was soon indicated by churn-drill holes that the slope to the south (fig. 3) would favor shovel operations, in that the line of contact between commercial ore and waste was about 45° from the horizontal. To the north (fig. 4), however, the line of contact is much steeper, approximately 70° from the horizontal, and here the ore is overlaid by the quartzite beds. To mine the deep ore to the north, it was apparent that much waste which extended to depths of several hundred feet in this section of the mine would have to be removed. Although the stripping ratios are excessive in sections of the deposit, the average ratio at present does not exceed the removal of 1 ton of waste for every 2 tons of ore.

The grade of the ore, generally speaking, is very even, and while the commercial values range from six-tenths of 1 per cent to over 2 per cent there is little variation in the run-of-mine average from month to month, owing to the distribution of many digging units throughout the operating area. Within the confines of the area stripped, three varieties of mineralized porphyry are exposed. One is a dark basic porphyry confined to the south end of the levels. To the north of this basic porphyry is a series of prominent faults striking to the southwest and dipping to the north. In the proximity of the fault zone, the porphyry is highly silicified and has a greater mineralization than that found in the basic porphyry to the south. North of the fault zone or silicified porphyry is a typical gray monzonite porphyry which is the most highly mineralized zone of the mine and is also largest in extent. Although large tonnages of 2/10 to 3/10 of 1 per cent material occur, the quantity of so-called "possible ore" containing between 3/10 and 6/10 of 1 per cent copper is relatively insignificant, in other words, the material excavated for the most part is definitely either ore or waste, which is fortunate in a mining sense for the operator; Of course there are exceptions to this rule, but in general this is the case.

METHODS OF PROSPECTING AND EXPLORATION

Prior to the advent of steam shovels at the Bingham property, many miles of underground work had been driven by the Boston Consolidated and other companies operating in the district, so that in the beginning most of the proved tonnage was developed by this means. After the organization of the Utah Copper Co., both diamond drills and churn drills were used in exploration. Unsatisfactory results were noted from the very few diamond-drill holes drilled in the porphyry, as there was such a wide variation between core and sludge samples that diamond-drill work was discontinued and prospecting continued solely by means of the churn drill. This prospecting has, with the exception of short intervals, been continued to the present day. Figure 5 shows the relative development of the ore body by underground workings and churn-drill holes.

Two types of churn drills are used at present: One, a modified standard rig for drilling holes to depths of 1,000 feet or less, and two standard oil rigs for drilling the deeper holes; all are operated by electric power. When any considerable depth is anticipated, the holes are started with a 26-inch bit and a 26-inch stovepipe casing is carried down 80 or 100 feet, depending upon the extent to which the ground caves. At this point a 23-inch casing is inserted and carried down until it is "frozen" in the hole by caving ground. Underreaming is often resorted to when the casing will not follow the bit. Smaller sizes of casing are inserted as required, and in some instances casings as small as 4 inches have been used. It became a rule early in this work not to run an open hole more than 50 feet in advance of the casing, to eliminate the danger of salting the bottom samples by cavings from the bore of the hole above, and to minimize the need of underreaming, consequent delays, and increases in drilling cost. It is thoroughly understood that an accurate sample is wanted, even at the sacrifice of depth of hole if necessary.

All churn drilling is done by contract on a footage plus labor and supply cost basis. Following is a cost statement of one of the recent holes completed:

Cost of drilling hole with Standard electric rig

Spudded in with 23-inch casing.
Elapsed time, 455 days.
Average progress per day, 3.23 feet.
Average drilling progress per day, 3.92 feet.
Total footage drilled, 1,468.8.

	Cost	Per foot
Moving and setting-up drill:		
Labor, power, and water supply	$1,667.08	$1.14
Actual drilling:		
Amount paid contractor plus labor and supplies	19,578.33	13.33
Casing hole:		
Labor and supplies	4,468.47	3.04
Sampling and assaying	4,367.55	2.98
Total	$30,081.43	$20.49

METHODS OF SAMPLING AND ESTIMATES OF TONNAGE

Sampling at the churn drills is under the supervision of the geological department. A sampler is assigned to each rig.

Samples are taken every 5 feet and the cuttings removed by a suction bailer. The

CHURN DRILL HOLE NO. 134 Cordinates S.973.75 W.2396.39 El.6708.0				
Casing Left in Hole	Casing Record	Adopted Assays CU	Character of Material	General Remarks & Minerals
23" Casing 70' with shoe 20" " 100' " " 15½" " 323' " " 12½" " 130' " " 200	23" casing 20" casing	Elev. 6698 0.88%	Silic. Porph. Silic. Porph Silic. Porph	8-14-28, Spudded in 23" Bit Pyrite, chalcopyrite, chalcocite, covellite, bornite, molybdenite Chalcopyrite, pyrite, chalcocite, covellite bornite, molybdenite Chalcopyrite, pyrite, bornite, chalcocite
400	8¼" casing	Elev. 6393	Silic. Porph.	Covellite, molybdenite Chalcopyrite, pyrite, chalcocite, bornite, covellite, molybdenite
600	15½" casing	1.16%	Silic. Porph.	Chalcopyrite, pyrite, chalcocite, molybdenite, bornite, covellite
800			Gray Porph.	Chalcopyrite, chalcocite, pyrite, covellite, bornite, molybdenite
1000	12½" casing	Elev. 5833	Dark Porph.	Chalcopyrite, chalcocite, pyrite, covellite bornite, molybdenite.
1200	10" casing	0.81% Elev. 5573 0.52% Elev. 5543	Limestone Dark Porph.	Altered, chalcocite, chalcopyrite, pyrite, covellite, bornite, molybdenite. Chalcopyrite, chalcocite, pyrite, covellite, bornite, molybdenite
1400	6¼" casing	0.78% Elev. 5468 0.54% Elev. 5423 0.80%	Limestone Mixed zone of limestone and porphyry Dark Porph.	Black, large amounts of chalcocite in limestone in porphyry primary minerals only, chiefly chalcopyrite, bornite, pyrite, unusal amount of bornite In limestone; chalcopyrite, chalcocite, covellite
		Elev. 5233	With limestone inclusions	In porphyry; chalcopyrite, bornite, pyrite, molybdenite 2-10-29 - Hole completed - Depth 1475'

Figure 6.- Condensed Churn Drill Log

content is discharged into a launder, passed over a screen 8 to 10 feet long, from which the large sizes are sent through a small gyratory crusher, then combined with the undersize, and the whole is passed into an inverted cone-shaped tank. Sludge in the tank is agitated for 20 minutes by a mechanical agitator that revolves near the bottom, aided by compressed air that enters through a 1-inch line near the bottom of the tank. From here the sludge passes to a cutter so constructed that three separate samples are obtained. One is sent to the mine assay office, one to the assay office at the mills, and the third and smallest sample is cut down to 5 pounds and placed in a 5-gallon wet sample can. This last sample is used as a part of a composite sample for every 100 feet drilled to serve as a check on the 5-foot samples; it is also used for making experimental flotation tests. Samples taken for immediate analysis are thoroughly dried on a large sheet-iron stove, care being taken not to burn or break down the sulphide. A 2-pound specimen is saved of every 25 feet of hole drilled in porphyry and of every 5 feet when the hole is near a porphyry quartzite contact. These specimens are examined by the geologist to determine the minerals contained and character of the rock.

Daily reports are made and a complete log of the hole is kept by the geological department (fig. 6). A record of the assay returns from the mine and mills is kept, and when a variation of over 0.05 per cent exists between the mine and mill assays, duplicate analyses are made of the pulp and averaged for the adopted assay.

Owing to the irregularity of the surface, holes were not drilled at the actual intersections of predetermined squares, but are drilled as nearly as possible at the corners of equilateral triangles. Where values are consistent, a spacing of 400 feet is considered safe, but as the limits of the ore body are approached, holes 200 feet apart have frequently been drilled.

In making the ore tonnage estimate, each level was treated as a separate mine; a plan map of each level was made showing the limits of the developed ore, underground workings, churn-drill holes perforating that particular level, and the ultimate location of the level. The specific gravity of the ore was determined, which gave a factor of 13 cubic feet per ton in place. The ore area was divided into blocks 100 feet square, with the height of the shovel bank in question taken as the depth of the block. Drill-hole assays within the segment of the hole between the top and bottom elevations of the bench were averaged, and the value was assigned to the 100-foot block perforated by the hole. The intervening blocks between drill holes were assigned assay values determined by proportioning the average of the intercepted drill-hole segments in accordance with their distances from the block in question. Where blocks were cut by the drifts and crosscuts of the old underground mine workings, the assays taken were also averaged for each block.

A detailed system of toe sampling is used at the mine to enable the operator to have a close check on the grade of ore loaded by each shovel. Samples are taken every 10 feet along the face of the level, following each shovel cut. The assay results are placed on a plan map showing the toe and edge of each level. New toe sample maps are made up every two weeks.

In the tonnage and grade calculations the toe assays were also averaged for each 100-foot block and compared with the assigned churn drill and underground mine assays in the same block. From this comparison a discount factor was arrived at for those levels where any decided difference was noted between toe assays and drill and underground mine assays, the toe assay being used as the basis for the discount. Differences in value were found to be confined largely to old stope areas, while the drill assays checked closely with the toe assays.

For estimating tonnage and grade of the ore located below the lowest level of the present working faces, namely 6,240 feet elevation, a plan map was made as of this elevation, showing all drill holes extending below this plane. Volumes were figured by triangular prisms bounded by drill holes, multiplying the area by the average depth of the hole and weighing the assays in accordance with the footage. Refinements of this method, using corrections for triangles that were not equilateral, were tried, but the differences did not warrant this procedure. The results obtained from the separate levels were combined with this lower calculation to give the total gross figure.

Preparatory to making detailed ore estimates, the cut-off between commercial ore and waste must be determined. In other words, a grade must be determined below which the material can not be mined and meet its mining and treatment costs and show a profit. To arrive at this cut-off grade. certain assumptions must of necessity be made, such as the selling price of copper, the estimated recovery in per cent of gross metal content, and the cost of producing a pound of copper, which includes all costs other than stripping. The stripping cost is kept separate for reasons that will develop later. Table 1 is set up to illustrate the method used to determine the point of cut-off:

Table 1.- Method of determining point of cut-off between ore and waste

(1) Per-cent copper	(2) Gross pounds per ton	(3) Assumed recovery, per cent	(4) Pounds recov-ered	(5) Assumed selling price per pound, cents	[1](6) Assumed cost per pound, less stripping, cents	(7) Partial net profit per pound, cents	(8) Loss per pound, cents
1.10	22	91	20.0	13.5	6.25	7.25	-
1.00	20	90	18.0	13.5	6.94	6.56	-
.90	18	89	16.0	13.5	7.81	5.69	-
.80	16	87	13.9	13.5	8.99	4.51	-
.70	14	85	11.9	13.5	10.50	3.00	-
.60	12	83	10.0	13.5	12.50	1.00	-
.50	10	80	8.0	13.5	15.62	-	2.12

Table 1.- Method of determining point of cut-off between ore and waste (Continued

(1) Per-cent Copper	(9) Stripp-ing cost per cubic yard	(10) [2]Ratio of waste to ore	(11) Proportionate stripping cost per pound this basis, cents	(12) (6+11) Total cost per pound, cents	(13) Profit per pound, cents	(14) Loss per pound,
1.10	$0.40	1/2 to 1	.50	6.75	6.75	-
1.00	.40	1/2 to 1	.56	7.50	6.00	-
.90	.40	1/2 to 1	.62	8.43	5.07	-
.80	.40	1/2 to 1	.72	9.71	3.79	-
.70	.40	1/2 to 1	.84	11.34	2.16	-
.60	.40	1/2 to 1	1.00	13.50	-	-
.50	.40	1/2 to 1	1.25	16.87	-	3.37

1 - Based on a cost of $1.25 per ton These figures can be varied to cover increasing costs due to increasing copper content, such as bullion freight, refining, selling, etc.

2 - Or one-fourth yd waste to 1 ton ore.

Under this set of conditions a copper content of .6 per cent would be the point of cut-off; any grade under this figure would be waste, and anything over should be classed as ore.

Other tables should be made with a variation of doubtful assumptions to assist the engineer in establishing a safe cut-off figure.

It is at this stage convenient to set up a table of grades showing the amount of stripping any given grade of ore will carry. This is usually worked up as follows:

If it costs 40 cents to waste a cubic yard of overburden weighing 2 tons, 1/2 ton will cost 10 cents. It then becomes necessary to determine what grade of ore will yield a return of 10 cents per ton under the given conditions, assuming the average recovery to be 85 per cent. The figure in this case is .0436 per cent copper, arrived at as follows:

.0436 per cent copper x 2,000 pounds = .872 pounds x 85 per cent recovery = .741 pounds;

.741 pounds x 13.5 cents equals 10 cents, the cost of moving 1/2 ton of stripping.

By adding this increment of grade (.044 per cent) to the ore, it will support the removal of an additional half ton of waste for each addition of the increment. One can then set up the following table:

Table 2.- Tonnage of stripping, plus all other costs, carried by 1 ton of ore of various grades

Grade of ore, per cent	Tonnage of stripping
0.6	0.5
.644	1
.688	1.5
.732	2
.776	2.5
.820	3

From the foregoing tables, graphs can be made from which it can be determined at a glance whether or not a certain block of material is ore or waste when the stripping ratio has been determined.

Dividends, of course, can not be paid on a grade of ore at or near the point of out-off; the average grade, therefore, must be well in excess of the cut-off grade, and no section of the ore body that will not pay its own way should be combined with higher grades for the purpose of increasing the reserve. Possible exceptions to this rule appear, of course, when a "horse" of waste or a small amount of low-grade material occurs that has to be removed in any case. These small quantities of waste may not be easily separated and may be milled at a loss (capacity permitting) which, however, will be smaller than the cost of removing the material as waste. In such cases, the low-grade tonnage is included in the ore reserve with its grade. The engineer's judgment will guide him (after he has made a complete analysis of the ore body) in rounding out an estimate where so many variables are concerned. It is well to remember that material which at the time of the estimate is waste, may come into the classification of ore by an increase in the price of copper, by an improvement in metallurgy, or by a lowering of costs with improved equipment.

CHOICE OF METHOD

A point that is never lost sight of is that the total cost of mining and stripping shall not exceed a reasonable underground cost. To determine this cost limit a series of trial sections was made up running normal to a tentative location of the stripping limits. On these sections detailed studies were made showing the ratio of ore to waste and the stripping limit for the particular section determined (see fig. 7 for a typical example of the problems involved). In this study it was necessary to determine the grade of the ore in each section and to ascertain just what amount of stripping could be moved and still show a commercial profit from the ore in question. Referring to the table of stripping ratios, it may be noted that under the costs and conditions upon which the table is based, ore having a grade of 0.82 per cent copper can carry a stripping ratio of 3 to 1 or of 1½ cubic yards of waste to each ton of ore. At 40 cents per cubic yard this means a stripping cost of 60 cents per ton of ore. In some sections the average grade of ore is in excess of 1 per cent copper and as far as grade is concerned could still show a profit for larger stripping ratios permitting the moving of the stripping limit to the increased ratio. But stripping

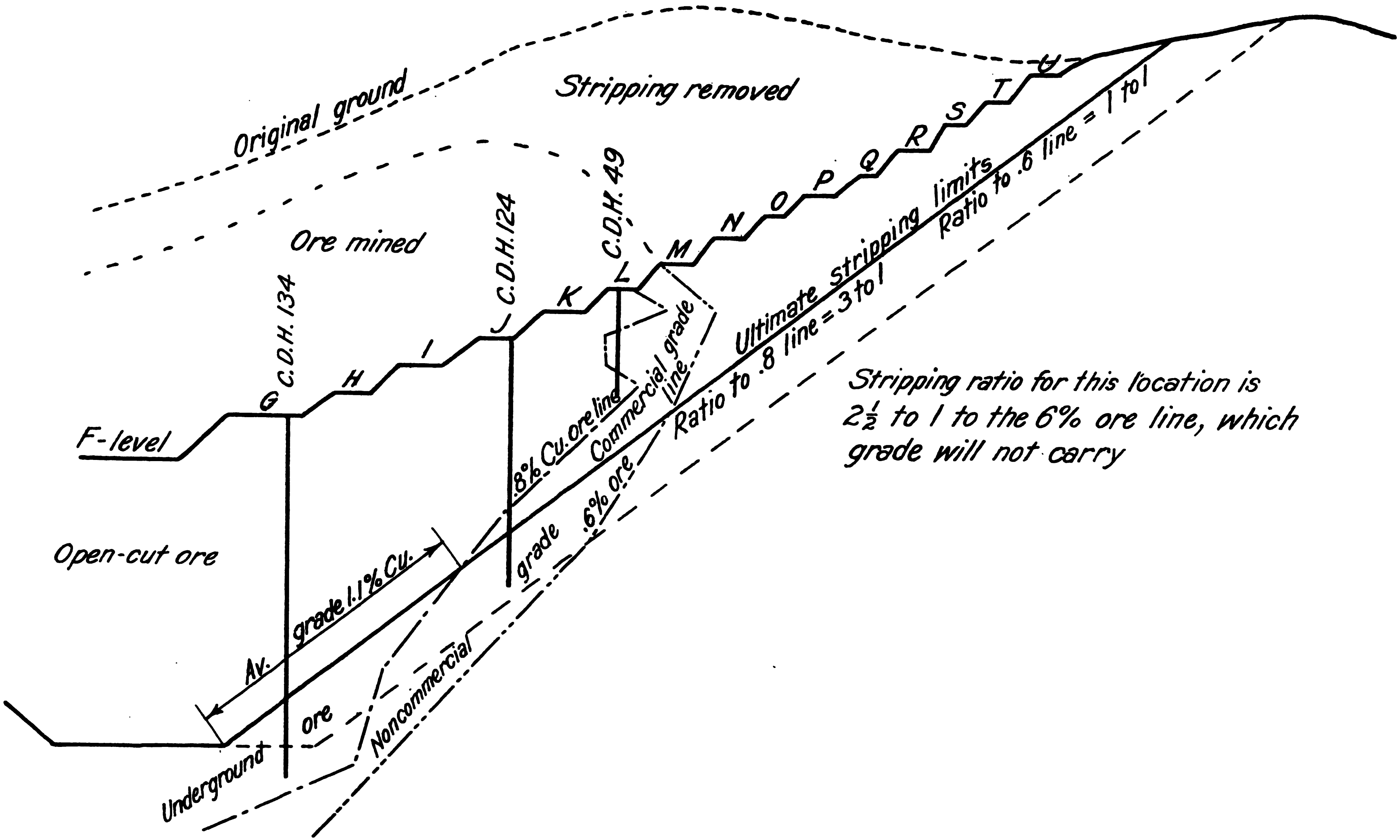

Figure 7. - Typical section showing method of fixing the ultimate stripping limits

costs in excess of 60 cents per ton under the heretofore mentioned conditions, together with other mining costs, exceed a fair underground mining cost; therefore the maximum stripping ratio should not be greater than 3 to 1 and ore outside such limits should be classed as underground ore. Referring again to the section, it may be noted that the .6 per cent ore line extends beyond the stripping limit, but such a grade is not considered profitable by underground mining and is therefore not included in ore reserve calculations. When located within stripping limits, this grade can be classed as commercial ore and included in the reserve.

The study was continued for each section; the ultimate location of stripping limits was then finally laid out, and it was found that of the 625,000,000 tons of ore reserve developed to date, approximately 580,000,000 could be removed by open-cut methods. Possibly some 40 years hence, after open-cut methods have ceased to be profitable, there will be a "mop-up" job to win the remaining tonnage by caving methods. This will entail considerable development work, and plans that are being made for shoveling below the present scene of operations are being laid out to tie in with the possible underground operations.

In addition to the above factors that determine the choice between open-cut and underground methods is the practical side involving the necessity for mass production of a low-grade ore to make it of maximum commercial value. To this must be added the value of flexibility of control of production. To illustrate: If occasion should demand an immediate increase in production to 60,000 tons from a mine ordinarily producing 50,000 tons of ore per day, practically all that is involved in open-cut work is to take two shovels working on stripping and place them on ore. In underground work, to increase the number of ore faces 10 per cent would present a serious problem.

Other issues involved that fix the limit to which open-cut operations can be carried are such factors as maximum degree of railroad curvatures, sufficient space for the efficient operation of power shovels, adjacent property rights, dump grounds for waste material, and above all the safe degree of over-all slope. Calculations of ore tonnages recoverable by open-cut methods and stripping are also dependent upon the slope.

The total ore removed to January 1, 1929, amounts to 175,007,974 tons, having an average grade of 1.21 per cent copper. During the same period 94,338,953 cubic yards of capping and low-grade material has been stripped and dumped in near-by gulches. This gives a stripping ratio to date of 1.1 tons of waste to 1 ton of ore, and is also the ratio being maintained at the present time. The final ratio is entirely dependent upon the ultimate over-all pit slope. Based upon a 40° slope this ratio will be approximately ½ to 1, but, if conditions make it necessary to use a much flatter slope, the ratio may be increased to equal amounts of waste and ore. These are general averages, as there are sections where the ratio reaches the maximum of 3 to 1, and it is in such places that slope is of paramount importance.

SLOPES

When it is considered that in one section of the Utah copper pit the question arose as to whether or not 38° instead of 40° should be used above a certain level and that the cost of removing this extra amount of waste would reach a sum of $2,000,000, it can readily be seen how vital a thing, at least to the Utah Copper Co., the ultimate over-all slope really is.

The individual bench slope - that is, the actual slope of one shovel face from the upper edge to the toe of the slope - is of less consideration. This is, of course, a function of the over-all slope, but, while an individual bench might stand at 60° from the horizontal, it is not to be concluded that a face 1,500 feet high will stand at the same angle of repose. When the material in any part of the ore body gradually declines in grade from ore to waste, there is always the question as to where the limit of excavation will ultimately be, and it therefore becomes necessary to maintain the shovel terraces on the pit face where, for the time being at least, operations are suspended. This shovel bench can be maintained at a minimum width to accommodate a shovel and loading track--say 30 feet. The slope we are most concerned with is the aggregato made up of the individual bench slopes plus the width of benches. In other words, the over-all slopes would be the angle from the horizontal from the top edge of the excavation to the bottom toe of the excavation, and it is this angle that the Utah Copper Co. has tentatively set at 40°.

There will always be local variations. At some points the slopes will doubtless take the angle of repose of broken material, say 35°. With others a slope of 50° may be safe. Geological conditions will enter into this. For instance, where stripping is being done against a face of quartzite that is dipping toward the operations at 30° from the horizontal, this slope will doubtless turn out to be the dip of the beds. On the contrary, where the beds are dipping into the bench, say from the shovels, a much steeper slope can be maintained.

MINING METHOD

When operations were first started, the initial point of attack was on the west side of the canyon. From the beginning of the first shovel cut, terraces were developed above this cut until there is now a shovel terrace for almost every letter in the alphabet beginning at the yard elevation with bench A and extending up the mountain side to bench W. Sublevels below bench A have also been cut which will be numbered 1, 2, etc., as the down ward excavation of the mine progresses. The actual bench heights vary from 40 to 70 feet as a maximum and bench widths from 30 to 450 feet, the average width being about 100 feet. From the experience of the Utah Copper Co. and under the operating conditions at this mine, a bench height of from 40 to 50 feet is deemed the most economical.

At the mine the elevation of the yard tracks (6,340 feet) practically represents the bottom of the mine as opened at present, and by far the greater part of the ore mined to date has come from above this elevation. The top edge of the open-cut is 1,500 feet vertically above the yard elevation and, since all of the ore from these heights above must come by rail to the assembly yards, a very difficult problem of switchbacks had to be solved. It is, where possible, always desirable to have two entrances to each of the shovel benches; to accomplish this, two sets of switchbacks were constructed, one on what is known

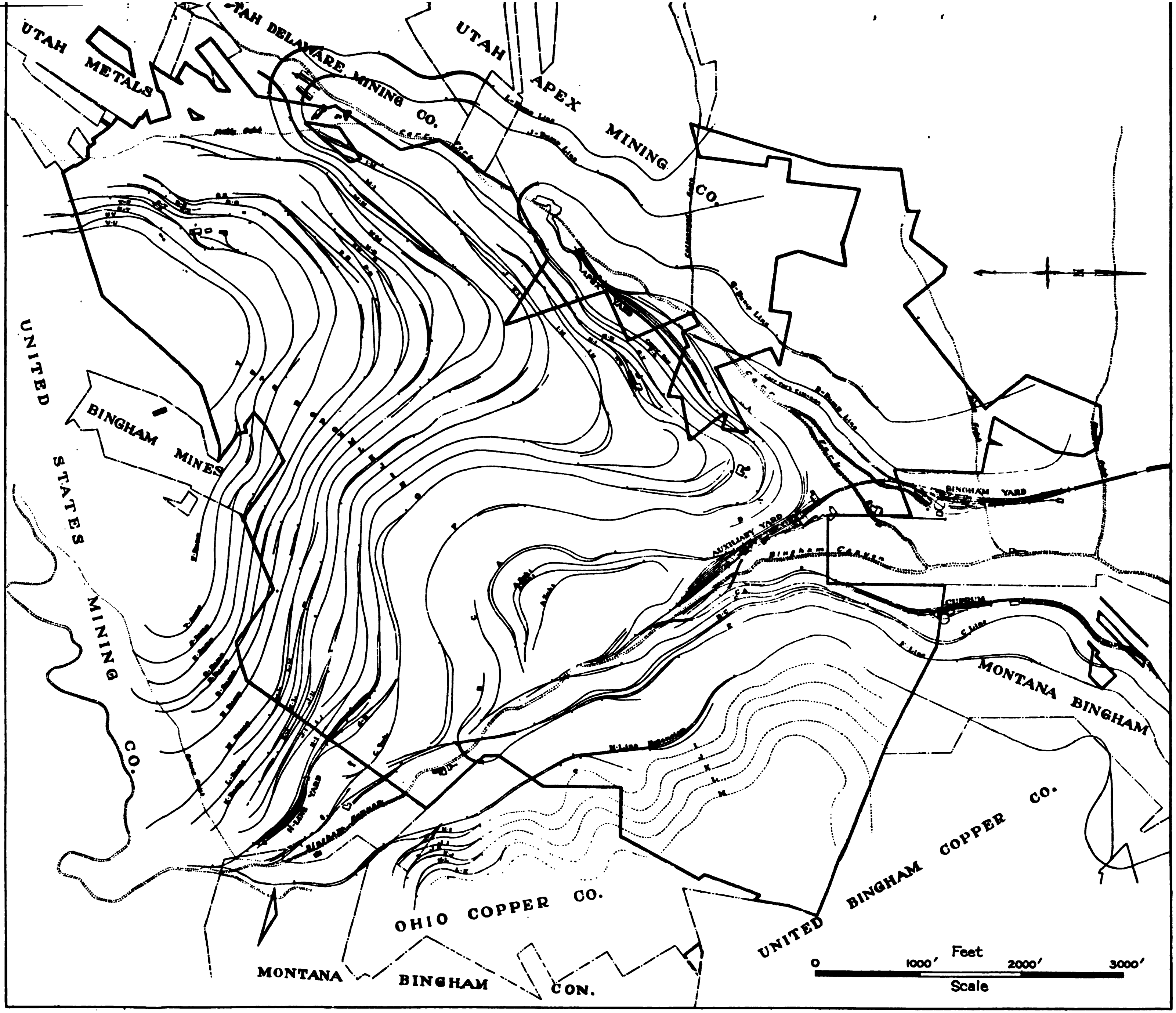

Figure 8. – Utah Copper Company Mine at Bingham Canyon, Utah

as the Carr Fork side of the mine and the other on the opposite or Copperfield side of the mine. The connecting switchbacks are located on maximum 4 per cent grades equated for curvature, and in locating these it was necessary to allow for a tail track of maximum train length. There are approximately 60 miles of trackage in the pits. The distance from the assembly yard to the most remote ore shovel is about 7 miles and requires about 1¼ hours to make a round trip. The average ore haul at the mine is 3 miles (6 miles round trip).

The shovel method in use is the usual power shovel procedure; that is, the ground is drilled and blasted well in advance of the shovel and the trains are loaded in a regular routine way. These trains are dispatched down over the switchbacks by flagmen.

At present the average daily production is 40,000 tons, but when occasion has demanded the production has been increased to 60,000 tons without any unusual stress. As far as shovel equipment is concerned, 90,000 tons a day could be loaded for short periods. In addition to this the present rate of stripping is 30,000 cubic yards per day.

Figure 8 gives a plan of the layout at the Utah Copper Co. mine at Bingham; Figure 3 is typical east-west cross section through the ore and overlying capping, and Figure 4 is a typical cross-section in a northwest direction.

By referring to these sections the terraced arrangement of shovel benches may be noted. These bench heights are not all the same for various reasons. The topography of the ground at the point of entry and the arrangement of switchbacks made it necessary to locate the approach at the most practical point, all things considered.

Shovel operations have been started on the east side of the mine across Bingham Canyon from the main workings. When operations have been completed the excavation will be 2,500 feet deep from the top-most edge to the bottom of the pit. The elongated bowl will be more than 8,000 feet long by 6,000 feet wide.

Waste tracks run out from the various waste levels to nearby gulches, and when the dumps are close to the shovels there is a waste disposal track for each level. Where the waste has to be transported greater distances, one of these disposal lines will serve about three benches, as the benches are connected by switchbacks before reaching the main waste line.

The subject of waste disposal again brings up the question of cut-off between ore and waste. Referring to Table 1, it may be noted that the commercial cut-off is .6 per cent copper under a stripping ratio of 1/2 to 1. When material having a value of .55 per cent copper is within the mining zone, the question arises as to whether it should be sent to the mills or to the waste dumps. Under the conditions upon which the aforementioned table is based, the cost of wasting this material is 20 cents per ton (40 cents per yard). If sent to the mills as ore, the cost would be $1.25 per ton and, as it has a metal content that would yield $1.18, there would be a loss of 7 cents per ton. The conclusion is, that to mine and treat this grade of ore will cost less than to waste it. This example is given to illustrate the point that when horses of waste are encountered within the ore zone, it is the policy of the company, capacity permitting, to mill a small amount of nonpaying material rather than to waste any commercial ore.

In the stripping operations at this mine the comparatively little material that is entirely barren of copper values is limited to the surface capping, which is from 50 to 200 feet thick. The quartzite (which is the country rock of the district), within the stripping limits usually carries values ranging from .2 to .5 per cent copper. The low-grade porphyry, which is also sent to the waste dumps, will average .4 per cent copper. In the early period of operations at Bingham, material up to 1 per cent copper was sent to the dumps, so that the older dumps contain considerable quantities of material that would now be classed as ore. Moreover, it has recently been demonstrated that the copper in such dumps can be very economically recovered.

Copper-precipitation plants have been constructed, and the copper-bearing waters from all the gulches in which dumps have been deposited are conducted to these plants. During the spring run-off, when water is abundant, as much as 50 tons of copper have been precipitated daily on de-tinned scrap. Recoveries are being held at about 98 per cent of the copper content in the water, so eventually the copper in the dumps will be recovered, limited, of course, to the extent to which the waters percolating through the dumps will take copper into solution.

In view of this, a particularly close check on shovel loadings in ore or waste need not be maintained. If a shovel is loading a train of ore cars and runs into small horses of waste (low-grade), it is all loaded and sent to the mills, as it obviously would be an expensive operation to leave it and later load it into waste cars. The same is true for a shovel loading waste cars; small amounts of milling ore are loaded and sent to the dumps, as in time at least a part of the copper in it will be recovered at the precipitation plants.

DRILLING AND BLASTING

When the first shovels were in operation during the summer of 1906, drilling was done by air drills mounted on tripods, and ordinary dynamite was used as the explosive. As the first stripping was done in the vicinity where caving to the surface had resulted from underground mining, it was found that much of the force of the explosion was lost in the broken material. To give a heaving effect to the charge, a portion of black blasting powder was added to the charge. Later, small coyote adits were driven into the toe of the banks for distances of 20 to 40 feet and charges of 10 to 25 tons of explosives were used in the coyote blasts. These large charges interfered with railroad operations because of caving the levels immediately above, so that this system of blasting was abandoned. Toe drilling was again adopted and is the method used at the present time.

The drilling equipment which is used for breaking the ground of the Utah Copper Co. is of very old design and consists of 3½-inch diameter reciprocating drills on tripods operated without the leg weights. Modern drills have been repeatedly tried, but this old-type drill which is used in drilling 22-foot holes still serves better than any that have been recently tried. The fact that the modern hammer machine using hollow drill steel requires water to keep the sludge from in front of the drill bit would make it necessary to maintain water lines on the levels, and this is impossible in winter weather at Bingham.

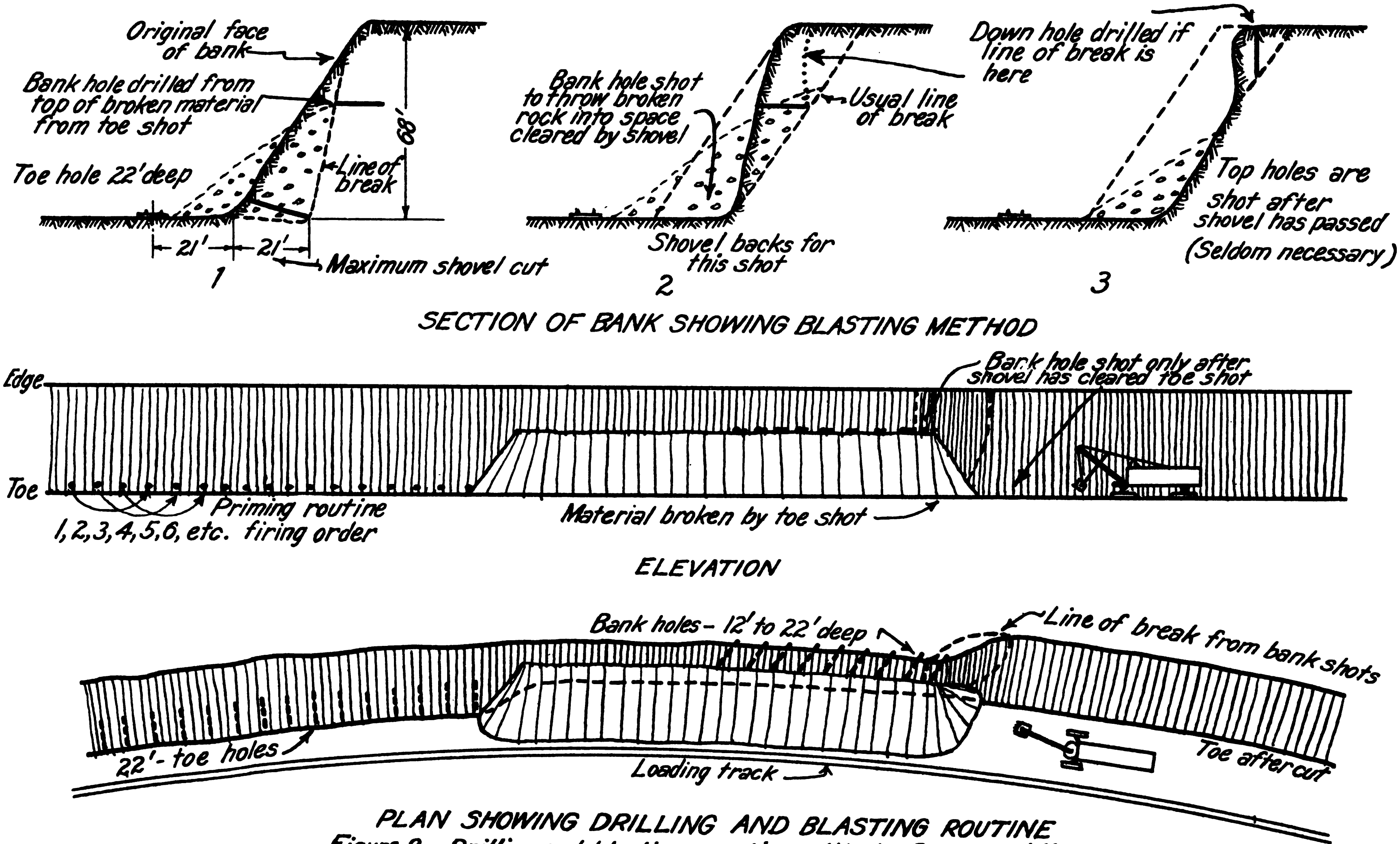

Figure 9.- Drilling and blasting practice Utah Copper Mine

Three compressors furnish 11,200 cubic feet of free air per minute, about 90 per cent of which is used for drilling. The air is supplied to the drills at 75 to 80 pounds pressure through a continuous air line going entirely around the workings and is delivered to each level through headers.

A hole is started with a 3-3/4-inch cross bit and finished with a 2¼-inch bit. The holes are then sprung until a chamber large enough to receive the blasting charge has been made. To chamber the holes usually requires four shots, using approximately 7, 15, 30, and 50 sticks of powder. Water is used for stemming these shots. The main powder charge varies between 150 and 250 pounds. A low-freezing ammonium nitrate powder with a rating of 60 per cent dynamite is used at present. The charge is put in the hole through a 1½-inch pipe and tamped in place. It is fired by means of a primer consisting of a stick of powder having attached to it a No. 6 cap and 6 feet of fuse.

A drilling crew consists of a machine man and two helpers. This crew also does the blasting and trims the bank after the blast when necessary. Two crews are maintained for each shovel, and two general powder foremen have direct charge of the work, one in the upper section of the mine and the other in the lower part. The general foreman tells the crews where to drill and also decides how much powder to use for each hole.

The usual procedure is first to blast the lower part of the bank well in advance of the shovel (fig. 9). The machine is set up on the usual talus 3 to 5 feet above grade and drills downward with an inclination varying from 5 to 15°, so that the hole will bottom at about grade. These holes are from 20 to 25 feet deep and are spaced about 15 feet apart; on the average 18 holes are drilled to the round. It is well at this point to call attention to the fact that the blasting cut should not exceed the reach of the shovel, as it is important that the toe of the cut shall be clean in order to facilitate the drilling of toe holes for the next blast. Another factor that limits the size of the blast is that the broken material must not be of such a quantity as to cover the loading track. A round is fired by three crews who work in tandem. No. 1 crew ignites the fuse and inserts the primer in the first hole, and then proceeds to the fourth hole. No. 2 crew follows the first crew by a few seconds, primes the second hole, and then goes to the fifth. No. 3 crew carries out the same procedure at the third hole then goes to the sixth, and so on until all holes have been primed (fig. 9). A 6-foot fuse gives them about four minutes to complete the operation which, for a round of 18 holes, is ample time. By firing the holes in succession the broken material from the first shot covers the second hole, serving as a blanket for that shot and preventing the material from being thrown over the loading track; and so on with the succeeding shots. After the toe of the bank has been blasted for a distance of 200 feet or more ahead of the shovel, the powder foreman selects those places that the toe shot did not break for the entire bank height in which to drill additional holes about three-fifths the distance up the bank. The holes are drilled horizontally into the bank but at an angle toward the shovel (fig. 9), and are loaded and shot just in advance of the shovel, so that the ore broken by the bank shot falls into the space that the shovel has just previously cleared of the broken ore from the toe blasting. This makes it necessary to back the shovel for the shot, which requires about 5 minutes. Occasionally it is necessary to drill "down-holes" at the top edge of the bank to break off projections of rock that do not break from the toe and bank shots. Bowlders too large to be handled by the shovel dippers are "dobe" blasted, or block-holed by jackhammers and blasted. After the shovel has quit loading at the end of the shift, the banks are trimmed of loosened rocks near the top to make safe working conditions at the toe of the bank. This is done by hand bars and so-called "pot-holing." Men are let over the top of the bank by ropes and pry down what loose rock they can with hand bars; the rest is shot down by digging small holes 12 to 15 inches deep and loading with a bundle of from 15 to 30 cartridges of powder. This shooting constitutes by far the greater bulk of secondary blasting at the mine.

An average of 45 feet of hole is drilled per drill shift, and 0.022 of a foot is drilled and 1/8-pound of explosive used per ton of material broken. Explosives cost about 1-1/2 cents per ton. Drilling and blasting costs--including labor, equipment, and maintenance of machines and pipe lines--added to the cost of explosives brings the total to a little less than 3 cents per ton. The powder is distributed as follows: 16 per cent is used for chambering, 69 per cent for the main blasts, and 15 per cent for secondary blasting. As has been stated, the ore is a soft porphyry extensively cut up by faults and fissures so that little difficulty is experienced in placing the blasting charges to insure finely-broken rock suitable for shovel-loading. Any difficulty in drilling because of hard ground is also eliminated. A set of drill steel is used for four or five 25-foot holes and then is replaced, not because of losing the cutting edge, but because of loss of gage. Secondary blasting is limited to "blocky" sections of the porphyry or quartzite.

It is the policy of the mine management to have an abundance of broken ground ready in advance of the shovels, as well-broken ground is of the first importance to high shovel efficiency.

LOADING

The first shovels placed in operation at Bingham were 60-ton steam shovels. Soon, however, the 90 to 100 ton railroad-type shovel, on tracks of 4-foot 8½-inch gage, was made the standard digging unit. In 1923 the first caterpillars were placed under these machines, consisting of one tractor under each jack-arm and one trailer tractor under the rear of the cab. This change is without a doubt the greatest advance made in shovel practice during the past 20 years. It meant that no tracks had to be maintained on which to operate the shovels, the short rail sections were dispensed with, and the pit crew was reduced from four or six men, depending on the condition of the shovel floor, to one man. More important than this saving in labor, perhaps, is the facility with which the shovel can by its own power move forward and backward at a moment's notice. It would be difficult to state just how many times a shovel has been saved from being buried by reason of the caterpillar equipment. With the shovels on rails it was necessary to block the machine in position so that it would not move backward while filling the dipper. This, of course, prevented any quick action on the part of shovel crew in getting their machine back out of danger. Every move-up of the machine would require from 10 to 15 minutes, whereas now no time is lost in this part of the procedure. Shovel-loading efficiencies have been advanced materially by the elimination of this delay. Average shovel loadings during 1923, when the old railroad-type steam shovels using 3-1/2-yard dippers were in use, was 2,350 tons of ore per shovel shift, as compared with an average of 3,966 tons for the year 1928 loaded with electric shovels mounted on caterpillars and using 4-1/2-yard dippers.

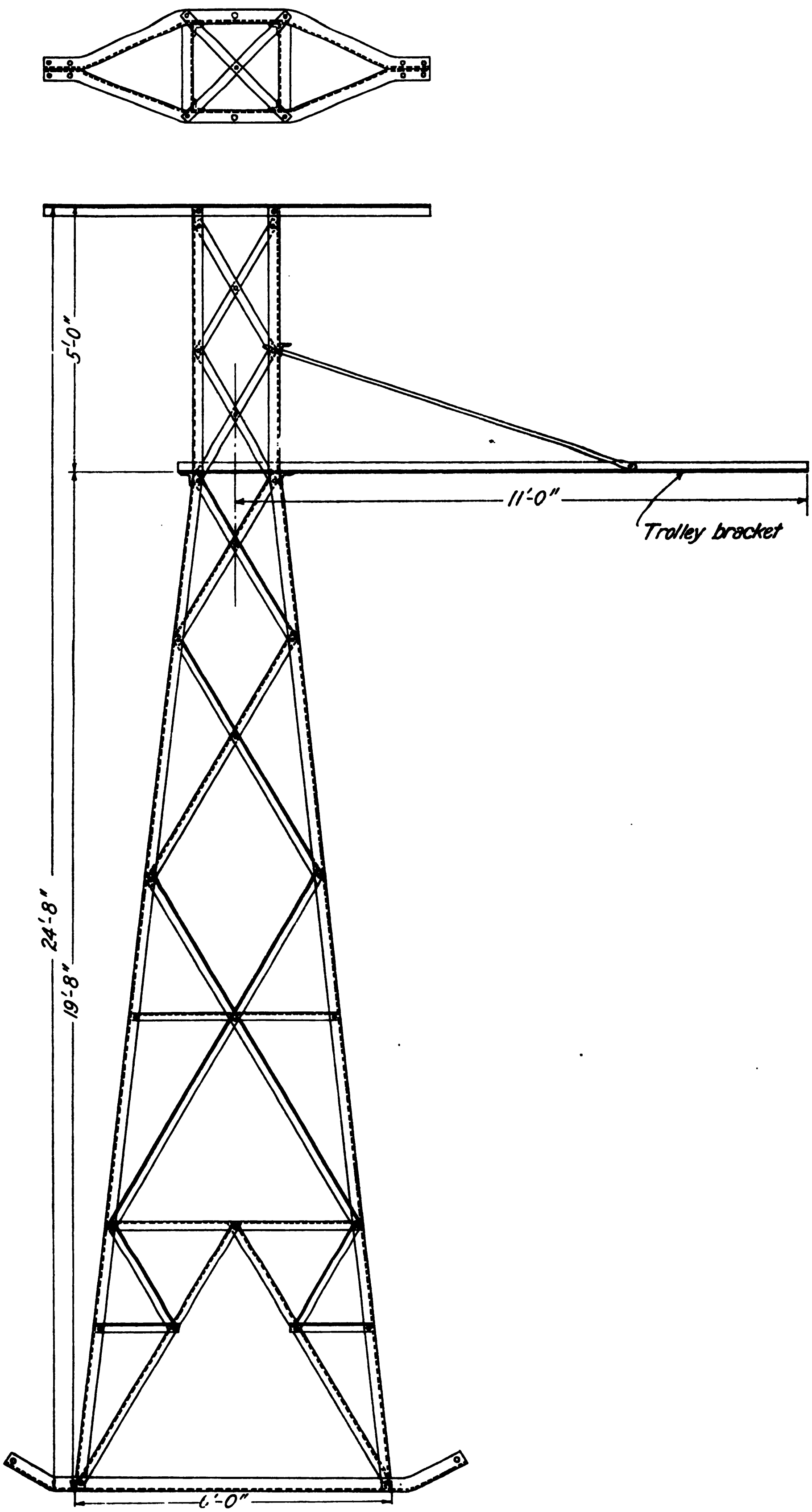

Figure 10.- Portable Transmission and Trolley Tower

In 1923 the first electric shovel was introduced in the mine, which was the same type shovel mounted on three caterpillar tractors as above described. All shovels are now electrified. The crew of these machines consists of the operator, the cranesman, one pitman, and one electrician who looks after two or three machines, depending on the proximity of one to the other. In the change from steam to electric power, the necessary equipment for changing 15 of the old machines was purchased from the manufacturer and installed on the shovels at the company's own shops in Bingham. In addition, eight new shovels were purchased. The savings anticipated as the result of this improvement have been more than realized; many items, such as the servicing of the machines with coal and water, delays due to boiler trouble, and low steam pressures, which, while of real moment, were difficult to set down in cents per ton, have been eliminated by electrification. Regardless of the weather, coal, water, or the vitality of operator, the machine is ready at full operating pressure the moment the switch is closed.

There are 23 electric shovels in service at the mine; 8 are of the alternating-current and 15 of the direct-current type. The alternating-current shovels have rheostatic control and the direct-current shovels use the Ward-Leonard control. The booms are 30 feet long and swing through an arc of 190°, which makes it possible for the shovel to make a 20-foot cut across the level. The dippers have a capacity of 4-1/2 cubic yards.

Electric power for the Utah Copper Co. operations at both the mine and mills is purchased from the local power company and is transformed to 44,000 volts at the company's central station at Magma. Two three-phase lines of No. 2 copper wire, 14 miles long, extend from Magma to Bingham, where at a central switch rack they split into a two-circuit line. These feed lines supply two outdoor substations, each of which contains six 400-k.v.a. transformers where the tension is reduced to 5,500 volts for shovel operations. The power from each substation is carried over two single-circuit lines, mounted on wood poles, which encircle the entire mining area. These lines are built close to the outlets of the different mine levels, and with this system of distribution it is possible to feed any level from any one of the four secondary trunk lines. The reason for such flexibility in the secondary distribution system is to provide insurance against delays, as continuity of power supply is of utmost importance to uninterrupted shovel performance.

The problem of getting power from the distribution lines to the shovels was a difficult one because of the danger of blasting the lines, and further because such power lines must be portable. Rubber-covered cable insulated for 7,500 volts was tried, but was supplanted by light, portable steel towers mounted on skids (fig. 10). These towers are approximately 25 feet high and 6 feet square at the base. They carry three No. 2 stranded copper wires in average spans of 125 feet. Each shovel receives power from this portable line through a connecting 500-foot length of four-conductor trail cable, three of these conductors being for power and the fourth for ground. This cable is wound on a reel mounted on the rear of each shovel and permits moving the shovel a distance of 400 feet before the connection must be changed.

The general plan for loading ore at the mine is to work full capacity during the daylight shift. At the end of the day shift the amount of ore loaded is computed, and enough shovels are detailed for loading ore on the night shift to make up the quantity required for that particular day. The night shift is also utilized for maintaining good working conditions at the waste dumps. When one of the dumps in a series is lagging behind the others, stripping is removed on the corresponding mine level at night and added to this dump.

A shovel will start making its cut at one end of a bench and will proceed across the bench to the other end, loading the material whether it be ore or waste. When it reaches the end of its travel, the operation is repeated. As the different shovels move across their respective benches, their progress is so arranged that one shovel will not be directly above another.

Usually ten to twelve 80-ton ore cars are brought in and spotted for loading, the locomotive remaining coupled. When the train is loaded and enroute to the yard, another locomotive will bring in the next train, and the process of loading continues. The procedure is the same in loading waste. The average capacity of the shovels is about 600 tons per hour, and an ore train with a capacity of from 800 to 960 tons will require from 1½ to 1¼ hours to load. Waste trains consist of from three to seven 30-yard cars, depending on the proximity of the waste dumps.

The following table gives the performance of the shovels for 1928:

Table 3.- Performance of Shovels

Type of shovel	Material	Shifts worked	Loaded: Tons	Loaded: Tons per shift
A.c.	Ore	929.87	3,613,720	3886
A.c.	Waste	1651.76	6,908,460	3692
	Total	2581.63	9,712,180	3762
D.c.	Ore	3244.74	12,942,350	3989
D.c.	Waste	2228.62	8,301,710	3725
	Total	5473.36	21,244,060	3881
A.c.	Ore	929.87	3,613,720	3886
D.c.	Ore	3244.74	12,942,350	3989
	Total	4174.61	16,556,070	3966
A.c.	Waste	1651.76	6,098,460	3692
D.c.	Waste	2228.62	8,301,710	3725
	Total	3880.38	14,400,170	3711
	Grand Total and Average	8054.99	30,956,240	3843

The table shows that both the alternating-current and the direct-current shovels loaded more per shift in ore than in waste, the average of the two being about 7 per cent more. It also shows a slightly higher performance of the direct-current over the alternating-current shovels. During the good-weather months the shovels loaded about 7½ per cent more than the average for the year. In 1928, from May to October, inclusive, the alternating-current shovels averaged 4,031 tons per shift as against 3,762 tons, the average for the year, while during the same months the direct-current daily average was 4,097 tons as compared with 3,881. Loadings as high as 7,000 tons have been made in a shovel shift.

The direct-current shovel will load 25 per cent more material than the alternating-current shovel with the same power, as shown by the following data for 1928. The cost for power was 15 per cent of the total cost of shovel operation.

	Alternating-current shovels	Direct-current shovels	Totals
Kilowatt-hours	2,380,041	4,202,539	6,582,580
Tons loaded	9,712,180	21,244,060	30,956,240
Tons per kilowatt-hour	4.081	5.055	4.703

During 1928 the shovels were actually loading 82.1 per cent of the total time they were on shift. The loss of time from electrical trouble is the smallest of all of the losses, and the most time lost is due to switching ore or waste cars. This loss of time is inherent in the method and could be avoided only by having passing tracks extending the full length of the levels so that an empty train could be spotted immediately upon the departure of the loaded train. It is, however, impracticable to have more than one track across a level.

A complete description of the shovel electrification appears in the September, 1927, issue of the Mining Congress Journal in an article written by C. W. and R. J. Corfield, electrical engineers for the Utah Copper Co.

TRANSPORTATION

Transportation equipment has shown marked change in size since the beginning of the Utah Copper Co. Twenty-ton locomotives, 6 cubic yard dump cars, and 30-pound rails have long since been abandoned. Prior to the recent electrification of the haulage at Bingham the steam locomotive weighed 85 tons and was modern in all respects, even to superheaters, etc.

The success in electrifying the shovels served to speed up the electrification of mine haulage, so that at this writing forty-one 75-ton electric locomotives are in service that have replaced a like number of steam units. The question as to whether or not the storage-battery type of locomotive should be used or whether a trolley should be strung along all the operating faces for a trolley-type locomotive was a matter of long consideration. Two experimental units were purchased, one a Deisel-electric 60-ton engine, and the other a combination trolley-storage battery locomotive. The result has been an almost exclusive use of the trolley type. For emergency purposes, seven locomotives of the combination type were selected which can be operated independently of the trolley. All locomotives are so constructed as to permit ballasting up to 90-tons. They are of the articulated type, and are capable of hauling 12 empty ore cars, each weighing 21 tons, up the 4 per cent switchbacks at 12 miles per hour.

Due to the complexity of the trackage system, it was necessary to provide for different methods of collecting power from the trolley, as wires can not always be placed over the center line of the track. Because of this the locomotives are equipped with a conventional main pantograph, two side-arm collectors and a cable reel containing 2,000 feet of cable. The pantograph and side-arm collectors are air-operated.

To date, approximately 40 miles of trolley have been erected, standard catenary construction being used for trolley line over permanent tracks and direct suspension elsewhere. Due to the necessity of operating spreaders to clear snow from the tracks, all poles are placed on the bank side of the track. On the working levels the steel towers used for the feeder lines to the shovels are also used to support the trolley, a standard bracket arm being attached to the tower for this purpose (fig. 10). These towers must be moved with each track change. Where dump tracks cross large fills a special type of construction is required because the dumps are continually settling. In such places the trolley support is made a part of the track to maintain a fixed clearance between rail and trolley wire.

In line with the company's policy of continually improving operating equipment, waste cars have been gradually increased in size until the 30-cubic-yard unit is now standard. In the matter of ore transportation the capacity of the cars has been 80 tons, but the new equipment recently put into service has been increased to a capacity of 90 tons on a running gear of 100-ton capacity. As ore tonnages have increased, the 65-pound rails first used for the mine track system have been replaced by 90-pound rails on the switchbacks, and the 65-pound rails thus released are used on the shovel levels and dump tracks. As replacements and new construction on the levels are made necessary, 90-pound rails are used.

Each shovel, whether loading waste or ore, is served by two locomotives. With two exceptions, all the switchbacks are single track, and the movement of the trains on them is controlled by flagmen placed at advantageous points. During cloudy or snowy weather, when it is impossible to employ visual signals, a telephone system is used which has been installed exclusively for this purpose. Whenever a train is ready to leave a bench the engineer asks for a signal by a whistle. If permission to leave is given by the flagman in charge of that section, he proceeds to the next flagman where he again has to whistle for signals. The mine locomotives take the ore cars to a main assembly yard where they are made up in 50-car trains and hauled to the mills by 320-ton Mallet locomotives.

In disposing of stripping in the gulches, the first dumping is at the end of the gulch farthest from the shovel bench. As this part of the dump is filled, the waste is dumped successively closer to the shovel bench until the entire length of the dump for that position of the track has been filled. The dump gang then levels off the dump and shifts the track to the edge. One dump gang, consisting of a boss and approximately 20 men, takes care of more than one dump where each dump is serving but one shovel. Where trains from two or more shovels run to one dump and dumps are high, the entire services of one gang may be required for the one dump.

Track shifting on the shovel levels is done by Petersen Track Shifters, of which there are nine in service. One of these machines requires a crew of six men to move the track. Three to five track gangs of 20 men each are also used in this work to line-up the track, etc. Track shifting on the dumps is done by track gangs using lining bars. Track shifters are used on the dumps only where the settling of the dump necessitates raising the track to grade.

DRAINAGE

Drainage at the mine is for the present a comparatively simple problem and requires no pumping. All mine levels are located on the side of the mountain, with the exception of the lowest level at the bottom of the pit. Water that drains into this level from the upper parts of the workings is passed through a churn drill hole in the pit to the Mascotte Tunnel which extends through the mountain on the east side of Bingham Canyon to Salt Lake Valley. Water from the higher levels is taken care of by the drainage system used for the town of Bingham, situated in the bottom of the canyon.

PLANT

Completely equipped machine, drill, blacksmith, carpenter, locomotive, and shovel repair shops are maintained at the mine and are located at the assembly yards. A mine warehouse is also located at the main assembly yard and supplies are delivered to different parts of the operation by a tramp engine and flat car. Water supply tanks are located at points of vantage outside the mine, from which water lines are run to all parts of the operation. A new compressor plant has been recently installed because the old plant was located too close to the mine and within the stripping limits. Air lines and power lines have been previously described. The water supply for the mine is taken from Middle Canyon at an elevation high enough to make possible distribution by gravity. No pumping is necessary.

PER CENT EXTRACTION

It can safely be said that the ore extracted from the Utah Copper mine to date is 100 per cent of the total in the area excavated. Some engineers might question this, but with the erecting of precipitation plants, copper existing in the waste dumps is being recovered. It has been said at the mine since the installation of these plants, that "We have no waste, but low and lower grade ore."

As stated previously, about 85 per cent of the total ore will be recovered by shovels and the remaining 15 per cent will be mined by underground methods which will probably not recover more than 90 per cent of it.

WAGES, CONTRACTS, AND BONUS SYSTEM EMPLOYED

The Utah Copper Co. mine is operated on a straight wage and salary basis, and no bonuses of any kind are paid. Isolated jobs are often done by contract, and the construction of trestles, houses, etc., is almost invariably given to the lowest bidder on a contract basis. With a production of 60,000 tons of ore daily, approximately 2,000 men are employed exclusive of railroad and mill hands.

SAFETY METHODS AND FIRST-AID ORGANIZATION AND TRAINING

The purpose of the safety department may be summed up under three headings:

1. Investigation of all accidents

All accidents, no matter how trivial, are thoroughly investigated and the causes determined so that the necessary changes in equipment, methods, and placing of men can be made to prevent their reoccurrence.

2. Distribution of safety propaganda

Safety educational activities include the maintaining of 52 widely distributed Bulletin boards upon which bulletins are changed weekly, safety campaigns, no-accident periods, etc., 100 per cent first-aid training, employees' safety publications, etc.

3. Safety inspections

Regular inspections are made of all equipment, buildings, methods of doing work, working places, etc., by a corps of safety inspectors.

The hill is divided into three sections, and a safety inspector assigned to each section. The inspector is responsible to the safety engineer and investigates all accidents that occur in his section. He also submits a written weekly inspection report to the safety engineer covering all conditions in his section.

Semimonthly meetings of all department heads are held in the superintendent's office, the superintendent acting as chairman. The report and recommendations of the safety engineer, the accidents which occurred during the previous two weeks and any other matters pertaining to safety and welfare are discussed at this meeting.

A monthly meeting of all foremen and bosses is presided over by the safety engineer. Although all accidents of the previous month are discussed and some first-aid training is given, the chief purpose of this meeting is to create good fellowship and morale among the foremen.

The following tabulation shows the extent to which accidents have been reduced at the mine by the foregoing program:

	1919	1920	1921	1922	1923	1924	1925	1926	1927	1928
Frequency rate										
Number of accidents per 1,000 men per month	23.6	22.3	16.5	29.1	19.8	20.5	18.0	18.5	6.0	3.7
Severity rate										
Days lost per 1,000 man hours	10.1	9.8	1.16	10.1	10.5	7.4	6.1	8.7	3.5	2.9

FORM OF ADMINISTRATIVE ORGANIZATION

The administrative and mine organization of the Utah Copper Co. is shown in detail in the accompanying organization chart (fig. 11).

Table 4.- Summary of Costs

Utah Copper Co. Mine Year of 1928
Cubic yards waste loaded during period - 7,220,034

Operating Cost per cubic yard of stripping

	(1)	(2)	(3)	(4)	(5)	(6)
Operating labor	$.0207	$.0128	$.0238	$.0565	-	-
Air	.0033	-	-	-	-	-
Explosives	.0337	-	-	-	-	-
Other operating supplies	.0006	.0007	.0006	.0142	-	-
Repair labor and supplies	.0021	.0197	.0226	-	-	-
Power	-	.0064	-	-	-	-
Bench transmission lines	-	.0020	-	-	-	-
Fuel	-	-	.0172	-	-	-
Dump cars	-	-	.0103	-	-	-
Supervision	-	-	-	-	.0132	-
General	-	-	-	-	.0133	-
Totals	$.0604	$.0416	$.0745	$.0707	$.0265	$.2737

Dry tons ore loaded during period: 16,558,500.

Direct operating cost per dry ton of ore

	(1) Drilling and blasting	(2) Loading	(3) Transportation	(4) Track maintenance	(5) Other mine charges	(6) Total
Operating labor	$.0094	$.0056	$.0104	$.0124	-	-
Air	.0015	-	-	-	-	-
Explosives	.0151	-	-	-	-	-
Other operating supplies	.0003	.0003	.0003	.0045	-	-
Repair labor and supplies	.0009	.0090	.0097	-	-	-
Power	-	.0031	.0003	-	-	-
Bench transmission lines	-	.0008	-	-	-	-
Fuel	-	-	.0074	-	-	-
Yard and train	-	-	.0080	-	-	-
Ore cars	-	-	.0006	-	-	-
Supervision	-	-	-	-	$.0058	-
Churn drilling	-	-	-	-	.0058	-
General	-	-	-	-	.0061	-
Totals	$.0272	$.0188	$.0367	$.0169	$.0177	$.1173

Table 5.- Summary of costs in units of labor, power, and supplies

Utah Copper Company Mine — Period - October, 1929

Material loaded during period:

Total material, cubic yard - 1,343,579
Ore, dry tons .... - 1,249,500
Waste, cubic yard .... - 741,990

	Stripping, per cubic yard	Mining, per dry ton	Total, units per ton ore
A - Labor (in man hours):			
Drilling and blasting	.032	.014	.033
Loading	.028	.011	.028
Haulage	.057	.024	.058
Track maintenance	.122	.031	.104
Repairs	.084	.025	.074
Miscellaneous	.004	.003	.005
Supervision	.024	.010	.025
Total labor	.351	.118	.327
Average yard or tons per man per shift	22.7	67.9	24.5
Labor, per cent of total cost[1]	63.6	56.9	60.3
B - Power and supplies:			
Explosives (lbs.)[2]	.241	.105	.248
Total power (kw.h.)-			
1. Shovels	.42	.20	.46
2. Locomotives	.50	.36	.66
3. Air compressors	.53	.23	.54
4. Shops	-	-	.04
5. Lighting and miscellaneous	-	-	.14
Total power	1.45	.79	1.84
Fuel (tons of coal)	.003	-	.002
Other supplies in per cent of total power and supplies[1]	58.0	49.5	53.8
Power and supplies, per cent of total cost[1]	36.4	43.1	39.7
C - Percent of total cost	100.0	100.0	100.0

1 - Based on ten months of 1929

2 - Blasting powder: Ammonium nitrate, permissible type, low freezing, rated equivalent to 60 per cent dynamite

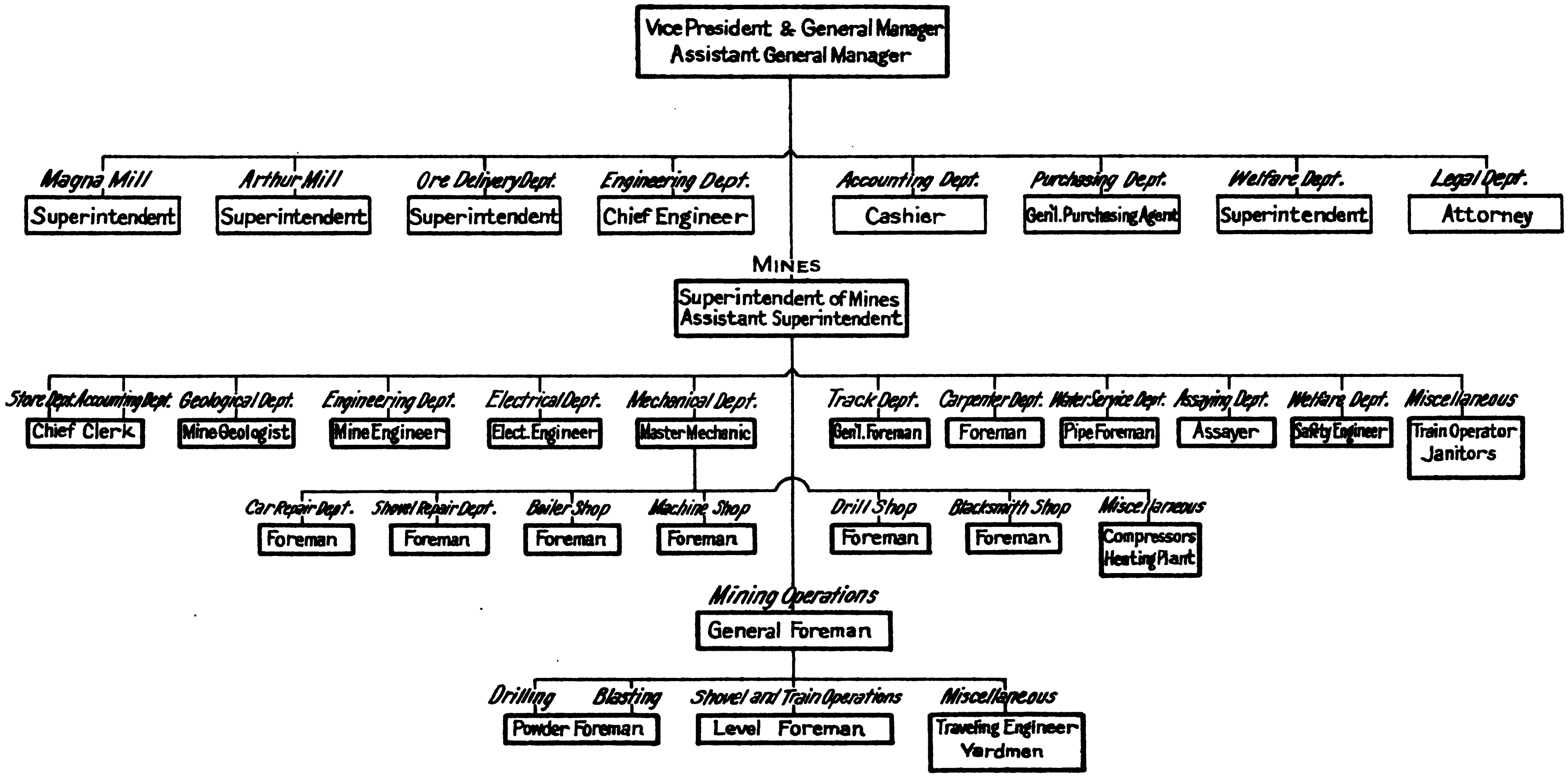

Figure 11.- Organization Chart