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PROSPECTING AND MINING OF COPPER ORE AT SANTA RITA, NEW MEXICO

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PROSPECTING AND MINING OF COPPER ORE AT SANTA RITA, N. MEX.

By Donald F. MacDonald and Charles Enzian.

INTRODUCTION.

It is a far cry from the small and uncertain mining efforts of 50 years ago to the splendidly equipped and solidly financed organizations that now handle thousands of tons of ore per day. The old mining was largely speculative; the new is practically on a manufacturing basis. In the old days of mining the aim was, chiefly, to find comparatively rich ore; modern practice looks more toward efficiency in mining and milling, so that vast tonnages of low-grade material may be worked at a profit. The old operators had faith and hoped for good returns; the new ascertain with a fair degree of precision what the available tonnage is, what the cost of mining and treating the ore will be, the time necessary to exhaust the deposit, the capital required, and the net profits to be made. The old mining was financed on a highly speculative basis, and the value of mining stock was largely controlled by the imaginations of the promoters and of the individual holders. In the new régime the financing of mines is often as conservative as the financing of manufactures or railroads, and the value of stock, quoted on the leading exchanges, is controlled by the carefully estimated profits in sight. Of course, with every legitimate industrial enterprise go certain business risks, and mining is not exempted from these. Then, too, valueless mining as well as valueless industrial stocks are sometimes offered for sale by the intentionally dishonest, or the ignorant. However, the prospective purchaser of a mine nowadays, if he is willing to pay for the services of reputable mining engineers, can obtain information on which to base an opinion as to the quality of his investment.

The relatively recent development of cheap and efficient methods of mining and of ore dressing has given value to low-grade ore bodies that a few years ago were almost valueless. The iron mines of northern Minnesota have been fruitful in developing efficiency in mining methods, notably the open-cut methods in which steam shovels strip off the overburden and then load the ore directly into railroad trains for transport to the furnace. In 1907 open-cut mining was applied to the low-grade copper ores of Bingham, Utah, with splendid results, and since then it has won a most important place in the mining of large masses of low-grade ores in several other mining regions, especially at Santa Rita, N. Mex.

Efficiency in mining as in other industries implies attention to detail, and the larger the scale of operations the more important may details become. For this reason the accompanying study of the methods used at Santa Rita will, it is believed, be of help to all engineers who have occasion to handle large masses of material.

ACKNOWLEDGMENTS.

Any merit that this report may have must be largely attributed to the splendid opportunity for study afforded the authors by the Chino Copper Co. To the manager, Mr. John M. Sully, the authors are deeply indebted for kindly interest and every facility to investigate the methods in use at the mines and mill of the company. For many courtesies and much help they are also under obligations to Horace Moses, superintendent; L. E. Foster, assistant superintendent; M. J. McGrath, chief engineer; Harry Thorne, explosives foreman, and many other officers of the Chino company's mine as well as to W. H. Janney, superintendent, and W. T. MacDonald, assistant superintendent, of the Hurley mill. Nearly two months were spent by the authors in field work at Santa Rita in the summer of 1914.

The outline of the geologic conditions that follows is largely from notes furnished by Spencer[a] and by Paige;[b] others[c] have also furnished valuable data bearing on the mining conditions of the region.

LOCATION AND GENERAL GEOGRAPHY OF SANTA RITA DISTRICT.

Santa Rita is on the mountain-bearing plateau of southwestern New Mexico. It is nearly 150 miles northwest of El Paso, Tex., and about 50 miles in the same general direction from Deming, N. Mex. It lies about a dozen miles east of Silver City, N. Mex., and is on Santa Rita Creek, not far from where the latter has its rise in the southeastern foothills of the Black Range. A branch of the Santa Fe Railroad connects Deming, N. Mex., on the Southern Pacific main line, with Silver City. From Whitewater, N. Mex., on this branch line, a subbranch leads to Santa Rita and to Fierro, N. Mex. (See maps, Pls. I and II.)

Santa Rita is in the central mining district of Grant County. In its vicinity the plateau is about 6,300 feet high, but some of the neighboring peaks reach altitudes of nearly 8,000 feet.

The climate is dry, bracing, and sunny. Even in the bright hot summer months the nights are cool. In winter, clear, cool weather, with occasional snow, adds to the stimulating effect of the climate.

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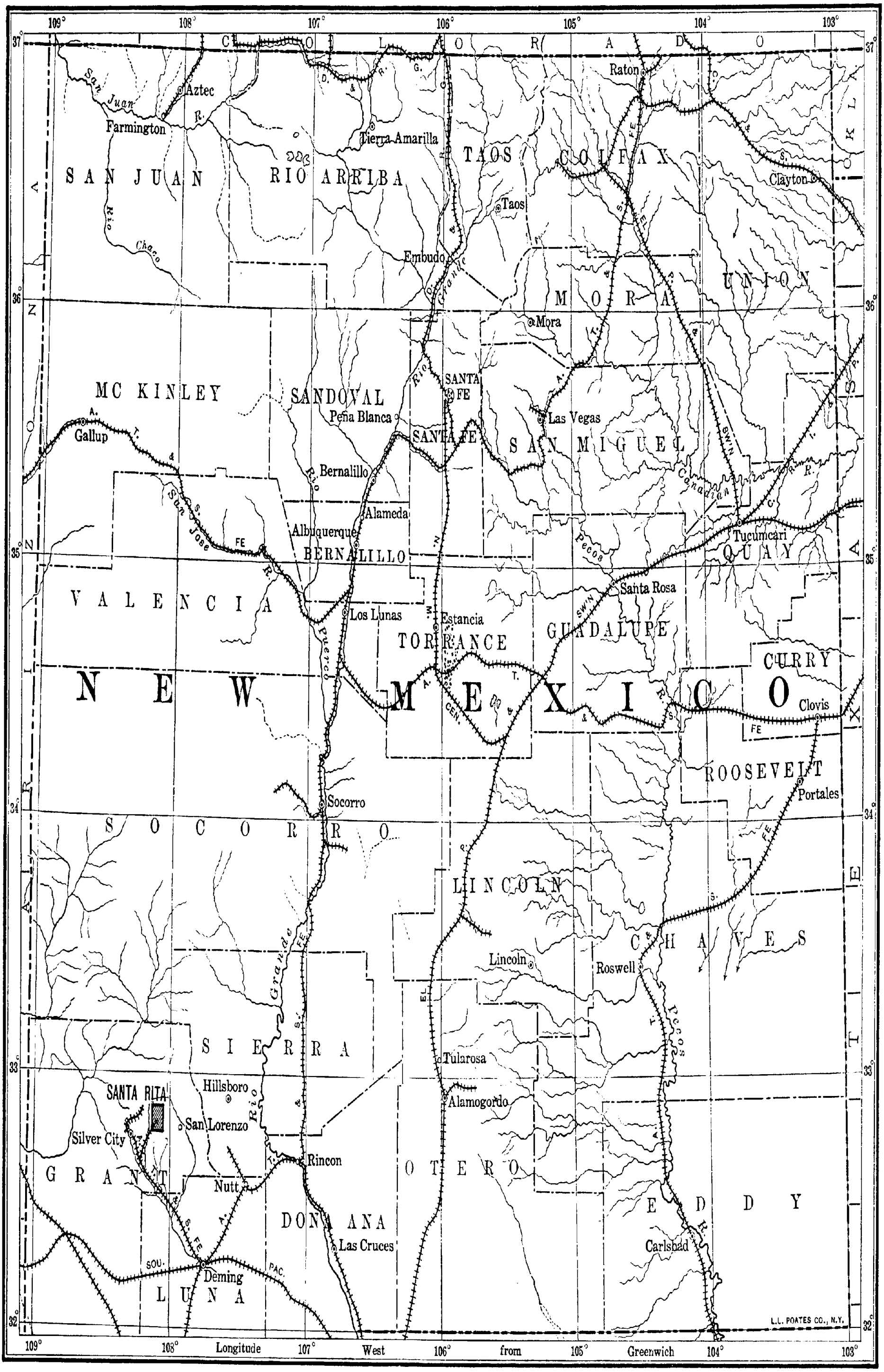
[a] Spencer, A. C., A detailed report on the geology of Santa Rita, in course of publication by the United States Geological Survey.

[b] Paige, Sidney, Silver City folio (No. 199), Geol. Atlas U. S., U. S. Geol. Survey, 1916, 20 pp.

[c] Lindgren, W., Graton, L. C., and Gordon, C. H., The ore deposits of New Mexico: U. S. Geol. Survey Prof. Paper 68, 1910, 361 pp.



MAP OF PART OF NEW MEXICO, SHOWING REGION AROUND SANTA RITA.

The following table gives the maximum and minimum temperatures and monthly evaporation for the year 1914. The monthly mean of the daily maximum and minimum temperatures is given under "mean maximum" and "mean minimum."

Data on temperature and precipitation at Santa Rita, N. Mex., during 1914, by months.

Month.	Temperature.[a]				Evaporation.	Precipitation.[b]		
	Maximum.	Minimum.	Mean maximum.	Mean minimum.	During month.	Total.	During month.	Total.
1914.	° *F.*	° *F.*	° *F.*	° *F.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*
January	64	14	52.27	28.15	4.329	4.329	0.58	0.58
February	62	22	51.39	28.95	4.266	8.595	0.43	1.01
March	70	24	57.63	32.66	7.448	16.043	0.66	1.67
April	78	30	66.88	41.18	11.507	27.550	0.02	1.69
May	84	31	74.23	48.27	12.416	39.966	1.00	2.69
June	91	48	81.82	57.50	12.253	52.219	1.51	4.20
July	85	52	78.11	58.76	7.612	59.831	7.51	11.71
August	88	53	80.98	58.66	9.151	68.982	2.98	14.69
September	83	46	76.77	54.12	8.214	77.196	0.75	15.44
October	78	37	65.27	43.03	5.964	83.160	3.58	19.02
November	64	30	57.73	36.48	3.584	86.744	0.93	19.95
December	50	15	42.94	27.15	1.636	88.380	4.70	24.65

[a] Elevation of gaging station, 6,311.5 feet. [b] Snow reduced to equivalent rainfall.

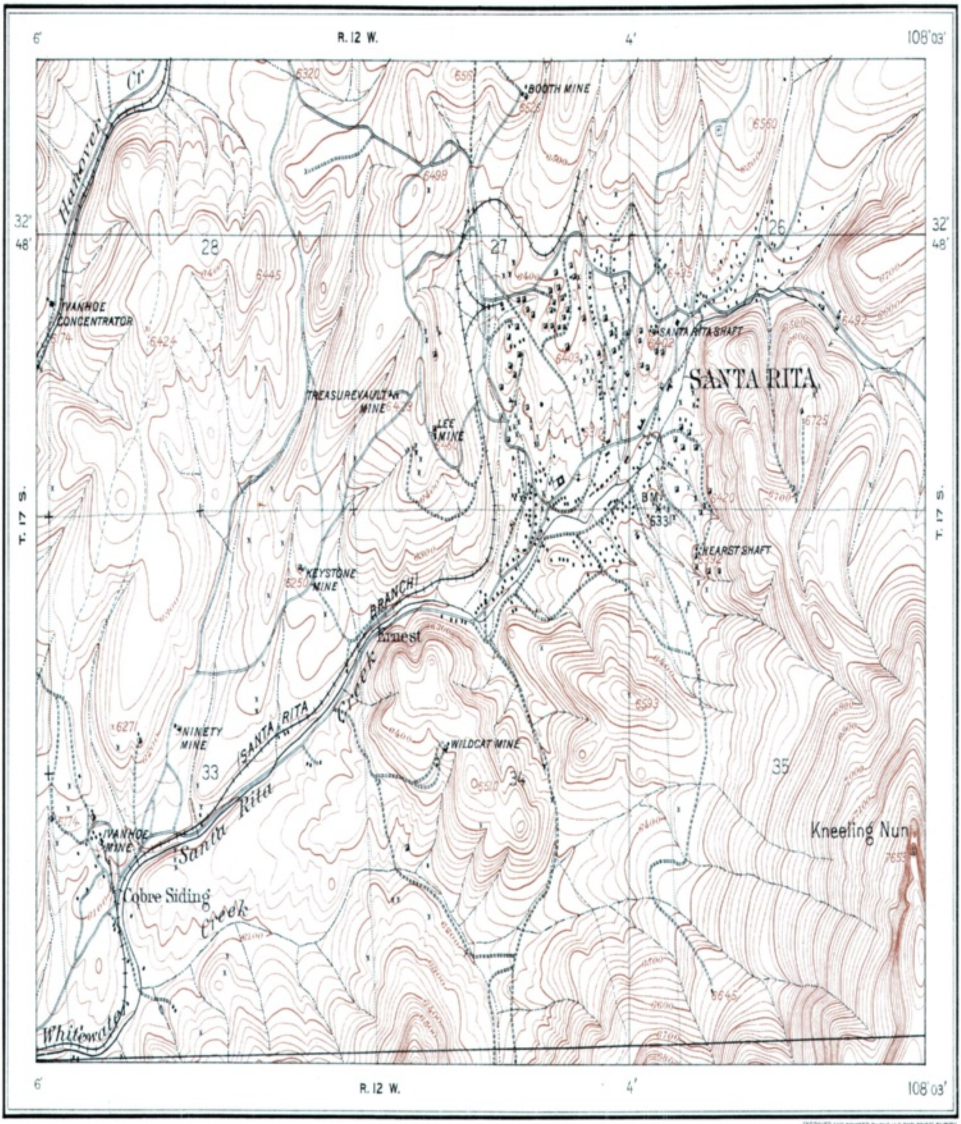
The streams are not large, and the country is not particularly well watered, either for mining or for agriculture. Marketable timber is scarce except in the higher mountains, and even there it is inclined to be scrubby. The timber used at the Santa Rita mines is brought in by rail, as the available local supply, never great, was exhausted some years ago.

Santa Rita and the neighboring regions offer an attractive field to the prospector; the climate is delightful, the country open and easy to traverse, and the rock outcrops fairly favorable.

GEOLOGIC CONDITIONS GOVERNING MINING IN SANTA RITA DISTRICT.[a]

The ore bodies at Santa Rita are associated with quartz-monzonite and quartz-monzonite porphyry intrusions, which cut limestones, sandstones and shales. These intrusives are of post-Cretaceous date, and are older than tuffs, rhyolites, and andesites in the vicinity, which are regarded as of Miocene age. Of the three types of intrusives near the ore, the oldest and the one constituting the largest mass is a quartz-porphyry; the next in point of age is an essentially even-grained quartz-bearing monzonite, containing idiomorphic feldspars and occurring as a stock of moderate size; the youngest is a porphyritic rock, locally characterized by large crystals of quartz, which occurs in the form of dikes. The sedimentary series cut by

[a] Notes on the geology of the region were furnished by Dr. A. C. Spencer, whose complete report on the geology of the Santa Rita district will be published by the United States Geological Survey.



CONTOUR MAP OF REGION AROUND SANTA RITA BEFORE MINING WAS COMMENCED

Scale $\frac{1}{24000}$ or 1 inch = 2000 feet.

1 ½ 0 1 Mile

2000 1000 0 2000 4000 6000 Feet

1 ½ 0 1 Kilometer

TRUE NORTH MAGNETIC NORTH 12½°

APPROXIMATE MEAN DECLINATION 1907.

Contour interval 20 feet.

Datum is mean sea level.

Surveyed in 1907 by the U. S. Geological Survey these intrusives consists of limestones of Carboniferous (Pennsylvanian) age, and of Cretaceous shales and sandstones.

The alteration of the rocks in the vicinity of the ore deposits has been marked. Within a rudely elliptical area, measuring about 7,000 feet east and west and more than 10,000 feet north and south, the prevolcanic rocks were thoroughly permeated by metamorphosing solutions. These solutions have sericitized the shales and have charged them with pyrite; they have silicified and pyritized the sandstones, and have changed the limestones in many places to chlorite-sericite rocks, carrying large amounts of pyrite and locally much magnetite. All the igneous rocks are somewhat mineralized, as shown by the presence of pyrite. The older quartz-porphyry is practically everywhere completely sericitized, but this sort of alteration is much less general, and is locally lacking in the younger intrusives.

In the main, the mineralization is of the type that is produced by general metasomatic alteration of the rocks. Highly individualized veins are essentially lacking, though reticulating fractures in the rocks are locally healed by quartz fillings. The date of the mineralization is not precisely known, but on the south the altered rocks are overlapped by unmetamorphosed volcanics, indicating that the mineralization was essentially complete before the extravasation of the latter in Miocene time.

ORE DEPOSITS.

The copper ores of Santa Rita are mainly low-grade deposits in which the copper minerals are disseminated through the rocks, and various rocks, both sedimentary and igneous, are contained in the ore bodies. Metamorphism and primary sulphide deposition resulted from the action of hot solutions of magmatic origin. Almost everywhere the altered rocks carry at least small amounts of copper, but material of ore grade has been formed mainly through the enrichment of relatively lean rock by the downward movement of water.

The principal copper mineral of the sulphide ores is chalcocite, but metallic copper and cuprite are locally present. Chalcopyrite, though seemingly not abundant in the ores, must have been the primary mineral from which most, if not all, of the secondarily deposited copper has been derived. Malachite occurs in much of the weathered capping, and is intermixed in varying proportions with other copper minerals in the upper parts of the sulphide ore bodies.

Rich copper ores that were mined in former years came from a series of ore bodies disposed along a curving belt 400 to 800 feet wide and about 6,000 feet long. This ore belt is comprised in the wider and more extensive zone which includes the principal ore bodies of much lower average grade that have been developed by the Chino

Copper Co. As now outlined (see Pl. III) the ore zone is an approximately elliptical annulus 500 to 1,000 feet wide, with an inside shorter diameter of 2,000 feet and a longer diameter of 3,000 feet. The principal axis of the ellipse trends northwest. The ores are of variable composition, as the different rocks mentioned have entered into the ore bodies, and each rock has been differently affected by the metamorphosing agents.

In the main the results of churn-drill prospecting show that the ore bodies are rather definitely though not sharply limited downward, and this, together with the fact that chalcocite replaces pyrite, proves that the ores owe their present copper content partly to processes of enrichment due to the downward movement of surface waters.

The weathered overburden that caps the sulphide ores varies in thickness from a few feet to 150 feet or more, the average being about 85 feet. In general the ore bodies measure more than 100 feet vertically, but what may be called the bottom surface is extremely irregular. Thus in one section of the ore area, material carrying chalcocite and metallic copper has been found continuing to depths as great as 1,300 feet, and in several places the ore bodies are 400 to 500 feet thick.

The variations in thickness, and the fact that the segregations that are large enough and of sufficiently high grade to constitute ore occur in a rather definite zone, are thought to be due to differences in the conditions of underground drainage.

The variations in the mineral constitution of the ores are wide. Different sorts and grades of ore are rather intricately intermixed, so that maintenance of the output of a mine at any average mineral composition is always difficult and at times impossible. The chief variations of practical moment are, first, in the relative amounts of copper minerals to pyrite and magnetite, and, second, in the proportions of heavy copper minerals to malachite. In the milling of the ores a high iron content—that is, a high proportion of pyrite and magnetite—tends to a low ratio of concentration, and any large proportion of malachite results in a marked decrease in the percentage of copper recovered.

ADAPTATION OF MINING METHODS TO GEOLOGY AND TOPOGRAPHY.

The methods of prospecting and mining adopted in the Santa Rita district have been determined by the geologic features of the ore deposits, and the general scheme of excavation, ore delivery, and waste disposal has been nicely adjusted to the topographic situation.

By reason of the wide extent of the ore bodies as compared with their depth, they were well suited to churn-drill prospecting for the determination of ore tonnages and grades of ore. The altered and generally soft nature of the rocks resulted in low drilling costs, and the relatively even grade of the ore favored accuracy in determining the copper content of the ores.

The feasibility of open-cut mining by means of steam shovels depends on the size of the ore bodies and the thickness of the overlying waste.

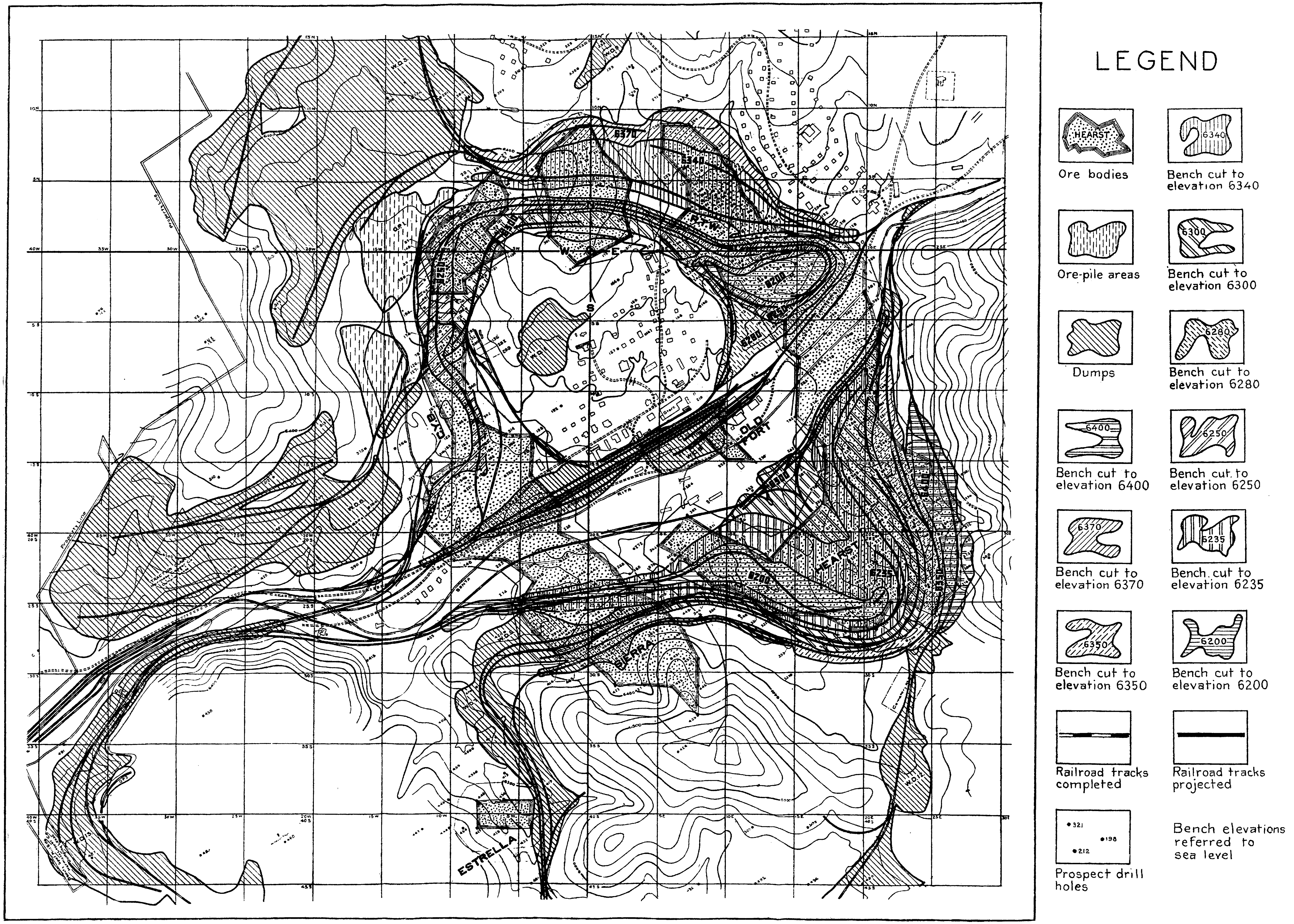
Because the altered rocks that contain the ore and constitute the gangue are soft, drilling is easy. These rocks are much jointed and break readily when blasted.

Any appreciable addition to the amount of waste material that must be moved, owing to large slides, is not anticipated. In general, the overburden of soil, or unconsolidated débris, is shallow, and everywhere such material is essentially free from percolating water. Overburden consisting of weathered rock is also well drained, and both materials stand well without "slaking" in any important degree. Although the ore and the weathered rock contain abundant sericite, and when finely broken develop much colloidal material, in place they are generally strong enough to stand with fairly steep slopes for all depths that are reached by open pits.

The mines are situated in a basin which is a broadened section of the valley of Santa Rita Creek.

In plan the outline of the Santa Rita mine workings is like that of a jew's-harp, two great cuts forming the flaring loop, the block left to carry the creek forming the tongue (see Pl. III). The creek flows southwest, leaving the ore-bearing area 4,000 feet beyond the point of entry and at an elevation 120 feet lower. The inclined approaches that permit locomotive haulage in removing the ore from the deep pits have grades opposed to the general surface slope. These approaches diverge at a point more than 1,200 feet down stream from the westerly edge of the ore area. The length of haul from the far end of each of the mine pits to the common track that serves the crushing plant is thus greater than 1 mile, so that considerable depths can be directly attained. Eventually spirals or switchbacks will be required and possibly the use of hoisting planes or Shay geared locomotives may be advantageous. At present the development plans of the company do not contemplate that any mining will be done by means of steam shovels below an elevation of 6,050 feet, which is approximately 150 feet below the points where the pit approaches intersect the ground surface.

The crusher is situated near the mouth of the canyon-like valley which drains the Santa Rita basin, the delivery track being at a level about 80 feet above the assembly yard. From this yard the ore trains go to the concentrating plant at Hurley by way of the Atchison, Topeka & Santa Fe Railroad.



TOPOGRAPHIC MAP OF ORE DEPOSITS AT SANTA RITA, N. MEX.

Shows relations of prospect drill holes to coordinates, placed every 200 feet, and to ore bodies, which are outlined; the methods of working the ore bodies by benches and the trackage on each bench are indicated.

Whereas the ore tracks necessarily converge for the purpose of delivery, the waste tracks are in the main divergent. The principal dumping ground is in the valley of Whitewater Creek, south of the mines, but most of the waste from the upper levels of the northern section has been deposited within the Santa Rita basin west and northwest of the mines.

Because of the semiarid character of the climate, current groundwater seepage is slight in comparison with the extent of the workings and pumping costs are correspondingly low. At times of heavy rains the lowest levels in the pits become flooded, but little time is lost from this cause. Continued rains, which are seldom experienced, result in reduced operating efficiency, mainly by causing track troubles.

For some time to come a channel for the creek will be maintained directly across the Santa Rita basin, but eventually, in order to remove large amounts of ore, the diversion of the stream will be necessary. Occasionally after heavy rains the volume of water carried by the creek is large, but the mine pits have been so planned that they are in no danger of being flooded from this source.

The Chino company has, for convenience of reference, grouped its property into six ore bodies, as follows: C. Y. B., Sierra, Hearst, R. W. T., Old Fort, and Estrella. This grouping, however, is somewhat artificial and the divisions do not correspond in every case with actual breaks in the mineralized area. In fact some of the ore bodies are somewhat merged with each other and some are separated by areas of mineralized but low-grade rock. Their annular grouping around a relatively barren core, their extent, and their general relations are shown by Plate III.

HISTORY OF MINING IN SANTA RITA DISTRICT.[a]

It is said that copper was first mined at Santa Rita by the Aztecs. It was probably one of the places where they procured the native metal. About 1800 a friendly Apache chief showed the croppings near the present Romero shaft to Col. José Manuel Carasco, the officer in charge of the Spanish military posts in New Mexico. Col. Carasco interested Don Manuel Francisco Elguea, of Chihuahua, a wealthy merchant banker and subdelegate to the Spanish Court, and the latter obtained a concession to the property under Mexican law. Four years later Elguea purchased Carasco's interests and began to open up the property. The copper produced found a ready market with the Mexican Government for coinage. Almost until the early nineties the region was infested with warlike Apache Indians, who were not subjugated until almost exterminated; hence the Spaniards

[a] The chief source of information regarding past conditions is a manuscript report by Mr. John M. Sully, manager of the Chino Copper Co.

had to build a fort and maintain a military force to protect the camp. The fort was a triangular structure with heavy built adobe towers guarding each of its apices, and these were joined by strong adobe walls. One of the old towers is still standing and until recently was used as a jail. Other remnants of these ancient fortifications are still extant.

Rough smelting works were built and from the sulphide and oxide ores mined copper metal was extracted in a crude way. This was carried on mule trains, under military convoy, to Chihuahua and on to Mexico City. It is said that Elguea left at his death, in 1809, a considerable fortune. Later, his widow leased the property to Juan Onis. After a few years it was leased to Siqueiros, and was worked intermittently, owing to Indian outbreaks, until the late fifties, when Indian hostility finally forced its abandonment.

Gen. Sibley, in command of Confederate forces from Texas, held this region for a short time. Efforts, mostly unsuccessful, were then made to work the mines.

In 1873 the dawn of a more prosperous period was at hand, for M. B. Hayes, connected with the first smelting works in Colorado, began work at the present Romero mine. A 248-foot shaft was sunk, and a furnace, which proved inadequate, was erected. A shipment of 40 tons of picked ore and imperfectly smelted copper was hauled by teams 700 or 800 miles to a railroad station in Colorado and forwarded to the Baltimore Copper Works and to the Revere Copper Works, at Point Shirley, Mass. In October, 1873, Mr. Hayes and associates obtained title to, and possession of, the property from the sole surviving heirs of the Elguea family, who were settled through Mexico and Europe. Until the heirs had transferred their interests the United States Land Office had refused to issue patent to the property, because the Elguea family had rights under the treaty between the United States and Mexico. Later, patent was taken out by the Hayes interests, so as to make the title more secure and to guard against future litigation.

In 1881 the property was sold to J. Park Whitney, and a stamp mill, later burned, was erected near the foot of Romero Hill. In 1882 diamond drilling was done. Although the records of some of these holes are still extant, their exact location has not been ascertained.

The year 1891 saw the railroad completed to Hanover, a few miles away, and in 1899 a branch into Santa Rita was finished.

In 1897 Mr. Whitney gave a lease and bond to the Hearst estate, but in 1899 sold the property to the Santa Rita Mining Co., before the lease had expired. The new company developed some large bodies of high-grade sulphides and considerable native copper. After extraction of these, active operations were carried on, mostly by lessees, on a basis of 35 per cent to the owners when copper was 15 cents or more per pound, and 25 per cent when it was less than 15 cents. The Santa Rita company, however, continued to operate the mill, treating ore purchased from the lessees and low-grade ore from the dumps, formerly thrown out as waste. It is said that this old mill, which had a capacity of only 120 tons per day, left copper in the tailings to the amount of 4 per cent of the rock treated. It is therefore a far cry from the old methods to present-day milling efficiency. Lessees continued work until about 1906. Ore containing 10 per cent copper was sold by them, but ore with a lower copper content they threw on the waste dumps.

A new era dawned on this ancient mining district in 1906, for in that year systematic sampling of the property was begun by Mr. John M. Sully, the present manager of the Chino company. The old dumps and the old workings were sampled, and churn-drill sampling was begun.

PRESENT CONDITIONS AND EXTENT OF PROPERTY.

After an extensive report by Mr. Sully concerning the results of the sampling mentioned, the present company acquired the property, which now consists of 147 mining claims, including fractional and full claims, comprising 2,645 acres; 131 of these, aggregating 2,412 acres, are patented. In addition, the company owns 160 acres of agricultural land adjacent to its mining claims. The property contained 19 different shafts, aggregating over 4,000 feet of sinking, and these were connected by several thousand feet of underground workings.

This, then, was the beginning of the development as a result of which an old and inefficient mill that treated 120 tons of ore per day has been replaced by a modern mill, with a capacity of about 7,000 tons per day, which successfully treats ore containing less than 0.8 per cent of copper.

Stripping operations were started by the Chino company in October, 1910, the ore taken out being sent to stock piles. During August, 1911, the first shipment of ore was made to the mill. The total shipments during that month were nearly 14,000 wet tons.

PROSPECTING AND LOCATING ORE BODIES.

Preliminary to the systematic prospecting and delimiting of the ore bodies with churn drills, an accurate topographic map of the region, on a scale of 100 feet to the inch and showing a contour interval of 5 feet, was made. This map was then laid off into north-south and east-west coordinates at intervals of 100 feet, and a datum point at a coordinate intersection was established, for the purpose of locating, by coordinates referred to this point, all the drill holes that were to be put down, as for instance: Hole 10, north 650 feet and west 355 feet.

CHURN DRILLING AND SAMPLING.

TYPE OF DRILL USED.

For prospecting the ore body, No. 6 deep-hole Cyclone churn drills were used. They were fitted with steam power and were of the walking-beam type, using 4-inch to 8-inch bits, depending on the depth of hole and other conditions. In sampling low-grade deposits of relatively even tenor over a large superficial area, where the rocks are not hard, these drills give much better satisfaction than do diamond drills. Of course, churn drills can sink only holes that are vertical or approximately so, whereas diamond drills can be used to bore at any angle.

METHODS OF DRILLING.

Drilling was begun as near to coordinate intersections as was convenient. Each drill was operated day and night by two crews, each working 12 hours, and consisting of a driller, his helper, and a sampler.

In the process of drilling, water was poured into the hole to form a sludge with the cuttings. As the drill was removed any mud adhering to it was washed back into the drill hole. In relatively barren surface material, trial samples were taken at intervals of about 5 feet, but below the surface material samples were taken every 3 feet, the distance being measured by marks on the drilling cable. When ready for sampling, a bailer with a valve in the bottom was lowered, and all the sludge cut in the sampling interval, 3 or 5 feet of drilling, was carefully bailed out. The sludge from the bailer was discharged into a launder, and from there was cut on what was in effect a Jones sampler. The material saved, about one-quarter of the total, was caught in a tub and, if too bulky, was again poured through the sampler; the bailer, launder, and tub were washed after use, and the washings were also run through the sampler. The final quartered sample, amounting to about a gallon, including the washings, was sent to an evaporating furnace, and, when dry, to the assay shop. The method of treatment there is described later. A part of each sample was caught and carefully panned, so as to indicate the kind and character of the rock and the minerals contained, and all the results were set down in the record or log of the hole.

ERRORS IN CHURN-DRILL SAMPLING.

The errors peculiar to churn-drill sampling are as follows: (*a*) Deviation of the hole from the vertical; (*b*) breaking off at the point of drilling of a surplus of rich brittle mineral, thus giving a sample that is too rich; (*c*) concentration of heavy minerals in the bottom of the hole, so that they are not reached by the bailer, or are recovered from the hole when it has reached a much lower depth where the rock may actually be barren; (*d*) caving of the rock from the sides of the hole.

Little trouble resulted from the deflection of holes, except in a few instances where the drills penetrated narrow zones of crushed rock contained between relatively hard, steep-sloping walls. The tendency was then for the drill to be deflected along the slope into the softer material. However, careful drilling usually remedied this tendency. A few deflections of as much as 4 or 5 feet per 100 occurred.

Concentration of heavy minerals in the bottom of a hole was not particularly troublesome, and was largely obviated by careful cleaning out of the bottom of the bore each time a sample was taken.

Caving was the most obvious cause of error, but was largely remedied by casing the hole and continuing it with a smaller bit. As a result of this procedure some of the holes that were 8 inches in diameter at the top were only 4 inches wide at the bottom.

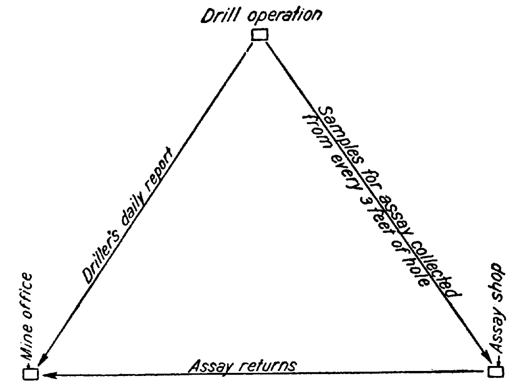


FIGURE 1.—Diagram showing character and distribution of reports made in connection with drill holes.

The accuracy of the churn-drill method of sampling was later tested by a few shafts and "raises" along some of the drill holes. The average results obtained from shaft samples and from drill-hole samples agreed closely. The churn-drill method of sampling other similar copper deposits in the Southwest has been proven comparatively accurate and satisfactory.

OPERATION AND EFFICIENCY OF PROSPECT DRILLING.

At the end of every shift, each driller submitted to the mine office an outline of what had been accomplished by his crew during the hours on duty. This was recorded on a special form, shown as Form 1 (p. 12).

In addition to the shift reports, each drill crew sent to the assay office samples quartered down from the total cuttings of every 3 feet of hole drilled. These were assayed and the returns were sent to the mine office as soon as completed. The different reports mentioned are more graphically shown by figure 1.

FORM 1.—*Form used by driller in reporting to mine office the work done by his crew.*[a]

DAILY CHURN-DRILL REPORT.

SAMPLES.

Depth.	Character of rock.		Minerals.	Sampling appliance.		Caving.			Is sample reliable? If not, state reasons under "General remarks."	True depth.	Length of sample.
Number of sample.	1. Quartz-diorite. 2. Andesite-porphyry. 3. Quartzite. 4. Limes or other rock. Leached or unleached. Altered or unaltered. Hard or soft. 5. Silicified porphyry. 6. Central city intrusive.	Color of sludge: Red or dark brown. Light yellow. Pink or purple. Light brown. Gray or white. Green or blue.	A Chalcocite. B Pyrite. C Metallic copper. D Oxides of copper. E Carbonates. F Chalcopyrite. G Magnetite. H Hematite. I Mica.	Split divider.	Tub.	No.	Yes; slightly.	Yes; considerably.			

GENERAL REMARKS.

....

....

....

WATER.—Depth reached............ Stands at depth of..........................

....

FAULTS AND CLAY SEAMS.—Depth below surface...ft. Width of fault material...ft.
SIZE OF BIT.......... CASING: Size..........; depth..........

EMPLOYMENT OF TIME.

Actual drilling hours..................... Moving and setting up hours.............
Repair hours.............................. Casing or removing casing................
Other delays..
Driller....................................... Shift..........................
Helper.. Date...........................
Sampler....................................... Hole No. ..... Drill No. .....

NOTE.—Enter time after each name.

[a] Size of form as printed by operating company, $8\frac{3}{8}$ by $13\frac{3}{4}$ inches; held in binder through two holes in top.

In the office the data from the daily drillers' reports were compiled into a monthly report (Form 2). The average performance and

FORM 2.—*Form*[a] *used in compiling monthly report from data in daily drillers' reports, as shown in Form 1.*

Date.	Shift.	Hole.	From—	To—	Number of feet drilled.	Hours drilling.	Hours moving.	Hours casing.	Hours reaming.	Hours repairing.	Other delays.	Total time.	Drill No.	
													Driller.	Helper.
1														
2														
3														
4														
5														
6														
7														
8														
9														
10														
11														
12														
13														
14														
15														
16														
17														
18														
19														
20														
21														
22														
23														
24														
25														
26														
27														
28														
29														
30														
31														
Totals														

[a] Size of form used by operating company, $8\frac{3}{8}$ by 14 inches; upper half only, showing space for recording data covering day shift, is shown; lower half provides for similar data covering night shift. Held in binder through four holes at left.

efficiency of the churn drills used in prospecting the Santa Rita ore bodies can best be shown by the table following. In this the performance of two Cyclone No. 6 deep-hole drills is averaged for a period of 10 months. The figures are fairly representative of what has been accomplished by prospecting drills on the somewhat softened and locally sheared granitic and other rocks of Santa Rita. A similar table, showing in a comparative way the performance of all the drills, was made out for each month.

Churn-drill record.

[March to December, 1912—10 months.]

DAY SHIFT.

Drill No.	Feet drilled.	Total working hours.	Per cent time drilling.	Per cent time delayed.	Per cent time repairing.	Per cent time moving.	Average feet per hour of drilling.
6	3,221	3,727	59.3	18.2	3.3	19.2	1.46
7	3,365	3,776	70.7	10.6	2.5	16.2	1.26
NIGHT SHIFT.							
6	3,071	2,598	78.9	16.4	3.5	1.2	1.50
7	2,947	2,850	84.4	12.3	2.2	1.1	1.23
TOTAL.							
6	6,292	6,325	67.0	17.5	3.4	12.1	1.48
7	6.312	6,626	76.4	11.3	2.4	9.9	1.25

COST OF PROSPECT DRILLING.

To the miner the cost of prospecting is of great importance; hence cost data regarding churn drilling these ore bodies are, through the courtesy of the Chino Copper Co., presented in some detail, as follows:

Cost data on churn or prospect drilling at Santa Rita, N. Mex.

Drill Nos.	Month.	Cost of—						
		Labor.	Supplies.	Teaming.	Administration.	Total.	Per foot.	Per shift.
	1912.							
6, 7, and 12.	May	$2,106.14	$494.03	$142.18	$349.28	$3,091.63	$2.31	$22.90
6 and 7	June	2,029.35	661.09	80.25	270.23	3,040.92	2.29	26.44
6 and 7	July	1,997.43	451.20	112.75	229.67	2,791.05	2.23	24.48
6 and 7	August	1,969.57	496.14	129.00	209.12	2,803.83	2.00	23.56
6 and 7	September	1,953.78	442.02	150.25	234.24	2,780.29	2.13	24.56
6 and 7	October	1,823.99	506.98	153.95	185.25	2,670.17	2.68	23.95
6 and 7	November	1,775.52	619.47	134.75	238.54	2,768.28	2.42	27.82

The above table is valuable as a general index of the cost of churn-drill prospecting. Of course, in other places local conditions might cause considerable variation from the figures given.

ASSAY RECORDS AND ORE CLASSIFICATION.

The assay returns were compiled on loose leaves (Form 3), which, when filled out, were put into strong permanent binders and kept as volumes. In the columns of this form the location of the sample in the hole was recorded under depth and its value when assayed by the permanganate method was shown in the "Perm. Cu." column.

FORM 3.—*Form used in assay record of drill-hole samples.*[a]

DRILL HOLE NO. ....

Coordinates..........................

Time drilling ....

Depth.	% Perm. Cu.	% Met. Cu.	% Cu. total.	% Elect. Cu.	Averages.	Rock.	Minerals.	Sludge.

The results of electrolytic check assays were recorded under the heading "Elect. Cu." In the column headed "Averages," the ore was divided into classes, as follows:

Ore classification used by Chino Copper Co.

Class.	Copper content, per cent.	Designation.
P	0.00 to 0.79	Poor.
L 1	0.80 to 0.89	Low No. 1.
L 2	0.90 to 0.99	Low No. 2.
F	1.00 to 1.24	Fair.
G	1.25 to 1.49	Good.
E	1.50 to 1.99	Excellent.
S	2.00 upward	Superlative.

Any native copper contained was recovered by screening or panning the crushed sample; this was weighed and the weight recorded in the column headed "Met. Cu." The sum of the native copper content and the copper determined by the permanganate method was then set down in the column headed "Cu. total." Notes on the kind of rock passed through, the minerals encountered, and the color and character of the sludge were set down in the columns headed "Rock," "Minerals," and "Sludge." It will be seen that this form gave a comprehensive record of each hole, including its location by coordinates, time of drilling, depth, values for every assay interval of 3 feet, and averages. However, to more thoroughly visualize the mineralized conditions indicated in each hole, its assay record was plotted on cross-section paper somewhat as shown on Form 4. Two ink lines to represent each drill hole were drawn as shown. The sections as determined by the groups of assay results were indicated by horizontal ink lines, as shown, each small square on the paper representing a depth, or sampling interval, of 3 feet; the figures opposite showed the percentage of copper that the rock contained.

In grouping the ore into the foregoing classes, every sudden and considerable change in values, if the record showed a change in the character of the sludge, was noted as a possible class boundary. Rock carrying less than 0.8 per cent of copper was considered of too low grade to be classed as ore, as is explained later. However, where small masses of waste occurred with the ore, in such a way that separation from the latter could not profitably be carried out, it was figured in as a diluent of the ore reserves.

CROSS-SECTIONING THE ORE BODIES.

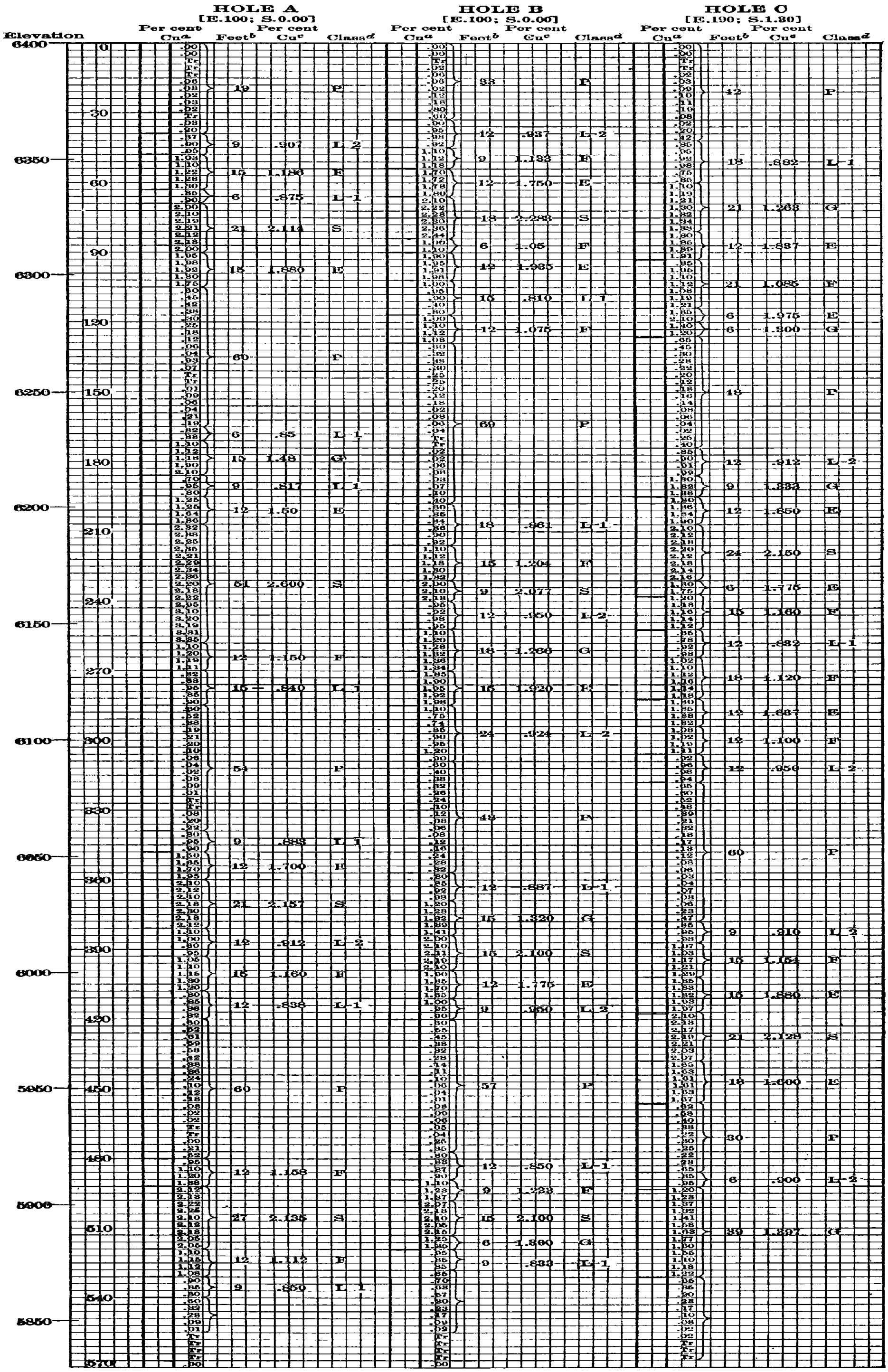
Following the assay-record sheets came the work of making cross sections of the ore body. These were compiled from the data shown on the assay records and the drill hole records and show both north-south and east-west cross sections of the ore bodies at intervals of 100 to 200 feet, depending on the spacing of the drill holes and the data that they furnished. Where the holes were several feet away from a cross section, the data from them were projected to the section and recorded there, with a statement of the distance and direction of the hole from the section. These cross-section sheets are used to show the general form and relations of the ore body and the relations of the different classes of ore with the barren areas and with each other.

A typical generalized cross section is shown in Form 5. The scale is the same as that used by the Chino company, namely, 1 inch equals 200 feet horizontally and 60 feet vertically. Each hole shows in place the number of feet of each class of ore, and of waste, that is encountered. Lines were drawn connecting points of approximately equal values; thus, by the law of averages, the relative quantities of each class were graphically shown on the section.

The figures *20, 21, 23*, etc., along the top of Form 5 are the numbers of the drill holes along the line of the cross section, the cross section being constructed from the data revealed by these holes. The numbers 2′ N, 3′ S, etc., under some of the hole numbers mean that the hole is 2 feet north, or 3 feet south of the actual cross-section line, but the data from holes that are so close to the section are projected onto the section just as though the holes were in line. The P areas (unshaded) show the relations of waste material to ore, etc. The shaded areas that lie inside and above the heavy dotted line are ore areas, and these are divided into two different classes by two different patterns of shading. The shaded areas are further subdivided into a total of five classes of ore by light dotted lines connecting points of similar change of value in each drill hole.

The ore below the 6,050-foot level can probably not be recovered by steam-shovel mining and the present methods of rail haulage, so that the cost of recovering it will be somewhat increased.

FORM 4.—*Specimen cross-section assay record of results of sinking three prospect churn-drill holes to determine copper content of ore bodies.*



[a] Percentage of copper present in each 3-foot sampling interval. The depth intervals or sampling intervals of 3 feet are represented by the side of a small square.

[b] Thickness in feet of each class of ore encountered.

[c] Average value in percentage of copper for each class of ore encountered.

[d] Symbol for each class of ore, see p. 15.

43503°—Bull. 107—16. (To face page 16.)

The notation 10 N., East and West crosssection, at the top of the cross section means that this is an east and west cross section, and is the tenth north from cross section 1. The figures 1000, 1500, etc., are the horizontal distances in feet. The vertical distances, referred to sea level, are shown by the figures 6050, 6200, 6350, etc., on the right margin.

The use of colors to denote the different classes of ores shown by the cross section adds greatly to its clearness. In the form as used at the mine two colors, indicated by shading in the form as presented, were used to bring out the special relations.

Figure 2 represents two drill holes. Hole *1* shows 3 feet of class E ore below 17 feet of barren rock. Hole *2* shows 10 feet of class E ore

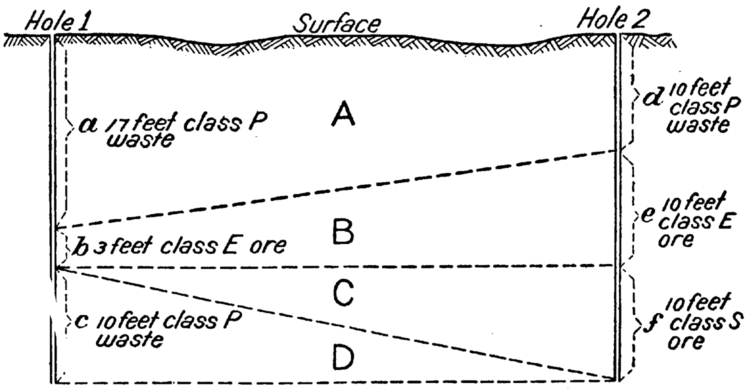


FIGURE 2.—Method of estimating classes of ore from drill records so as to show them on cross sections. *a*, hole *1*, 17 feet of class P rock or waste; *b*, 3 feet of class E ore; *c*, 10 feet of class P rock; *d*, hole *2*, 10 feet of class P rock; *e*, 10 feet of class E ore; *f*, 10 feet of class S ore. The area A would be waste, the area B would be class E ore, and the triangular area C would be S ore. It would be figured as extending up to hole *1* though no S ore had actually been encountered in that hole. The D area would be waste rock that would not be moved.

and 10 feet of class S ore below 10 feet of barren rock. A straight (dotted) line connects the top boundary of class E ore in hole *1* with the top boundary of this class of ore in hole *2;* and similarly for the bottom boundary. Although no class S ore was found in hole *1* yet the class S ore body found in hole *2* may maintain its full thickness out nearly to hole *1*, so, by the law of averages, it is approximately correct to figure the ore as occupying the triangular space C.

Cross sections are very helpful in showing the trend of ore bodies and their form and general relations. Where geological conditions can be shown on the cross sections they are especially valuable in studying ore deposits. Such cross sections as are presented are not quite accurate enough for use in making final estimates of ore tonnages. A more accurate method of ore estimation is discussed in a subsequent section.

GENERAL CONSIDERATIONS IN DELIMITING ORE BODIES.

An inspection of the completed cross-section sheets made fairly evident just what ground could profitably be mined under the conditions. With the cross-section data as a basis, the limiting outlines of the ore bodies were drawn on maps showing the surface configuration, drill holes, etc. The chief considerations that governed the mapping of the horizontal limits to which ore could be mined were as follows: (1) The depth of the overburden and the slopes at which its banks would stand; (2) the local size and thickness of the ore body and its character and value; (3) the method of mining—whether the ore body could all be taken out by steam shovels at about the same average cost as the main part of the ore body, or whether extra expense would be involved.

Of course, it was recognized that there must necessarily be a limit to the depth to which steam-shovel mining with locomotive haulage could be carried. After consideration of various factors it was believed that below the 6,050-foot level (measured from sea level) the great open-cut mining pits would be so deep and so restricted in area at the bottom that certain modifications in the methods of mining the ore below that level—a relatively small percentage of the total ore—would have to be adopted. The chief factors regulating the depth to which it would be convenient to mine with steam shovels were believed to be as follows: (1) The horizontal extent of the ore body, in relation to its depth, would have to be considered because a certain amount of "elbow room" would be necessary for steam shovels, car tracks, etc.; (2) the maintenance of economical transportation grades for hauling out the ore and waste would limit the depth of working by steam shovels and locomotive haulage; (3) if the horizontal dimensions of the deposit were great enough, steam shovels might load the ore even in very deep pits, and it might be hauled out by means of Shay or rack-geared locomotives, or over inclined planes with cable haulage.

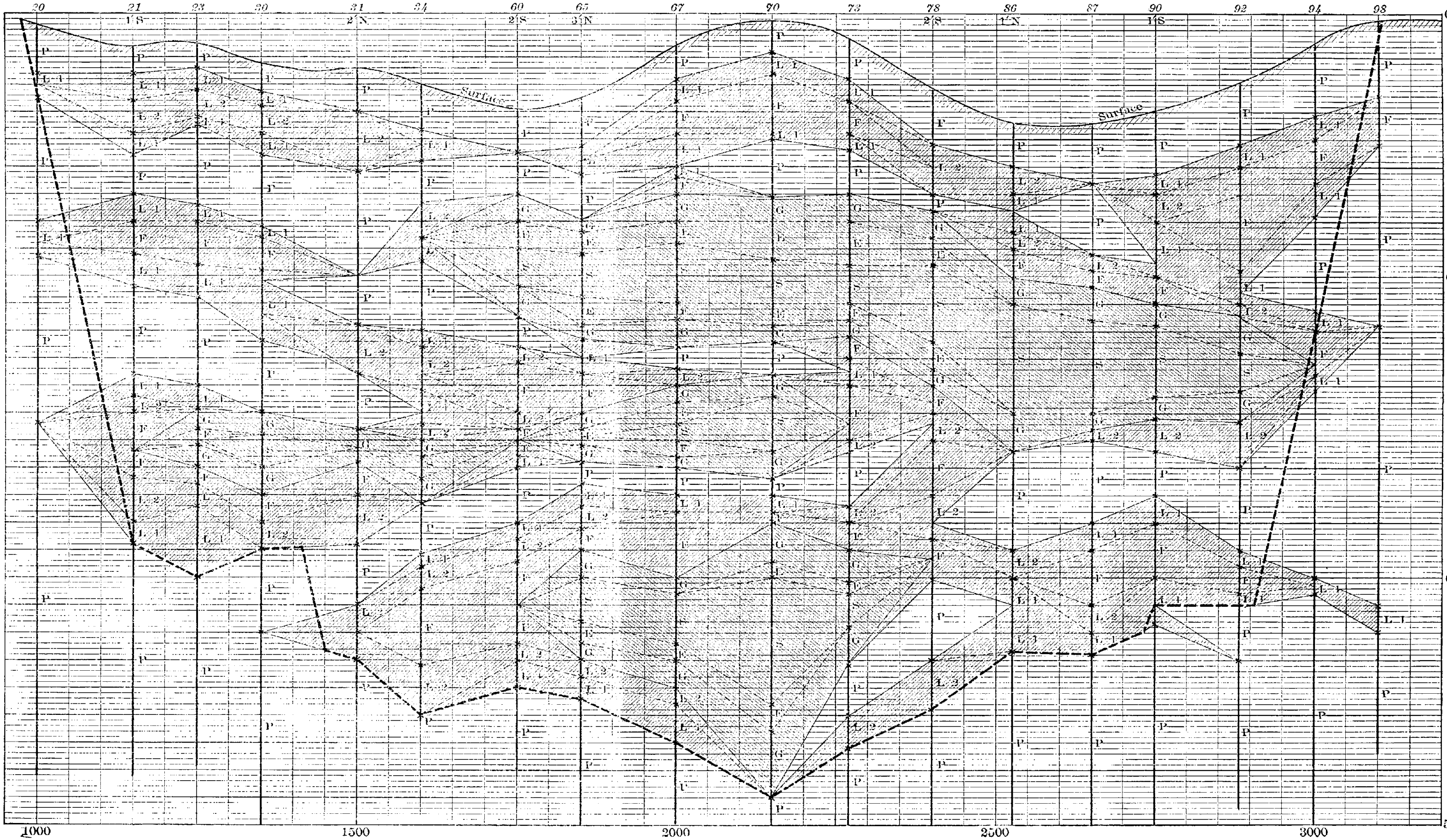
Estimates of the ore above the 6,050-foot level were kept separate from those of the relatively small proportion of ore below that level, as the cost of taking out the lower ore would be slightly greater.

METHODS OF ESTIMATING ORE.

When the facts collected in prospecting and sampling the ore bodies were supplemented by geologic and by general mining cost data, the whole body of information thus systematically gathered was used in estimating the tonnage of rock rich enough to be classed as ore and available for mining.

FORM 5. *Typical cross section of ore body.*

[This section was designated "Cross Section 10N, East and West." One inch represents 200 feet horizontally and 60 feet vertically. Letters, P, L1, L2, F, G, E, S, showing grades of ore, are explained on page 15.]



ASSAY PLAN.

In proceeding toward a methodical solution of the problem of ore estimation the first step was to make an assay plan. This consisted of a copy of the original drilling map showing each hole, by number, its location, the number of feet of ore encountered in it, the average value of the ore, the number of feet of waste that would have to be moved from the ore there, etc.

PRELIMINARY ESTIMATES.

Some general idea of the form and size of the ore body, as well as possible methods of working it, had to be obtained before definite figures on the amount of minable ore available could be made. For instance, if only a quarter of a million tons of ore was in the deposit, the average cost per ton for mining would be more, other things being equal, than if it contained 5,000,000 tons.

After determination had been made, in a general way, that the ore body was properly situated and extensive enough to warrant mining it with steam shovels on a large scale, an estimate was made of the probable cost of removing and dumping the valueless stripping material and mining the ore by open-cut methods on a scale commensurate with the great size of the deposit. The estimated cost, including concentrating, smelting, marketing the product, all wastage, and all expenses was $2 per ton for moist ore. From the cost data on mining, shipping, and treating the ore from the somewhat similar ore bodies of the Utah Copper Co.'s properties, it was known that the figure mentioned was a conservative one on which to base estimates of the ore reserves.

In order to facilitate the estimation of ore, the critical worth, below which mining at a profit could not in general be counted on, was expressed in units or values of copper. At 13 cents per pound for copper, ore that assayed 0.8 per cent, or 16 pounds of metal per ton, valued at $2.08, meant at least 8 cents per ton profit, if, as above stated, all costs and wastage allowances amounted to $2 per ton.

After figures of this kind had been prepared in this general way, it was decided to adopt 0.8 per cent of copper as what might be called the critical content. All rock containing a less percentage could not on this basis be estimated as ore. However, it was realized that if the price of copper went up considerably, then wherever practicable much of the leaner rock would undoubtedly be mined as ore, but such a contingency could not, of course, be made a factor in a conservative estimate. Wherever a mass of lean rock happened to be surrounded by ore, so that separating it was unprofitable, then of course it was figured in as a diluent, which reduced somewhat the average tenor of the ore.

A critical copper content of 0.8 per cent having been fixed as the content below which the rock could not conservatively be called ore, there being very little gold or silver in it, the estimate was made first by the cross-section method, which consisted in estimating the mass of ore by means of square prisms and then by figuring its value from the drill-hole and cross-section data. Later, as a check, the more exact triangular-prism method was used. As this method is better adapted for close and careful ore estimation, though involving about the same general principle as the cross-section method, it is explained in detail below.

ESTIMATING ORE BY TRIANGULAR-PRISM METHOD.

From the assay record of each drill hole the ore reserves were calculated. For these calculations three well-bound books with blank pages were used. They were each about 9½ by 12 inches in size and were cross ruled on a scale of 4 spaces to the inch. Book 1 may be called the log book of prospect drill holes. On page 1 were recorded the ore calculations for hole 1, on page 2 the calculations for hole 2, and so on.

The following examples are to show how the drill-hole records, prepared on Form 4, were calculated in Book 1:

Specimen calculation sheets from drill-hole records shown in Form 4.

HOLE A (E. 100; S. 0.00.).

LOG OF HOLE.

Feet of ore.	Class.	Per cent copper.	Foot per cent.
19	P[a]	.....	......
9	L 2	.907	8.16
15	F	1.186	17.79
6	L 1	.875	5.25
21	S	2.114	44.39
15	E	1.880	28.20
60	P[a]	.....	......
6	L 1	.850	5.10
15	G	1.480	22.20
9	L 1	.817	7.35
12	E	1.500	18.00
51	S	2.600	132.60
12	F	1.150	13.80
15	L 1	.840	12.60
54	P[a]	.....	......
9	L 1	.883	7.94
12	E	1.700	20.40
21	S	2.157	45.30
12	L 2	.912	10.94
15	F	1.160	17.40
12	L 1	.838	10.05
60	P[a]	.....	......
12	F	1.158	13.89
27	S	2.135	57.64
12	F	1.112	13.34
9	L 1	.850	7.65
520			519.99

ORE OF CLASS S.

Feet of ore.	Per cent copper.	Foot per cent.
21	2.114	44.39
51	2.600	132.60
21	2.157	45.30
27	2.135	57.64
120		279.93

ORE OF CLASS E.

15	1.880	28.20
12	1.500	18.00
12	1.700	20.40
39		66.60

ORE OF CLASS G.

15	1.480	22.20

ORE OF CLASS F.

15	1.186	17.79
12	1.150	13.80
15	1.160	17.40
12	1.158	13.89
12	1.112	13.34
66		76.22

ORE OF CLASS L 2.

9	.907	8.16
12	.912	10.94
21		19.10

ORE OF CLASS L 1.

6	.875	5.25
6	.850	5.10
9	.817	7.35
15	.840	12.60
9	.883	7.94
12	.838	10.05
9	.850	7.65
66		55.94

CLASS P WASTE.

193	.....	......
520		519.99

[a] Waste.

HOLE B (E. 210; S. 0.00.).

LOG OF HOLE.

Feet of ore.	Class.	Per cent copper.	Foot per cent.
33	P[a]	.....	......
12	L 2	.937	11.24
9	F	1.133	10.19
12	E	1.750	21.00
18	S	2.283	41.09
6	F	1.050	6.30
12	E	1.935	23.22
15	L 1	.810	12.15
12	F	1.075	12.90
69	P[a]	.....	......
18	L 1	.861	15.49
15	F	1.204	18.06
9	S	2.077	18.69
12	L 2	.950	11.40
18	G	1.266	22.78
15	E	1.920	28.80
24	L 2	.924	22.18
48	P[a]	.....	......
12	L 1	.887	10.64
15	G	1.320	19.80
15	S	2.100	31.50
12	E	1.775	21.30
9	L 2	.950	8.55
57	P[a]	.....	......
12	L 1	.850	10.20
9	F	1.233	11.10
15	S	2.100	31.50
6	G	1.300	7.80
9	L 1	.883	7.95
528			435.83

[a] Waste.

ORE OF CLASS S.

Feet of ore.	Per cent copper.	Foot per cent.
18	2.283	41.09
9	2.077	18.69
15	2.100	31.50
15	2.100	31.50
57		122.78

ORE OF CLASS E.

12	1.750	21.00
12	1.935	23.22
15	1.920	28.80
12	1.775	21.30
51		94.32

ORE OF CLASS G.

18	1.266	22.78
15	1.320	19.80
6	1.300	7.80
39		50.38

ORE OF CLASS F.

9	1.133	10.19
6	1.050	6.30
12	1.075	12.90
15	1.204	18.06
9	1.233	11.10
51		58.55

ORE OF CLASS L 2.

Feet of ore.	Per cent copper.	Foot per cent.
12	.937	11.24
12	.950	11.40
24	.924	22.18
9	.950	8.55
57		53.27

ORE OF CLASS L 1.

15	.810	12.15
18	.861	15.49
12	.887	10.64
12	.850	10.20
9	.883	7.95
66		56.43

CLASS P WASTE.

207	.....	......
528		435.83

HOLE C (E. 190; S. 130.).

LOG OF HOLE.

Feet of ore.	Class.	Per cent copper.	Foot per cent.
42	P[a]	.....	......
18	L 1	.882	15.88
21	G	1.263	26.52
12	E	1.837	22.04
21	F	1.085	22.79
6	E	1.975	11.85
6	G	1.300	7.80
48	P[a]	.....	......
12	L 2	.912	10.94
9	G	1.333	12.00
12	E	1.850	22.20
24	S	2.150	51.60
6	E	1.775	10.65
15	F	1.160	17.40
12	L 1	.832	9.98
18	F	1.120	20.16
12	E	1.837	22.04
12	F	1.100	13.20
12	L 2	.950	11.40
60	P[a]	.....	......
9	L 2	.910	8.19
15	F	1.154	17.31
15	E	1.880	28.20
21	S	2.128	44.69
18	E	1.600	28.80
30	P[a]	.....	......
6	L 2	.900	5.40
39	G	1.397	54.48
531			495.52

[a] Waste.

ORE OF CLASS S.

Feet of ore.	Per cent copper.	Foot per cent.
24	2.150	51.60
21	2.128	44.69
45		96.29

ORE OF CLASS E.

12	1.837	22.04
6	1.975	11.85
12	1.850	22.20
6	1.775	10.65
12	1.837	22.04
15	1.880	28.20
18	1.600	28.80
81		145.78

ORE OF CLASS G.

21	1.263	26.52
6	1.300	7.80
9	1.333	12.00
39	1.397	54.48
75		100.80

ORE OF CLASS F.

21	1.085	22.79
15	1.160	17.40
18	1.120	20.16
12	1.100	13.20
15	1.154	17.31
81		90.86

ORE OF CLASS L2.

Feet of ore.	Per cent copper.	Foot per cent.
12	.912	10.94
12	.950	11.40
9	.910	8.19
6	.900	5.40
39		35.93

ORE OF CLASS L 1.

18	.882	15.88
12	.832	9.98
30		25.86

CLASS P WASTE.

180	.....	......
531		495.52

In explanation of the calculation sheets shown it may be said that the number of the hole and its location by coordinates are set down at the top of the page, as Hole A (E. 100; S. 0.00). In the left-hand column the number of feet of each class of ore or waste, according to the classification mentioned on page 15, is shown in proper relative position, having been taken direct from the assay record of the drill hole (Form 4). The average assay of each class of ore in percentage of copper is shown in the second column.

Now 10 feet of 1 per cent ore and 20 feet of 2 per cent ore would not give 30 feet of 1.5 per cent ore, but 30 feet of 1.667 per cent ore. This correct average is obtained by taking account of the fact that there is twice as much of the rich 2 per cent ore as of the 1 per cent ore. The calculation follows:

Feet of ore.		Per cent copper.		Foot per cent.
10	×	1	=	10
20	×	2	=	40
30				50

Dividing the total foot per cent, 50, by the total feet of ore, 30, gives 1.667 per cent of copper as the average value throughout the 30 feet.

In like manner the number of feet of each class of ore is multiplied by its assay value, and the product is set down under the column marked "Foot per cent." The depth of the hole to the bottom of the lowest ore divided into the total feet per cent would give the average value for that hole. On the right-hand side of the page are segregated the figures for the different classes of ore for the given hole, the number of feet of each class, and its foot per cent.

Book 2, used in the ore calculations, may be called the "triangular-prism book." In its records the ore body is regarded as composed of a series of vertical triangular prisms, and the ore grade of each prism is separately calculated. The first step in this calculation is to take a copy of the drill-hole map and connect, by straight lines, each drill hole with its nearest neighbors, in such a way that the whole ore-bearing area is divided into triangles, with the apices of each triangle, or rather the edges of each triangular prism, marked

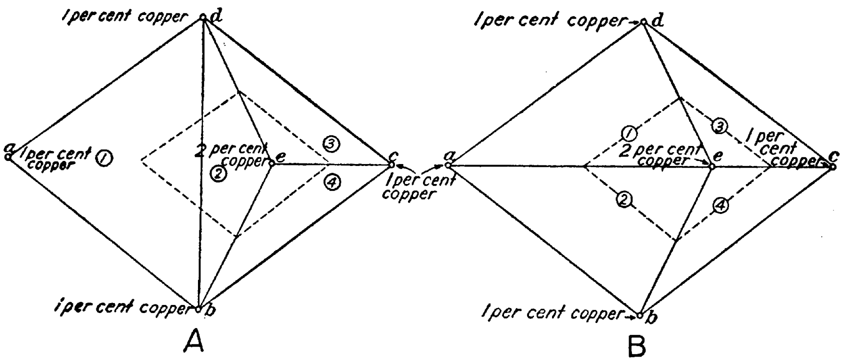


FIGURE 3.—Methods of dividing ore area into triangles. A, incorrect method; B, correct method.

by a drill hole. The ore grade in each triangular prism is thus closely approximated by the average grades found in the three drill holes forming its edges.

There is a right way and a wrong way to lay off these triangles. Suppose five drill holes were placed and showed values about as in figure 3.

Figure 3, A, shows the wrong way to triangulate these holes, if divided in the way shown, triangle *1* would show only 1 per cent of ore, whereas in reality the 2 per cent ore should be calculated as extending halfway from *e* to *a*, as indicated by the dotted lines. Of course, in any particular case, the rich ore might not extend halfway toward *a*, but, according to the law of averages, it should be calculated as though it did, so as to make proper apportionment of the high as well as of the low values. If the drill-hole area were divided as in figure 3, A, triangle *1* would be undervalued and triangle *2* overvalued.

Correct apportionment of the values to each of the triangles is shown in figure 3, B. The data on which to base properly evaluated triangulation are obtained not only from the average assays of each hole, but largely from the geologic information, such as the continuity of the ore values, shown on the cross sections.

When all of the ore-bearing drill holes have been connected by lines so that each hole forms the apex of one or more triangles, each triangle so formed is given a number. On page 1 of the triangle book are found the calculations for triangle 1, as follows:

Specimen page from triangle book.

TRIANGLE 1. AREA, 7,150 SQUARE FEET. HOLES A, B, AND C.

[Data taken from calculations on pages 20, 21, and 22.]

Hole.	Class of ore.												P, feet of waste.
	S.		E.		G.		F.		L 2.		L 1.		
	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	
A.......	120	279.93	39	66.60	15	22.20	66	76.22	21	19.10	66	55.94	193
B.......	57	122.78	51	94.32	39	50.38	51	58.55	57	53.37	66	56.43	207
C.......	45	96.29	81	145.78	75	100.80	81	90.86	39	35.93	30	25.86	180
a[a].......	222	499.00	171	306.70	129	173.38	198	225.63	117	108.40	162	138.23	580
b[b].......	74	166.33	57	102.23	43	57.79	66	75.21	39	36.13	54	46.08	193.33
c[c].......	529,100		407,550		307,450		471,900		278,850		386,100		......
d[d].......	1,189,260		730,945		413,199		537,752		258,330		329,472		......

[a] Line a shows sum of the number of feet and foot per cent of the different classes of ore for the three holes.
[b] Line b shows one-third of a, or the average of the three holes.
[c] Line c shows average thickness multiplied by area, giving cubic feet of each class of ore in prism.
[d] Line d shows average foot per cent multiplied by area giving cubic foot per cent or "volume assay" of each class of ore in prism.

P.
A=193
B=207
C=180
580

$\frac{580}{3}\times 7{,}150=\frac{4{,}147{,}000}{3}=1{,}382{,}333$ cubic feet, or amount of waste contained in triangular prism 1.

The sample of the calculations set forth in the triangle book shows that the area of the triangle and the number of the holes from which its ore content is calculated are set down at the top of the page. Line a shows the total number of feet of ore and the foot per cent of each class of ore in each hole. The average thickness and the average foot per cent of each class for the prism (one-third of a) are entered in line b. Line c shows the average thickness of each class of ore multiplied by the cross-sectional area of the prism. This gives the number of cubic feet—the volume—of a particular class of ore. The d line shows the cubic foot per cent, or the volume assay, for each class of ore in the triangular prism. It is obtained by multiplying the average foot per cent by the cross-sectional area of the prism.

The third book for ore calculations is of the same type as the other two, and may be called the book of ore totals. In it are recorded the volume of ore and the volume assay, or cubic feet per cent, of each class in each triangular prism for each ore body. Let us assume that ore body A contains 100 triangular prisms of ore, each exactly equal to prism 1, already recorded. The ore in such an ore body would be summed up in the book of ore totals somewhat as follows:

Specimen page from book of ore totals.

TOTAL ORE IN ORE BODY A.

Class of ore.	Cubic feet of ore.	Volume assay or cubic feet per cent.	Per cent copper.	Tons of ore, at 13 cubic feet per ton.
S	52,910,000	118,926,000	2.247	4,070,000
E	40,755,000	73,094,500	1.793	3,135,000
G	30,745,000	41,319,900	1.343	2,365,000
F	47,190,000	53,775,200	1.139	3,630,000
L 2	27,885,000	25,833,000	.926	2,145,000
L 1	38,610,000	32,947,200	.853	2,970,000
	238,095,000	345,895,800	1.453	18,315,000

Dividing the volume assay, or cubic feet per cent, of ore by the cubic feet gives the average assay value, in per cent, for each class and for the total ore.

In order to convert the cubic feet of ore into tons, the average specific gravity of the ore body must be determined.[a] If found to be 2.5, then a cubic foot of ore would weigh 2.5 times the weight of a cubic foot of water, or $2.5 \times 62.5 = 156.25$ pounds; thus 13 cubic feet of such ore would weigh approximately a ton. Thirteen cubic feet of Chino ore in the solid and 21 cubic feet of broken ore were found to average practically a ton, and this average was used in the calculation of ore reserves. Dividing the cubic feet of ore by 13 gives the tonnage shown above—18,315,000 tons of ore, containing an average assay value of 1.453 per cent copper, or practically 266,117 tons of copper. The actual amount of this copper that could be recovered would of course depend on the percentage of recovery made by the mill and smelter.

To summarize, holes were drilled through the ore body every 100 or 200 feet, and these were sampled at depth intervals of 3 feet. The assay values from each sample were arranged in order, and grouped into classes as shown in Form 4. The classes were entered in the log book of prospect drill holes, and the drill holes were grouped together in sets of threes, each set of three marking the edges of a vertical triangular prism. The average values for each prism were calculated in the triangular prism book; and finally the total ore for each ore body was summed up in the book of total ore. These books, with all calculations, are preserved with other company records.

[a] For diagram and formula convenient for ore calculations see Mead, W. J., The relation of density, porosity, and moisture to the specific volume of ores: Econ. Geol., vol. 3, June-July, 1908, pp. 319-325.

The results from the book of ore totals were arranged into tables as follows: A table of total ore developed up to March 1, 1912, and the yardage of stripping that would have had to be moved to recover it; a table of the total steam-shovel ore developed up to the same date, and the yardage of stripping that covered it; a table of additional steam-shovel ore developed after March 1, 1912. Of these tables the one covering the amount of steam-shovel ore developed up to the date mentioned follows:

Data on amount of steam-shovel ore developed up to Mar. 1, 1912.

Name of ore body prospected.	Steam-shovel ore.																Stripping.[b]		Ratio of ore to waste.
	Class S.[c]		Class E.[c]		Class G.[c]		Class F.[c]		Class L2.[c]		Class L1.[c]		Class P.[c]		Total tons.	Average assay.	Slope.		
	Tons.	Average assay[a]	Tons.	Average assay.	Tons.	Average assay.	Tons.	Average assay.	Tons.	Average assay.	Tons.	Average assay.	Tons.	Average assay.			Cubic yards broken.	Tons.	
C. Y. B.	3,441,710	2.85	2,617,770	1.67	1,758,932	1.37	2,084,220	1.13	1,034,788	0.95	625,754	0.86	458,342	0.55	12,021,516	1.72	6,527,129	8,392,023	1:0.70
R. T. W.	4,681,412	2.91	3,320,939	1.71	2,219,884	1.37	3,282,042	1.11	1,690,213	.94	1,260,122	.85	916,712	.58	17,371,323	1.68	10,989,870	14,129,833	1:0.81
Hearst	13,324,098	3.41	4,926,392	1.66	4,429,134	1.35	4,110,055	1.10	2,784,595	.94	2,353,135	.84	1,512,191	.61	33,439,599	2.08	21,028,461	27,036,593	1:0.81
Old Fort	31,195	3.30	61,913	1.59	133,116	1.32	76,826	1.12	31,464	.91	6,788	.82	19,616	.48	360,918	1.41	325,180	418,088	1:1.16
Sierra	1,930,258	2.71	4,058,965	1.70	2,938,513	1.35	2,895,046	1.10	1,440,814	.93	994,511	.85	699,634	.63	14,957,741	1.46	13,229,328	17,009,136	1:1.14
Estrella	1,120,984	2.66	1,206,484	1.69	555,521	1.35	858,734	1.10	210,882	.94	193,593	.85	242,991	.51	4,389,190	1.64	9,367,943	12,044,498	1:2.74
Total	24,529,657	3.14	16,192,463	1.68	12,035,100	1.36	13,306,923	1.11	7,192,756	.94	5,433,908	.85	3,849,486	.59	82,540,287	1.81	61,467,911	79,030,171	1:0.96

a Each average assay is expressed as per cent of copper per ton of ore.
b The slope at first was ½ horizontal to 1 vertical, but in 1915, after the stripping had been nearly completed, a slope of ¾ to 1 was adopted.
c For explanation of ore classification, see p. 15.

FORMULAS FOR ESTIMATING ORE.

FORMULA FOR DETERMINING LOWEST GRADE OF ROCK THAT MAY BE CLASSED AS ORE.

The formulas following may be largely credited to Mr. L. E. Foster, assistant superintendent of the Chino company's mine. The authors merely developed a little further his fundamental formula. Formula *1* is for use in determining what grade of rock might be mined as ore under varying conditions of costs and of copper prices. The factors in the equation are explained below. Certain cost figures are given to be substituted in the equations. These figures, however, are being modified by more efficient methods as time goes on.

$$(1) \qquad \frac{M+F_0+m}{AC\times2000\times0.95}+\frac{f_o+S}{D\ 2000\times0.95}+R+E+c=B\frac{100}{105}$$

Simplifying equation *1* we get:

$$(2) \qquad \frac{M+F_0+m}{1900\ C}+\frac{A\ (f_o+S)}{1900\ D}+A(R+E+c)=AB\frac{100}{105}$$

Transposing equation *2* gives:

$$(3) \qquad \frac{M+F_0+m}{1900\ C}=A\left[\frac{100\ B}{105}-\frac{(f_o+S)}{1900\ D}-(R+E+c)\right]$$

Transposing equation *3* gives:

$$(4) \qquad A=\frac{\dfrac{M+F_0+m}{1900\ C}}{\dfrac{100\ B}{105}-\dfrac{(f_o+S)}{1900\ D}-(R+E+c)}$$

Simplifying equation *4* gives:

$$(5) \qquad A=\frac{M+F_0+m}{1900\ C\left[\dfrac{100\ B}{105}-\dfrac{(f_o+S)}{1900\ D}-(R+E+c)\right]}$$

Values substituted for factors in foregoing equations.

Factor.	Definition of factors.	Value.
M	=Mining costs per dry ton of ore	\$0.4683
F_0	=Freight from mine to mill per dry ton of ore	.1111
m	=Milling costs per dry ton of ore	.5776
f_o	=Freight on concentrates per dry ton from mill to smelter	1.1191
S	=Smelting charges per dry ton of concentrates	6.352
R	=Refining and delivering charges per pound of copper paid for	.0106
E	=Export freight and insurance charges per pound of copper paid for	.0014
c	=Commission on sales	$\frac{1}{100}B$
B	=Value per pound of copper	.13
C	=Per cent of extraction by the mill	65
D	=Per cent of copper in the concentrates	25
A	=Per cent of copper per dry ton of ore.	

The constant 2,000 stands for the 2,000 pounds in each ton of ore. The constant 0. 95 stands for the 95 per cent of the copper content, or the net copper that the smelter actually pays for. The dry ton of ore figured in these calculations is the ore with the weight of its moisture content deducted. The moisture content in the ore averages about 10 per cent.

About eight months elapse from the time the ore is mined until the copper is sold. During this period capital is tied up in the metal. Interest on this capital is allowed at the rate of 5 per cent for the eight months. This interest factor is introduced into the equation by multiplying B, the value of copper, by $\frac{100}{105}$, a fraction that represents the lessened value of money by reason of the interest deduction.

In the expression $AC \times 2000 \times 0.95$, used as denominator in the first member of equation *1*, 2,000 pounds of ore multiplied by A, the percentage of copper contained in the ore expressed decimally, multiplied by C, the percentage of copper that is saved by the mill, multiplied by 0.95, the percentage of the copper from the mill that is paid for by the smelter, gives the net amount of copper in pounds recovered from each ton of ore. The expression $\frac{M+F_0+m}{AC \times 2000 \times 0.95}$, means simply the net number of pounds of copper recovered, divided into the cost of recovering it. In the same way, $\frac{f_0+S}{2000D \times 0.95}$ means the net copper recovered from the concentrates at the smelter divided into the cost of recovering it. The whole equation then represents the total net cost of copper with the different factors expressed as mathematical quantities, equated to $B\frac{100}{105}$, which is the value of the copper lessened by an interest charge of 5 per cent. As the equation stands, nothing is charged for the interest lost on the deferred profits. In general, application of the formula interest on the capital tied up in the enterprise should be charged.

By substituting in equation *5* the different costs selected as being well within the factor of safety for the first estimate of the ore values, and simplifying, we get:

$$(6) \qquad A=\frac{1.157}{1235\ (0.1238-0.0157-0.0133)}=\frac{1.157}{1235\ (0.0948)}=\frac{1.157}{117.078}$$

$$=0.00988, \text{ or } 0.988 \text{ per cent.}$$

Now by eliminating M, the cost of mining, from equation *5* and then solving for A, we find what low-grade material, which had to be removed in order to get at richer ore, could profitably be sent to the mill rather than to the waste dump. The figures follow.

$$(7) \qquad A=\frac{0.6887}{117.0533}=0.005882, \text{ or } 0.5882 \text{ per cent.}$$

Equation *7* is equation *6* with the M values—the cost of mining—eliminated. Some rock that contains between 0.5 and 0.8 per cent of copper has to be moved to get at good ore; the cost of sending it to the dump is about the same as that of sending it to the mill. The question to be solved under such conditions is: Having this material mined and loaded because of the necessity of getting it out of the way, what content of copper will decide whether it shall be sent to the mill or to the waste dump. Equation *7* gives the answer—all rock that contains more than 0.5882 per cent of copper should, under the conditions given, go to the mill, and all rock that contains a lesser proportion of copper should go to the waste dump.

FORMULA TO DETERMINE WHETHER ORE SHOULD BE SMELTED IN THE CRUDE, OR MILLED AND THE CONCENTRATES SMELTED.

The formula used to determine what grade of ore from the mine should be shipped direct to the smelter, or to show where the cleavage between smelting ore and concentrating ore should be, is discussed below. When the total cost per pound of copper paid for is the same, whether the ore be smelted in the crude or milled and the concentrates smelted, then the following equation holds:

$$\frac{M+F_s+S_0}{Ay}+R+E+c=\frac{M+F_0+m}{ACy}+\frac{f_c+S_c}{Dy}+R+E+c. \tag{8}$$

The values substituted in this formula are in general fairly representative of the costs at Santa Rita up to the middle of 1914. Since then improvements have been made.

Values substituted for factors in equation 8.

Factor.	Definition of factor.	Value.
M	=Mining cost per dry ton of ore	\$0.4683
F_s	=Freight on crude ore from mine to smelter per dry ton	1.1111
F_0	=Freight on ore from mine to mill per dry ton	.1111
m	=Milling costs per dry ton of ore	.5776
f_0	=Freight on concentrates, mill to smelter, per dry ton	1.1191
S_0	=Smelting charges for crude ore per dry ton	7.25
S_c	=Smelting charge per dry ton of concentrates	6.352
R	=Refining and delivering per pound of copper paid for	.0106
E	=Export freight, insurance, etc., per pound of copper paid for	.0014
c	=Commission	$\frac{1}{100}B$
B	=Value per pound of copper.	
C	=Percentage of extraction by mill.	
D	=Percentage of copper in the concentrates.	
A	=Percentage of copper per ton of ore (expressed decimally).	
y	=(2000 multiplied by 0.95)=1900.	
0.95	=Percentage of the copper content which is recovered by the smelter (expressed decimally).	

Multiplying equation *8* by Ay we get:

$$(9)\qquad M+F_s+S_o+Ay(R+E+c)=\frac{M+F_o+m}{C}+\frac{A(f_o+S_o)}{D}+Ay(R+E+c)$$

Transposing:

$$(10)\qquad \frac{A(f_o+S_o)}{D}+Ay(R+E+c)-Ay(R+E+c)=M+F_s+S_o-\left(\frac{M+F_o+m}{C}\right)$$

Hence:

$$(11)\qquad A=\frac{\frac{M+F_s+S_o}{1}-\frac{(M+F_o+m)}{C}}{\frac{f_c+S_o}{D}}=\frac{D[C(M+F_s+S_o)-(M+F_o+m)]}{C(f_c+S_o)}$$

If the value of the mill recovery (C) and the percentage of copper the concentrates contain (D) are known, these values can be substituted in equation *11* and the value of A obtained. Below are a few values of A for given values of C and D.

Values of A for given values of C and D.

C	D	A	C	D	A	C	D	A
P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
60	20	18.47	60	25	23.09	60	30	27.71
65	20	18.87	65	25	23.59	65	30	28.31
70	20	19.21	70	25	24.01	70	30	28.82

In other words, with C at 60 per cent and D at 20 per cent, the rock would have to contain 18.47 per cent of copper before the cost of recovering that copper would be the same whether the ore was smelted in the crude or milled and the concentrates smelted. For given values of C and D, the corresponding values for A represent critical values below which the ore should be milled and the concentrates smelted and above which it should be shipped direct to the smelter.

When the profits on every dry ton of ore are equal whether the ore is smelted in the crude or milled and the concentrates smelted, then the following is true: The value of the copper recovered from each dry ton of ore by direct smelting (VCu_s) minus the total cost of recovering it (C_s) is equal to the value of the copper recovered from each dry ton of ore by milling and smelting the concentrates (VCu_{m+s}) minus the cost of all the expenses of treating and handling each ton of ore (C_{m+s}). Putting this in equational form:

(*a*) $VCu_s-C_s=P_s=$profits from direct smelting.

(*b*) $VCu_{m+s}-C_{m+s}=P_{m+s}=$profits from milling and smelting the concentrates.

But if $P_s=P_{m+s}$, then

(*c*) $VCu_s-C_s=VCu_{m+s}-C_{m+s}$.

Now, substituting in equation *c* the mathematical expressions for the value of the net copper extracted from each ton treated and the expressions for the cost of treating by both methods, selected from equation *9*, we get:

$$(d)\qquad \begin{gathered} ABy-[(M+F_s+S_o)+Ay(R+E+c)]= \\ ABCy-[(M+F_o+m)+\frac{(f_e+S_c)}{D}AC+(R+E+c)ACy] \end{gathered}$$

By substitution of the more fixed costs, etc., we get:

$$(e)\qquad 1900AB-8.8294-22.8A-19AB=1900ABC-1.157-\frac{7.4711AC}{D}-22.8AC-19ABC$$

Simplifying:

$$(f)\qquad 1881AB-22.8A=1881ABC+7.6724-\frac{7.4711AC}{D}-22.8AC$$

Factoring and transposing, we get:

$$1881AB(1-C)=7.6724+22.8A(1-C)-\frac{7.4711AC}{D}$$

Transposing, we get:

$$1881AB(1-C)-22.8A(1-C)=7.6724-\frac{7.4711AC}{D}$$

Multiplying through by $-\frac{1}{A}$ and transposing, we have:

$$\frac{7.6724}{A}=1881B(1-C)-22.8(1-C)+\frac{7.4711C}{D}$$

Factoring and solving for A, we get:

$$\frac{7.6724}{A}=(1881B-22.8)(1-C)+\frac{7.4711C}{D}$$

Therefore:

$$(g)\qquad A=\frac{7.6724}{(1881B-22.8)(1-C)+\frac{7.4711C}{D}}$$

Now, by substituting values for B, C, and D, we can solve for A. The value of A thus obtained will be the critical grade or percentage of copper per dry ton of ore, at which the profits will be equal whether the ore be smelted in the crude or milled and the concentrates smelted. If we let D = 20 per cent, C = 60 per cent, and B = 12 cents, we can substitute these values in formula *g* and solve for A, thus:

$$(h)\qquad A=\frac{7.6724}{(1881\times 0.12-22.8)(1-0.60)+\frac{7.4711\times 0.60}{0.20}}=0.0741, \text{ or } 7.41 \text{ per cent.}$$

If the copper content be greater than 7.41 per cent under the conditions given, the ore should be sent direct to the smelter; if less, it should be sent to the mill.

From the formula represented by equation h a table may be made showing the values of A for several values of B, C, and D, thus·

Values of A for several values of B, C, and D, in equation h.

D=20 per cent; C=60 per cent. B	A	D=20 per cent; C=65 per cent. B	A	D=20 per cent; C=70 per cent. B	A
12 cents	7.41 per cent	12 cents	8.05 per cent	12 cents	8.82 per cent
14 cents	6.47 per cent	14 cents	7.07 per cent	14 cents	7.80 per cent

By giving other values to D, as well as by giving B and C a large range, the table can be made much more extensive. In all cases the value of A thus obtained is the approximate grade of ore at which profits, per dry tone of ore, equalize, whether the treatment is direct smelting or milling and smelting the concentrates.

The expression $\frac{M+F_s+S_o}{Ay}+R+E+c$, taken from formula *8* (p. 30), represents the cost per pound of the copper sold when the ore is treated by direct smelting. By substituting values as given on page 30, simplifying, and rearranging, we get:

$$\frac{0.004647}{A}+0.012+\frac{B}{100}=0.012+\frac{0.004647}{A}+\frac{B}{100}=\text{cost per pound of copper sold when the ore is smelted direct.}$$

Now, by substituting values for A and B, and solving, we get the cost of copper per pound. Subtracting this figure from the selling price gives the profit per pound. If considerable range be given to the values of A and B, a table may be made somewhat as follows:

Cost and profit per pound of copper paid for when ore is smelted in the crude.

Price of copper per pound.		Per cent of copper in the ore. 4	5	6	7	8	9	10
Cents.								
12	Cost	$0.129	$0.1061	$0.0907	$0.0796	$0.0713	$0.0648	$0.0597
	Profit	[a].009	.0139	.0293	.0404	.0487	.0552	.0603
14	Cost	.1296	.1063	.0909	.0798	.0715	.0650	.0599
	Profit	.0104	.0337	.0491	.0602	.0685	.0750	.0801
16	Cost	.1298	.1065	.0911	.0800	.0717	0652	.0601
	Profit	.0302	.0535	.0689	.0800	.0883	.0948	.0999
18	Cost	.1300	.1067	.0913	.0801	.0719	.0654	.0603
	Profit	.0500	.0733	.0887	.0999	.1081	.1146	.1197
20	Cost	.1302	.1069	.0915	.0803	.0721	.0656	.0605
	Profit	.0698	.0931	.1085	.1197	.1279	.1344	.1395

[a] Loss of 9 mills per pound of copper.

By substituting 0.05 for A, or 5 per cent, and 0.12 for B, or 12 cents per pound, and solving, we get $0.1061 as the cost per pound of copper by direct smelting of 5 per cent ore, as shown in the above table; subtracting this value from the price of copper, $0.12, we get $0.0139 as the profit per pound. An extensive table may be compiled in this way.

In like manner, from formula *d* (p. 32) we get the expression $[M+F_s+S_o+Ay(R+E+c)]$, which stands for the total cost of mining and treating each dry ton of ore when the process is direct smelting. By substitution of the values given on page 30 the expression reduces to $8.8294+22.8A+19AB$, the cost per dry ton of ore treated. Substituting values and solving gives the cost per ton. By subtracting the figure for cost from the product of the expression $2000\times0.95\times0.05\times0.12$, which represents the net pounds of copper recovered from each ton of 5 per cent ore smelted, multiplied by its selling price, we get the profit per ton treated. A table as extensive as may be necessary can be made thus:

Cost and profit per dry ton (2,000 pounds) of ore smelted in the crude.

Price of copper per pound.		Per cent of copper in the ore.						
		4	5	6	7	8	9	10
Cents.								
12	Cost	$9.833	$10.083	$10.334	$10.585	$10.836	$11.087	$11.337
	Profit	[a].713	1.317	3.346	5.375	7.402	9.433	11.463
14	Cost	9.848	10.102	10.357	10.612	10.866	11.121	11.375
	Profit	.792	3.198	5.603	8.008	10.414	12.819	15.225
16	Cost	9.863	10.121	10.380	10.638	10.897	11.155	11.413
	Profit	2.297	5.079	7.860	10.642	13.425	16.205	18.987
18	Cost	9.878	10.140	10.403	10.665	10.927	11.189	11.451
	Profit	3.802	6.960	10.117	13.264	16.438	19.591	22.750
20	Cost	9.893	10.159	10.426	10.692	10.957	11.223	11.489
	Profit	5.307	8.841	12.374	15.897	19.450	22.977	26.512

[a] Represents a loss of 71.3 cents per ton of ore treated.

By giving a larger range of values to A and B, this table can be greatly extended.

From formula *8* (p. 30) we get the expression $\frac{(M+F_o+m)}{ACy}+\frac{(f_c+S_c)}{Dy}+(R+E+c)$, which represents the cost per pound of the copper actually marketed where the ore is milled and the concentrates smelted. Now, substituting cost values given on page 30 in this expression we get $\frac{1.157}{1900\ AC}+\frac{7.4711}{1900\ D}+0.012+\frac{B}{100}=$cost per pound of copper sold when the treatment is milling and smelting the concentrates. By giving A, B, C, and D different values a whole series of tables may be compiled showing the cost and the profit per pound of copper sold.

In like manner, from formula *d* (p. 32), the expression $[M+F_o+m+\left(\frac{f_c+S_c}{D}\right)AC+(R+E+c)ACy]$ represents the cost of treating each dry ton of ore by milling and smelting the concentrates. Substituting costs gives $1.157+\frac{7.4711AC}{D}+22.8AC+19ABC$. By substituting a range of values for A, B, C, and D an extensive series of tables can be compiled showing the cost and profit on each dry ton of ore by this method of treatment.

SAMPLING AND ASSAYING METHODS.

PROSPECT-HOLE SAMPLES.

In the development of "ore in sight" by drilling, samples were taken about every 5 feet of hole drilled in the relatively barren material near the surface; but in all other material they were taken every 3 feet. The mixture of water and drill-hole cuttings that composed each sample was run through what was, in effect, a Jones sampler, and quartered down so that sample and washings constituted about a gallon. This ultimate sample was evaporated to dryness on a specially arranged home-made furnace, consisting of a fire box and

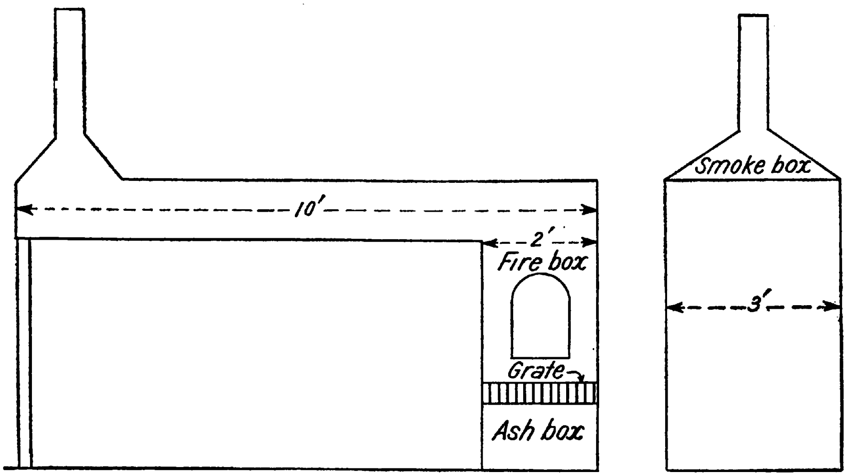


FIGURE 4.—Side and end views of improvised furnace for evaporating drill samples of ore. Can be made by blacksmith out of $\frac{3}{16}$-inch or $\frac{1}{4}$-inch boiler plate. Smoke box of heavy sheet iron; smokestack, 6 feet of ordinary 6-inch stovepipe; evaporating pans, about 18 by 18 inches and 6 inches high, made of heavy sheet iron.

a heating chamber built from $\frac{1}{8}$-inch boiler plate, somewhat as shown in figure 4.

Some of the rejected part of each sample was carefully panned and the concentrates examined to get information regarding the character of the rock through which the hole was passing. When notes covering these data had been made, the panned material was put into paper envelopes and filed away.

The dried sample was tagged with the number of the hole and the depth at which it was taken, and sent to the sample room. There, if necessary, it was run through a small gyratory crusher, the largest pieces from which were about one-fourth of an inch in diameter; then after careful rolling on a mixing cloth it was reduced on a Jones sampler to about 150 or 200 grams. After that came grinding in a Braun disk pulverizer, followed by more rolling on a mixing sheet. Screening through a 150-mesh to 200-mesh screen removed any metallic copper, and this, if present, was weighed and figured into the ore values as 90 per cent pure. The pulp was then sent to the assay shop, where the determination was made, mostly by Lowe's potassium permanganate method. Each assay was given a serial number in addition to the number of the hole and the depth below the surface from which it had been taken, and the part of the pulp not required in the assay was put into a paper envelope and carefully filed away.

GOLD ASSAYS.

Since the summer of 1914 a large number of gold assays have been made. As a rule, the amount of gold present is small, so that charges of 4 assay tons [a] have had to be used to get a gold bead of a size that might be weighed conveniently. The smelter pays for the gold in the concentrates when its value is 60 cents or more per ton. Assuming a concentration of 10 into 1, and a saving of 70 per cent in the mill, a gold content of only 9 cents per ton in the ore would give a revenue from the smelter of 63 cents per ton of concentrates. Obviously the presence of even small amounts of gold favors the practice of making the concentrates as clean as possible, and during the winter of 1914–15 important progress was made in this direction at the Hurley mill.

BLAST-HOLE AND PIT SAMPLES.

Where more detailed information is necessary as mining proceeds, samples are taken from the blast-holes. Such sampling shows with considerable exactness the value of the rock in place ahead of the steam shovels. Samples are taken—a small shovelful here and there—from the broken rock in front of each steam shovel. These are called pit samples and may be gathered every hour or so. Rough determinations, by the colorimetric method, are made on these, as they are only for the purpose of showing whether the rock is running about as it has been indicated by the cross sections and the estimates. Samples are also taken from every car of ore loaded, and these are checked against other assays of the ore body and against mill returns, tailings assays, etc. Formerly, samples were also taken by carefully cutting across the face of many of the benches with a small pick or a hammer and moil; a ladder was used to get at the upper part of the face. Few bench samples are taken now. Moisture determinations were made at various times on the ore and rock.

STEAM-SHOVEL SAMPLES.

Steam-shovel samples are occasionally taken and consist of a "grab" shovelful from each steam-shovel dipperful after it has been put on the car.

---

[a] An assay ton contains 29.166 grams.

To summarize, all four kinds of samples are currently taken at Santa Rita, as follows:

1. Prospect-hole samples.
2. Pit samples.
3. Car and steam-shovel samples.
4. Blast-hole samples.

Moisture determinations are made on many of the samples. On the average, about 2,400 assays and determinations are made per month. The assay staff consists of an assayer, one assistant, and two men in the sample shop. The latter attend to the grinding of the samples, preparing the pulp, etc.

SAMPLE SHOP AND ASSAY SHOP.

The sample shop, a one-story structure, having a floor space of about 500 square feet, is fitted with the following appliances: 1 Blake jaw crusher, with opening 10 by 5 inches, crushing to 2 inches, to which a revolving sampler is attached; a small Blake crusher, with an opening 3 by 4 inches, crushing to ¼ inch; a gyratory crusher giving a maximum size of ¼ inch; a Braun disk pulverizer; a gasoline stove for drying, and many other accessories. Power is furnished by an Allis-Chalmers 7.5 horsepower motor.

The assay shop, with almost 1,000 square feet of floor space, contains a balance room, a laboratory for wet assays, and a furnace room. In the latter are a smelting and a cupelling furnace, each fired by gasoline, vaporized under about 50 pounds of air pressure, and blown in through Cary hydrocarbon burners.

NUMBER AND COST OF ASSAYS.

The following table gives some idea of the number of assays made per month and the cost of making them:

Cost of sampling and assaying at Santa Rita.

Month.	Number of assays.	Cost per assay.	Mining costs.			Stripping costs.		
			Labor.	Supplies.	Cost per ton.	Labor.	Supplies.	Cost per cubic yd.
1914.								
January	3,610	$0.46	$610.73	$106.69	$0.00366	$802.72	$128.39	$0.00302
February	2,159	.80	575.72	112.69	.00365	882.56	149.88	.00365
March	3,001	.66	623.36	151.46	.00349	997.96	193.41	.00345
August	[a] 1,335	.86	370.84	182.67	.00454	408.74	182.38	.00317
September	[b] 1,475	.42	207.90	58.03	.00255	276.14	78.74	.00223

[a] 66 of these assays were for gold.
[b] 130 of these assays were for gold.

As is indicated by the table, the figures cover ore and waste. The ore assay costs are charged against mining, and the costs of assaying waste are charged against stripping. The cost of the labor for assaying and sampling and the cost of the supplies used in connection with the assays are kept separate, as they are in all the other accounts.

The assay cost is distributed per ton according to the tons of ore mined, and per cubic yard according to the cubic yards of waste moved, as shown by the table.

The permanganate method is used for prospect drill holes and for car samples. The colorimetric method is used for pit and shovel samples where absolute accuracy is not considered necessary. The determination of iron in many samples is made as it is indicative of the degree to which the ore will concentrate.

The wide variation in the cost of the assays is caused by the wide variation in the amount of crushing and mixing to be done on them—whether permanganate or colorimetric determinations or fire assays are made; the relative number of each, and the total number of all assays for the month; the amount and cost of the chemicals used, etc.

FORMS USED IN ASSAY SHOP.

Forms 6 to 9 following are used in the assay shop and are convenient aids to bookkeeping.

FORM 6.—*Form of general assay certificate issued by assayer.*[a]

ASSAY CERTIFICATE.

Per ton of 2,000 lbs. SANTA RITA, N. MEX., ..., 191...

Sample No.	Date.	S. S. No.	Ore body.	Bench.	Description.	% Silica.	% Sulphur.	% Iron.	% Copper.	Oz. gold.	Oz. silver.				

...., *Assayer.*

[a] Size of form as used by operating company, 5½ by 7½ inches.

FORM 7.—*Form used by assayer to report on steam-shovel samples.*[a]

STEAM-SHOVEL SAMPLES—LABORATORY REPORT.

Sample No.	S. S. No.	Date.	Shift.	Cars waste.	Cars ore.	Wt. of met.	Wt. of pulp.	% Met. Cu.	% Perm. Cu.	% Cu. total.	

[a] Samples consisted of a small shovelful taken from each steam shovelful that went onto a car. Samples of this kind are not now usually taken. Form used by operating company is 6 inches wide by 9 inches long.

FORM 8.—*Form for presenting comparative data regarding steam-shovel samples.*[a]

COMPARATIVE STATEMENT—STEAM-SHOVEL SAMPLES.

DAY SHIFT....................., 191..

Steam Shovel No...................

.... Ore body.

............................Bench.

Time.	Pit sample % Cu.	Mined as—		Bench sample—		Mined as—	
		Ore.	Waste.	No.	% Cu.	Ore.	Waste.
Average % Cu. Car sample % Cu. Car sample % Cu. Ore shipped to mill.							

SPECIAL PIT SAMPLES.

Time.	Taken from.	Ore % Cu.	Waste % Cu.
Remarks:			

[a] Only one form is shown; the record used by the operating company has five of these forms (3¾ by 8¾ inches) on a sheet, the sheet with margins being 10 by 20½ inches. Each sheet, bound through two holes in the left margin, is kept in a heavy binder.

FORM 9.—*Form used in making daily report on mine samples.*[a]

DAILY REPORT OF MINE SAMPLES.

....................................19....

Mine sample number.	Description.			Per cent copper.			Remarks.
	Mine.	Level.	Working.	Pulp.	Met.	Total.	

....

[a] Size of form used by operating company, 8¼ inches wide by 10¾ inches long.

METHOD OF SAMPLING OLD DUMPS.

Sampling on the present property, on an extensive scale, was first begun by Mr. John M. Sully, in 1906. In gathering samples from the large dumps left from many thousands of feet of old workings, he used the following method:

The dumps were laid off into 25-foot squares, with levels run to the corners of each square, to determine its elevation. The contour of the ground on which the dump rested was estimated as nearly as possible, from the general surface configuration of the land. From the surface elevation and the probable elevation of the bottom of the dump, the yardage in each square prism was estimated. Each square, as laid off on the dump surface, was given a number, and about 4 holes were dug into it, as deep as could be conveniently made with a long-handled shovel. Every second shovelful was retained; the sample thus gathered was rolled and quartered on a canvas and put into a tagged canvas sack. Mr. Sully gathered 1,143 samples from the old dumps, and a total of 8,190 from the dumps and the old workings.

DETAILS OF MINING IN SANTA RITA DISTRICT.

GENERAL ORGANIZATION OF OPERATING COMPANY.

In opening an open-cut mine on an extensive scale, the initial cost is heavy. The equipment necessary is expensive, including steam shovels, locomotives, and cars, and railroad track and a water-supply system must be constructed, to say nothing of the concentrating plant to handle vast tonnages of ore, the power plant to furnish the motive force, and the machine and other shops to keep the whole vast plant in repair.

Any large and successful organization is developed slowly. The factor that wins in business or in sport is practically always well-organized "team work," or efficiency of production. If a mistake occurs the organization must be such that the responsibility for the mistake can be traced to its source at once, and the right remedy be quickly applied in the right place. The management of the Chino company has been successful in building up an efficient and loyal organization, one illustrating the benefits that may be derived from the application of science and of business methods to the solution of mining problems. The company's organization chart, or "flow sheet" of responsibility, as worked out by Mr. L. E. Foster, the assistant superintendent, is shown in Plate IV.

MINING AND ESTIMATION OF ORE BY BENCHES.

In stripping and mining, each steam shovel took off a slice of ground about 25 feet wide, and as the work progressed terraces or benches 100 feet or more wide were formed around the sides of the great open cuts (Pl. V, *A*). It was found that where the bench was less than 30 feet high, the material could not be excavated as economically as where it was, say, 50 feet or more high.

There were three chief factors that regulated the height of each bench: (1) If a bank stood fairly steep for some time and then caved, it was found best to have the bench not more than 30 feet high. The caving of 30-foot banks caused little delay or damage to the shovels working below them, whereas the caving of higher banks often caused serious trouble. Where a bank caved slowly and kept gradually coming to the shovel at about the same general slope, it could be worked in benches of 100 feet or more if necessary. (2) If the waste material on top of the ore was, say, only 15 feet thick, then, of course, the bench at that place was only 15 feet high. On the other hand, if the overburden was only, say, 1 or 2 feet thick, it was probably found cheaper to let it go in with the ore without stripping. (3) Where the drill holes indicated considerable relatively barren rock between upper and lower ore masses, then, as far as possible, the benches were planned so that the top ore was taken off the waste rock in one cut or bench, and the waste rock off the lower ore in the second bench, and the lower ore was taken out clean in a third bench. If the ore bodies and waste mass were thick enough, they were generally taken out in two or more separate benches. Barren areas of considerable size were also considered in planning the benches.

In other words, benches and tracks were planned (see Pl. III) to conform, as nearly as practicable, to the shape of the ore body, so that ore could be separated from waste and mined as economically as possible. In general, however, it was found best to have the benches about 50 feet apart; that is, one bench about 50 feet above the other.

The different classes of ore and the waste were calculated for each level, or bench, in order to have convenient information as to just what production should be expected from each.

Tables were also made to show the ore tonnage by benches; that is, the tonnage of the first bench plus the tonnage of the third bench, etc. Such tables permit convenient comparison of what has been taken out from the different benches plus the material calculated to remain with the total ore for each bench, etc. These bench calculations were made from the same data and by the same general method as the triangular-prism estimates. Summing up the ore classes, calculated by benches, served as a sort of check on the other calculations of total ore.

BLAST-HOLE DRILLING.

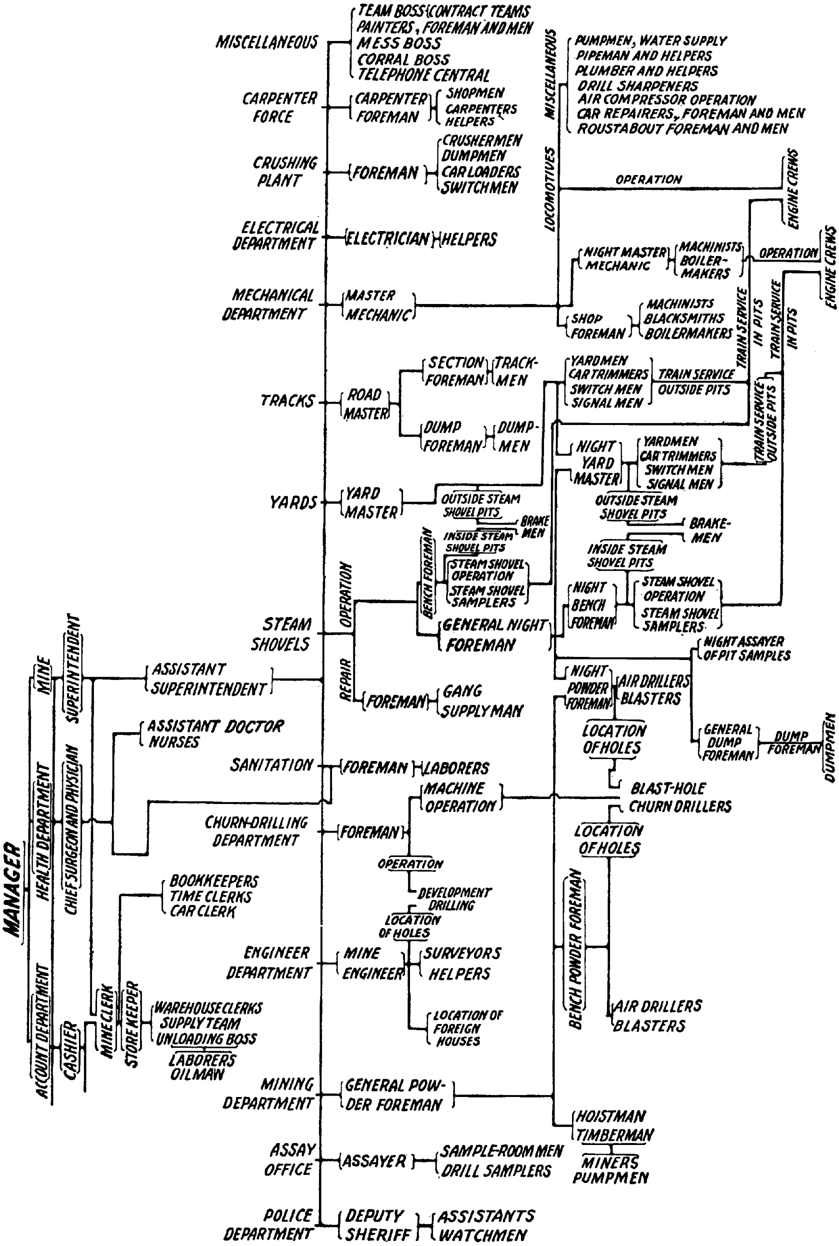
The blast-hole drilling at Santa Rita aggregates several linear miles per month. Most of it is done by churn drills. However, toe holes at the foot of slopes are locally necessary, and these, together with the shallow, levelling-off bottom holes occasionally needed in front of steam shovels, are put down with compressed-air tripod drills. Small hammer drills have been used, to some extent, for drilling short holes to blast bowlders too large for the steam shovels to handle. However, their use has been practically discontinued, because it is generally cheaper to use more powder and to "dobe" or "bulldoze" the rock than to keep a shovel idle while it is being drilled.

TYPES OF CHURN DRILLS USED.

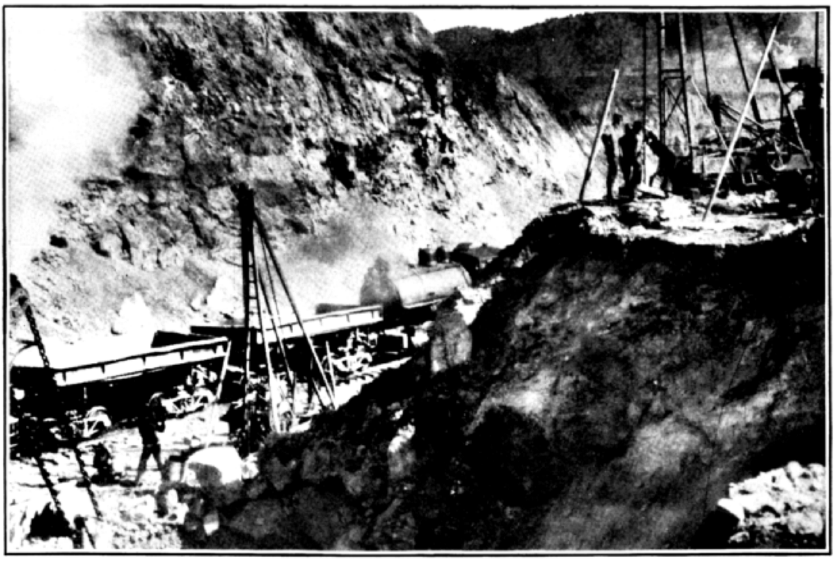
The types of churn drill mostly used at Santa Rita are the No. 12 and the No. 14 Cyclone drills for big blast holes. They are fitted with Cook 12-horsepower vertical gasoline motors, have traction gearing, and use 4-inch to 8-inch bits. Gas-engine oil, instead of gasoline, is used, and the consumption averages about 6½ gallons per 10-hour shift. The speed of drilling varies widely with variation in the hardness of the rock; however, the general average for blast holes not more than 65 feet deep is said to be about 20 to 30 feet per 10-hour shift. The crew of each machine consists of a driller and a helper; these two men can move the drill, even over rough ground, by turning on the traction gearing.

Each drill is fitted with equipment that enables the driller to sharpen his own bits. In general, the rock is soft enough so that one to two sharpenings per shift suffice. The blast holes are sampled if special information from particular localities is needed.

A few of the older steam-driven churn drills are now operated by air, piped from the compressor. In general, they are not as satisfactory as the oil-motor machines. There have been as many as 18 churn drills, both prospect and blast-hole, at work on the property at the same time.

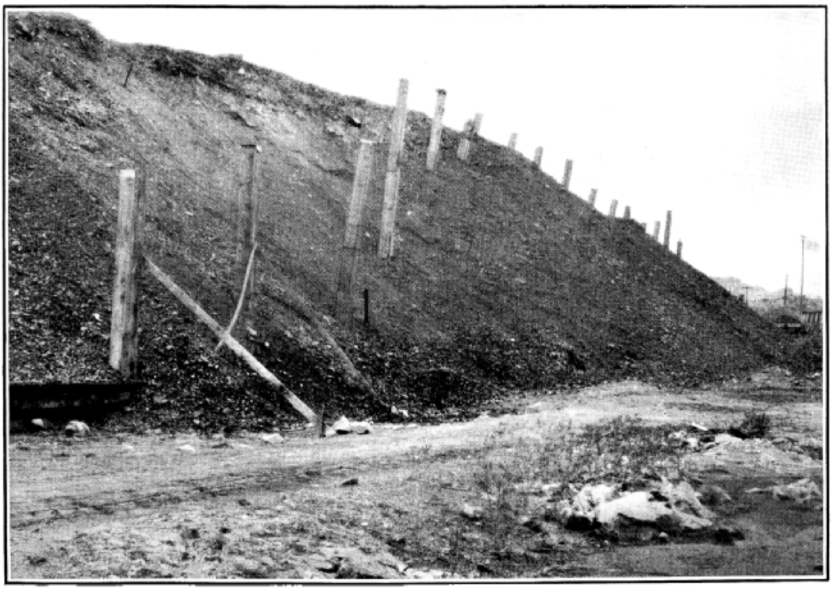


ORGANIZATION CHART OF THE CHINO COPPER CO.



A. VIEW OF A BENCH OR TERRACE OF ORE, SHOWING METHOD OF MINING.

On the right, drills are at work on a bench just ahead of a steam shovel. The crain of the shovel can be seen at the left. A train of cars is on the loading track below the bench that is being drilled.



B. PILE OF COAL CONTAINING WOODEN VENTILATING STACKS TO PREVENT SPONTANEOUS COMBUSTION.

The hydrothermally altered and considerably sheared character of the ore-containing rock facilitates the use of churn drills and is favorable for low drilling costs. Little trouble is encountered in putting down blast holes. Even in cutting across sloping sheared zones, a little care generally guards against much deviation from the vertical.

OPERATION AND EFFICIENCY OF BLAST-HOLE CHURN DRILLS.

Cyclone churn drills, models Nos. 12 and 14 for deep blast holes, are used to put down blast holes. Records of the performance of these are kept and constitute valuable information on the efficiency of churn drilling in general. Records for two typical months, January and August, 1914, are given below:

Record of blast-hole churn drilling for January, 1914.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
DAY SHIFT.							
				Per cent.	Per cent.	Per cent.	
1	630	280	2.25	72.8	19.3	7.9	3.09
2	876	310	2.83	61.9	25.2	12.9	4.56
10	1,235	327	3.78	84.1	11.9	4.0	4.49
11	974	310	3.14	75.2	16.1	8.7	4.13
12	1,212	310	3.91	86.8	8.4	4.8	4.51
13	941	322	2.92	73.9	15.8	10.3	3.95
14	924	311	2.97	78.5	17.0	4.5	3.79
15	952	310	3.07	79.0	20.0	1.0	3.88
NIGHT SHIFT.							
1	670	260	2.58	86.5	13.1	0.4	2.98
2	1,144	298	3.84	84.2	13.8	2.0	4.56
10	1,272	321	3.95	88.2	7.8	4.0	4.50
11	928	266	3.49	89.9	8.6	1.5	3.88
12	1,244	290	4.29	98.0	1.7	.3	4.38
13	1,000	329	3.04	83.6	15.8	.6	3.64
14	953	275	3.47	90.5	9.5	....	3.83
15	1,151	310	3.71	90.3	9.4	.3	4.12
TOTAL.							
1	1,300	540	2.41	79.4	16.3	4.3	3.03
2	2,020	608	3.32	72.8	19.6	7.6	4.56
10	2,507	648	3.87	86.1	9.9	4.0	4.49
11	1,902	576	3.30	81.9	12.7	5.4	4.03
12	2,456	600	4.09	92.2	5.1	2.7	4.44
13	1,941	651	2.98	78.8	15.8	5.4	3.78
14	1,877	586	3.20	84.1	13.5	2.4	3.81
15	2,103	620	3.39	84.7	14.7	.6	4.01
Total for month	16,106	4,829	3.34	82.5	13.4	4.1	4.04

Record of blast-hole churn drilling for August, 1914.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
DAY SHIFT.							
				Per cent.	*Per cent.*	*Per cent.*	
1	725	186	3.90	95.2	4.8	....	4.09
2	1,073	235	4.57	95.3	4.7	....	4.79
10	1,049	244	4.30	91.8	2.0	6.2	4.68
11	1,082	256	4.23	92.6	4.7	2.7	4.56
12	788	187	4.22	89.8	7.0	3.2	4.69
13	814	232	3.51	83.2	2.1	14.7	4.22
14	383	185	2.07	75.1	18.4	6.5	2.75
15	963	246	3.91	86.6	8.5	4.9	4.52
16	988	246	4.02	94.3	2.0	3.7	4.26
NIGHT SHIFT.							
1	....	....	....	....	....	....	....
2	125	30	4.17	100.0	....	....	4.17
10	477	129	3.76	97.7	0.8	1.5	3.79
11	82	39	2.10	74.3	23.1	2.6	2.83
12	182	44	4.14	91.0	4.5	4.5	4.56
13	304	81	3.75	95.1	4.9	....	3.95
14	75	39	1.92	69.2	28.2	2.6	2.78
15	....	....	....	....	....	....	....
16	352	94	3.74	95.7	4.3	....	3.92
TOTAL.							
1	725	186	3.90	95.2	4.8	....	4.09
2	1,198	265	4.52	95.8	4.2	....	4.72
10	1,526	373	4.09	93.8	1.6	4.6	4.36
11	1,164	295	3.95	90.1	7.2	2.7	4.38
12	970	231	4.20	90.0	6.5	3.5	4.67
13	1,118	313	3.57	86.2	2.9	10.9	4.14
14	458	224	2.04	74.0	20.2	5.8	2.76
15	963	246	3.91	86.6	8.5	4.9	4.52
16	1,340	340	3.94	94.6	2.7	2.7	4.16
Total for month	9,489	2,473	3.84	90.0	5.9	4.1	4.26

COSTS OF BLAST-HOLE CHURN DRILLING.

The cost of drilling blast holes with churn drills are fairly well illustrated by the figures for the representative months of January and August, 1914, as given in the tabulation following:

Record of costs of blast-hole churn drilling.

AUGUST, 1914.

Churn drill No.	Cost of labor.	Cost of supplies.	Total.	Feet drilled.	Cost per foot.
1	$229.43	$73.36	$302.79	725	$0.4176
2	311.52	76.49	388.01	1,198	.3239
Total for steam drills using air	540.95	149.85	690.80	1,923	.3592
10	383.42	58.25	441.67	1,526	.2894
11	317.99	100.68	418.67	1,164	.3597
12	302.49	89.93	392.42	970	.4046
13	361.37	147.90	509.27	1,118	.4555
14	274.08	106.00	380.08	458	.8299
15	261.02	34.17	295.19	963	.3065
16	377.13	70.19	447.32	1,340	.3338
Total for gasoline drills	2,277.50	607.12	2,884.62	7,539	.3826
Total for all churn drills	2,818.45	756.97	3,575.42	9,462	.3779

Record of costs of blast-hole churn drilling—Continued.

JANUARY, 1914.

Churn drill No.	Cost of labor.	Cost of supplies.	Total.	Feet drilled.	Cost per foot.
1	$595.59	$183.40	$778.99	1,300	$0.599
2	637.90	193.08	830.98	2,020	.411
Total for steam drills using air	1,233.49	376.48	1,609.97	3,320	.485
10	668.24	195.21	863.45	2,507	.344
11	594.89	242.66	837.55	1,902	.440
12	622.02	176.92	798.94	2,456	.325
13	653.24	190.99	844.23	1,941	.435
14	607.87	135.12	742.99	1,877	.396
15	618.97	244.88	863.85	2,103	.411
Total for gasoline drills	3,765.23	1,185.78	4,951.01	12,786	.387
Total for all churn drills	4,998.72	1,562.26	6,560.98	16,106	.407

The total cost of drilling per ton of ore mined is about 1 cent; and per cubic yard of material stripped, about 1.3 cents. The reasons for the cost per foot of blast-hole drilling being so much less than the cost of prospect drilling (p. 14) are that the prospect holes were larger and averaged eight to ten times as deep as those for blasting; also the cost of sampling was included in the prospect-hole costs.

TYPES OF TRIPOD DRILLS USED.

Two types of tripod drill, the Sullivan and the Ingersoll-Rand, are used where the rock is hard enough so that toe and shallow bottom holes are necessary. Both these makes, having 3¼-inch and 3⅝-inch cylinders, were satisfactory. They are operated by air under about a 90-pound pressure, piped from the central compressor plant. The steel used is mostly of the 1¼-inch crescent brand. It is sharpened by a No. 3 Leyner drill-sharpening machine. Machine-sharpened drills have given better service at a smaller cost than was obtained from hand-sharpened drills.

COSTS OF DRILLING BLAST HOLES BY AIR DRILLS.

During January, 1914, air-driven tripod drills drilled 9,875 linear feet of holes at a cost of 46½ cents per foot; during August they made 6,951 feet of holes at the rate of 32½ cents per foot. This great difference in rate of cost seems to be due to greater ease of drilling during August.

USE OF HAMMER DRILLS.

Where bowlders, too large to be handled by a steam shovel, remain after a bench blast, and where loading will not be too much impeded by drilling, or "block holing," instead of by "dobe" blasting, the "block holing" is done by small air hammer drills of the Sullivan D. C., 19 type, using 2-inch bits. Air is supplied to them through flexible armored hose from a supply line running along the top of the cuts. The exhaust from the machine passes out through a hole in the bit, thereby automatically cleaning the hole of cuttings. From 1½ to 3 inches of hole can be drilled per minute with this machine, and it can be operated by one man. The operator is paid $3.75 per day. Only one of these drills was in use at the time the mine was visited, because it had been found that "dobe" blasting of bowlders could be done more quickly and interfered less with loading operations, and hence, in the long run, was more economical.

BLASTING.

The object of blasting in metal mines where there are rich brittle sulphides and carbonates is generally, as in coal mines and quarries, to break material loose from the main mass without pulverizing it greatly. On the other hand, if the broken rock is subsequently to be crushed for recovery of its values or for engineering operations the aim then is that the blasting shall break the rock loose and at the same time pulverize it as much as is possible, consistent with the efficient use of explosives. At Santa Rita, of course, the aim in blasting the ore-bearing rock is to pulverize it. Any fragments of broken rock that remain too large to enter a 3-yard steam-shovel dipper have to be broken by secondary blasting at considerable extra cost.

EXPLOSIVES USED.

In breaking the rock in a safe and efficient manner the following kinds of explosives are those most extensively used:

(*a*) Dupont FF and FFF black blasting powder.

(*b*) Dupont quarry powder.

(*c*) Dupont Repauno low-freezing dynamite, 30, 40, and 60 per cent strengths.

(*d*) Trojan powder No. 2 in bags, and 40 per cent strength, in sticks.

Black powder is used for breaking and loosening up soil, hardpan, or soft rock; and quarry powder, instead of the 40 per cent dynamite, is used for soft or fissured rock. Repauno 40 per cent strength low-freezing dynamite is used to spring holes for primers and to blast holes that contain water; it is also used for holes in medium hard rock, and for "tight holes" in broken formations. For "dobe" shots 40 per cent Repauno dynamite is used almost exclusively. Repauno 60 per cent strength dynamite is used only in the hardest rock formations. Trojan powder, packed in 12½-pound waterproof paper bags, four bags to the box, is used most in medium hard or broken rock. This powder is understood by miners to be of equal strength to 40 per cent dynamite, but on account of its loose, granulated form it fills in the cracks and crevices in broken formations, thus excluding air spaces and increasing the blasting efficiency. Trojan sticks are also used for "springing" or chambering wet holes.

Detonation by fuse and detonator is used for springing holes and for "dobe" or "bulldoze" blasting. All the heavy bank-hole blasting and some of the heavy springing charges are detonated with electric detonators. The brands of fuse used are "Sylvanite" and "Cactus." Sylvanite is an asphaltum-covered, triple-wound, cord fuse, which has a burning rate of 16 inches per minute. Cactus is a triple-wound cord fuse, with a white clay-compound waterproof covering. It has a burning rate of about 20 inches per minute, and on account of its white color is generally used for night work. No. 7 detonators are used with these fuses. Where the detonation is by electric current, No. 7 electric detonators are used.

STORAGE AND HANDLING OF EXPLOSIVES.

The main magazines have each a capacity of about 130 tons. Each is formed by a drift into the hillside so that a chamber about 30 by 25 feet, and 8 feet high, is cut into the solid rock (fig. 5). Each chamber is secured by double sets of timbers, with stulls or legs set about 5 feet apart, and with the roof and sides lagged. From the entrance to the back wall a 5-foot passageway is kept clear. The explosives are segregated according to kind and are piled in tiers in the chamber. These underground magazines afford safety against stray bullets, etc., and give what is necessary in the storing of explosives—a comparatively even temperature that remains above freezing during summer and winter. The floor of each of these chambers is covered with several inches of mill tailings. Thus a soft smooth surface, which acts as a cushion to the cases of explosives, is insured.

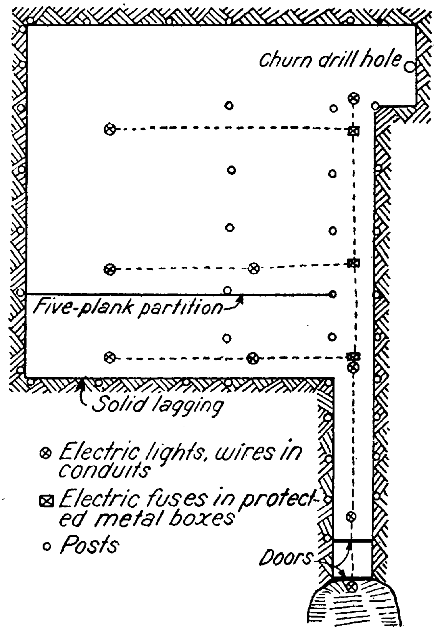


FIGURE 5.—Plan of typical underground explosives magazine in Santa Rita district.

The magazines have to be kept well ventilated in order that they may not become overheated in summer and that gases from the stored materials may not accumulate in them. To this end, a 6-inch churn-drill hole connects the inner part of the passage in each magazine with the surface (fig. 5). These bore holes are protected on the surface with hooded stacks about 12 feet high, the upper 3 feet of which are perforated.

The entrance to the magazines is guarded by two double-planked, steel-faced, swinging doors, one about 6 feet inside the other. The outside door is perforated in the upper half, and the inside one in the lower half, for the admission of air (Pl. VI, *A*).

From the magazines the explosives are distributed by wagons to several small portable powder stations. There are two of these stations to every steam shovel—one for the high explosives and one for black powder—so they are known as shovel powder houses. They consist of skeleton frames, made of 2½-inch by 2½-inch material, 7 by 4 feet by 4½ feet high, and covered with corrugated sheet iron. Each of these little houses has a capacity of 3,500 pounds of high explosives, or 4,000 pounds of black powder. However, they seldom contain more than a wagon-load—2,000 to 3,000 pounds. Carrying handles, consisting of scantlings 2 by 4 inches in size, are nailed to the sides.

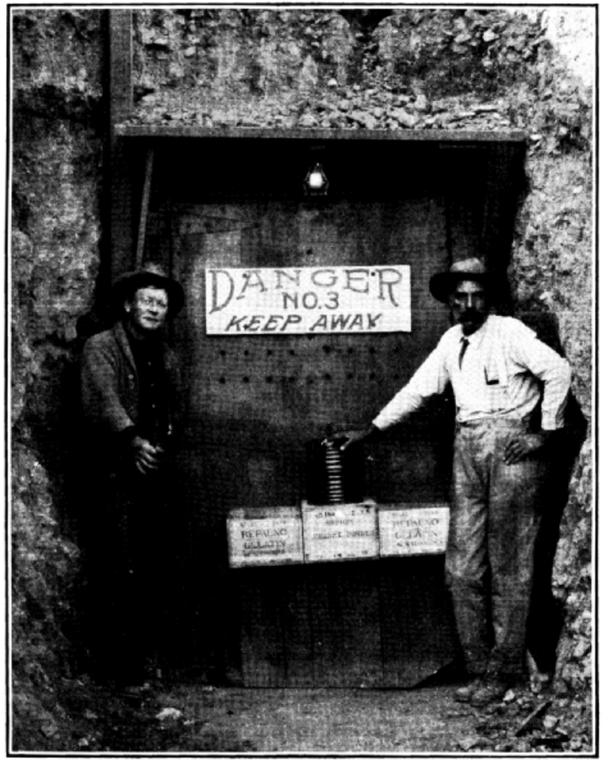
When black powder is needed in large quantities at any point, a sled with a pointed corrugated sheet-iron roof and drawn by a team of horses, is used to transport it from the shovel powder house to the place of use.

Little trouble is experienced from the freezing of explosives, as the operating company generally buys only the low-freezing or nonfreezing brands, and as the magazine temperature is always above freezing. During exceptionally cold weather hot-water thawing chambers are used and then the "powder" has to be carried a considerable distance from the magazines.

"SPRINGING" OR CHAMBERING HOLES.

Before the drill holes are loaded and blasted, one or more small charges are exploded in the bottom of each, to make an enlarged chamber for the final charge. This is known as "springing" or chambering the holes. The aim is to make so large a chamber at the bottom of the hole that the charge may be concentrated into so compact a mass that none of it will fail to detonate, and that the explosive energy may be concentrated where it will give the maximum effect. The springing eliminates any water that may be in the hole, but it generally increases the liability of an inflow of water by shattering and fissuring the surrounding rock. If granulated powder can be forced into these fissured places the effectiveness of the blast is greatly increased.

Where the rock to be broken is relatively soft and weak, and the drill holes not more than, say, 30 feet deep and not over a third of that distance back from the face of a steam-shovel cut, one "springing" is usually all that is necessary. Where the rock is hard and tough, or where the holes are very deep, as many as four successive springing charges may be required before a chamber large enough for



A. ENTRANCE TO UNDERGROUND EXPLOSIVES MAGAZINE.

B. EXPLODING A SPRINGING CHARGE IN A BANK HOLE.

Note how the water and rock shoot out at the surface.

the final blasting charge is made. After each successive "springing" the hole must be cooled off, generally with water, before another charge is introduced.

The following are average charges for each successive "springing," in medium-hard rock:

Average charges for "springing" or chambering drill holes.

Springing No.	Bank holes.		Toe holes.	
	Number of primers used.	Number of sticks of dynamite used.	Number of primers used.	Number of sticks of dynamite used.
1	1	5	1	2
2	1	11	1	5
3	2	21	1	9
4	2	28		

For "springing" holes 40 per cent strength dynamite or other high explosive, cut into quarter or third sticks, is generally used. It is detonated with a No. 7 detonator crimped onto 24 inches of fuse.

Both before and after "springing" the drill holes are sounded by a specially constructed plummet—a billet of hardwood about 30 inches long and having a metal weight in the lower end. This sounding apparatus, known as a "springing gage" (fig. 6), is attached to a rope and lowered into the hole. By tilting the upper end of it from side to side while the lower end rests on the bottom (fig. 7) the experienced chargeman can estimate the size of the hole. With this gage he also measures the depth of water in the hole. A small hand mirror is at times very useful for reflecting

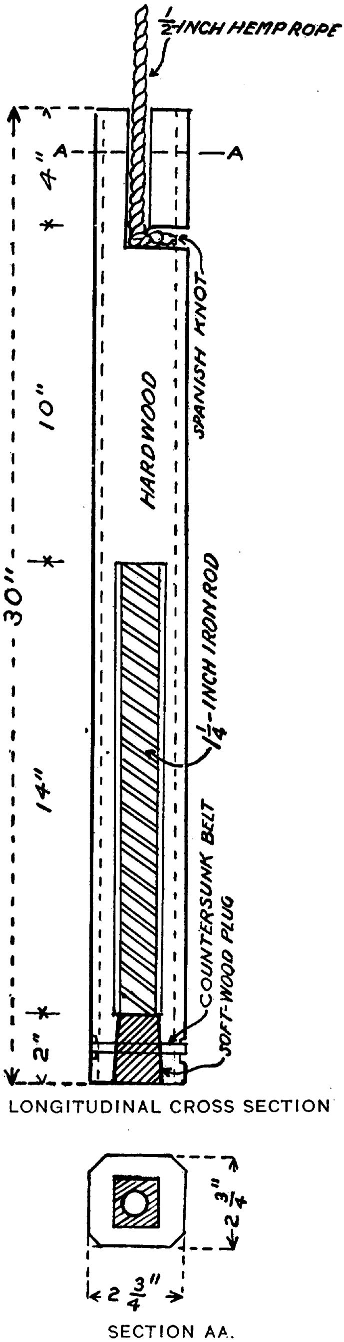


FIGURE 6.—Longitudinal and transverse cross sections of "springing gage."

light into the hole, thus enabling inspection of the upper 10 to 30 feet.

The procedure in "springing" holes is somewhat as follows: The chargeman lowers the springing gage into the hole and notes its size and depth, the depth of water, etc. He then drops in the springing charge, and again lowers the gage to determine whether the charge has reached the bottom. A fuse, passed diagonally through a stick of dynamite about 2 inches from one end, extends along the side of the stick to the other end, into which the end containing the detonator is inserted about 2½ inches (fig. 8, A).[a] This stick of dynamite, with cap and fuse attached, is called a "primer." The fuse is then lighted and the primer is dropped or lowered into the charged

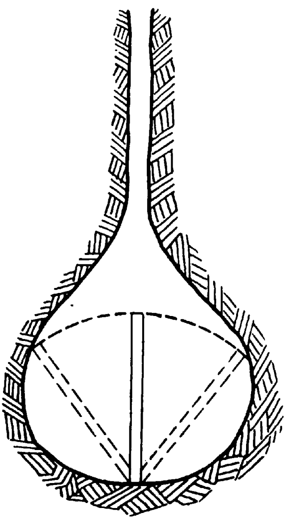


FIGURE 7.—View of chambered drill hole showing how "springing gage" is used.

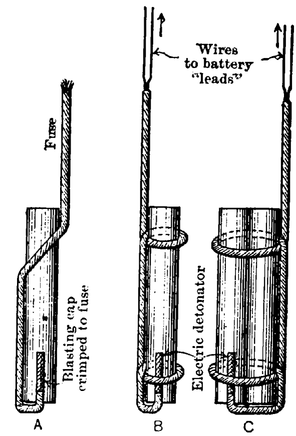


FIGURE 8.—Arrangement of fuses for different primers. A, for detonation with fuse, as practiced in Santa Rita district; B, for electrical detonation, single primer; C, for electrical detonation, double primer.

hole, followed by a hurried lowering of the gage to show whether the primer has reached the bottom. After this 2 to 4 gallons of water is poured in as "stemming," the gage is quickly withdrawn, and the chargeman retreats to a safe distance. After the detonation, the gage is again lowered and the size of the chamber estimated. If necessary other "springing" charges are used, each about twice as large as the preceding one, until the hole is sufficiently widened or chambered out at the base to receive a proper blasting charge (fig. 9). For the third "springing," where the holes are over 30 feet deep, and often for the second "springing," if much water is present, electric detonation is used (fig. 8, B and C). Where there are several holes to be fired simultaneously electricity is employed instead of fuse.

[a] The Bureau of Mines does not recommend the lacing of a fuse through a primer.

The lead wires or detonating wires are connected in series and the detonators tested with a galvanometer. The lead wires are then connected with a battery and the charges are fired (Pl. VI, *B*).

CHARGING AND FIRING.

The blasting squad usually consists of a charge man or powder man and four helpers. The charge man places all explosives in the drill holes, except in the "springing" of short holes, when the work may

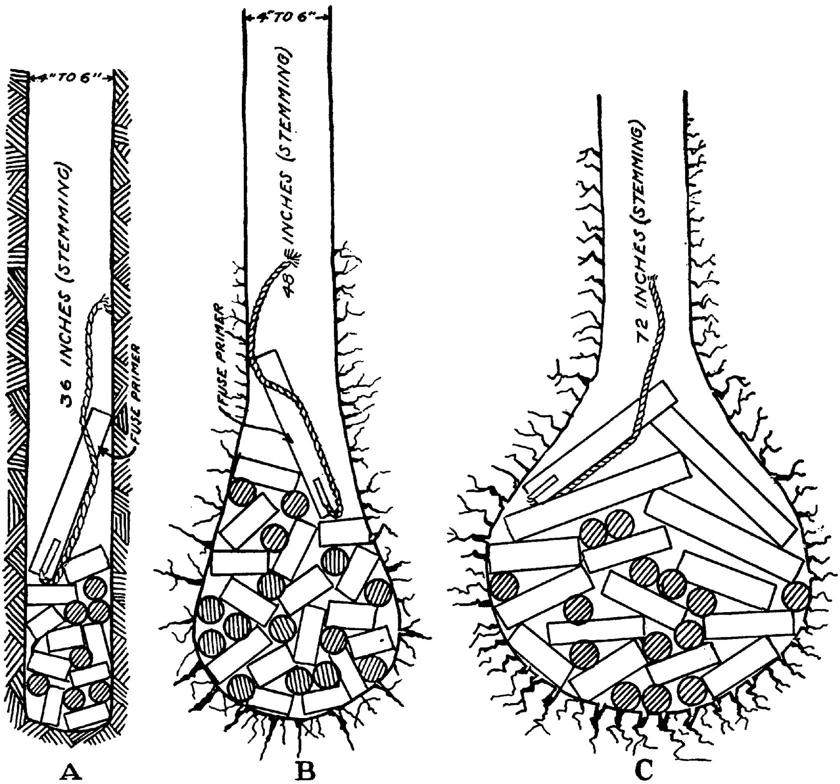


FIGURE 9.—"Springing" charges in bank holes in dry soil or soft rock. Water stemming only. A, charge for first "springing"; B, charge for second "springing"; C, charge for third "springing."

be done by the assistant charge man. The helpers carry the explosives and accessories to the charge man while he is gaging the hole and charging it.

The amount of the charge for each hole varies according to the toughness of the rock, the position of the hole with relation to the face and to other holes, and its depth and extent of chambering. The drill holes to be blasted are "bank holes"—those drilled vertically on each bench and situated back approximately one-third of their depth from the edge of the bench—and "toe holes"—those driven into the face of the bench, say, 4 to 6 feet above the shovel level at the foot of the bench. The toe holes are driven with a downward slope of 20° to 30° below the horizontal.

Average charges for bank holes in medium-hard rock are about as follows:

Average charges for bank holes in medium-hard rock.

Depth of hole.	Sticks of dynamite in primer.	Charge.		
		40 per cent strength dynamite.	Trojan powder No. 2 (granulated).	Stemming.
Feet.		*Pounds.*	*Pounds.*	*Feet.*
28	1 and 2	138	100	24
40	1 and 2	....	267	36
50	1 and 2	170	200	40
70	3	....	550	60
[a] 50	2	[b] 375	....	30

[a] Driven into soil.
[b] FF black powder.

Average toe-hole charges for medium-hard rock are:

Average toe-hole charges for medium-hard rock.

Depth of hole.	Number of sticks of dynamite in primer.	Charge.	
		40 per cent strength dynamite.	Stemming.
Feet.		*Pounds.*	*Feet.*
8	1	12.5	4 to 6
18	1 and 2	60.0	12 to 14

The following are average mixed charges from a number of observations:

Average mixed charges.

BANK HOLES.

Depth of holes.	Quantity of mixed charge.	Charge per foot of hole.
Feet.	*Pounds.*	*Pounds.*
28	188	6.72
40	267	6.68
50	369	7.38
[a] 50	[b] 375	7.50
70	550	7.86

TOE HOLES.

8	12.5	[c] 1.56
18	60.0	3.33

[a] In soil and soft rock.
[b] Black powder.
[c] Holes not chambered.

When the holes are sufficiently chambered or sprung and are ready for loading, an electric-detonator primer, prepared as shown in figure 8, C, is carefully lowered to the bottom of the hole. The first part of the charge, varying from 100 to 300 pounds, is dropped in after it. Another primer is then placed in the hole and the remainder of the charge, 75 to 250 pounds, is added (fig. 10). During the process of loading, the charge man frequently drops in his gage, to determine how rapidly the hole is filling and to guard against the clogging of the powder. Often granulated Trojan powder is poured in to fill the interstices between the sticks of dynamite and the fissured places in the rock (fig. 10). Thus air spaces are minimized and the efficiency of the blast greatly increased. When the charge is all placed, screened clayey sand and loam are carefully shoveled into the hole and packed down for stemming.

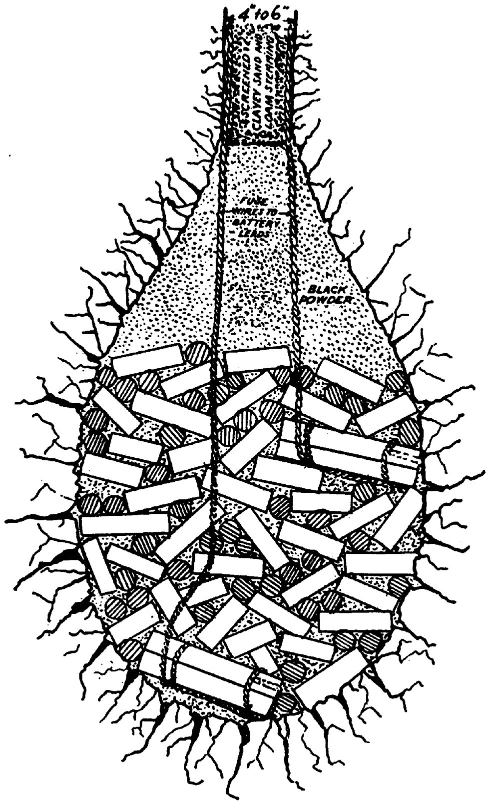


FIGURE 10.—Typical blasting charge in bank hole.

All blasting is done at such time as will least interfere with the operation of steam shovels, trains, drilling machines, and general work. Wherever practicable, the blasting is done at the end or before the beginning of the shift or during the noon period. A great deal of blasting is also done by the night squad.

The rules of the company prohibit more than one independent charge in any hole; that is, the explosives must be so placed that no stemming separates any part or parts of the total charge in any hole. Double or triple charges, separated by stemming, are not allowed to be put into one hole, owing to the danger from incomplete detonation. A part of the charge under such conditions might not explode, thus rendering subsequent steam-shovel operations dangerous, for unexploded dynamite is sometimes set off when dug into by steam shovels.

The toe holes are loaded in much the same way as the bank holes, except that the charge has to be pushed into the relatively flat toe holes, a little at a time, with the aid of a charging stick—a long, slender wooden pole (Pl. VII, *A*).

In general the equipment of the charging squad is as follows:

"Springing" gage.
Charging sticks.
Pocket reflecting glass.
Galvanometer.
Rheostat.
Blasting battery.
Explosives.
Fuse.
Electric detonators.
Combination detonator crimper, pliers, and punch.
Wooden mallet.
Large funnel.
Pocket knife.
Wooden axe handle.
Detonators.
Battery lead wire.
Connecting wire.

After the holes have been charged the detonator wires are connected, generally in series, with the wires that lead to the battery (fig. 11). A galvanometer is then attached to the lead wires, and if the whole connection makes a complete circuit the galvanometer needle will be deflected. If there is a break in the circuit it is located by attaching the wires so that first one hole, then two holes, and so on, are left out of the circuit; as soon as a complete circuit is found, it is known that the break must be between the completed circuit and the hole last cut out of circuit. When a complete circuit is established, the wires are ready for connection with the battery. The current used in the testing galvanometer is about one-thirtieth of that necessary to detonate the charge, so that there is an ample margin of safety.

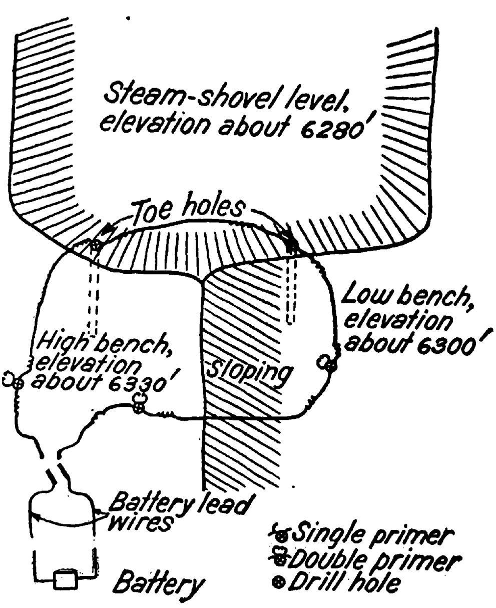


FIGURE 11.—Arrangement of detonator and electric wires for exploding charge.

Five or ten minutes before the charge man finishes connecting the detonator wires he notifies the engineer of the nearest locomotive or steam shovel, who toots the whistle 40 or 50 times. This warning is understood by all workmen, and they immediately retreat to a safe distance. The final connection with the battery is then made, and after an interval of a few minutes the charge man sets off the round

A. LOADING A TOE HOLE. THE CHARGE IS BEING PUSHED IN WITH A LONG SLENDER WOODEN POLE KNOWN AS A "CHARGING STICK."

B. FACE OF SHOVEL CUT AFTER FIRING BLAST HOLES, SHOWING LARGE BOWLDERS ON WHICH "DOBE" CHARGES ARE BEING LAID.

of shots. After the explosion he notifies the steam-shovel or locomotive engineer, who, when he returns to his post, sounds a long blast of the whistle, indicating that the danger is over.

If a charge misses fire, of course no attempt to withdraw it is made. Such a proceeding would be entirely too dangerous. The remedy is to drill a new hole within 6 or 8 feet of the unexploded one. The new hole must not be so close to the old one that deflection may cause it to break into the unexploded charge. The distance is gaged by the charge man who chambered the old hole and knows the size of the chamber, and by the driller who drilled it. During the drilling of the new hole, as a precautionary measure, nobody is allowed to work in the vicinity of the unexploded hole except the men necessary to operate the drill.

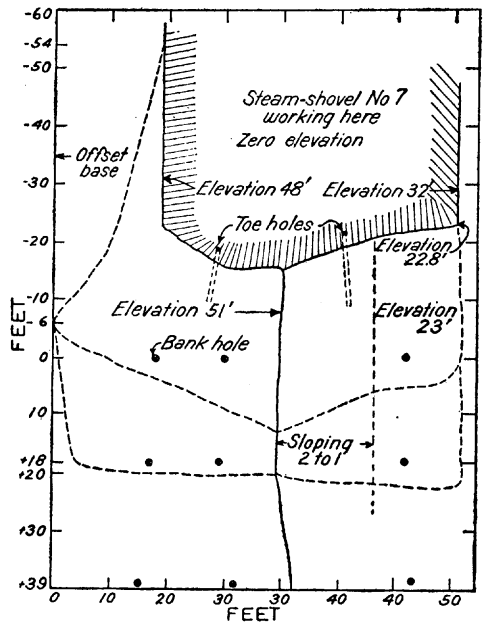


FIGURE 12.—Results of firing bank and toe holes in positions shown. Arrangement of holes and superficial area cut by each series are indicated by dotted lines.

FACTORS AFFECTING BLASTING EFFICIENCY.

The quantity of material loosened by a given amount of explosive depends on many factors, chief among which are: Toughness of the rock; degree of jointing and fissuring; number and extent of free faces of the bench to be blasted; depth and spacing of the holes—their distance apart and from the free faces; character of chambering and of charging of the holes; character of detonation of the charges—whether they are all detonated simultaneously.

A factor of special significance in determining the charges to be used is the degree of solidity or "tightness" of the rocks at the base or toe of the blasting face. Where the rocks are solid at the toe, toe holes with fairly heavy charges greatly increase the efficiency of the blast, and cause the rock to break down to an even surface. Data regarding several bank and toe holes projected are shown in figure 12. The conditions seemed to be about average for the mine. The results from the blast with these holes were as follows: About 3.2 cubic yards of soil and weathered surface rock was broken per pound

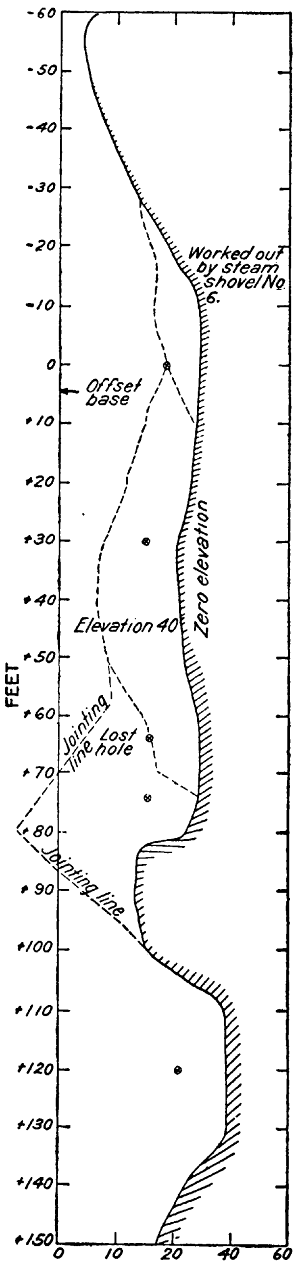


FIGURE 13.—Results of firing blast holes in firm rock. Shows arrangement for single-line bench holes. The areas cut by each are indicated by dotted lines.

of black blasting powder. Approximately 3.1 cubic yards of somewhat weathered dioritic rock was averaged per pound of Trojan powder. Where this rock was firmer and more solid the results averaged about 1.5 cubic yards for a pound of Trojan and 40 per cent strength Repauno gelatin dynamite (fig. 13). In the tougher areas only about 0.4 cubic yard was broken per pound of Trojan powder. Average results from a number of blasts are given in the following table:

Average results from several blasts.

Character of rock and ore.	Depth of holes.	Explosives used.	Rock broken.	Rock broken per pound of explosives.
	Feet.	*Pounds.*	*Cubic yards.*	*Cubic yards.*
Soil and weathered rock	50	378	1,222	3.233
Partly weathered dioritic rock	70	700	2,168	3.097
Do	28 and 50	2,235	3,249	1.454
Relatively soft porphyry	40	825	2,264	2.744

For a three-month period in 1913 the average quantity of ore and rock broken per pound of explosives used was 2.55 cubic yards. For a corresponding three-month period in 1914 the average was 2.67 cubic yards. The quantity of both kinds of explosives used during these periods was as follows:

Quantity of explosives used in blast holes during two three-month periods.

Kind of explosive.	Three-month period in 1913.		Three-month period in 1914.	
	Quantity of explosive used.	Per cent of total.	Quantity of explosive used.	Per cent of total.
	Pounds.		*Pounds.*	
Black powder	192,870	44.56	136,950	39.71
Higher explosives	240,000	55.44	207,950	60.29

SECONDARY OR "DOBE" BLASTING.

When the bank holes are exploded, 5 to 15 per cent of the material broken is so coarse (Pl. VII, *B*) that it has to be blasted again in order to be handled easily by the buckets of the steam shovels, which have a capacity of 3 to 3½ cubic yards. This secondary blasting is done by placing a number of sticks of dynamite and a primer, with 42 inches of fuse attached, on the rock to be broken, the size of the charge being regulated by the judgment of the powder man. Formerly it was the general practice everywhere to cover charges of this kind with a stemming of clay or adobe, giving rise to the term "adobe" or "dobe" blasting. This manner of breaking rocks is also referred to as "bulldozing."

When the "dobe" charges are all set, the nearest steam whistle toots a warning for the workmen to get out of the way. The chargeman and one or two assistants, each using a piece of fuse equal to the length of the primer fuses (42 inches) as an igniting medium, light the primer fuses and then run to safety. A series of cuts half through the igniter fuse indicate to the men doing the igniting just how fast their fuses are burning and how much time they will have to finish the lighting and get to safety. The number of "dobe" shots placed is counted, and the number of explosions heard is carefully observed so that any missed or delayed shots will be detected.

Prior to the installation of the plant for crushing coarse material, the cost of blasting bowlders, or of "dobe" shooting, was about 1.54 cents per ton of ore loaded. It is estimated that the new crushing plant saves 80 to 90 per cent of that cost.

BLASTING COSTS.

During a three-month period in 1913, when the mine was working at normal capacity, the cost of explosives was 3.75 cents per cubic yard of material loaded. For a corresponding period in 1914, when operations were conducted at only about 60 per cent capacity, the charges for explosives had dropped to 3.56 cents per cubic yard. The difference was due largely to the fact that fewer bowlders had to be blasted after the coarse crushing plant had been installed.

The average cost per pound of all the explosives used in 1913 was 9.55 cents, and in 1914 it amounted to 9.64 cents. "Dobe" blasting is an inefficient method of breaking rock, because the charge is not confined and much of its energy is lost; also because the loading operations of the nearest shovel or shovels must cease while the "dobe" charges are being placed and fired.

BLASTING ACCIDENTS.

Under normal mining conditions about 1,500,000 pounds of explosives is used annually by the Chino company. These figures are so large that, in comparison, the accidents have been few. During 1914 only two men were killed as a result of blasting operations. They miscounted the number of "dobe" shots lighted, and walked back before the last one, which was delayed, had exploded. In 1913 three men were killed. A party of five men had taken refuge under the back end of a steam shovel, against the general orders of the company. The blast was so heavy that the concussion moved the shovel back on the men, fatally crushing three of them.

SAFETY RULES.

To help safeguard its employees the company has posted rules, printed in English and Spanish, so that everyone, especially those that handle explosives, may become familiar with them. Some of the principal rules so posted follow:

Before exploding a blast, care should be taken to give the blasting signal and to wait until every person has reached a point of safety. Do not explode a blast until at least three minutes have elapsed from the last note of warning.

The powder "whistle off" signal must be given after each and every time of blasting.

All persons are warned not to return to the vicinity of "dobes" until at least one-half minute has elapsed from the last shot heard.

When dobying (bulldozing) is done directly in front of a shovel no one should remain on or under that shovel for protection.

In firing bank shots or toe holes the battery wires must not be connected to the battery until the person immediately in charge walks back to the battery and gives the word to connect.

Do not smoke while handling explosives, and do not handle explosives near an open light.

Do not carry loose detonators or electric detonators in the clothing. Carry them in special boxes.

Do not transport caps or cartridges containing caps to the blasting place with the supply of dynamite to be used, and do not place them side by side until ready for explosion.

Do not tamper with caps. If the caps are damaged, discard by loading into some bank hole.

Inspect the place where bulldozing has been done before work is resumed. Recover all powder and caps possible and return to their proper places.

Carelessness will not be tolerated. The careless laborer imperils not only himself but he endangers his fellow workman as well.

All blasting machines must be tested with rheostat at least twice a week. If blasting machine is defective, this must be reported. The machine must not be used until it has been repaired.

During cold weather no dynamite but the nonfreezing powder shall be used unless by order of the general foreman.

Powder boxes and powder cans must not be opened otherwise than with the wooden tools provided (which are mallet and axe handle).

Black powder cans should not be rolled down the bank.

Black powder should not be opened near a light or fire.

Dynamite must not be rolled down the banks tied to a rope except in places and manner designated by the head powder foreman.

It is absolutely prohibited to store powder or caps anywhere about a steam shovel. All such recovered material must be gathered up and returned to the boxes.

Do not allow explosives to become scattered. Immediately after each finished period of loading all unused powder and caps must be returned to their respective places.

Detonators and powder must not be stored in the same place.

Fuse, firing machines, and all tools necessary for blasting must be kept in a box separate from the powder. All material must be kept under lock and key.

Place all powder boxes so that the cartridges will rest in a level position.

Do not fasten a detonator with the teeth or by flattening with a knife; use a crimper.

Do not explode a charge to chamber a hole and then immediately reload the hole; the hole will be hot. Water must be used to cool it.

Do not put two independent charges of explosives in the same drill hole.

Do not attempt to draw nor to dig out the charge in case of a misfire.

No person shall blast a hole alone; he must have some one near him.

For dobying, fuse must not be less than 30 inches; for springing holes not less than 18 inches.

All electric detonators must be tested with galvanometer before placing stemming in the hole.

Not more than three men are to load a hole at one time.

In loading bank holes one primer must be placed in the hole before adding any of the charge of powder.

After primer or powder has been placed in the hole no metal shall be used either to tamp or clear the hole.

All tamping used in holes shall be screened dirt.

Caps and electric detonators shall be stored in separate box, which shall be marked "Cap Box." Nothing else shall be stored in these boxes.

One man is never to trim a bank alone; at least two men should work together.

While trimming, bank men are not to go down the bank unless with a rope securely fastened above.

"GOPHER" BLASTING.

At the present stage of development "gopher" blasting has been practically discontinued. However, in the past it was advantageously used to break down some high banks that were difficult to drill and load in the ordinary way. A good account of one of the largest gopher blasts has been written by R. I. Kirchman,[a] bench foreman for the Chino company. It is so instructive that extensive quotations from it follow. Kirchman first explains that in the preliminary work at Santa Rita, owing to the uneven surface of the ground, the height of the benches varied from a few feet to 150 feet or more. Some of the high benches were maintained because the expense of opening intervening levels was not warranted by the conditions. Locally the rock was considerably fissured and jointed with a tendency to "hang up." Hanging brows of the rock, where the bank was 75 to 100 feet high, came down at uncertain intervals,

[a] Kirchman, R. I., Gopher blasting: Colorado School of Mines Mag., vol. 4, May, 1914, pp. 103-107.

sometimes endangering the steam shovels near the base. When toe holes were sprung they often caved, thus rendering them useless and necessitating new holes. Because of these difficulties it was decided to handle the bank with a large gopher blast. Kirchman describes the operations as follows:

Three adits were determined upon as shown in figure 14. The north adit was numbered 1, the middle 2, and the south 3. Adits 1 and 2 required two tunnel sets each. These sets were lagged. Adit 3 was not fractured as much as 1 and 2 and did not require timbering. The cross-section dimensions were 3 feet by 5 feet high, driven part by hand drill and part by air-drill machine.

The muck from the main adits was dumped directly in front and around their entrances. The broken earth from the crosscuts was thrown back and leveled to a height just sufficient to allow the passage of a man—that is, an opening about 2 by 3 feet. The excess muck was carried to the entrance. The corresponding crosscuts were thus left half full of muck back to the main adits.

The ends of the crosscuts were chambered to a size sufficient to accommodate the powder predetermined by the amount of yardage over the "gophers." The gopher was assumed to break from a line 10 feet beyond the breast of the main adits, and along a 1-to-1 slope to the surface.

The Mexican laborers were divided into classes of miners and muckers. The miners were paid at the rate of $2.75 per day of 9 hours and the muckers $2 per day. At the start only two 9-hour shifts were employed, but as greater progress was desired three 8-hour shifts were put on.

The latter change left 2 men in each face, or a total of 18 men per 24 hours. The progress was about 4 feet per shift. The total linear feet of gopher driven was 352. The time occupied in driving was 31 days. Loading commenced immediately after completion and required 3 days of 10-hour day shifts, and was accomplished by 4 powdermen and 10 laborers for loading and tamping.

On the bottom of the drifts a walk was laid, composed of single boards placed loosely end to end; no nails were used. The powder boxes were slid along this path to the chambers. Ordinary pocket electric flash lights gave fair light for about 24 hours.

Crosscut 1 was loaded first. The ends of the powder boxes were opened and placed end to end. The boxes were pyramided in a single tier down the middle of the pocket. An electric fuse was placed in the middle box of each layer of the tier, making three fuses to the chamber. Extreme care was taken not to injure the light fuse wires. An important precaution to observe in loading these initial charges is the avoidance of undue pressure upon the fuses or stick powder. The stick powder used was Du Pont 40 per cent Repauno gelatin. The fuses were Victor No. 7, 10-foot lengths, having a resistance of 1.2 ohms.

The black powder, in 25-pound cans, was next placed about the powder boxes. About three-quarters of the black-powder charge was loaded from the breast to the front of the charge and one-half way up to the back. Judson powder, in 12½-pound paper bags, was placed upon the powder cans as compactly as possible. This latter charge filled the pocket clear to the back. The final face of the chamber was sealed with the remaining one-quarter of black powder in cans. The latter precaution was taken to avoid any danger or damage to the dynamite and Judson powder when tamping.

All connecting fuse wires were firmly contacted. The charges were connected in series with No. 14 lead wire, and the fuse wires were wrapped around the lead wires. All fuses were tested with a Du Pont galvanometer, which shows the presence of a circuit and resistance contained therein. All completed circuits in the different chambered pockets were tested before the chambers were sealed with carefully placed

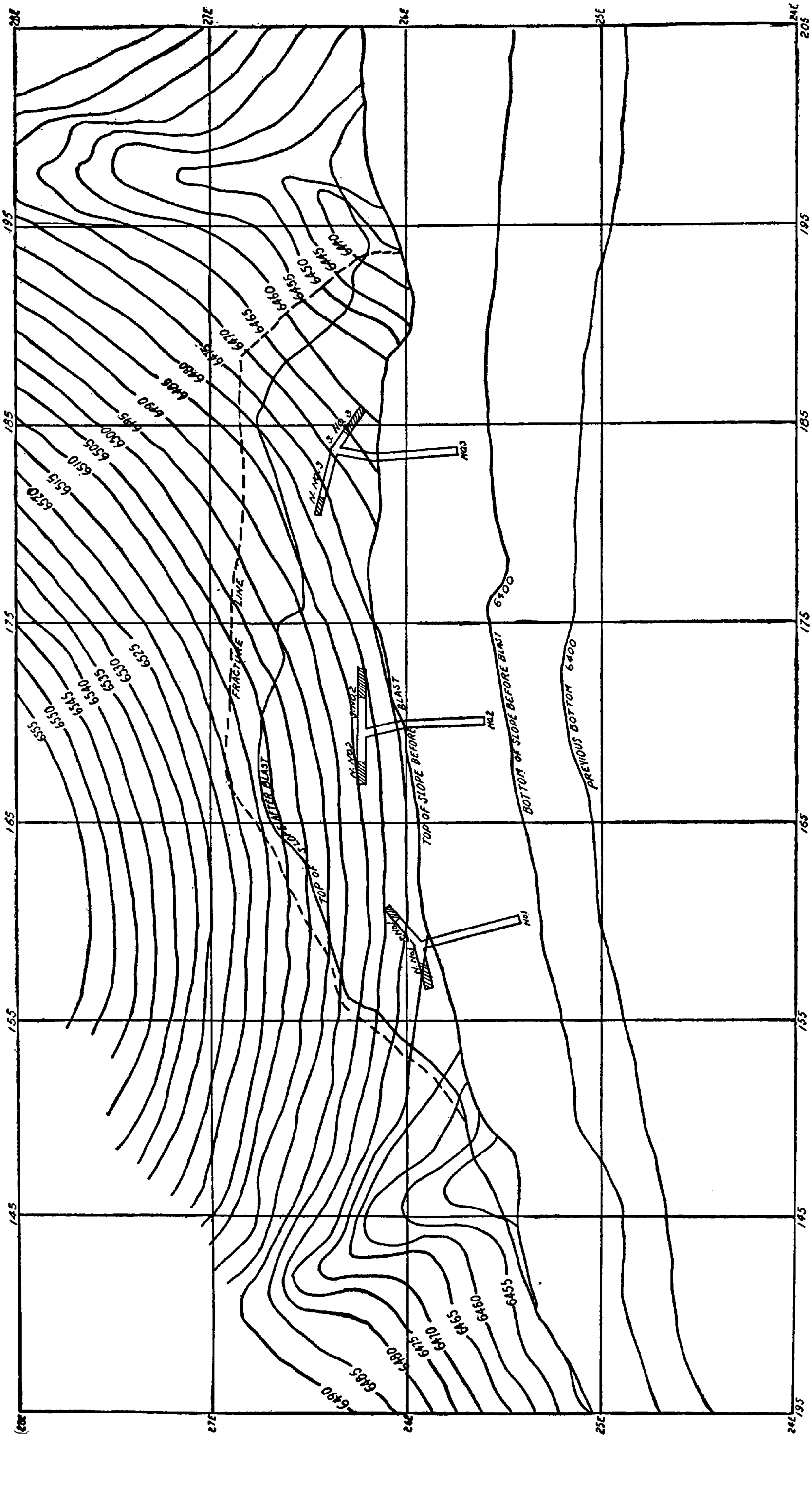


FIGURE 14.—Placing of "gopher" holes for large blast.

muck. The lead wires from each chamber were laid on top of the muck and in one corner of transverse cross section. Pieces of wood broken from the refuse powder boxes were placed at an angle above these wires, thus affording a protection from the rocks. These lead wires were uncoiled as fast as the tamping proceeded.

The tamping was a slow and laborious process. The muck had to be carried in powder boxes and slid along the board path, and dumped into the receding filled faces. At intervals of 3 feet of fill the circuits were tested in all drifts.

During the loading of the gopher the men suffered from powder headaches. These were sharp and annoying, but soon passed away in most cases when brought to the fresh air. The tamping progressed more rapidly when the main adits were reached. An unusual source of danger occurred for a few days during the presence of electrical storms. The precaution of keeping all lead wires well within the adit entrances was observed.

The lead wires from the several entrances were then connected in series. A pair of main lead wires was then laid a distance of 1,500 feet north of the entrance of adit 1. These were not connected until the proper time for blasting. The blasting battery, a Du Pont 1–75 fuse machine, was cleaned and tested.

The final circuit was tested with the galvanometer and a resistance of 24 ohms noted; this indicated a good connection. The eighteen 10-foot fuses used should have had a total of 21.2 ohms; the remaining 2.8 ohms was apparently due to imperfect connections. The resistance of the lead wires was practically zero. On June 25, 1913, the gopher was fired. Some bowlders were thrown 400 to 600 feet by the concussion of the sharp blast. The main mass of the material was not thrown over 200 feet.

The yardage of the gopher was figured on a 1-to-1 slope. The steam shovels removed 60,000 cubic yards (in solid), but the total has not yet been removed. It is estimated that 40,000 cubic yards, in solid measurement, of loosened material is still to be removed.

It was the opinion of some that the gopher was loaded a trifle too heavy. For the class of overburden a load of 1 pound of powder for 3 or 3½ cubic yards (solid) would have yielded better results; but as an experiment for future reference this was a successful gopher. No serious delay occurred, no damage was done, and the practicability of a gopher bank was demonstrated.

The data were summarized by Mr. Kirchman as follows:

Data regarding one of largest "gopher" blasts used by Chino company to break down high bank.

Chamber.	Powder loaded (pounds).					Rate loaded.		
	Black powder.	Judson.	Forty per cent gelatin.	Ratio of explosion to ignition charge. [a]	Total powder charge.	Engineers' estimate.	Pounds per cubic yard.	Cubic yards broken per pound of explosives.
						Cu. yds.		
N. No. 1	7,725	2,400	800	12.65	10,925	25,664	0.41	2.34
S. No. 1		5,500	500	11.00	6,000	14,948	.40	2.49
N. No. 2		5,250	600	8.75	5,850	18,122	.32	3.10
S. No. 2		9,250	800	11.56	10,050	25,197	.40	2.50
N. No. 3		7,050	500	14.10	7,550	12,446	.67	1.65
S. No. 3		9,050	900	10.05	9,950	22,283	.45	2.24
Total or average	7,725	38,500	4,100	11.27	50,325	118,660	.425	2.36

[a] These data are of interest in that they indicate the proper ratio of the ignition charge to the explosion charge. An idea of the ratio of the overburden in cubic yards to the powder in pounds is suggested. A large ratio in this character of ground is recommended.

LOADING ORE.

After the ore and waste have been broken by blasting they are loaded onto cars by Marion steam shovels. Of these, there are six of the 91-ton model, two of the 92-ton model, one of the 100-ton model, and one of the 40-ton model—ten in all.

The 40-ton and the 100-ton shovels are used for digging first or pioneer cuts, also called "thorough" cuts. The 40-ton machine has a large arc of swing to its boom, so that it can carry a cut into a bank, swinging its dipper back to one side far enough to dump into a car close to the side of the shovel and back from the working face of the cut. The other shovels when making first, or pioneer cuts, have to load onto cars that stand on the bank above, outside the excavation that is being made. The 91-ton and the 92-ton shovels are designed to have a maximum lift of 19 and 21½ feet, respectively; hence, in general, they can not make a first cut more than 10 to 12 feet below the level of the loading track. The 100-ton shovel, however, is built with a long boom and dipper arm, so that it has a maximum lift of 35 feet. Its function is primarily to make pioneer or "thorough" cuts, which the other shovels enlarge. It can cut a wide ditch about 22 feet deep, lifting the material up onto cars that stand on the banks outside the cut. It has a dipper with a capacity of 3 cubic yards, whereas the capacity of the 91-ton and of the 92-ton shovels is 3½ cubic yards. This smaller dipper is necessary because the longer boom and dipper arm—92-foot extreme radius—greatly increases the strain on the machinery.

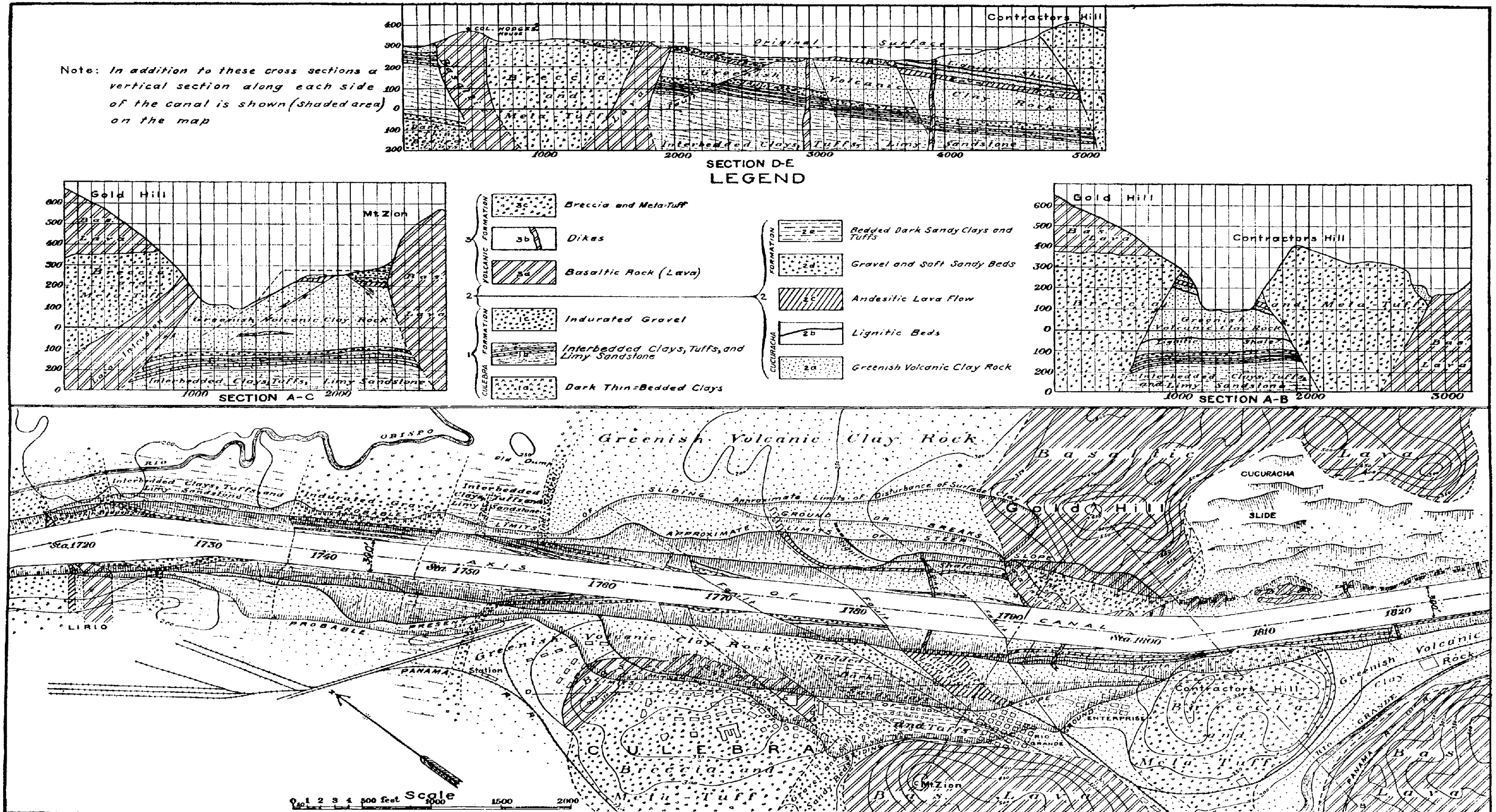
The crew of each shovel consists of an engineer at $190 per month, a craneman at $135 per month, a fireman at $3.50 per 10-hour shift, an oiler at $2.25 per shift, and 6 to 8 pitmen at $2.25 per 10-hour shift, who assist in loading operations and in moving the shovel.

The material handled per day of two 10-hour shifts by each shovel, except the small 40-ton machine, is 1,400 to 2,800 cubic yards, measured in the solid. This rather wide variation is due to occasional shortage of cars, local difficulties of digging, or temporary breakdowns.

The coal, oil, and cotton waste used per 10-hour shift is as follows:

Coal, oil, and cotton waste used per 10-hour shift.

Material.	Price.	Amount used by different sized shovels.		
		40-ton.	91-ton and 92-ton.	100-ton.
Black oil	$0.14 per gallon	3 pints	2 quarts	2 quarts.
Cylinder oil	.37 per gallon	3 pints	3 quarts	1½ gallons.
Engine oil	.22 per gallon	½ pint	1 pint	1 pint.
Crude oil	.10 per gallon	1 quart	2 quarts	1½ gallons.
Cup grease	.05 per pound	½ pound	½ pound	½ pound.
White cotton waste	9.39 per hundredweight.	1 pound	1 pound	1 pound.
Run-of-mine coal	4.30 per ton	3½ tons	5½ tons	7½ tons.



By Donald F. MacDonald.

MAP AND SECTIONS TO SHOW GEOLOGIC CONDITIONS AND PROBABLE LIMITS OF SLIDING GROUND, CULEBRA AND VICINITY.

Heavy castings and repair parts for all the shovels are kept on hand so that a shovel can, without much delay, be almost rebuilt in the company's machine shops.

The cost of loading is in general 6 to 12 cents per ton. In a few cases, owing to exceptionally unfavorable circumstances, the loading costs for one shovel for a month have averaged as high as 29 cents per ton. On the other hand, exceptionally favorable conditions have at times reduced the monthly average to a little more than 4 cents per ton.

TRANSPORTATION OF ORE.

In the matter of transportation, it was planned to have the loaded cars moved from and the empties moved to the shovels with as little delay as possible. To handle the trains of ore and waste fourteen 42½-ton and seven 50-ton, 4-wheeled Porter locomotives are used. These have cylinders 15 by 24 and 16 by 24 inches. They are each manned by a locomotive engineer, at $4.25 per 10-hour shift and a fireman, at $3 per 10-hour shift. The quantities of oil and fuel used by each locomotive per shift of 10 hours are about as follows:

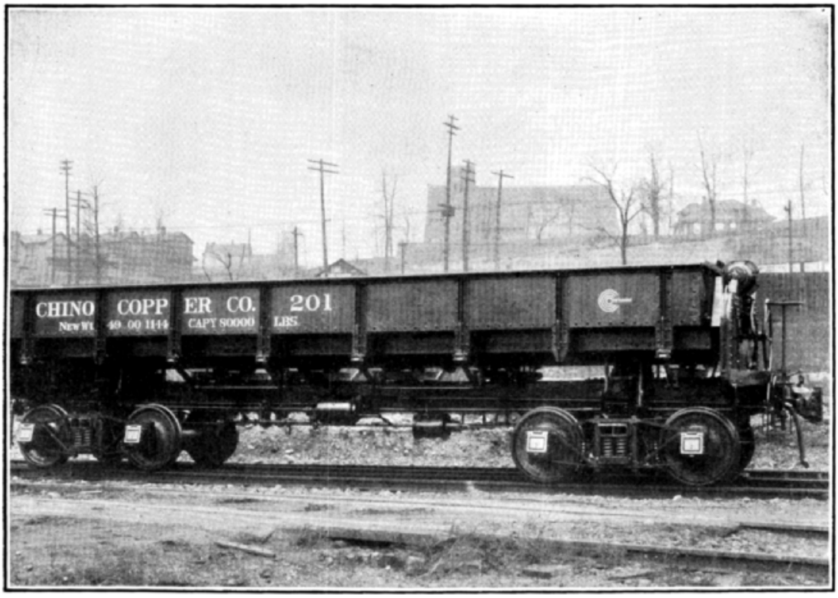
Quantities of oil and fuel used by 42½-ton and 50-ton Porter locomotives with side or saddle water tanks.

Engine oil	1⅓ pints, at 22 cents per gallon.
Cylinder oil	1 pint, at 37 cents per gallon.
Cup grease	½ pound, at 5 cents per pound.
White cotton waste	½ pound, at $9.39 per hundredweight.
Run-of-mine coal	3 tons, at $4.30 per ton.

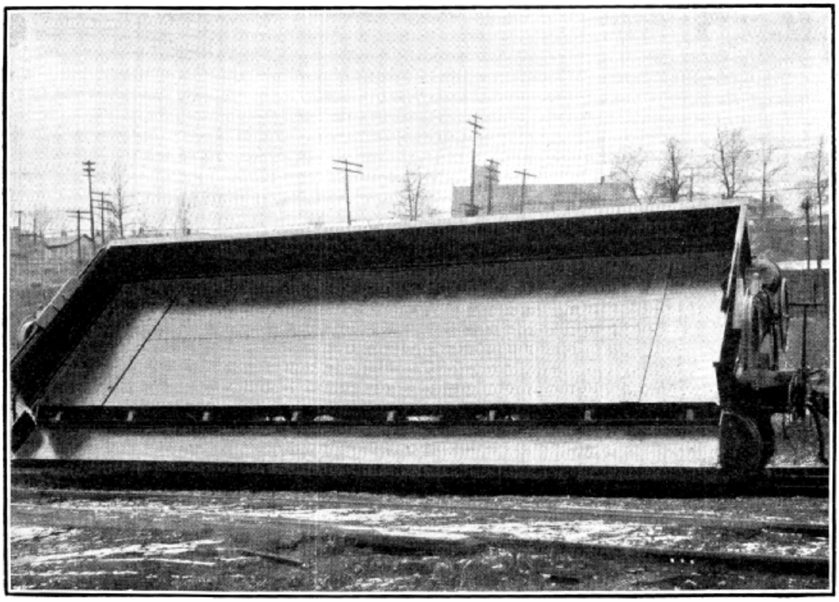
In general, there are always about two locomotives in the shop undergoing repairs. In addition, one locomotive is held just outside the shop with steam up ready for instant service. The total repair, upkeep, and all other expenses of the locomotives amount to about 0.2 cent per ton of material handled.

The ore cars in use are about as follows: Forty-five 12-yard, side-dump, Oliver steel cars; forty 6-yard steel cars not equipped with air brakes; four 20-yard steel cars fitted with air brakes, air dumping appliances, etc. The four last-mentioned cars are leased from the Santa Fe Railroad, and have given such good satisfaction that the Chino company has ordered several of the new Clark 20-yard extension dumping steel cars (Pl. VIII). These dump on either side, by air or by hand power, and are said to be satisfactory.

The experience of the Chino company has been that 6-yard cars are too light for loading with heavy shovels and too expensive to keep in repair. Not being fitted with air brakes, they are rather dangerous to handle on heavy grades. The large, heavy, air-brake



A. STEEL ORE CAR USED AT SANTA RITA. DUMPS ON EITHER SIDE BY AUTOMATIC AIR CONTROL OR BY HAND, CAPACITY 20 CUBIC YARDS.



B. STEEL ORE CAR IN DUMPING POSITION.

and air-dump cars, though expensive, are cheaper in the long run, under the conditions of heavy service that prevail at Santa Rita.

Ore that is fine enough not to need crushing is generally loaded directly into 50-ton steel railroad cars, the property of the Santa Fe Railroad Co., and is sent direct to the mill at Hurley without having to pass through the crushing plant just below the mine.

The following table shows the number of cars of each size loaded per month, during five typical months, and the cost of keeping them in repair. The repair cost per dry ton of ore handled and per cubic yard of waste is also given. The 50-ton cars are the property of the Santa Fe Railroad Co., but are kept in repair by the Chino company while used by them, and so are included in the repair account.

Cost of car repairs for five typical months.

CARS USED IN MINING ORE.

Month.	Cost of repairs.			Number of cars loaded.			
	Labor.	Supplies.	Cost per dry ton.	20-yard.	12-yard.	6-yard.	50-ton.
1914.							
January	$188.81	$196.19	$0.00196	....	100	....	4,028
February	148.99	85.01	.00124	....	4	....	4,066
March	138.48	103.40	.00109	....	898	....	4,435
August	94.93	78.72	.00142	107	2,103	....	1,837
September	86.24	91.87	.00171	12	2,015	....	1,438

CARS USED IN STRIPPING WASTE.

			Cost per cubic yard.				
January	$1,699.33	$1,765.73	$0.01122	....	27,124	15,711	....
February	1,340.88	765.08	.00741	....	29,573	13,043	....
March	1,246.36	930.61	.00629	....	34,525	16,105	....
August	854.35	708.48	.00838	1,017	10,169	10,778	12
September	776.17	826.86	.01008	1,548	9,511	9,065	14

The material loaded onto cars by the steam shovels is hauled out of the pits to one of the following destinations: The mill, direct; the crusher; the ore stock pile; or directly to the smelter. The waste goes to the waste dumps. The lowest grade of ore, containing about 0.8 per cent copper, which has to be moved to get at other higher grade ore and to develop the workings properly, is sent to the ore stock pile if the copper market happens to be unfavorable. It is left there until such time as it can be most favorably utilized. Only the ore that contains many bowlders too large to pass the 12-inch by 13-inch grizzly at the mill is sent to the crusher. The other ore is sent direct to the mill, except the occasional carloads that average high in copper; these are shipped direct to the smelter.

The grade of the pit tracks varies considerably. The average grade of all the tracks to the waste dump, the stock pile, the crusher, and the railroad yards is estimated to be 2¼ per cent. Some idea of the amount of material handled over the company's tracks during the month of July, 1914, when the mine was working at normal capacity and during the month of September, 1914, when it had greatly curtailed its output owing to the European war, is given in the table following:

Data showing material handled on Chino company's tracks.

Month.	Ton-miles of haul.					Grade miles, haul to waste and ore piles and to crusher.
	Waste.	Ore piles.	Crusher.	Mill or smelter.	Total.	
1914.						
July	953,055	39,911	....	228,771	1,221,737	2,533
September	447,486	3,045	36,517	78,830	565,878	2,055

The figures in the second, third, fourth, and fifth columns of the above table are obtained by multiplying the average length of haul by the tonnage moved, thus giving the ton-miles; the figures in the last column are the product of the linear miles and the rise in feet per mile for hauls to the waste and ore piles and to the crusher. The cost of hauling out and dumping waste averaged a little less than 3 cents per cubic yard.

The tracks are of standard gage and consist of 60-pound rails laid on 7-inch by 9-inch ties 8 feet 2 inches long and spaced 20 ties for each length of 33-foot rails. A few 75-pound rails are used in the pits. The Chino company constructed at its own expense and is operating about 24 miles of railroad track. This trackage connects with the railroad yards of the Santa Fe branch line to Santa Rita. It is provided with ample turnouts and crossovers and is systematically operated under the supervision of a yardmaster and his assistants, switch tenders, etc.

The maintenance of the pit tracks is under the supervision of an assistant roadmaster. Subordinate to him are 4 track foremen, each in charge of a gang of 13 men. These gangs shift track and keep the track lines up to the steam shovels. The waste and ore-dump tracks are looked after by 7 track foremen, each in charge of a gang of 13 men. These crews level off and trim down the dumps and keep the tracks shifted out toward the edge of the dump slope. The pay of the laborers is about $2.05 per 10-hour shift. The track foremen receive $4.55 per 10-hour shift.

CRUSHING OF COARSE ORE.

The coarse ore grizzlies at the Hurley mill have openings 12 by 13 inches. It is intended that the ore shipped to the mill shall have few fragments too coarse to pass through these openings. Up to 1914 ore fragments coarser than this had to be broken by "dobe" blasting, or by hand, either at the shovel, on the cars, or at the mill. These methods of breaking the large fragments were relatively expensive. It was therefore decided to build near the mine a plant for crushing coarse ore, the plant to accommodate any bowlder that could be handled by the 3 to 3½ cubic yard dippers of the steam shovels.

A site for the new crushing plant was selected on a side hill, so that the ore cars could be run on a track about 50 feet higher than the feeding hopper of the large crusher (Pl. IX, *A*). It is situated about 1½ miles below the mine, and is convenient to the branch line of the Santa Fe Railroad which leads to the mill at Hurley. The cars of coarse ore from the mine are dumped onto the steel grizzlies (Pl. IX, *B*), the bars of which are about 8 inches apart. The oversize pieces roll down into a hopper that feeds to a 48-inch by 60-inch Superior jaw crusher. This vast crusher weighs 100 tons, and is driven by a 150-horsepower motor. It crushes to 6 inches and smaller, and has a capacity of 3,000 to 5,000 tons per 10 hours. A 48-inch overlapping steel pan conveyor, driven by a 50-horsepower Westinghouse induction motor, conveys the crushed product, and the fines that have gone through the grizzly bars (Pl. IX, *B*), up a 15½° incline to the top of the loading bin (Pl. IX, *A*). A number of observations showed that one-fortieth to one-fourth of the contents of each car dumped onto the grizzly went over into the crusher. The remainder went through the bars as undersizes or fines.

The loading bin, of steel, is 40 by 14½ feet at the base, and 23 feet high. The upper 9 feet of the longer dimension tapers in to 26 feet 8 inches instead of 40 feet as at the base. It has a capacity of 567 tons, on the basis of 21 cubic feet to the ton of ore. The bin has six loading chutes on each side, and is between two tracks, so that two large steel cars can be loaded at the same time. The loading chutes are high enough above the rails so that the loading is all by gravity. From 10 to 20 minutes is required to load a car having a capacity of 100,000 pounds. The coarser the material the easier the loading, after it is once started. Fine material sometimes chokes the chutes, until prodded loose by the bars of the car loaders.

About four 12-yard cars at a time are pushed by a dinkey locomotive from the mine to the up-grade side of the grizzly. The locomotive engineer then gives three blasts of his whistle. If the crusher man is ready for more ore he answers with two whistles. The first car is set opposite the grizzly and dumped by hand, then the other cars are set and dumped as fast as the crusher can take the ore. After all the cars in a trip have been dumped, the locomotive engineer sounds two blasts of his whistle, and pulls the dumped cars past the grizzly; after the cars have been uprighted and fastened, he proceeds to the mine for more material.

Five men are required at the top of the grizzly to dump and upright the cars. A trip of 4 cars can be dumped and uprighted in about 5 minutes. After the train has pulled out, the men shovel onto the grizzly any material that has fallen aside. The crew operating the crusher consists of a foreman, two mechanics, two crusher runners, four car men, one loader, and three laborers. The total operating wages amount to about $35 per 10-hour shift.

The total cost of building and equipping this crushing plant was about $127,000, of which $42,000 was for labor and $85,000 for supplies.

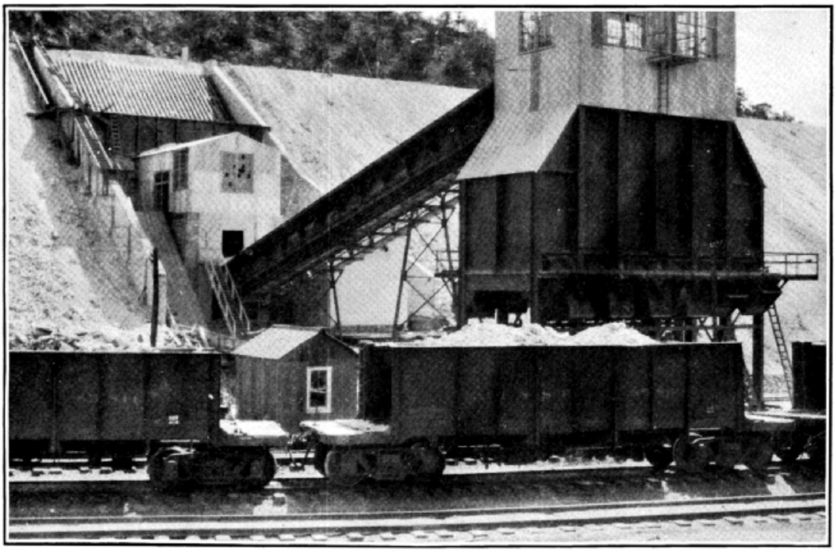
The total cost of operating the plant for crushing coarse material for a part of August, 1914, was $1,231.84, or at the rate of 1¼ cents per dry ton of ore. For September, 1914, the total operating cost of the plant was $1,994.84, or at the rate of 1.9 cents per dry ton of ore. This higher cost was due to some important reinforcements required for the retaining walls and additional equipment charges.

MINE DRAINAGE.

The precipitation in the Santa Rita region is so light that the amount of water that gets into the workings is relatively small. The slopes in some of the open cuts are such that they drain directly to the Santa Rita Creek. In the deeper open cuts sumps with electrically driven pumps take care of the water. The 300-foot level of the underground workings, which are old workings kept open mainly for drainage and prospecting, is drained by two Aldrich triplex pumps and one Byron-Jackson centrifugal pump, all operated by electric power. These pumps have a total capacity of about 1,000 gallons per minute, against a head of 400 feet. A reserve pumping plant on this level consists of two Cameron pumps, 16 by 7 by 18 inches. They are operated by compressed air, and are held in readiness in case of accident to the electric pumps or to the supply of electric power. On the 400-foot level the small amount of seepage is collected in a sump, and pumped to the 300-foot level by a 16-inch, three-stage, electrically driven Byron-Jackson centrifugal pump, having a capacity of 900 gallons per minute.

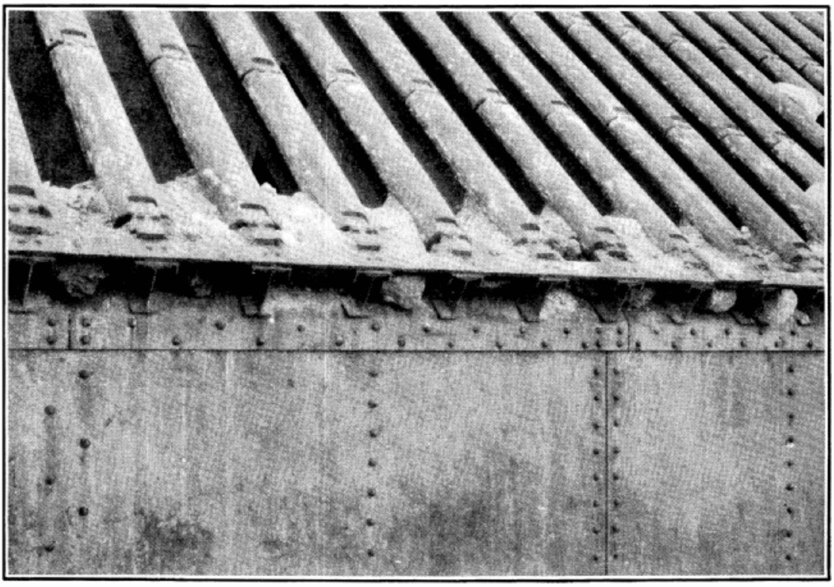
EFFECT OF SLOPES OF OPEN CUT.

The stability of the slopes of an excavation has an important bearing on the cost of the excavation. Where the slopes stand steeply the cost is much less, other things being equal, than where they slide to flat angles, owing to softening by ground water or other causes. If the slopes of the excavations at Santa Rita were as unstable as, say, those of the Culebra Cut, much would be added to the cost of mining.



A. PLANT FOR CRUSHING COARSE ORE, AND LOADING BIN, SHOWING CARS AND LOADING TRACKS.

The coarse material is hauled to this plant over the upper track, the grade of which is visible in the upper part of the picture, and dumped over the grizzly, from which the oversize goes to the crusher situated in the structure below and in front of the grizzly. An inclined belt conveyor with overlapping steel pans elevates the fines from the crusher and grizzly to the loading bin, from which the cars are loaded on the lower tracks.



B. VIEW OF PART OF THE GRIZZLY SHOWN IN PLATE IX, *A*. THE OPENINGS BETWEEN THE STEEL BARS ARE ABOUT 8 INCHES WIDE.

At Santa Rita great open cuts have been made and from these ore and waste are being excavated by steam shovels. These vast open pits are thus constantly being widened, from the central area outward, and are being deepened to the bottom of the ore body. The steam shovels operate on benches, as already explained, one or more to a bench, depending on the location of the ore bodies, the requirements of mining, etc. The bench levels, in this process of being widened, are cut back into the rising ground, and into the sides of the hills, necessitating slopes that, locally where ridges are cut, are several hundred feet high.

These high slopes, in places, may be largely in waste material that overlies valuable ore bodies; hence their stability has much to do with the cost of mining. If they stand steeply, much less excavation is necessary than if they slough down to flat grades. At Santa Rita the open-cut slopes have stood remarkably well, chiefly because of the following factors:

(1) There is little ground water in the soil, which covers the rocks to depths of a few inches to 30 feet or more. This is because of the relatively small precipitation and the good drainage conditions. The latter are due to the porosity of the soil and the slope of the ground toward the open cut.

(2) The upper or weathered zone, which varies much in thickness, depending on the character of the rocks, the degree of jointing, etc., is considerably cut by jointing (Pl. X), and some shearing. This weathered zone, together with most of the mineralized rocks, is locally much altered, not only by weathering, but also by hydrothermal action. The effect on the slopes is as follows:

(*a*) The weathering and mineralizing solutions have much sericitized the feldspars, thus weakening the rocks and causing a considerable degree of pulverization when they are blasted. With excessive ground water this condition would also tend to give them muddiness and mobility. However, ground water being largely absent, as already explained, they are strong enough to stand at fairly steep slopes without crushing or movement, except some sloughing, for all heights of slope that are likely to be encountered.

(*b*) The jointing has cut the rock mass into irregular blocks (Pl. X), and these will, of course, slough off where the slopes are rather steep, especially after they have been somewhat loosened and jarred by blasting. However, this jointing is not great enough to give a flowage-like motion, or "creep," to the jointed material, even on the highest and steepest slopes. Where faulting has sheared the rocks, and slikensided them, there is considerable local sloughing, but such zones are too restricted to cause a noticeable increase of mining cost.

The rocks below the weathered zone are less jointed and are more stable than those just described for the weathered zone; hence they stand fairly steeply.

The steepest slopes of the cut are those in the quartzite and other hard rocks. They range from 55° to 85°, depending on whether the principal joint planes dip toward the excavation. Probably the average slope in such rocks is about 70°, with the soil covering and loose rock on top sloping at about 40°. In the softer rocks slopes of 40° to 70° are maintained. In calculating the yardage of stripping to be removed, a slope of 1 on ½ was first used, meaning one unit vertical to one-half unit horizontal. However, in 1915 the use of a slope of 1 on ¾ was adopted as being safer for making ore estimates.

The flattest slopes are those at which the material in the waste dumps comes to rest. Several of the large dump slopes, some showing about 200 feet of elevation were measured, and they all approximated 37°, in spite of the fact that they contained much fine material. The upper half of each of the dump slopes was about 37½° and the lower half about 36½°. The increased flatness below is due to the tendency of the larger fragments to roll down and flatten the slope at the bottom.

COMPRESSOR PLANT.

The air-compressing plant contains two Nordberg compound compressors, each driven by a 200-horsepower electric motor. The air, when compressed to 90 pounds per square inch, is stored in two steel cylindrical receivers 4½ feet in diameter and about 13 feet long. One of these is a discarded steam boiler that serves its purpose reasonably well; the other was built as an air storage tank. From these reservoirs pipes distribute the air to the places in the workings where it is needed to operate the tripod and small hammer drills, and the few churn drills that formerly used steam but now use air. The power charges against the compressor are $550 to $650 per month.

CENTRAL POWER PLANT AND POWER DISTRIBUTION.

The Chino company has its power-generating and power-distributing center at Hurley, where the concentrating plant is situated. Power is distributed in the form of electrical energy generated in a large modernly equipped steam-driven plant. When the plant was visited by the authors in 1914, steam was supplied from eight Heine safety boilers, each of 445 horsepower capacity. One or two of these were always in reserve, or being cleaned, or undergoing repairs, while the others were in active operation. Two additional 419-horsepower boilers were being added. The boilers were equipped with superheaters, and with mechanical stokers of the Greene and American types. They used natural draft, except the two new ones, which had forced draft.

A. FACE OF EXCAVATION, SHOWING JOINTING.

B. ANOTHER VIEW SHOWING JOINTING.

The plant operated with steam at a pressure of 180 pounds and heated to 370° F. One fireman and two helpers, per eight-hour shift, operated the plant. The additional boilers called for two more helpers per 24-hour day. In the boiler room were also two boiler washers and one repairman. Feed water heated to a temperature of 190° to 195° F. by Stilwell heaters, and measured by V-notch weirs, is supplied to the boilers by three automatically controlled pumps, each 8 by 12 by 7 by 12 inches.

The boilers supplied steam to three Nordberg compound engines, each 27 by 60 by 48 inches. Each engine drove a 1,250-kw., 480-volt, Allis-Chalmers generator. Steam was also supplied to a 150-horsepower Ballwood tandem compound engine, which drove a 100-kw., 250-volt, Allis-Chalmers generator. Another 100-kw., 250-volt, generator, of the Westinghouse type, was driven by a 150-horsepower Westinghouse steam turbine, and one 100-kw., 250-volt, Allis-Chalmers generator was driven by a 150-horsepower Allis-Chalmers alternating-current motor. There were other smaller motors about the plant. A Worthington surface condenser, operated by steam-driven dry-vacuum and motor-driven wet-vacuum pumps, provided water for each dynamo engine.

The water from these condensers, the condensate, was pumped into a series of baths, in which the cylinder oil and graphite was removed. The cleaned water was then returned to the feed-water storage tanks in the boiler house. The exhaust from the other engines went direct to the feed-water heater. The water for cooling the condensers was circulated by two 16-inch Worthington volute pumps driven by 200-horsepower Allis-Chalmers induction motors operated at 690 revolutions per minute against a pressure of 37 pounds. Only one of these pumps was used at a time, the other being held in reserve.

During July, 1914, the cost for the coal used at the Hurley plant was 33.2 cents per 1,000 pounds of steam generated, and during September it was 32.7 cents.

In the transformer room the plant contained two banks, each of three single-phase, 400-kilovolt-ampere, 60-cycle, Allis-Chalmers transformers, oil-insulated and water-cooled. These transformers could be run in multiple or separately, and they stepped the current from 480 to 24,000 volts.

Attached to the power plant there was an oil-filtering plant for treating used oil. It had a capacity of 25 gallons per minute and a storage capacity of 450 gallons. The used oil drained by gravity to the filtering room and three small double pumps circulated it to the pressure filter tanks. After filtration, the oil was forced by air pressure to the engine room for reuse. All oil pipe lines leading to and from the filter room were colored. For example, yellow indicated engine oil, brown indicated the cylinder-oil drain to the filter, green the crank-pin oil drain, etc. The cylinder oil required two filterings. The cotton waste was cleaned by a centrifugal machine.

During the year 1913 the power-generating plant had a net output of 24,198,633 kilowatt hours. The average cost for the year for current delivered to the feeder circuits on the switchboard was $0.012 per kilowatt-hour, not including the distribution losses. An analysis of this cost is given in the following table:

Analysis of cost of generating power at Santa Rita plant in 1913.

Item.	Total cost.	Cost per kilowatt-hour.
Overhead expense, direct supervision	$6,081.68	$0.000251
Labor, operating	27,970.48	.001156
Labor, repairs	8,864.88	.000366
Supplies, operating	167,649.56	.006928
Repair materials	4,597.14	.000190
Total direct cost	215,163.74	.008890
Overhead expense, general administrative (estimated)	7,600.00	.000314
Taxes (estimated)	5,530.00	.000229
Fire insurance (estimated)	1,383.00	.000057
Interest on plant, $553,000, at 6 per cent	33,180.00	.001371
Depreciation, 4 per cent (amortization fund based on 15-year life, 6.66 per cent depreciation at 6 per cent interest, compounded semiannually)	22,120.00	.000914
Gross cost of generation	284,976.74	.011776

During the month of September, 1914, the Santa Rita operations consumed 105,529 kilowatt-hours. The distribution of power for the month of July, 1914, when the mine was operating at normal capacity, and for September, 1914, when it was operating at about 50 to 60 per cent capacity, was as follows:

Distribution of power in Santa Rita operations during two months.

Item.	Kilowatt-hours.	Electrical horsepower-hours.	Kilowatts.	Electrical horsepower.
JULY, 1914.				
Milling	1,521,101	2,039,011	2,044.49	2,740.60
Pumping	478,514	641,439	643.16	862.15
Current for town of Santa Rita	202,246	271,107	271.84	364.39
Shops	12,150	16,286	16.33	21.89
Lights	30,560	40,965	41.07	55.06
Total	2,244,571	3,008,808	3,016.89	4,044.09
SEPTEMBER, 1914.				
Milling	981,331	1,315,459	1,362.97	1,827.04
Pumping	342,971	459,746	476.34	638.54
Current for town of Santa Rita	105,529	141,459	146.57	196.47
Shops	12,150	16,286	16.87	22.61
Lights	22,913	30,714	31.82	42.65
Total	1,464,894	1,963,664	2,034.57	2,727.31

SANTA RITA POWER AND SHOP EQUIPMENT.

A high-tension transmission line from the company's electric power station at Hurley, about 11 miles south, furnishes power to operate the shop machines, compressor, plant for crushing coarse material, pumps, etc., and to light the town of Santa Rita. The line is designed to carry a pressure of 24,000 volts, but this high pressure is stepped down by transformers at Santa Rita to 440 volts for use in motors and to 110 volts for lighting.

In general, the repair plant may be said to consist of a machine shop, with blacksmith and boiler shops attached, and a carpenter shop. At the time of the authors' visit in 1914, the working force of the mine had been considerably reduced, owing to war conditions. At that time the machine shop employed 3 blacksmiths, 4 blacksmith helpers, 1 drill sharpener and helper, 9 machinists, 5 machinist's helpers, 4 boilermakers, 4 boilermaker's helpers, and 9 car repairers. The blacksmiths and machinists were paid $4.50 for 9 hours and the boilermakers $5. The car repairers got $3 to $4 for 9 hours. All the helpers were paid $2.50 to $3 for 9 hours.

The shop equipment was as follows:

Shop equipment of Chino Copper Co. in 1914.

1 Sellers power hammer, operated by air.
1 coke furnace for extra-heavy forging.
2 blacksmith forges.
3 lathes: 1 Davis, 12-inch; 1 Bradford, 18-inch; and 1 Pond, 36-inch.
1 Acme bolt cutter.
1 Williams pipe machine.
1 Barnes drill press.
1 Miles radial drill.
1 Steptoe shaper.
3 emery wheels: 1 wet, 1 dry, and 1 drill grinder.
1 American planer.
1 Long & Allstatler punch and shears.
1 Watson Stillman portable wheel press.
1 6-ton hand crane.
1 power hack saw.
2 Buffalo forge blowers, operated by a Allis-Chalmers 25-horsepower motor.

In the drill-sharpening shop, a No. 3 Leyner drill-sharpening machine, operated by two men, took care of all the steel for the tripod drills. In this shop there was also a boiler flue scaling machine, an oil furnace for heating flues, a flue welder, a flue swedging machine, and a circular saw for cutting wood, all driven by a 15-horsepower motor.

Before the water-softening plant was installed the locomotive and steam-shovel flues had to be taken out and the scale removed about every three months, at great expense. Subsequently the frequency of scaling has been reduced to about once each year. Flues are welded, swedged wherever necessary, and used over again. The composition of the boiler scale taken out was about as follows:

Composition of boiler scale from flues of Chino Copper Co.

Constituent.	Per cent.
Volatile matter	33
Insoluble	13
Al_2O_3	6
Fe_2O_3	14
MgO	1
CaO	28
CO_2	3
S	2
Total	100

Locomotive, steam-shovel, drill, and pump parts were kept in stock. Car axles, car wheels, and all castings were received in the rough and were machined in the company's shops when needed. Locomotives and steam shovels were practically rebuilt on the premises when necessary.

The carpenter shop, a large building, had a lumber yard attached in which about 100,000 feet of material was always kept on hand. A 25-horsepower motor drove a circular saw, a handsaw, a swing cut-off saw, a lathe, and other accessories. Nine carpenters and four helpers were employed in 1914. Carpenters were paid $4.50 per day of 9 hours and their helpers up to $3 per day. Mexican laborers working around the carpenters' shop got $2 per 10-hour shift. The construction and repair of all the company's dwelling houses and other buildings, as well as bridge building, timber framing, etc., came under the charge of the foreman carpenter.

POWER COSTS.

During the month of September, 1914, after the mine work had considerably slowed down, 105,529 kilowatt hours of electrical energy was used at Santa Rita at a total cost of $1,075.31. Of this cost $247.10 was for labor, and $828.21 for supplies. The distribution of the cost was as follows:

Distribution of cost of power generated for town of Santa Rita by Chino Copper Co.

Item.	Per cent.	Cost.	Item.	Per cent.	Cost.
Compressor	40	$430.12	Carpenters' shop	1	$10.75
Machine shop	3	32.26	Light service, town lights, etc.	8	86.03
Assaying and sampling	1	10.75	Crusher operation	26	279.58
Mine drainage; electric pumps.	6	64.52	Total	100	1,075.31
Water supply; 3 pumping stations	15	161.30			

This cost amounts to 1.02 cents per kilowatt-hour, or 0.76 cent per horsepower-hour. Assuming that the power was used for 720 hours, the number of hours in the month, the cost per horsepower per month was $5.47.

WATER SUPPLY AT SANTA RITA.

The somewhat arid climatic conditions of the region give particular importance to a water supply adequate to the needs of such extensive mining operations as those at Santa Rita. However, a series of wells and some small streams supply ample water and a water-softening plant eliminates the excess of calcium and other salts and renders the supply satisfactory for domestic and boiler use.

The largest well at Santa Rita is the Pinder. It is near the bed of Santa Rita Creek, and much of its supply comes from the subsurface flow of that creek. A Dean triplex 9-inch by 12-inch electric pump lifts the water from this well and forces it through 3,600 feet of pipe against a difference in elevation of about 410 feet. The diameter of the discharge aperture of the pump and of the first 30 feet of the pipe is 7 inches; the next 2,500 feet is 8-inch pipe, and the last 1,100 feet is 12-inch pipe. The gage on the water pipe near the pump shows a pressure of about 200 pounds per square inch. Against this pressure the pump throws 175 to 200 gallons per minute, more or less. It is operated by an Allis-Chalmers, 75-horse power, three-phase motor, using a 440-volt current. The curvature of the pipe line and the lengths of the different sizes of pipe are shown in figure 15. The well is also equipped with a steam auxiliary plant for use in case of necessity. During October, 1914, 4,965,230 gallons was pumped from the Pinder well to the water-softening plant, or at the average rate of 10,558 gallons per hour of pumping. In addition, 1,449,782 gallons was pumped for domestic purposes from "drill hole I" and 1,440,450 gallons from the Booth well for general supply.

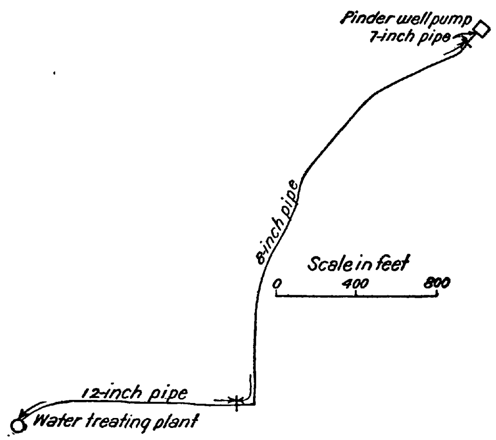


FIGURE 15.—Horizontal curvature and lengths of different sizes of pipe of Pinder well pipe line.

Each blast-hole churn-drill machine uses about 200 to 300 gallons of water per eight hours. The larger prospecting churn drills each use about 500 gallons per eight hours for all purposes. Steam shovels each require 6,000 gallons, and mine locomotives about 4,000 gallons each per 10 hours.

WATER-SOFTENING PLANT.

A Reisert automatic water-softening plant, with a capacity of 9,000 gallons per hour, takes the excess of calcium and allied salts out of the water so as to make it suitable for boiler use. The plant is situated on a hill, about 300 feet above the town and about 350 feet above the mine, where the water is mostly required.

Figure 16 shows the flow sheet of the plant. The raw water is admitted to the bottom of the saturator, where it comes into contact with cream of lime previously supplied from the lime-mixing or slaking compartment. In rising and mixing with the cream of lime, the raw water reaches the top of the tank in a saturated condition and nearly clear. From the lime saturator it flows into the mixing pan at the top of the settling tank, coming into contact there with raw water and with soda ash solution from the soda compartment. The proper proportions of these inflowing solutions are automatically regulated by float valves. After the raw water, the saturated lime water, and the soda ash solution have been thoroughly mixed they pass from the mixing pan through the downtake pipe to the bottom of the settling tank. From there the mixture slowly rises to the top, and the precipitates resulting from the chemical reaction slowly settle out. At the top the water is fairly clear, but it overflows into filter tanks where the last of the suspended matter is removed. Emerging from these, it goes to a storage tank ready for use. The filter tanks are washed out, whenever necessary, by reversing the flow. The solid matter may be drawn from the bottom of the settling tank by opening a valve.

The plant is operated by one man under the supervision of the mine assayer. The mechanical regulation and operation of the plant can be taught to the average operator in a few days.

FUEL.

The fuel used by the Chino company comprises bituminous and subbituminous coals from northern New Mexico. Run-of-mine coal and No. 2 lump are principally used. Under normal working conditions about 63,000 tons is used annually at Santa Rita. The average cost of coal delivered there is $4.35 per ton, more than half of which is for freight charges.

When received the coal is unloaded to storage piles. To keep the temperature down and prevent fires from spontaneous combustion, the piles are built around ventilating stacks, which are wooden conduits a foot square made out of 1-inch or 1½-inch boards in which many holes have been bored (Pl. V, *B*).

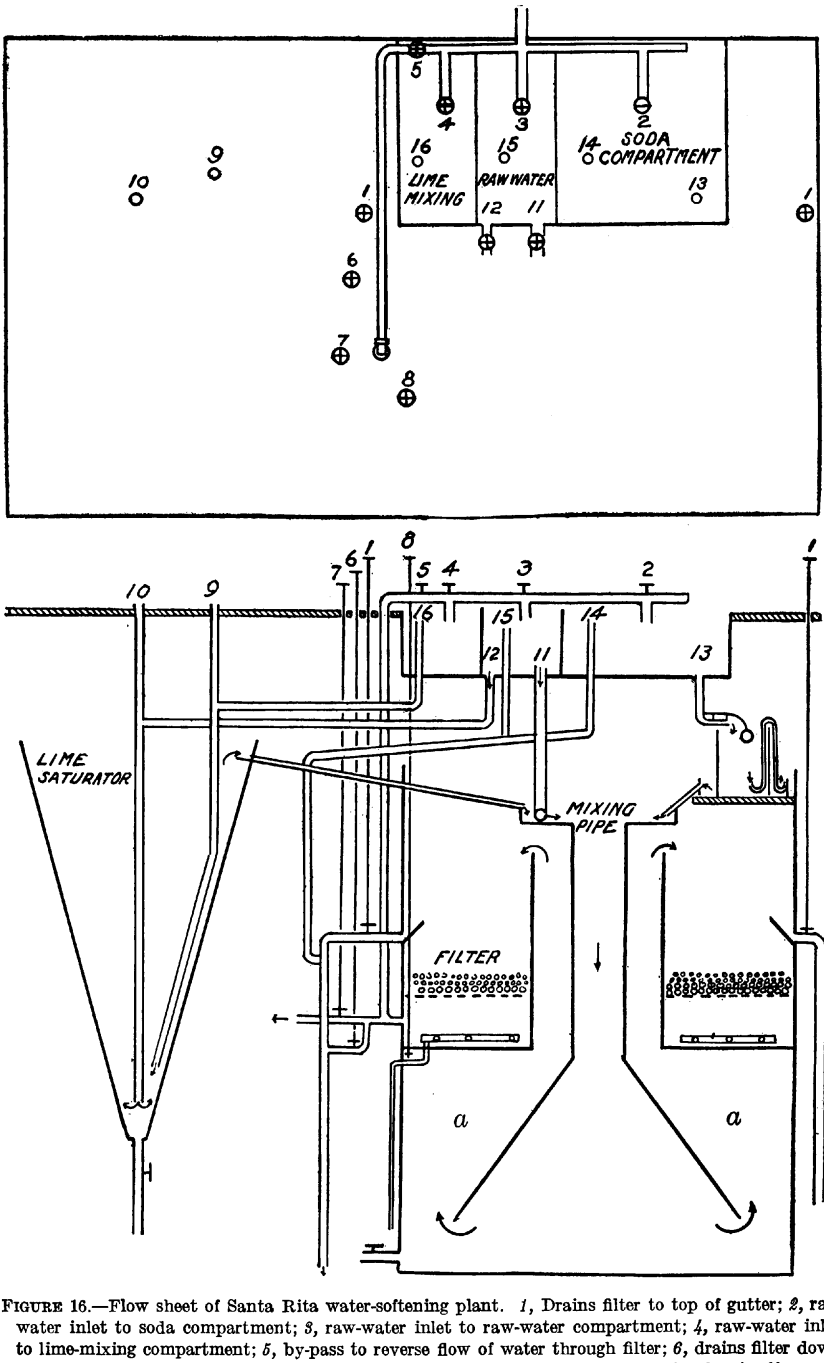


FIGURE 16.—Flow sheet of Santa Rita water-softening plant. *1*, Drains filter to top of gutter; *2*, raw water inlet to soda compartment; *3*, raw-water inlet to raw-water compartment; *4*, raw-water inlet to lime-mixing compartment; *5*, by-pass to reverse flow of water through filter; *6*, drains filter down to tight head; *7*, treated-water outlet; *8*, air valve for cleaning filter; *9*, open pipe for cleaning lime passage; *10*, open pipe for cleaning lime-saturator outlet; *11*, controls flow of raw water through treating plant; *12*, controls flow of saturated lime water; *13*, soda-solution outlet; *14*, drain for cleaning out soda compartment; *15*, drain for cleaning out raw-water compartment; *16*, passage from lime-mixing compartment to saturator; *aa*, *settling tank*.

MINING COSTS.

Economy of production is the policy that, other things being equal, offers the greatest opportunity for success in mining, as in most other lines of business. The cost figures established by the Chino company should be of interest, not only to mining men, but also to engineers and contractors who have to do with excavations of various kinds.

The total average cost for blasting, loading, and hauling per ton of ore for the period October, 1910, to September, 1914, was about 20 cents; of this, about 12 cents was for labor and about 8 cents for supplies. The lowest average monthly cost was 12.9 cents per ton, 7.8 cents of which was for labor and 5.1 cents for supplies. The highest average monthly cost was 60.5 cents for July, 1911; 43.5 cents of that cost was for supplies and 17 cents for labor. The purchase of an unusually large store of supplies, charged against this month, caused this abnormally high average.

The total average cost of stripping per cubic yard, from October, 1910, to August, 1914, was 31.5 cents, equivalent to 15.2 cents per ton. Of this, 20.1 cents was for labor and 11.4 cents for supplies. During the same period, the lowest monthly average cost per cubic yard was 23.4 cents, 15.2 cents of which was for labor and 2.8 cents for supplies. The highest monthly average cost was 36.8 cents per cubic yard, of which 24.3 cents was for labor and 12.5 cents for supplies.

During the year 1913, the average cost of mining and hauling to the mill and dumps all classes of material was 36.87 cents per cubic yard. The cost of handling waste alone was 33.43 cents per cubic yard in place, or a little over 16 cents per ton. The cost per ton for blasting, loading, and hauling the ore was 23.13 cents per ton. In general the cost per ton is greater for ore than for waste because of the limited areas in which ore is loaded and the consequently more intermittent character of the loading. Other delays arise from occasional shortage of standard railroad cars for loading ore. The breaking of large blocks of ore by blasting formerly added something to the ore cost. The new plant for crushing coarse material at the mine is intended to obviate such blasting.

MILLING.

About 10 miles southwest of the terminus of the railroad line at Santa Rita, it emerges from the narrow winding Whitewater Valley and gains the open plateau. Near here, with many miles of open country in which to expand, the mill town of Hurley was laid out, and the concentrating plant was built. The discussion following is not intended to be a complete description of this splendid modern mill, but it calls attention to certain details not mentioned in the account of the mining operations.

The ore is hauled from the mine to the mill by the Santa Fe Railroad Co. Each train, handled by one locomotive and composed of 25 to 27 50-ton steel cars, carries 1,250 to 1,350 tons of ore. At Hurley the cars are run out on a very high trestle over the top of the crude-ore bins, into which they discharge by gravity. The course of the ore through the crushing plant at the mill is shown by figures 17 and 18.

Description of parts shown in figure 17.

1. Fifty-ton side-dump ore cars. Weight of car, 44,700 pounds. Lengtn, 35 feet.
2. Grizzlies of 60-pound rails over bins; openings, 12 by 13 inches.
3. Crude-ore bins. Total storage capacity, 14,500 tons; available storage capacity without poking, 5,000 tons; each bin 34 feet wide by 28 feet 6 inches deep by 300 feet long inside.
4. Forty feeders of the caterpillar type. Speed of apron, 10 feet per minute; capacity of each feeder, 1.25 tons per minute—75 tons per hour.
5. Two 30-inch conveyor belts moving 245 feet per minute; about 325 feet from center line to center line.

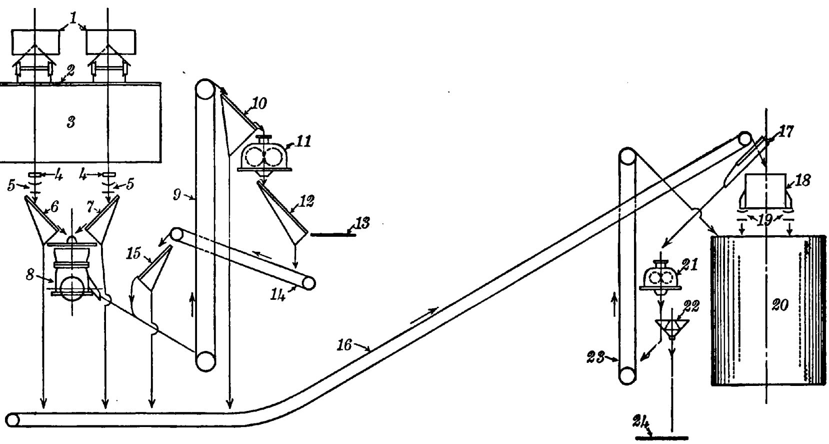


FIGURE 17.—Flow sheet of plant for crushing coarse material, Hurley mill.

6. Grizzly, 1 inch by 4 feet by 12 feet; bars $\frac{3}{4}$ inch by $\frac{1}{2}$ inch by 3 inches.
7. Grizzly, 1 inch by 4 feet by 11 feet; bars $\frac{3}{4}$ inch by $\frac{1}{2}$ inch by 3 inches.
8. No. 8 McNally crusher; 375 revolutions per minute; crushing to 3 inches maximum size.
9. Elevator; belt 36 inches, 12 ply; head pulley, 60 by 38 inches; boot pulley, 48 by 38 inches; 49 feet 4 inches from center line to center line; belt speed, 440 feet per minute; buckets, $17\frac{1}{2}$ by 8 by 10 inches; projection, 18 inches between buckets.
10. Grizzly, 1 inch by 4 feet by 9 feet; bars, $\frac{3}{4}$ inch by $\frac{1}{2}$ inch by 3 inches.
11. Rolls, 72 by 20 inches; peripheral speed of roll shells when new, 1,000 feet per minute.
12. Grizzly, $2\frac{1}{2}$ inches by 3 feet by 8 feet 9 inches; for removing large pieces of native copper, wood, etc.
13. Steel platform at bottom of grizzly.
14. Twenty-inch conveyor belt; 20 feet from center line to center line; inclination, 20°; speed, 330 feet per minute.
15. Grizzly, 1 inch by 3 feet by 7 feet.
16. Conveyor belt, 36 inches wide; about 275 feet from center line to center line; maximum inclination, 20°; speed, 300 feet per minute.
17. Primary sampler; cuts sample every five minutes.
18. Receiving hopper.
19. Two 24-inch shuttle conveyors; 24-inch belts, 120 feet long; speed, 350 feet per minute.
20. Ten steel tanks, 24 feet 6 inches in diameter by 40 feet high.
21. Rolls, 14 by 27 inches.
22. Vezin sampler, cuts 20 per cent of (*21*) roll product.
23. Elevator, returning material rejected from samplers to bins for fine ore; 6-ply belt, 6 inches wide.
24. Floor on which sample is quartered by hand.

Description of parts shown in figure 18.

1. Fine-ore bins; two cylindrical steel tanks 24 feet 6 inches in diameter by 40 feet deep; approximate capacity 1,000 tons each.
2. Four feeders of the caterpillar type.
3. Two 20-inch belt conveyors, about 20 feet from center line to center line; speed, 250 feet per minute.
4. Two 24-inch wet elevators; 24-inch, 12-ply belts; speed, 400 feet per minute; about 62 feet from center line to center line; buckets staggered on belts; pitch of alternate buckets, 9 inches; buckets 12 by 7 by 8 inches.
5. Six 36-inch by 48-inch impact screens; 600 impacts per minute.
6. Two 42-inch by 16-inch rolls; peripheral speed of shells when new, 1,000 feet per minute.
7. Four 36-inch by 48-inch impact screens; 600 impacts per minute.
8. Two jigs with two 6-mesh screens each; screens 24 by 36 inches; 112 revolutions per minute; 1½-inch stroke.
9. One 30-inch wet elevator; 30-inch, 12-ply belt; speed, 450 feet per minute; 60 feet from center line to center line; buckets, 15 by 8 by 10 inches, staggered on belt; pitch of alternate buckets, 9 inches.
10. Four 4-spigot Richards-Janney classifiers.
11. Two desanding tanks, 5 feet 3 inches by 7 feet 2 inches by 6 feet 9 inches deep.
12. Chilean mill feed tank, 3 feet by 24 feet by 6 feet 6 inches deep.
13. Three Garfield Chilean mills, 32½ revolutions per minute.
14. Two three-spigot Richards-Janney classifiers.
15. Two three-spigot Richards-Janney classifiers.
16. Four five-spigot Richards-Janney classifiers.
17. Six 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ⅞-inch stroke.
18. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ⅞-inch stroke.
19. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ⅞-inch stroke.
20. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ⅞-inch stroke.
21. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ⅞-inch stroke.
22. Six No. 5 Wilfley tables; 245 revolutions per minute; ⅞-inch stroke.
23. One double three-compartment jig; 188 revolutions per minute; 2-inch stroke; 26-inch by 38-inch screen; first and second compartments, 7-mesh, 0.035 brass wire; tail screen, 9-mesh, 0.035 brass wire.
24. Four Isbell vanners; corrugated belts, 6 by 10 feet; 2-inch side stroke; 130 revolutions per minute; 7½-inch slope.
25 and *26*. Each consists of four vanners same as *24*, but having 7-inch slope.
27. Four vanners same as *24*, but having 6½-inch slope.
28. Four vanners same as *24*, but having 6-inch slope.
29. Four vanners same as *24*, but having 7-inch slope.
30. Four vanners same as *24*, but having 6-inch slope.
31. Four vanners same as *24*, but having 6-inch slope.
32. Twelve vanners same as *24*, but having smooth belts; 120 revolutions per minute; 4½-inch slope.
33. Thirty-two vanners same as *32*.
34. Twenty-eight vanners same as *32*.
35, *36*, *37*, and *38*. Each consists of ten steel-cone tanks; 9 feet 5 inches in diameter; 60° slope.
39. Fifteen double wooden tanks; water-recovery plant, equivalent to 50 cone tanks or 10 cone tanks per section; serving five sections.
40. Two slime pumps.
41. Four pumps; 10-inch, special, class "B" volute; two sets of two each, series connected; serve five sections.
42. Two muddy-water sumps, serving five sections.
43. Four pumps; 10-inch discharge; special, class "B" volute, serving five sections.
44. Two sumps for all vanner concentrates, serving five sections.
45. Two 4-inch centrifugal pumps, series connected, taking material from sumps *44*, and serving five sections.
46. One 4-compartment Richards-Janney classifier.
47. One 4-foot by 12-foot Garfield table, modified.
48. One 4-foot by 12-foot Garfield table, modified.
49. One 4-foot by 12-foot Garfield table, modified.
50, *51*, and *52*. One vanner same as *24*, but having 6-inch slope.
53. Concentrate sump, serving five sections.
54. Three 6-inch centrifugal concentrate pumps, serving five sections.
55. One primary concentrate sampler, Vezin type, serving five sections.
56. One secondary concentrate sampler, Vezin type, serving five sections.
57. Sixteen concentrate bins, 22 feet 6 inches by 15 feet by 12 feet deep. All bins are served by a 4-ton Gantry crane with a 24 cubic foot clamshell bucket.
58. One tailings sampler, pendulum type, cutting the tailings stream once in 6 minutes; serves five sections.

NOTES.—*46* to *52* inclusive, constitute concentrate cleaning plant. Three such plants serve five sections.

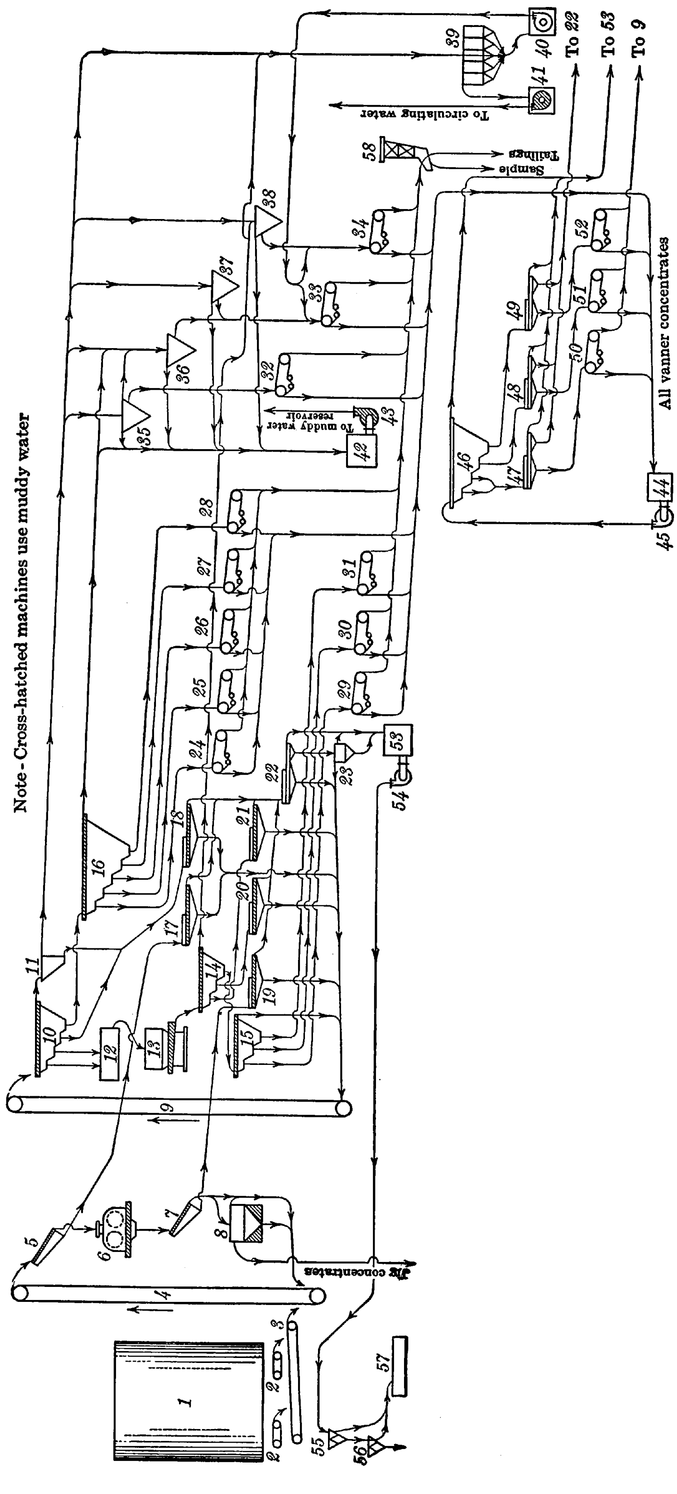


FIGURE 18.—Flow sheet of a section of operations at Hurley mill.

Since the installation of the crushing plant for coarse material at the mine, the grizzlies (*2*, fig. 17) across the top of the crude-ore bins, consisting of 60-pound rails giving openings 12 by 13 inches, are seldom needed. The capacity of the receiving bins *3* is 5,000 tons, and they discharge, through 40 caterpillar feeders, *4*, arranged along the bottom, 20 on a side. These supply two 30-inch conveyor belts, *5*, which carry the ore to the second set of grizzlies, *6* and *7*, from which the undersizes drop onto the main belt conveyor *16*, and the oversizes go into a No. 8 McNally crusher, *8*, breaking to a maximum size of 3 inches. From this the total product is raised by an elevator, *9*, and passes over a third grizzly, *10*, with openings ¾ by ½ inch by 3 inches. The oversize from this goes to rolls *11* and the undersize to the main belt conveyor *16*.

Another grizzly gets the product from the rolls *11*. This treatment is mostly to take out, as oversize, flattened pieces of native copper, wood, etc. The undersize goes to another elevator, *14*, then over another grizzly, *15*. The oversize from this is elevated and again passes through the rolls *11*, the undersize going to the main belt conveyor *16*. The latter elevates the crushed ore to a receiving hopper, *18*, which distributes it onto shuttle-belt conveyors, *19*. These discharge into one of 10 steel ore tanks, 24½ feet in diameter and 40 feet high. A primary sampler, *17*, cuts a sample from the crushed-ore stream every five minutes. This sample passes through rolls, *21*, and is then resampled by a Vezin sampler, *22*, the rejected material being returned to the fine-ore bins by an elevator, *23*. The sample retained is quartered by hand on a table, *24*.

From the fine-ore bins (*1*, fig. 18) caterpillar feeders, *2*, deliver the ore to belt conveyors, *3*. From them it is raised by an elevator, *4*, and delivered to impact screens *5*, with the oversizes going to rolls *6*, and the undersizes to Garfield tables *17*. The product from the rolls goes to other impact screens *7*, the fines from which go to Garfield tables *19*, and the oversizes to jigs *8*. The jig oversizes go back to the elevator *4*, and the concentrates finally reach the concentrate bins. From the Garfield tables the concentrates go to Wilfley tables *22*, and the tailings go back by elevator *9* to classifiers *10*. Of the five products from these, the two coarsest go to Chilean mills *12* and *13*, the finest to desanding tanks *11*, the next finest to fine classifiers *16*, and the intermediate to Garfield tables *18*. All the products from the fine classifiers *16* go to vanners *24* to *28*. The products from the Chilean mills *12* and *13* go to classifiers *14*, which give three products, the two coarser going to Garfield tables *20* and *21* and the finest to a fine classifier, *15*. The three products from the latter go to vanners *29* to *31*. The Wilfley tables *22* give three products, the coarse tailings going back by elevator *9* to classifiers *10*, the intermediate to jigs *23*, and the concentrates to concentrates sump *53* and thence to storage bins *57*. All the concentrates from vanners *24* to *31* go to sumps *44*, from which pumps *45* elevate them to the cleaning plant *46* to *52*. The tailings from the vanners go out onto the waste dump, being automatically sampled by samplers *58* on the way. The overflow from the desanding tanks *11* and from the classifier *16* goes to a series of steel-cone settling tanks *35* to *38*. From these the overflow goes to muddy-water reservoirs *42*, where it is somewhat clarified by settling, so as to be used over again. The slimy sediment from the cones goes to smooth-belt vanners *32* to *34*, which give tailings that go to the waste dump and concentrates that go to the cleaning plant *46* to *52*. This comprises a Janney classifier, modified Garfield tables, and vanners, designed to handle very fine material. The concentrates from the vanners of this cleaning plant are mixed with the concentrates from all the other vanners, and go through the cleaning plant again; the tailings go back to the classifiers *10*. The intermediate product from the modified Garfield tables of the cleaning plant goes back to Wilfley tables *22* and the concentrates from the latter go to the concentrates sump *53* and bins, as do the concentrates from the Wilfley tables and the overflow from the classifiers *46* of the cleaning plant.

The foregoing outline, in conjunction with figures 17 and 18, gives in brief form the chief operations of the mill. Since the time of the authors' visit in 1914 alterations and improvements have been added to the plant, but these are perhaps too technical to be treated in any but a report addressed to metallurgists.

The mill has a maximum capacity of almost 8,000 tons per 24 hours, when the character of the ore is favorable, but 6,000 tons is considered a fair daily average. To operate the total milling plant, consisting of five concentrating sections similar to that shown in figure 18, and the coarse-crusher section (fig. 17), about 3,500 horsepower of electrical energy is required.

The average extraction for 1913 was 67.31 per cent from 1,942,700 dry tons milled, equivalent to 5,322.4 tons per day. During a period of about five months a section of the mill was idle owing to modification in the scheme of treatment; hence the tonnage for the year does not represent full capacity. The ratio of concentration for the year was 10.61 to 1, with the concentrates averaging 14.518 per cent copper, or a recovery of 27.37 pounds per ton of ore. The ore treated for the year averaged 2.033 per cent copper. The ratio of concentration was lower than in 1914 because of the high iron content of certain of the ores mined during the year.

The ores treated during 1914 carried less iron and gave a better concentration. For the month of September, 1914, the ratio of concentration was 17.32 to 1, from 101,300 dry tons milled. The lessened tonnage treated was due to the war conditions. The actual recovery was 73.438 per cent of the copper content of the ore. The concentrates carried 27.67 per cent of copper. Any excess of silica over iron is penalized by the smelter. Toward the end of 1913 a concentrate recleaning plant was added to the mill and this helped to make a cleaner product, thereby lessening the freight and smelter charges on the copper actually produced.

HURLEY WATER SUPPLY.

A most important question which had to be solved before a site for the new mill could be decided on was the matter of an adequate water supply. Such a supply, it was found, could be obtained from springs, wells, and the subsurface flow of creeks within piping and pumping distance of the mill site at Hurley. An electrical pumping station on the "B Ranch" has a capacity of 750 gallons per minute. The pump is an 11-inch by 12-inch Aldrich triplex, driven by a 75-horsepower motor, which has to overcome a head of 250 feet. Similar pumping plants are on the Cameron and Whiskey Creeks. The Apache Tejo pumping station has two 14-inch by 18-inch quintuplex Aldrich pumps, each having a capacity of 2,500 gallons per minute; each is driven by a 400-horsepower electric motor, which has to overcome a total head of 400 feet. Only one of the pumps at the Apache Tejo station is operated at a time, the other being held in reserve. The supply of water for domestic use in the town of Hurley comes from this station.

Two reservoirs are in use at Hurley, one for clear and the other for muddy water. The clear-water reservoir gets the water from the pumping stations above described. It is 210 by 144 feet, and 12 feet deep. The muddy-water reservoirs get the settled water, already used in the mill, and caught and partly settled by the Whitewater No. 2 Dam. The size of this reservoir is about 60 by 70 feet and 12 feet deep. The muddy water is handled by six 10-inch, Class B, Worthington volute pumps, each having a capacity of 2,500 gallons per minute against a head of about 95 feet, and each driven by a 75-horsepower motor. The Whitewater No. 2 Dam and the settling reservoirs near it have a storage capacity of about 15,000,000 gallons. The water that flushes the tailings out of the mill is there ponded and settled to be used again. When the new spillway at the Whitewater No. 1 Dam is completed there will be a storage capacity for muddy and flood water of 500,000,000 gallons. The muddy water, after partial clarification by settling, is used over again on certain of the tables and classifiers. It may contain up to 4 or 5 per cent of solids.

The total amount of new water used in the mill during the month of July, 1914, was 52,569,637 gallons. With this about 560,000,000 gallons of muddy water was pumped to be used over again. The total ore treated during the month was 188,400 tons. The water used per ton of ore treated was 325.6 gallons of new water and 2,961.8 gallons of muddy water, a total of 3,287.4 gallons. The water used for domestic purposes in the town of Hurley during July amounted to 8,768,775 gallons.

MILLING COSTS.

The total average cost for all the ore treated in the mill up to September, 1914, was 58 cents per dry ton. In 1914, the lowest average cost for one month was 47 cents per dry ton, for June; during that month 201,000 dry tons was treated. The highest monthly average was 58 cents for the month of September, 1914, with only 101,300 tons milled. Since the mill opened, the total average cost of crushing coarse material (fig. 17) to a maximum size of 1¼ inches, has been 4 cents per dry ton, including supplies, operation repairs, remodeling, etc. During September, 1914, in spite of the lessened tonnage handled, the crushing of coarse material amounted to only 1.8 cents per dry ton. The crushing of fine material to a maximum size of 40 mesh, has cost, on the average, 11 cents per dry ton. For September, 1914, it averaged 9.7 cents. The concentration of the crushed material on the different tables, vanners, jigs, etc., has averaged about 13 cents per dry ton.

The total average milling expense, since the mill opened up to September, 1914, expressed in cents per dry ton of ore treated, is the resultant of the following charges: Light and power, 7.9 cents; water service, 4.8 cents; laboratory and sampling, 1.3 cents; Hurley office expense, 1 cent; engineering and surveying, 0.2 cent; tailings expense, ponding tailings, etc., 0.9 cent; mill heating, 0.1 cent; warehouse expense, 0.5 cent, making a total direct operating expense of 16.7 cents per dry ton of ore treated. A general mill-expense charge of 6 cents, probably covering experimental mill work, etc., together with 6 cents for administration and 1.4 cents for maintenance, raises the cost of mill-operation to 30.1 cents. This, when added to the cost of crushing coarse and fine material and concentrating, 4, 11, and 13 cents, respectively, makes the grand total of 58.1 cents per dry ton treated.

When the mill again begins to handle ore up to its full capacity, the recent adjustments and improvements made in it should considerably better these general average costs.

In connection with the Hurley plant there is a large assay shop where about 100 assays per day are made. Experimental work on the ores is also carried on in a well-equipped laboratory, with the object of still further reducing the cost and increasing the efficiency of concentration.

REPAIR SHOPS AT HURLEY.

At Hurley there are also storehouses and extensive machine and repair shops. The completeness of the latter is indicated by the following list of equipment:

Equipment in Chino company shops at Hurley.

Equipment in mechanical department.

One Woodward & Powell, jr., patented 42-inch by 42-inch by 12-foot planer.
One J. G. Blount No. 5 double-head emery grinder.
One New Haven 50-inch by 24-foot triple-geared, screw-cutting engine lathe.
One Prentiss 24-inch by 22-foot engine lathe.
One Davis 12-inch by 8-foot quick-change gear lathe.
One Steptoe 20-inch back-geared crank shaper complete.
One Blount 20-inch wet tool grinder, complete, with 20-inch by 2½-inch emery wheels.
One Yankee twist-drill grinder.
One Williams No. 5 12-inch pipe machine.
One Niles 5-foot semiuniversal radial drill.
One Hoefer 21-inch vertical drill press.
One Owen No. 2 milling machine.
One Acme 2-inch single-end bolt-threading and nut-tapping machine, class "A."
One Cleveland Punch and Shear Co. No. 0 plate-bending rolls.
One Robertson No. 2 rapid power hack saw.
One Ryerson 1,200 pound single-frame steam hammer.
One American 36-inch lathe, with 18-inch bed, double back-geared.
One American No. 3 high-speed radial drill.
One Higley No. 12 cold metal saw.
One Ryerson-type, radial drill, with 8-foot arm.
One Ryerson & Son Cleveland universal splitting shear.
One American Tool Works 18-inch, new-pattern, heavy-duty engine lathe.
One Cleveland Punch & Shear Co., style C, solid-frame single end punch-and-shearing machine.
One Acme 1-inch single-bolt cutter, class "A."
One Green River No. 55 opening die bolt cutter.
One Racine high-speed power hack saw, 6-inch capacity.
One Brown & Sharpe cutter grinding machine No. 2.
One Lodge & Shipley screw-cutting heavy-duty engine lathe.
One Racine high-speed draw-cut hack saw, 6-inch capacity.

Carpenter-shop equipment.

One power mortising machine.
One 24-inch single-cylinder planer and matcher.
One 24-inch hand planer and jointer.
One 16-inch variety saw.
One Universal 16-inch woodworker's saw.
One single-spindle vertical boring machine.
One 36-inch band saw.
One pattern maker's wood lathe, 24-inch swing.
One Buffalo knife grinder.
One Universal No. 8-f trimmer.
One double emery-wheel stand.
One 36-inch grindstone with iron frame.
One 24-inch rip saw.
One cut-off saw, 36-inch swing.

During the month of June, 1914, 33 mechanics and helpers were employed in the mechanical department, and 8 carpenters and helpers in the carpenter shop of the Hurley shops.

During the month of June the total cost of repairs at Hurley, including labor, supplies, and overhead charges, was 1.96 cents per dry ton of ore milled. The cost for September, 1914, was 1.89 cents, and for April, 1915, 1.32 cents per dry ton milled.

GENERAL ENGINEERING DETAILS.

Where operations are so extensive as those at Santa Rita, necessarily a wide range of engineering problems is presented for solution. Not only is much field work and mapping necessary, but a great deal of yardage measuring and computing must also be done each month. Railroad-engineering problems are common, and jobs in machine design, architecture, reinforced concrete work, hydraulics, etc., make calls on the engineering staff from time to time. This staff embraces a chief engineer and five to eight assistants.

MONTHLY ESTIMATES AND STATEMENTS.

At the end of every month stripped and mined areas are measured, and the yards and tonnage are computed. The measurements are made by stadia, and the information is plotted on a coordinated and contoured working map, which has a scale of 30 feet to the inch. The stadia stations are accurately located by transit from triangulation points. A field party consists of a transitman, a recorder, and two rodmen. The field notes are recorded in the transit book in the following form:

Specimen record in transit book.

Date ....

Low Line No. 8 Dump.

Party: ...., Transitman; ...., Recorder; ...., Rodman (top).

.........., Rodman (bottom).

[Field page.] [Office page.]

Station.[a]	B. S.[b]	Horizontal angle.[c]	Stadia distance.[d]	Vertical angle.[e]	High or low sights.[f]	Horizontal distance.[g]	Vertical distance.[g]	Elevation.[h]
	△ N							
12 (H. I. 4.9); elevation.	....	352°00′	408	+3°26′	....	408	+24.5	6370.4
6,345.9	....	355°15′	388	+4°45′	+2ft.(B.L.)	386	+32.1	6376.0
	....	358°15′	370	+3°43′	....	370	+24.0	6369.9

[a] Transit station.
[b] Backsight (on triangulation station N).
[c] Read to nearest 15 minutes.
[d] Stadia distance read to the nearest foot.
[e] Read to nearest minute.
[f] High or low sights read (called "boot leg").
[g] Actual, obtained from stadia reduction tables.
[h] True elevation of ground calculated.

When information similar to that contained in the specimen statement is put on the map, a color distinctive from that used the previous month is employed. Sections across the entire cut or dump are taken every 25 feet and are plotted as shown in figures 19 and 20.

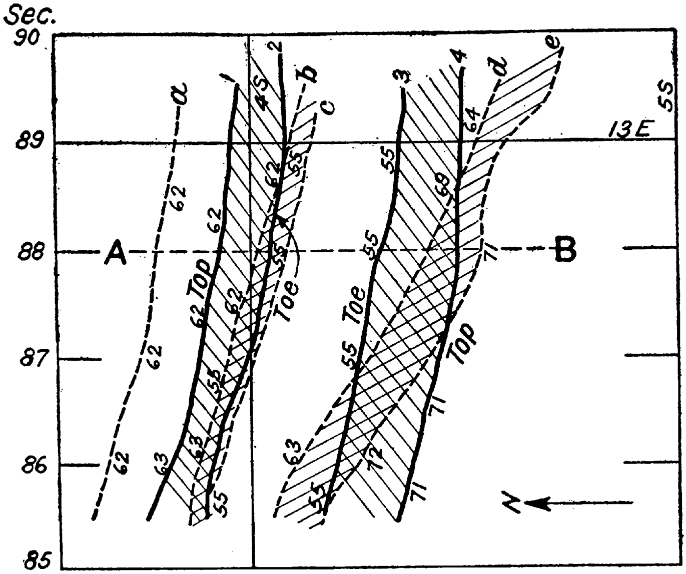


Figure 19.—Plan map of excavation and of filling during one month. The dotted lines *a* to *e* are each drawn through points having approximately the same elevation, on surface of previous month's excavation; solid lines *1* to *4* show points having same elevation on present surface. Elevation is indicated in feet. The figures *85* to *90* refer to sections on plotting map; *5S*, 13*E*, refer to coordinates of drill holes shown on engineer's records of yardage of excavation.

The surface as measured the previous month is shown on the cross section (fig. 20) by dotted lines, and the more recent surface is shown by solid lines. A plan map of excavation and filling for one month

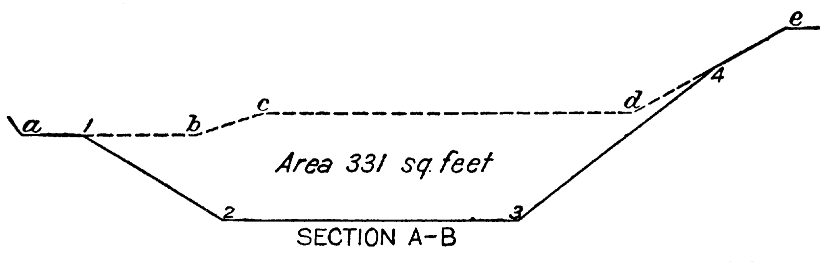


Figure 20.—Plotted cross section of month's excavation. Area represented is that of a vertical cross section through A B, in figure 19. The dotted line represents surface as measured in previous month.

is shown in figure 19, which represents an area of which figure 20 shows a cross section.

The end areas are scaled directly off the cross sections, and the average of the two end areas, multiplied by the distance between the cross sections, gives the cubical contents of each excavated block.

The plotting and computations are checked, and then the results are entered in the "Steam-Shovel Computation Book" in the following manner:

Specimen entries in "Steam-shovel Computation Book."

Section No.	Total cut.[a]		Ore estimate.		
	Area.	Volume.	Area.	Average end area.	Volume.
(Wedge, 15 feet) to 95...........	*Sq.ft.* (Noted if any.)	*Cu.ft.* (Noted if any.)	*Sq.ft.* 14	*Sq.ft.* 7	*Cu.ft.* 105
95..............	....	....	14		
94..............	....	....	56		
			70	35	875
94..............	....	....	56		
93..............	....	....	174		
			230	115	2,875
93..............	....	....	174		
92..............	....	....	284		
			458	229	5,725

[a] Stripping and mining.

Under the column headed "Section No." is recorded the wedge of ground excavated during the preceding month between cross sections 96 and 95. Under "Total cut" is shown the total excavation of waste and ore; under "Area" the areas of the cross sections in square feet are given, cross section 95 had an area of 14 square feet, and cross section 94 one of 56 square feet. The average area of the two sections, 35 square feet, is given under the column designated "Average end area." Multiplying this by 25 feet, the distance between the two cross sections, gives 875 cubic feet, the volume of the prism between sections 94 and 95, removed during the month. In like manner the cubical contents of all the other prisms are calculated. In the calculations 13 cubic feet of rock in place, and 21 cubic feet of broken rock, are used as the equivalent of a ton of 2,000 pounds. Additions to and subtractions from ore and waste dumps are calculated in the same manner.

The records of excavation for each bench are reported separately, as are the records of the excavation work of each steam shovel. If a shovel excavates from two or more benches during the month, the proportion moved from each is determined from the bench and the shovel records, and the proper apportionment made. The assay records and the number of cars sent to the crusher, direct to the mill, or direct to the smelter, determine the amount of the material classed as ore. The number of cars sent to the waste dumps determines the amount of the material excavated classed as waste. Of course, these data are checked against the engineers' estimates of solid cubic yards moved, etc.

A monthly comparative statement of the work done by each shovel is compiled for each month. This table also shows the number of cars of material loaded by each steam shovel and the capacity of each car. Records are kept by each steam-shovel foreman of the number and capacity of cars loaded and whether they contain ore or waste. The table following shows the results of steam-shovel operations during July, 1914:

Results of steam-shovel operations at Santa Rita during July, 1914.

Steam shovel No.	Solid cubic yards of waste handled.[a]	Tons of ore handled.[b]	Material in place handled, solid cubic yards.[a]	Shifts worked.[c]	Average solid yards per day (two shifts).
1	14,222	40,415	33,681	50.5	1,334
2	57,437	....	57,437	56.2	2,485
3	44,243	26,665	57,082	56.0	2,482
4	74,469	....	74,469	55.9	2,664
5	57,648	2,451	58,828	49.7	2,367
6	28,953	8,909	33,242	45.5	1,461
7	20,590	38,305	39,033	56.8	1,668
8	47,356	1,466	48,062	51.6	1,863
9	11,691	108,010	63,696	57.1	2,231
10	....	4,862	2,341	11.2	418
Total	356,609	231,083	467,871	488.5	1,916

[a] Compiled from engineers' cross-section estimates.
[b] Taken from mill returns.
[c] From steam-shovel foreman's report.

A table showing the broken-rock yardage capacity of all the different ore cars used is a convenience in making estimates. It shows, for instance, that a car having a capacity of 100,000 pounds can carry 38 cubic yards; a car having a capacity of 80,000 pounds is loaded with 30 cubic yards; a car having a capacity of 60,000 pounds carries 23 cubic yards; and a car of 55,000-pound capacity contains 21 cubic yards of broken rock. For convenience of calculation this table is extended so that one can see at a glance that a train of, say, 50 cars of 100,000-pound capacity can carry 1,900 cubic yards of ore.

The engineering department also posts a special form that serves as a daily indicator of steam-shovel operations for the information of the manager and superintendents. A specimen form follows:

Specimen report of day's steam-shovel operations, for information of mine manager and superintendent.

Date.	Shift (D=day N=night).	Accumulative car yards per shift.[a]	Total car yards per day.[a]	Accumulative car yards per day.[a]	Accumulative for month.[b]	Comparison with previous month.[c]	Remarks.
8	D	480	480	480	622	x 142	Shovel No. — shut down for repairs.
9	D	1,056	576	1,056	622	+ 434	

[a] From individual shovel records.
[b] From previous month's record.
[c] Shows gain or loss as indicated by difference between figures in two preceding columns. x indicates car-yard shortage, as compared with the previous month; + indicates car-yard increase.

The engineering computations in the office are greatly facilitated by detailed and comprehensive tables and diagrams, prepared for reducing the stadia observations, giving elevations, horizontal distances, etc.

On a plain coordinated wooden surface, about 4½ by 6½ feet, the drill holes are accurately located, and each is shown as a ⅛-inch hole bored into the wood. In these holes ⅛-inch round sticks are placed. The height of the stick above the surface represents the depth of the hole. On each stick, the ore body or bodies that the hole cuts are painted, according to scale, showing the proper thickness and spacing. This gave a fairly good chart, or model, of the ore bodies, showing their depth, thickness, etc. However, this chart proved much less useful than the cross sections of the ore bodies prepared from the drill-hole data.

MONTHLY, QUARTERLY, AND ANNUAL STATEMENTS BY ENGINEERING DEPARTMENT.

Monthly, quarterly, and annual statements and tables are compiled by the engineering department to give the following information: The amount of ore excavated, milled, and smelted; the total waste excavated and hauled to the dumps; the yardage of ore and waste taken from each bench; the costs of the various operations; the amount of ore remaining; its content of copper and value under given conditions, and other such data. In order to illustrate the manner in which these data are tabulated the following self-explanatory forms are given:

ORE AND STRIPPING.

Steam-shovel mining .... ore body.

SANTA RITA, N. MEX., ...

For month of ....

Bench.	Solid rock, cubic yards in place.	Earth and dumps, cubic yards in place.	Total cubic yards.	Cubic yards of stripping.	Per cent of copper in stripping.	Tons of ore.	Per cent of copper in ore.	Pounds of copper.

Remarks:

[This form is amplified to contain accumulative records of totals to date from Jan. 1.]

....

STEAM-SHOVEL OPERATION.

SANTA RITA, N. MEX., ...

For .... quarter, 19....

Steam shovel No.	Cubic yards of waste handled.	Tons of ore handled.	Yards in place handled.	Shifts worked.	Average solid yards per day (two shifts).

Remarks:

....

TOTAL STEAM-SHOVEL WORK.

SANTA RITA, N. MEX., ...

For month of ....

Ore body.	Solid rock, cubic yards in place.	Earth and dumps, cubic yards in place.	Total cubic yards.	Cubic yards of stripping.	Per cent of copper in stripping.	Tons of ore.	Per cent of copper in ore.	Pounds of copper.

Remarks:

[This form is amplified to contain accumulative records of totals to date from Jan. 1. A form with similar headings is used for quarterly reports.]

....

WASTE DUMPS.

SANTA RITA, N. MEX., ...

For month of ....

Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Cubic yards per car.	Totals to date from January 1.			
								Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

Remarks:

....

ORE PILE.

SANTA RITA, N. MEX., ...

For month of ....

Pile.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Cubic yards per car.	Totals to date from January 1.			
								Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

Remarks:

....

MILL SHIPMENTS.

SANTA RITA, N. MEX., ...

For month of ....

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of of copper.	Pounds of copper.	Cars.	Tons per car.	Totals to date from January 1.				
								Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper

Remarks:

....

SMELTER SHIPMENTS.

SANTA RITA, N. MEX., ...

For month of ....

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of of copper.	Pounds of copper.	Cars.	Tons per car.	Totals to date from January 1.				
								Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper

Remarks:

....

ORE AND WASTE DUMPS.

MILL SHIPMENTS.

SANTA RITA, N. MEX., ...

For .... quarter, 19...

Ore dumps.					Waste dumps.				
Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

SMELTER SHIPMENTS.

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.

Remarks:

....

HEALTH AND SANITARY MEASURES.

HOSPITAL AT SANTA RITA.

When the new company acquired the Santa Rita property and began operations on an extensive scale, a suitable hospital for their employees seemed highly desirable. Consequently, a well-ventilated one-story frame structure with wide open porches was built and adequately equipped. The hospital has been in existence four years, and in that time with a staff of two doctors and four nurses, it has successfully treated 1,314 cases. It is maintained by fees of $1.50 per month from all single employees and $1.75 from all married employees. The fees entitle the employees and their dependents to hospital services, medicine, and treatment during illness. Separate charges are made, however, for childbirth cases and also for cases of illness resulting from drunkenness or fighting, and for treatment of venereal diseases. Patients not entitled to free treatment give $2 a day in addition to nominal charges for surgical operations.

The following data regarding cases treated during the two years ended February 15, 1913, show how useful and successful the hospital has been:

Data regarding cases treated in company hospital during two years ended Feb. 15, 1913.

Cases in hospital	665
Deaths	21
General operations	106
Abdominal operations	32
Deaths following operations	[a] 2
Pneumonia cases	34
Deaths from pneumonia	5
Accident cases	157
Deaths from accident	2
Typhoid cases	37
Deaths from typhoid	1
Obstetrical cases	35
Deaths from obstetrical cases	0

CAUSES OF DEATHS.

Tuberculosis	2
Organic heart disease	1
Pneumonia	5
Cancer	1
Enterocolitis	2
Hookworm	1
Following injuries	2
Meningitis	1
Typhoid	1
Cerebral hemorrhage	2
Gunshot wound	1
Peritonitis	1
Syphilis	1
Total	[b] 21

ACCIDENT REPORTS.

When an accident occurs to an employee his foreman at once issues a report on the following form to the general office:

NOTICE OF AN ACCIDENT TO AN EMPLOYEE.

Name of person making report.................... Occupation..................
....................Street. ....City. ....State.
Date and hour of accident..191.., ...m.
Date of this report.................... 191...
Injured person:
Name............................... Address..............................
Age............... Family............... Weekly wages, $............
Occupation..
How long employed in this work?...
How long employed prior to accident?..
General duties..

[a] One was advanced appendicitis and one was gunshot wound of stomach, liver, and intestines.
[b] Of these, 7 did not live 2 hours after admission to hospital.

The injury:
Nature and extent..
Was surgical aid rendered?.......... By whom?........................
Taken home or to hospital?.......... Probable length of disability.........
Has injured returned to work?.......................................
The machine or appliance causing the accident:
What was it?..
State condition..................... When last inspected?................
Whose control at time of accident?..................................
Was there any defect in the machine or appliance? State fully: ....
....
Was light good?..................... Have broken parts (if any) been preserved?...
The accident:
Place?..
Was accident due to carelessness of injured or negligence of fellow workmen? If so, of whom?..
Statement of injured..
Name and address of foreman in charge of work.......................
Names and addresses of witnesses: ....

DESCRIPTION OF THE ACCIDENT.

(If necessary, to show cause of accident, draw rough sketch on back hereof.)

[Space for description.]

This notice made out by: ....
(State occupation.)

When the injured person has recovered, the following form certificate is executed:

Form for certificate of injury.

Claim No. ....

LIABILITY DEPARTMENT.

SURGEON'S REPORT AND CERTIFICATE OF CLAIM.

1. Name of injured...................... Occupation.......................
2. When first seen after accident.......................................
3. Precise nature, location, extent of injuries and how received.................
4. Were such injuries the direct result of said accident.......................
5. I attended claimant from .... to ...., 19...., during which time the injuries above described constituted the sole and only cause of disablement; and I certify that he was totally incapable of following his usual occupation, because of such injuries, from the .... day of ...., 19...., to and including the .... day of ...., 19.....

....
Surgeon.

CERTIFICATE OF EMPLOYER, SUPERINTENDENT, OR TIMEKEEPER.

This is to certify that .... herein referred to, is insured under policy .... and was in my employment at a remuneration of .... per hour; that he was injured at the time and in the manner stated and, in consequence thereof, sustained continuous and total loss of time for the period of .... hours, amounting to $...........

Date ....

....
(Position.)

RELEASE.

Received of ....the sum of .... ($................) dollars, in full satisfaction and discharge of any and all claims I have or may have against said .... because of a certain injury or injuries resulting from an accident sustained by me on or about the .... day of ...., 19...., while in the employ of the said ...., and further for any benefits that may be accrued to me by reason of said accident.

....
(Claimant.)

Date ....
Witness:

....

....

The number and the nature of accidents at the Santa Rita mine and at the Hurley plant for the years 1912 and 1913 are given in the following table:

Data regarding accidents at Santa Rita mine and Hurley plant, 1912, 1913.

	Year.	Killed.	Per cent.	Seriously injured.	Per cent.	Slightly injured.	Per cent.	Average number of employees per day worked.
Santa Rita mine......	1912	3	0.05	20	0.30	117	1.75	553
	1913	12	.11	25	.24	234	2.21	880
Hurley mill..........	1912	3	.036	15	.20	123	1.50	697
	1913	1	.01	5	.05	104	1.09	797

SANITARY EQUIPMENT.

A large number of the new company's houses are provided with modern plumbing. The sewage from these houses is discharged into cesspools. The smaller houses of the laborers are on large sloping lots and the waste water from the kitchens, baths, etc., is run out into the sand and gravel of the sloping surface and absorbed. Privies for these houses are placed at the rear of spacious lots, usually 100 feet away from the house. They are fitted with metal receptacles, which are removed periodically, and the contents thrown on the waste dumps and covered by carloads of waste. As a part of his general duties, the company's peace officer under the supervision of the chief physician maintains a vigilant watch over the sanitary condition of the camp.

RESCUE AND FIRST-AID WORK.

The Santa Rita main office has provided a suitable locker section in which three Draeger apparatus are stored. The equipment also includes duplicate oxygen bottles with water gage and liter bag. Hand flash lights are used exclusively.

Five employees of the company have been trained by Bureau of Mines field men, and received the bureau's certificates during the early spring of 1914; these five men have had periodic practice drills. It is the intention to train at least 10 more men in the use of the breathing apparatus.

The headquarters of the camp physician and his assistant is the company hospital, and on account of its close proximity to the mining operations no first-aid teams have been organized. The physician delivers periodic lectures on sanitation and elementary first-aid methods.

RECREATION.

The company officials encourage and contribute to the support of suitable forms of amusement for the employees and their families. Baseball and basketball teams are organized and participate in games arranged with similar teams from Hurley and Silver City. A rifle team enjoys the privileges of such teams as come up to the standards in the United States Army Regulations, obtaining healthful and instructive diversion. Two or three annual picnics give delightful outings to the men, women, and children of the camp.

TIME KEEPING, ACCOUNTING, AND WAREHOUSE METHODS.

A good working organization is one of the fundamentals of success in any business. A few years ago this factor was largely ignored by mining companies, but to-day it finds place in most of those that are successful. It, therefore, seems well worth while to call attention, in a general way, to the efficient system of time keeping, accounting, and warehouse methods in use at Santa Rita.

TIME KEEPING.

When skilled laborers are employed by the company, they are asked to fill out Form 10 so that some idea may be had of their responsibility and past achievements. From an unskilled laborer, no references are required other than a statement as to his last employment. Each foreman, on hiring any one, turns into the office a slip (Form 11). If the person employed is a skilled laborer, this form is yellow in color, and has the notice: "Bearer to furnish references." Each employee is given a number and is furnished with a cardboard check, 1½ by 2 inches, bearing that number.

FORM 10.—*Form filled in by applicant for position as skilled laborer.*[a]

APPLICATION.

Date .... 191...

MINING CO.,

Santa Rita, New Mexico.

GENTLEMEN: I hereby make application for work as .... and refer you to .... for my previous
(Give name of company and foreman, also address.)
record, and I hereby authorize and request .... to furnish same, including the cause of leaving .... service, if known. And I hereby release said .... from all liability for any damage whatsoever on account of furnishing such record.

....

Witness: ....

The above applicant .... was employed by the undersigned as .... from .... to ....
Cause of leaving ....
Ability ....
Habits ....
Services were ....satisfactory.
Remarks: ....
Dated: .... Signed: ....

FORM 11.—*Form used in notifying timekeeper of the hiring of a new employee.*[b]

.... 191..

TO TIMEKEEPER:
I have hired ....
as a .... at ....
per day.

....

Foreman.

At the time office, which is conveniently situated, each employee, except the office force, the locomotive engineers, the firemen, and the steam-shovel crews, check in at the beginning of every shift, by calling out his number to the clerk at the window of the time office. The clerk puts opposite each man's number a horizontal black mark if the man is on the day shift and a red if on the night shift. The numbers are arranged on a daily pay-roll balance sheet, a section of which is shown as Form 12. The next check on the working time of the employees is that obtained by the time inspector, who goes around checking up the men found on the work. Forms 13*a*, 13*b*, and 13*c* indicate the character of the records kept in the time inspector's note book. These are left at the timekeeper's office to be checked with a perpendicular mark against each man's number on the daily pay-roll balance sheet. After the morning horizontal mark and the time inspector's perpendicular mark, each workman checked in has a plus (+) sign after his name on the sheet.

[a] Size of form used by operating company, 8½ by 11 inches. [b] Size of original form 3½ by 4½ inches.

FORM 12.—*Register kept at time office.*[a]

DAILY PAY-ROLL BALANCE SHEET.

[Check men on shift with horizontal line in check column. Use black ink for day shift and red ink for night shift. Cross check from timekeeper's taken on the works. Enter hours and amount from work slip turned in at the end of each shift.]

0	Check.	Hours.	Amount.		1	Check.	Hours.	Amount.	
1 2 3 4 5 6 7 8 9 10									

[a] Form used by operating company, 19¼ inches high by 36¼ inches wide. Provision made for 1,500 names on a sheet, 100 names in a column. Horizontal blue lines separate each number, every tenth number being followed by a purple line.

FORM 13*a*.—*Form used by time inspector for recording "time" of miscellaneous employees.*[a]

MISCELLANEOUS LABOR.

................................, 191..

Day Night	No.	Hrs.	Remarks.
Car trimmers			
Switch tenders			
Switch tenders			
Flagman			
Water boys			

[a] Size of original form, 4¼ inches wide by 7¼ inches high, with three punch holes on left margin for binding.

FORM 13*b*.—*Form used by time inspector for recording "time" of steam-shovel crews.*[a]

SHOVEL NO. .... **Ore**..........**Hrs.** **Strip**.........**Hrs.** ...., 191..

Day Night	No.	Hrs.	No.	Hrs.
Engineer				
Fireman				
Craneman				
Coalman				
Pitmen				
Dumpmen				

[a] Size of original form, 4¼ inches wide by 7¼ inches high, with three punch holes on left margin for binding.

FORM 13c.—*Form used by time inspector for recording "time" of locomotive crews.*[a]

........................, 191..

ACCOUNT:

		No.	Hrs.	No.	Hrs.
Foreman					
Eng. No.	[b] E				
	F				
	B				
Eng. No.	E				
	F				
	B				

[a] Size of original form, 4¼ inches wide by 7¼ inches high, with three punch holes on left margin for binding. Original form provides for record of 11 engines instead of 2, as shown.
[b] E, engineer; F, fireman; B, brakeman.

For the men engaged on special jobs that are to be charged to certain accounts, and also as an additional check on working time, Form 14 is used. This form is issued in different colors for the different occupations, and the occupation and the rate of pay of the employee are entered on each. For special jobs by skilled laborers, chargeable to certain accounts, Form 15 is used. If a special gang is engaged on work chargeable to different accounts, the distribution of the charge against each account is entered on Form 16.

All these forms, by keeping track of the proper distribution of charges, greatly facilitate bookkeeping. Each man's time record is kept in a large loose-leaf book composed of pages similar to that indicated by Form 17. At the end of the month this is turned in to the main office.

When an employee is discharged or quits the company's service, he is given a slip (Form 18) by his foreman. On the back of this his store, board, and rent bills are accounted for, or marked "none." The indorsements are made in ink and are signed by the officers responsible for them. The timekeeper then fills out Form 19, puts it in a sealed envelope, and hands it to the employee, who takes it to the cashier and receives his pay.

In common with most large employers, the mining company keeps personal record cards for each employee; the scope of these is shown by Form 20.

FORM 14.—*Form used in recording "time" of men employed on special jobs, the expense of which is to be charged to certain accounts.*[a]

Occupation .... Rate ....

Date ....

P. R. No. .... Hrs. Worked .... Amount $ ....

Hrs.
......
......
......

Charge to ....

....

....

Foreman ....

PUNCH HERE HOURS WORKED

1	14
2	15
3	16
4	17
5	18
6	19
7	20
8	21
9	22
10	23
11	24
12	25
13	26

FORM 15.—*Form used in connection with special jobs by skilled laborers, chargeable to certain accounts.*[b]

CARPENTERS.

SANTA RITA MINE,
Date .... 191..

Name .... Check No. ....

Give full description of work done.	Hours.	Rate.	Amount.
Total			

Examined and approved:

....
Foreman.

[a] Size of original form, 3 by 5 inches.
[b] Size of original form, 4½ inches high by 7 inches long, on heavy flexible stock.

FORM 16.—*Form used in connection with work done by a special gang and chargeable against each of several accounts.*[a]

SANTA RITA MINE,

.................., *191* .

Occupation.	Number.	Hours.	Rate.		Amount.	

Charge to:

....

Examined and approved:

....

Foreman.

FORM 17.—*Form used in time office for keeping each man's time for a month.*[b]

TIME RECORD.

Name.................................... No.............................

Occupation.	1	2	3	29	30	31	Total time.	Rate.	Amount.

	Remarks.		Total.
Deductions. Doctors and hospital		$	
Insurance		$	
Store		$	
Board		$	
Rent		$	
		$	
Check No.		Bal. due	

FORM 18.—*Slip given by foreman to employee who is discharged or quits.*[c]

SANTA RITA, N. MEX., ..., 191 .

TIMEKEEPER:

Give bearer..

(No. ....) his time for services to date.

Quit } Mark
Discharged } with
No work } an X.

Signed...................................

IMPORTANT.

Get this cleared at Santa Rita store, Santa Rita boarding house, and bunk house, then present at the office for payment between the hours of 1 p. m. and 3 p. m. daily except Sunday.

(Over.)

[a] Size of original form, 4 inches wide by 7 inches long, on heavy flexible stock.
[b] Original form is 12¼ inches high by 14¼ inches long. Space for five names instead of one is provided, and columns for each day of the month. Each sheet is kept in time office until the end of the month, when it is turned in to the main office for balancing and checking.
[c] Size of original form, 3 by 5 inches.

[On reverse side.]

	Amount.		Signed.
Store Board Rent			

NOTICE.—The above must be filled in with ink; if there is no amount write "None" and sign name.

FORM 19.—*Slip handed by timekeeper to employee who leaves during month.*[a]

SANTA RITA MINE,
.... 191....

Name.. P. R. No. ....

		Amount.
.... shifts, @ $..........	....	$.......
.... shifts, @ $..........	....	....
.... shifts, @ $..........	....	....
.... shifts, @ $..........	....	....
Total amount earned	....	....
DEDUCTIONS.		
Doctor and hospital..	$..........	
Insurance..	....	
Store..	....	
Board..	....	
Rent..	....	
....	....	
....	....	
Total deductions	....	....
Balance due	....	....

The above statement is correct.

Signed....................................

FORM 20.—*Personal record card.*[b]

Name	Date employed	No.
Address	Boards	
Dependents		

Age	Weight	Height	Eyes	Hair	M. or S.

Nationality	Engaged by
Last employer	
Education	

Date Dept. Occ. Rate			
Date Dept. Occ. Rate			

Left employ—Date Reason:

[a] Size of original form, 5½ by 6 inches.
[b] Original card, ruled cardboard, 6 inches wide by 4 inches high.

ACCOUNTING.

When the labor-distribution cards (Form 15) have been signed and punched by the foreman and handed in to the main office they are entered on loose sheets similar to Form 21, which are kept in binders, forming large loose-leaf books. The labor data are all grouped, so that one or more of these sheets is used for firemen of all kinds, one or more for engineers, one or more for drillers, and so on. Each fireman, engineer, driller, etc., is charged against the particular locomotive or steam shovel fired by him for each day of his service during the month. These forms thus filled out present a complete daily record of labor charges classified and distributed against certain accounts. When this daily record has been summed up at the end of the month, the totals from it are entered on the Santa Rita labor-distribution sheets, the character of which is indicated in Form 22. This shows the total labor charges for the month properly distributed and arranged. One or more of these sheets is used for accounts that may be grouped under the general term, "Operation." In this group the columns are headed: Steam shovel 1, etc.; locomotives 1, etc.; drill 1, etc., and dump cars. The next group of headings on these sheets cover the following accounts: Rents and repairs, light service, automobile expense, prospect-drill expense, hospital fund, special water investigation. These are general ledger accounts and are carried in the general office at Hurley.

FORM 21.—*Form used in recording data on labor-distribution cards.*[a]

Month.....................191..   Account.....................

Date.	Shifts.	Amount.	Shifts.	Amount.	Total amount.
1					
2					
3					
4					
5					
6					
29					
30					
31					
Total					

[a] Original form 11 inches high by 26 inches wide, with wide margin on left for binding; 20 instead of 2 of the "Shifts-Amount" columns appear on the original, and space is provided for 31 days of the month.

FORM 22.—*Form used in recording total labor charges for each month, showing distribution.*[a]

SANTA RITA LABOR DISTRIBUTION.

Sheet No.......... ....191..

Occupation.					
1 Assayers and samplers					1
2 Boilermakers and helpers					2
3 Brakemen					3
4 Blacksmiths and helpers					4
5 Blasters					5
6 Compressormen					6
7 Chauffeur					7
8 Car repairers					8
9 Central					9
10 Carpenters and helpers					10
11 Cranemen					11
12 Drill foreman					12
13 Drillers					13
14 Drill helpers					14
15 Drill laborers					15
16 Drill mechanics					16
17 Dumpmen					17
18 Engineers and surveyors					18
19 Electricians					19
20 Engineers, locomotive					20
21 Engineers, steam shovel					21
22 Engineers, stationary					22
23 Firemen, locomotive					23
24 Firemen, steam shovel					24
25 Firemen, stationary					25
26 Foremen, steam shovel					26
27 Foremen, tracks					27
28 Foremen, surface					28
29 Laborers and bosses					29
30 Masons and plasterers					30
31 Master mechanics					31
32 Machinists and helpers					32
33 Machinemen and helpers					33
34 Miners					34
35 Oilman and helpers					35
36 Officemen					36
37 Pumpmen					37
38 Plumbers and pipe fitters					38
39 Painters					39
40 Pitmen					40
41 Repairmen					41
42 Samplers					42
43 Switchtenders					43
44 Stableman					44
45 Sandmen					45
46 Teamsters					46
47 Topmen					47
48 Trammers					48
49 Timber framers					49
50 Trackmen					50
51 Watchmen					51
52 Water and coal tenders					52
53 Yardmasters					53
54					54
55					55
Total					
1					1
2					2
3					3
4					4
5					5
Total					
Ore					
Waste					

[a] Original form 20¼ inches wide by 34 inches high. Original form has 20 instead of 4 distribution columns.

Another group of headings on these sheets indicate the following accounts: Water supply, Santa Rita boiler, compressor expense, heating, machine shop, blacksmith shop, carpenter shop, light service, engineering and surveying, assaying and sampling, office expense, pit drainage, stable expense, superintendent and general labor, and a few other accounts. Another group of accounts comes under the general term "Additions to plant," and the individual columns are headed "Crusher bins," "Hospital addition," etc. A fifth group is known as surface-development accounts. They include surface drainage, additions to dump lines, company roads, etc. The accounts-recoverable group comprises mess accounts, county roads built or repaired, etc.

Each column on each of these sheets (Form 22) represents a separate account. When these columns have been added, the total for each is divided between ore and waste according to the amount of time spent by each shovel in digging ore and the time spent in digging waste. These time figures are obtained from the reports of the shovel engineers, and are checked against foremen's reports, etc.

From the data recorded on the other forms the operating-cost sheets for the month (Form 23) are made out. One of these is used to show mining costs and another to show stripping costs. The expense apportioned to each is obtained from the total time spent by the shovels in digging waste and the total time spent by them in digging ore. The percentage of the total time spent in digging ore multiplied by the total cost for the month gives the cost of mining for the month. The remainder is the total cost of stripping. The recapitulation at the bottom gives a complete summary of the cost of production to date as well as costs for drilling, blasting, loading, and hauling. Thus, Form 23 provides a convenient and concise summary of all the costs.

FORM 23.—*Operating-cost sheet.*[a]

.... REPORT .... ORE. .... 191..

	Steam shovel No. 1.	Steam shovel No. 10.			Locomotive No. 1.	Locomotive No. 20.			Dump cars.	Air drilling.	Churn drilling.				Total cost.	Average per......
LABOR.																
Bankmen																
Boilermakers																
Blacksmiths																
Blasters and powdermen																
Brakemen																
Carpenters																
Car trimmers																
Car repairers																
Cranemen																
Drillers and helpers																
Dumpmen																
Electricians																
Engineering																
Engineers																
Firemen																
Floating gang																
Foremen																
Machinemen and helpers																
Machinists																
Officemen																
Oilmen																
Pipefitters																
Pitmen																
Pumpmen																
Sandmen																
Switchtenders																
Teamsters																
Track gang																
Watchmen																
Water and coal tenders																
Yardmaster																
Assaying and sampling																
Stable expense																
Superintendent and general labor																

Compressor expense																
Water supply																
Surface expense																
S. R. pumping and repairs																
Total labor																
Cost per																
SUPPLIES.																
Caps, fuse, and explosives																
Drill steel																
Electrical supplies																
Fuel																
Gasoline																
Miscellaneous																
Oil, waste, and packing																
Pipe and fittings																
Repairs and renewals																
Tools																
Teaming																
Office expense																
Engineering expense																
Assaying and sampling																
Stable expense																
Superintendent and general labor																
Compressor expense																
Water supply																
Surface expense																
S. R. pumping and repairs																
Total supplies																
Cost per																
Total labor and supplies																
Cost per																
No. cars loaded																
No. .... loaded																

[a] Original sheet, 17 by 31¼ inches.

FORM 23.—*Operating-cost sheet*—Continued.

RECAPITULATION.

	Previously reported.				Cost this month.				Total to date.				Cost per					Remarks.
	Amount.	No. of cars.	No. of	Cost per	Amount.	No. of cars.	No. of	Cost per	Amount.	No. of cars.	No. of	Cost per	Drilling.	Blasting.	Loading.	Hauling.	Total.	
Labor																		
Supplies																		
Total																		
Administrative expense																		
Grand total																		

WAREHOUSE METHODS.

The importance of the warehouse or storehouse division will be grasped from the fact that it always carries on hand and accounts for about a quarter of a million dollars' worth of stock of most varied character, say from turpentine to teeth for steam shovels.

All invoices received during the month are recorded in triplicate, Form 24, a "statement of invoices." The original of these is sent to the cashier's office, a duplicate goes to the supply agent, and the triplicate is retained in the warehouse. The invoices are then entered on the "warehouse-record" sheets (Form 25). These give a record of the freight charges, the material actually received, and the material invoiced. From this form the material is entered on large "commodity distribution" sheets, where the data covering different commodities are segregated in different columns, so that the amount of each received is easily determined. The commodities are then entered in the "price-and-invoice" book (Form 26), each page being given over to a different article, such as cylinder oil or steel rails. The price and balance on hand is kept posted for each month, and in addition stock is taken in December of each year and checked against this book.

Data covering the supplies issued are carefully recorded in several ways. Form 27 is an order on the storekeeper for material. The data on each of these forms received are entered in the "supplies issued record" book (Form 28), which shows the kind, quantity, and cost of supplies issued for every day of each month. At the end of the month these daily records are summed up on a monthly record known as the "Santa Rita supply distribution" sheet (Form 29). This summarizes the supplies charged during the month and furnishes part of the data for the work of the cost accountant in the main office.

The warehouse also keeps a record of the powder supply. Every week the powder supply is balanced in pounds of weight and every month in money. Form 30 is used for this record.

Many other labor-saving forms are used, but as they pertain more to general warehouse accounting than to mine accounting, they are not introduced here.

FORM 24.—*Form used in making statement of invoices.*[a]

STATEMENT OF INVOICES.

SANTA RITA MINE, ..., 191..

Date.	Favor of—	Amount of invoice.	Over-charges or deductions.	Amount taken into account.	Local freight bill No.	Amount of freight.
	Total					

[a] Original form 8¼ inches wide by 14 inches long.

FORM 25.—*Warehouse-record sheet.*[a]

WAREHOUSE RECORD.

SANTA RITA MINE, ... month, 191..

Freight bill.							Material received.			Invoice.		
Date.	Local freight bill No.	Car initial and No.	Weight.	Rate.	Amount.	Date passed, month entered.	Date received.	Quantity and weight.	Consignor, shipping point, and description.	Net amount.	Date of invoice.	Date passed, month entered.

[a] Original sheet 13¾ inches wide by 16 inches long.

FORM 26.—*Specimen sheet from price-and-invoice book.* [a]

Nuts, bolts, and washers.

	Jan.	Feb.	Mar.	Oct.	Nov.	Dec.	Bal. for'd.
Avg. price New price Bal. on hand Rec'd Total Used Order No							

FORM 27.—*Form used in order on storekeeper for material.* [b]

REQUISITION FOR SUPPLIES.

No.......[of foreman].

SANTA RITA, N. MEX.,.......................191..

TO STOREKEEPER:

No.	Material.	For—

Storekeeper Req. No. Approved.

....

Signed.

FORM 28.—*Form used in recording daily issues of supplies.* [c]

SUPPLIES ISSUED RECORD.

Month of....................191.. ....Account.

1	2	3	4	5	6	29	30	31	Articles.	Total.	Price.		Amount.		Total.	
											$	¢	$	¢	$	¢

[a] Original sheet 15 inches wide by 12½ inches high; sheet printed on both sides. Eight spaces instead of one space appear for "Order No." entries; also columns for each month of the year.

[b] Original form 8½ inches wide by 10½ inches high, printed on white paper. A duplicate set of forms is printed on yellow paper, a yellow duplicate of each order being retained by foreman making requisition.

[c] Original form, 17 inches wide by 11 inches high. Margin and punch holes at left provided for binding; also columns for each day of the month.

FORM 29.—*Form used in summarizing daily records from Form 28.*[a]

SANTA RITA SUPPLY DISTRIBUTION.

Sheet No.......... ....191..

	On hand. [b]	Received. [c]	Used. [c]	On hand. [d]	
1 Air-drill repairs					1
2 Air-brake repairs					2
3 Babbitt and metals					3
4 Belting					4
5 Boiler flues					5
6 Boiler compound					6
7 Blacksmith coal					7
8 Burlap and canvas					8
9 Candles					9
10 Cement, brick, and lime					10
11 Castings					11
12 Caps					12
13 Coke					13
14 Drill repairs					14
15 Dump-car repairs					15
16 Electrical fixtures and supplies					16
17 Fuse					17
18 Fire brick and clay					18
19 Fuel					19
20 Furniture and fixtures					20
21 Grease and lubricants					21
22 Hay, grain, and feed					22
23 Hose and fittings					23
24 Hardware					24
25 Iron and steel					25
26 Illuminating oils					26
27 Laboratory supplies					27
28 Lumber					28
29 Locomotive repairs					29
30 Labor					30
31 Machinery repairs					31
32 Metal lath					32
33 Nuts, bolts, and washers					33
34 Nails					34
35 Packing					35
36 Pipe and fittings					36
37 Pulleys and shafting					37
38 Powder					38
39 Paint, tar, and glass					39
40 Pump repairs					40
41 Rail and fittings					41
42 Rope and cable					42
43 Screens					43
44 Steam-shovel repairs					44
45 Stationery					45
46 Structural steel					46
47 Sand					47
48 Switching					48
49 Ties					49
50 Teaming					50
51 Timber and lagging					51
52 Tools					52
53 Waste					53
54					54
55					55
Total					
Santa Rita store supplies					
Electric power from Hurley					
1					1
2					2
3					3
4					4
5					5
Total					
Ore					
Waste					

[a] Original form, 24 by 24 inches. Original has 20 instead of 2 blank columns for headings showing distribution of supplies.
[b] At first of month.
[c] During month.
[d] At first of succeeding month.

FORM 30.—*Form for keeping record of powder supply.*[a]

POWDER SUPPLY.

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Powder.	1st to 7th.			8th to 14th.			15th to 21st.			22nd to			1st of			
															Orders not rcd.	
Grade and size.	On hand 1st.	Rcd. during period.	Used during period.	On hand 8th.	Rcd. during period.	Used during period.	On hand 15th.	Rcd. during period.	Used during period.	On hand 22nd.	Rcd. during period.	Used during period.	On hand 1st.	Used during month.	Order No.	Amount.
Bulk and Total black Total stick Grand total																

[a] Original form, 16 inches wide by 8½ inches high.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF METAL MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of special interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23–26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 75. Rules and regulations for metal mines, by W. R. Ingalls, James Douglas, J. R. Finlay, J. Parke Channing, and John Hays Hammond. 1915. 296 pp., 1 fig.

BULLETIN 80. A primer on explosives for metal miners and quarrymen, by C. E. Munroe and Clarence Hall. 1915. 125 pp., 15 pls., 17 figs.

BULLETIN 101. Abstracts of current decisions on mines and mining, October, 1914, to April, 1915, by J. W. Thompson. 1915. 138 pp.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 15. An electrolytic method of preventing corrosion of iron and steel, by J. K. Clement and L. V. Walker. 1913. 19 pp., 10 figs.

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TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 32 pp.

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