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MINING METHODS AND COSTS OF MINING COPPER ORE AT THE VERDE CENTRAL MINES, INC., JEROME. ARIZ.

BY

Robert H. Dickson

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHODS AND COSTS OF MINING COPPER ORE AT THE VERDE CENTRAL MINES, INC., JEROME, ARIZ.[1]

By Robert H. Dickson[2]

INTRODUCTION

This paper describing the mining practices at the Verde Central mine at Jerome, Ariz., is one of a series being prepared by the United States Bureau of Mines on mining practices, methods and costs in various mining districts of the United States.

ACKNOWLEDGMENTS

The costs contained in this paper were prepared by R. J. Huddleston, chief clerk, Verde Central Mines, Inc.

HISTORY

Prospecting and mining have been carried on in the Verde mining district for the past 50 years. The Verde Central was one of a number of prospects operated with the idea of finding ore in the vicinity of the large United Verde ore body; there are practically no surface outcrops on Verde Central ground. A company guided by W. F. Staunton, formerly manager and at present vice-president of Verde Central, started explorations to the south of the United Verde in the hope of finding ore in the shear zones extending southward from the main ore body. Verde Central Mines, Inc., was incorporated in 1916. Thomas H. Collins and Thomas Hoatson became interested and, through their influence, the Calumet & Arizona Mining Co. obtained control of the property in July, 1921. Ore was discovered in 1923. From then until 1928 all the work done consisted of that required to locate and develop the ore bodies.

In 1928 a 300-ton concentrator was built and operation started January 1, 1929. Stoping operations commenced during the latter part of 1928.

GEOLOGY

The rocks of the Jerome district are Pre-Cambrian schists and intrusives covered by Paleozoic sediments and Tertiary lava flows (fig. 1). The rocks of economic importance are the Pre-Cambrian formations locally known as greenstone and quartz porphyry. In the neighborhood of the Verde Central mine the term greenstone is applied to a series of flows, tuffs, and agglomerates, the flows ranging from rhyolite to amygdaloidal basalt, which have been highly folded and locally metamorphosed into schists. Quartz porphyry in the form of a

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6464."

2 One of the consulting engineers, U. S. Bureau of Mines, and manager, Verde Central Mines, Inc.,

batholith (about one-half mile wide at the Verde Central) has been intruded into the greenstone series. Small dikes of quartz porphyry are found in the greenstone some distance away from the main mass. More recent andesite dikes cut the ore bodies and all formations.

Almost all workings at the Verde Central are on the eastern contact of the quartz porphyry batholith with greenstone, or in greenstone (fig. 2). Ore has been found at the contact and in shear zones in the greenstone about 400 feet east of the contact. All ore bodies are associated with either the large mass or smaller dikes of quartz porphyry.

A number of ore zones have been found but the most important are those of the Rock Butte fracture and the Silver Cliff contact vein. The Rock Butte fracture zone is a mineralized shear zone, over 1,000 feet long, ranging from 5 to more than 100 feet in width, and extending vertically throughout the whole range of the mine (fig. 3).

The principal minerals are quartz with disseminated pyrite and small amounts of chalcopyrite. The north end of the fracture contains more quartz and less pyrite than the south end. Ore bodies, in which chalcopyrite exists in commercial quantities, occur as lenses in this larger mineralized zone and range from 5 to 40 feet in width, with an average of about 15 feet. They are from 50 to 300 feet long with vertical dimensions greater than the lengtns.

The Silver Cliff contact vein is a shear zone on the contact of quartz porphyry and greenstone, in which the rocks are metamorphosed partly to schist and carry chalcopyrite and quartz in small amounts.

Faults parallel to the famous Verde fault extend at intervals across the whole district. Several faults of this series cut across Verde Central workings; individual slips displace the ore bodies (fig. 2).

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

Both greenstone and quartz porphyry are highly siliceous, hard and firm, and permit openings to be driven without the necessity of timbering.

There is very little fracturing in the ore bodies. The walls of the stopes stand well except in the proximity of faults. At several places faulting cuts the ore lenses at oblique angles, making the walls heavy and creating a tendency for blocks to slide on the fault planes as the ore is mined from the vein.

Andesite dikes, ranging from 1 to about 6 feet in width and dipping vertically, cut almost at right angles across the ore bodies. This rock is very soft with a tendency to crumble in breaking. The dikes, however, do not interfere with mining except in diluting the ore.

The ore is not always as wide as the mineralization. In places the ore body is on one side of the mineralized fracture, in other places covers its whole width, and in other places consists of two lenses with a horse of waste between. In individual stopes the ore may follow one wall for some distance and then turn over to the opposite wall. In these cases one wall of the stope is quartz containing some values.

There is no ore above the 600 level. The ore does not oxidize in the stopes sufficiently to affect milling operations.

The grade of ore mined varies with the market price of the metals. Since stoping operations have been carried on, the market price of copper has been higher than usual, permitting the mining of lower-grade ores. During this period the ore mined has averaged about 2.7 per cent copper and 0.4 ounce of silver per ton.

METHODS OF PROSPECTING AND EXPLORATION

Early prospecting disclosed ore on the quartz porphyry-greenstone contact (the east side of the quartz porphyry batholith) and also found the Rock Butte fracture zone. Development work in the Rock Butte fracture has consisted largely of running crosscuts from the shaft on various levels to the mineralized zone or vein and then drifting along it. Crosscuts are run from the drifts at intervals to determine the width of the mineralization and the location of mineable ore. Raises are run wherever necessary.

The Silver Cliff contact vein on the quartz porphyry-greenstone contact is developed by drifting on this contact.

Holes up to 50 feet in length have been drilled from the drifts into the walls by using a heavy drifter and sectionalized steel. Diamond drilling has been used to get general geologic information and to determine the existence of mineralized fractures.

A considerable portion of the surface is capped with limestones which cover all signs of the Pre-Cambrian surface and the mineralization. The 1000 level was used as an exploratory level where work was done to determine the geologic conditions in the whole area. Use was made of all available geologic data in directing the exploratory work. The main drift followed the quartz porphyry-greenstone contact with a few crosscuts into the greenstone and drifts along mineralized fractures.

Development work is kept well ahead of stoping. The availability of empty stopes for disposal of waste is a determining factor in the time of doing work in barren rock.

SAMPLING AND ESTIMATING

Samples are taken across the faces of all drifts and raises as they advance. A grab sample is taken from the cars as they are loaded. Car and face samples are used to determine the location of ore. Before any estimate is made, the engineering department takes channel samples across the vein at the back of the drift at 10-foot intervals; raises are channel-sampled in the same way. The channel samples amount to about 100 pounds and are taken either by hand or by using a stoper machine. Either a moil or a chisel bit with a slightly convex edge is used for hand-sampling. A moil bit is sometimes used for cutting samples with a stoper machine, but more commonly the regular drill steel serves the purpose.

Estimates are made on the basis of these samples, which are weighted according to the amount of ore represented by each, rather than simply combined arithmetically. As the ore shoots generally have two or three times the vertical dimension that they have length, the height of ore is assumed to be as great as the length developed; however, no such estimate is carried above or below an adjoining level. A factor is applied for dilution which adds to the tonnage but decreases the grade. This factor is obtained from experience and is usually greater for the narrow stopes than for the wide ones; it also depends on the character of the walls and on the presence of the andesite dikes mentioned above. For example, in a stope 5 feet wide, with good walls, a dilution factor of 20 per cent was used. In a stope 20 feet wide, also having good walls, the factor adopted was only 5 per cent. In another stope, 15 feet wide, in which andesite dikes made up about 10 per cent of the area, a factor of 25 per cent was based on the assumption that normal dilution would have been about 10 per cent, but that the presence of the andesite dikes within the stope would add 10 per cent, and that the dikes would in addition break outside the stope limits and cause sloughing of the walls to the extent of 5 per cent more.

Samples are taken daily in the stopes, every fresh face being sampled. A hand sample is taken from the top of each car loaded and these are combined to make the stope sample for the day. The grab sample from any car is very inaccurate, but the summation of a number of such samples approaches a very accurate average as the number is increased.

METHODS OF DEVELOPMENT

The mine is developed by a three-compartment vertical shaft 1,930 feet deep. Each compartment is 4½ by 5 feet in the clear. Two are hoisting compartments and the third is used for manway, water columns, power cables, and the signal system. Hoisting is done by double-deck cages. The hoist is driven by a direct-current motor direct-connected to the hoisting drum shaft. The hoisting speed is 1,100 feet per minute.

Levels are 150 feet apart from the 1000 level (850 feet below the collar of the shaft) to the 1900 level (1,900 feet below the collar). Above the 1000 the levels are a little more than 150 feet apart.

A 60° inclined shaft extends from the surface to the 1000 level. This has two compartments about 5 by 5 feet inside. It is used for air and as a manway between levels. In addition, it carries one set of electric cables and a water column. This was the original shaft on the property.

At the vertical shaft each level is provided with a station, usually about 35 feet long and 15 feet wide. The stations are equipped with three tracks - one in the center for empties and two outside, running directly to the two hoisting compartments, for loaded cars. Landing chairs are installed in the shaft at each level.

Every level is used as a haulage level and any crosscut or drift may be used as a haulageway.

It is estimated that about 1 foot of development is required for every 60 tons of ore extracted. This figure includes prospect work in new areas, and driving drifts and raises to tap known ore bodies. It does not take account of raises driven from stopes to levels above for passageways, or of short crosscuts driven from stopes to test the walls; these are charged to stoping.

Shaft Sinking

The vertical shaft was sunk in three lifts between May, 1914, and October, 1926. Rock dimensions are about 8½ by 18 feet. It is timbered with 8 by 8 inch Oregon fir with sets at 5-foot centers. Bearers of 8 by 10 inch timber are placed about 150 feet apart throughout the length of the shaft.

The upper 850 feet of the shaft was completed in about 90 days. The old inclined shaft was in operation and there were cross-cuts at five levels to the new vertical shaft. To make the shaft, raises were run simultaneously from level to level. When the raises were connected they were enlarged to shaft size and timbered, beginning at the collar.

Below the 1000 level, sinking was done in the usual manner. The shaft crew consisted of a jigger and four shaftmen, at the bottom. A hoisting engineer and lander were on the landing level. Three crews of shaftmen followed each other, performing whatever operation confronted them. The average progress in sinking was 91 feet per month. Water caused some delay.

A 36-hole, V-cut round broke about 5 feet in depth. Electric delay blasting caps were used connected to a 440-volt circuit. Drilling was done with 65-pound, turbine-rotated sinking machines. The air and water hose were connected to a manifold; when drilling was finished, the machines were disconnected and the manifold with hose attached was hoisted up to a safe place for shooting. Broken rock was shoveled into 12 cubic foot buckets. Spare drilling machines and an extra sinker pump were always kept on hand. All machines and hose, as well as the pump, were tested in the shop before being sent underground. The sinker pumps were tested against a pressure equivalent to their working load in the shaft.

Drifting and Crosscutting

Crosscuts are usually driven 5 by 7 feet in section and are untimbered. All openings are driven on a grade of 0.5 per cent in favor of the loads and are provided with a ditch for water, if there is any likelihood it will be needed. The rock is hard and the average advance per round is about 4 feet. Drilling is done with 145-pound drifters mounted on columns and cross-arms. Oil-line lubricators are used. Drill steel is 1¼ inches round and the change is ⅛-inch gage. The drilling crew consists of one miner who operates the machine and one mucker supplied at the beginning of the shift to help set up the machine and at the end of the shift to help tear down and blast. Shoveling is done on the following shift, the rock being loaded by hand into 16 cubic foot cars. Muck piles are always sprayed with water to keep down any gas that may be in the pile.

Drifts are driven in the same manner as crosscuts, but in ore are usually a little higher and wider than a crosscut in order to make room for stoping operations. Drifts generally follow one or the other wall of a vein.

Drifting and crosscutting costs for 13,052 feet in 1927 averaged as follows:

	Per foot
Labor (drilling, mucking, and tramming to station)	$ 6.19
Explosives (powder, fuse and caps)	2.71
Miscellaneous (carbide, shovels, pipe, track, etc.)	0.54
Compressed air (drilling and ventilation)	1.23
Steel (material and sharpening)	.64
Drilling machines	.39
Total	$11.98

Raising

As the ore lenses are vertical or nearly so, most of the raises are vertical. They are usually driven about 5 by 9 feet in section and timbered with stulls or cribbing; the chute compartment is generally untimbered. The manway is 4½ by 4½ feet in the clear with a 22-inch timberway in one corner. When stulls are used they are of 8 by 8 inch timber, spaced 5 feet apart with 3-inch lagging on the chute side. Ordinarily cribbing is easier to put in than stulls; it is of 4 by 10 inch timber, lagged with 3-inch plank on the chute side. In cribbed manways the landings are about 10 feet apart. The ladders are staggered first on one side of the timberway and then on the next but in such a way that one does not have to make more than a quarter turn at each landing.

Drilling is done with 108-pound, self-rotated stopers or heavy hand-rotated machines. Quarter-octagon steel, 1-inch size, is used. A tugger hoist is provided to handle steel and supplies. In high raises, the round is fired by delay electric blasting caps. Two men make up a raise crew, each man operating a machine. Unless there is reason for speed, a raise is driven on only one shift.

STOPING

Most of the ore bodies are in veins with firm walls that stand well. These lend themselves readily to shrinkage stoping. Where the walls are bad owing to faulting, the ore is mined by cut-and-fill methods.

The Rock Butte fracture was developed to the 1750 level and a large tonnage indicated before stoping was started. The following conditions governed the sequence in which the various sections or ore bodies were to be mined:

From the standpoint of hoisting the center of gravity of the ore reserves was between the 1100 and 1200 levels. It was therefore desired to start stopes so that the average distance hoisted would be from between the 1100 and 1200 levels, thus insuring the hoisting of a constant amount of ore over a long period of time with the present equipment.

It was desirable to work out the ore at the extremities of the property so that later ore would not have to be trammed through the worked-out sections.

It was desirable to maintain a constant average grade for the mill heads. To make this possible, stopes of varying grades were operated simultaneously.

Certain development work was deferred until there was an open worked-out stope nearby where the waste could be gobbed. This factor influenced, to a small degree, the sequence of bringing in new stopes.

Stopes in some of the smaller ore bodies at a distance from the shaft are easier to ventilate in the winter than in the summer. These were worked during colder weather.

Shrinkage Stoping

The stopes were laid out and timbered (see fig. 4) in the manner developed by the author at the Eighty-Five mine, Valedon, N. Mex. Double chutes were installed at 25-foot centers with a blasting chamber above each chute. Cribbed manways were carried up at intervals of 100 feet or less through the broken ore. As stoping progressed, only enough ore was drawn off to provide working room at the top. When stoping was finished the broken ore in the stope was drawn out.

During the development stage a drift had been driven along the vein and raises run to determine the vertical extent of the ore.

In order to make room for chutes and stulls, the back of the drift is shot down for the length of the stope. "Cousin Jack" sets (see fig. 5) are erected to catch the broken ore and obviate mucking in this stage of the work. The ore above the sets is taken out only wide enough to make room for the chutes and blasting chamber. Above the blasting chamber the stope is gradually widened out to include all commercial ore from wall to wall.

The stulls at the back of the drift are of 10 by 10 inch timber; hitches are cut in both sides. A compressed-air clay digger fitted with a moil is successfully used for cutting hitches in all but the hardest ground. In very hard ground holes are drilled and shot to make the hitches. Posts, 8 by 8 inches in size, are used under the stulls to reduce any span to 5 feet or less. The stulls are covered with 3-inch lagging to form the floor of the stope. Double chutes (see fig. 6) are next built at 25-foot centers; after this the temporary "Cousin Jack" sets are taken down and broken ore falls directly into the stope chutes or onto the stope floor.

Three stulls of 10 by 10 inch timber are next placed 7 feet above the chutes and floored over. This is for the purpose of making the broken ore enter the chutes at the side. These stulls are usually less than 10 feet long and are supported in the middle by 8 by 8 inch posts. The posts limit the width of the opening to about 4 feet. Any bowlders too large to go through the chute are shot in this room.

Breaking ore starts at one end of the stope and progresses in benches over the whole length. The intention is to keep the back nearly level until near the top of the stope where the working face is carried at an angle to improve ventilation. Occasionally lean spots are encountered in the vein; these are left in place as pillars. When shooting is done directly over the manways, these are cribbed about 3 feet above the floor of the stope, covered with a bulkhead of 8 by 8 inch timbers, and banked with bowlders of ore.

Some secondary breaking is done in the stopes. In wide stopes there is usually one man using a plugger machine for each three men drilling in the back. This man not only drills and blasts bowlders, but helps the miners with taking steel in and out of the stope, and in setting up and tearing down their machines at the beginning and end of the shift.

Drawing of ore is carried on concurrently with mining and is so regulated as to maintain a space of about 7 feet above the broken ore for miners to set up stopers and drill the back. About 40 per cent of the broken ore is withdrawn during mining operations. If possible, some ore is withdrawn from every chute daily to keep the whole mass of ore moving uniformly. Very little trouble has been caused by ore hanging up in narrow stopes. In any stope less than 7 feet wide or wherever a narrow spot exists below, the miners use a 3 by 12 inch plank 16 to 20 feet long as a base to drill from and stand on. Each miner has one plank, which is stored away during blasting and used over and over. In case of a sudden slump, this plank prevents the man from falling or being drawn into the ore. In several cases the ore has slumped over an area 5 to 8 feet in diameter, and the plank has functioned as intended.

The selection of drills that would cut very hard rock has been a serious problem. After trying many drills a 145-pound drifter and a heavy hand-rotated stoper were adopted as standard. The drifter type is used in breast work (see fig. 4) and has been used in breaking the back, but with it there is a tendency to make an excessive amount of bowlders. Stopers are used in drilling the back. Self-rotating stopers have worked very well but their upkeep in hard ground was excessive. Machines of the stoper type break the ore finer and have more effective drilling time as they do not have to be set up.

The supply of steel in hard rock is another problem. One-inch quarter-octagon vanadium steel is used. This is a carbon steel which has had vanadium used in its manufacture as a deoxidizer and a purifier, though little or none remains in the finished steel. The adoption of this steel has reduced steel breakage to a remarkable extent. Tugger hoists in the form of "Ingersoll Rand EU Utility" and "Sullivan Turbinair" are used to hoist steel into and out of the stopes. These hoists are set up in a recess in the drift at the bottom of the timber slide. A square sheet-steel bucket, 12 by 12 inches in section by 4 feet long, is used to hold the steel. A tripod made of 2-inch pipe is set up at the top of the timberway and is high enough to permit landing the steel in the stope. At times a hole is drilled in the back and an eyebolt is inserted to do the same work as the tripod. Bronze-bushed sheave wheels and 5/16-inch cast-steel wire rope are used. It has been found that in these sulphide stopes cast-steel rope rusts and deteriorates less than the plough steel or other better grades of material.

The air and water hose are carried up each manway. The manways are 100 feet apart or less, so that a machine is always within 50 feet of an air connection. Normally the hose are connected to valves in the manways. In places where more than two machines are being operated, a manifold is used. The manifold consists of two pieces of pipe - one 4-inch, for air, and the other 2-inch, for water - about 2 feet long and fastened together so they can be easily handled. Six connections to which valves are attached are welded on each. One of the connections on each piece of pipe serves as the inlet. The manifold is connected by a 2-inch air hose to the 2-inch air line in the manway and by a $\frac{3}{4}$-inch water hose to the $\frac{3}{4}$-inch water connection in the manway. Uniform hose nuts are attached to the stems of all hose; thus any hose can be attached to any valve spud.

The cribbed manways are lagged on the outside with 3-inch plank and stand very well, usually lasting until breaking of ore is completed. However, as the working face approaches within 50 feet of the level above, raises are started upwards at about 25-foot centers. After connecting to the level these are used as entrances to the stope and the cribbed manways below are abandoned.

A floor pillar generally of a height a little more than the width of the stope is left to support the walls during the final drawing. If the ore in this pillar were to be broken down into an empty stope, following drawing, it would mix with considerable waste in the bottom, which has sloughed from the walls and followed the ore down, and the concussion would bring down more waste on top of it. When the grade of ore justifies it, the pillar can be removed by first filling the stope with waste through the raises mentioned. If this is done, one of the old cribbed manways can be used as a manway and ore-pass. These cribbed raises, being reinforced outside with long, overlapping plank, stand up surprisingly well in a stope that has been drawn empty; if the upper part of a raise topples over, it can be rebuilt as filling proceeds.

When breaking ore has been completed in a stope, the broken ore is withdrawn as rapidly as tramming will permit. The top of the broken ore is kept as nearly level as is possible. This tends to prevent any waste which has fallen on top of the ore from mixing with it. Weekly or oftener a survey is made of the position of the top of the broken ore in the stope by dropping a tape through the various holes below the level or by examination through existing manways, and the drawing of the chutes is adjusted accordingly.

Often broken ore hangs up on the pillars during this drawing stage. Bombs consisting of several sticks of powder tied to a pole are used to dislodge this ore.

When the ore has been pulled down until the chutes are empty, there still remain piles of ore on the drift lagging between the blasting chambers. If the drift is to be kept in use, this ore is left until some later date. If it is to be abandoned, this ore is recovered by raking it into the chutes, using the blasting chamber for cover, or extracted through "Cousin Jack" sets installed for the purpose.

Large bowlders are broken in the blasting chamber by drilling and blasting or by plastering, depending on the shape of the bowlder. This work is done by a chute blaster who takes care of all the chutes in one stope. The timbers in the blasting chamber last the life of the stope, but at the end many are badly worn.

Gelatin dynamite of 40 per cent strength and No. 8 blasting caps are used. In very hard ore 60 per cent strength gelatin is used when it is desired to break the rock finer. Timberite, a permissible explosive, is used to blast timber, such as comes through the chutes.

Cut-and-Fill Stoping

One stope where a transverse fault slip made the walls bad is being mined by the cut-and-fill method (see fig. 7). This stope averages about 20 feet in width and 200 feet in length. Chutes are originally spaced at 15-foot intervals along the drift, but as waste is introduced all are abandoned with the exception of those at the ends of the slides. A central raise to the level above serves to introduce waste fill. The ore is mined in 9-foot cuts, one-half of the stope furnishing ore while the other half is being gobbed. The working face is parallel to a floor laid on the natural slope of the waste fill. The floor is of 3-inch plank laid on the gob which has been leveled off to a plane surface. Stoper machines are used. Grizzlies of 8 by 8 inch timber spaced 10 inches apart are built over the tops of the ore chutes.

This method is more expensive than shrinkage stoping, but it affords cleaner mining and a place to gob low-grade material. In this particular stope the chief advantage of the method is that the open space at any one time is small and the waste fill holds the broken wall so that it does not move.

UNDERGROUND HAULAGE

Ore cars are 16 cubic foot end-dump with roller bearings. The track is of 16-pound rails laid to 18-inch gage. Grades are maintained at 0.5 per cent in favor of the loads. Hand-tramming is used where distances do not greatly exceed 600 feet. Tonnages from any one stope are comparatively small, and experience in this mine had tended to prove that for short distances hand-tramming is most economical. Trammers load their own cars and switch them at the station. The average number of cars per man-shift is 23, trammed an average distance of about 500 feet.

A $1\frac{1}{2}$ ton storage-battery locomotive is used when the tram is more than 600 feet. It handles from six to eight cars.

The loading chutes under the stopes are of wood with iron doors (see fig. 6). The trammer stands on a platform above the top of the car when loading, which places him out of the way of a possible rock spill.

Cars of ore are hoisted in double-deck cages. The hoist handles about 300 cars in an 8-hour shift besides men and supplies. A cager and helper handle the cars. Waste is caged from level to level for stope filling or disposal; this is done on the graveyard shift to avoid interference with ore hoisting, which is done on the regular day and night shifts.

DILUTION

Normally the walls of stopes stand well. There are barren areas and andesite dikes, however, that tend to dilute the ore. Some wall rock also falls in, especially after the ore is being drawn down past it. The dilution from all these sources probably amounts to 10 per cent.

A very small amount of waste is picked out of the chutes underground and discarded into old open stopes. All rock hoisted passes through a sorting plant on the surface, where barren material is hand-picked from a conveyor belt. About 4 per cent of the material hoisted is discarded as waste at this point.

COMPARISON OF MINING METHODS

Shrinkage stoping is being used because of its low cost and its allowance for an even production. In this district labor is more scarce and less efficient in summer than in winter. Therefore, the intention is to build up the broken ore surplus in the winter. Shrinkage stoping permits this very readily. Cut-and-fill stoping is more expensive, but would apply very well to sections where waste or low-grade material could be left in the mine.

WAGE, CONTRACT, AND BONUS SYSTEM

All employees are paid according to the scale in effect in the district at the time. This is a sliding scale varying with the selling price of copper. The daily wages corresponding to the costs accompanying this paper are:

Shaftmen	$ 6.27
Timbermen	6.27
Miners	5.69
Muckers	5.06
Trammers	5.06
Blacksmiths	6.27
Steel sharpeners	6.27
Hoistmen	7.59

Foremen and shift bosses are paid by the month.

All miners on development work are given a bonus for footage made above a certain standard. The standard is set by the mine foreman and approved by the manager. Measurements are made by the engineering department and the bonus is paid semi-monthly.

Breaking ore in stopes is contracted for at so much per square yard along the vein. The width of the vein does not enter into the calculations except in setting the standard. This is the same measurement scheme as is used in Colorado and elsewhere where stoping is by the fathom. The stopes are measured by the engineering department on the first and fifteenth of each month and bonus is paid on the pay day following. This system has worked very well and has resulted in a minimum amount of misunderstanding on the part of the miners in regard to their bonus.

VENTILATION

The mine is ventilated by natural draft. Rock temperatures are moderate and the ore does not oxidize readily. The vertical main shaft is upcast. Air enters the mine through an adit about 140 feet lower than the collar of the shaft, then drops down an inclined shaft to the 1000 level and thence passes through raises to the lower levels. A certain amount passes through doors at each level to the stoping areas and from there travels upward through the vertical shaft. There is always an ample supply of cool fresh air except at the face of long drifts or crosscuts. Some stopes are so cool that the men must wear coats to keep warm.

In long drifts, blowers driven by direct-connected motors are used to furnish air to the face and blow out the smoke. Galvanized pipe of 12-inch size is used and is hung from the back of the crosscut or drift.

SAFETY METHODS AND FIRST AID ORGANIZATION

A modern hospital is owned and maintained in Jerome by the United Verde Copper Co. It handles all hospital cases.

First-aid instruction is given the bosses and key men periodically. The Bureau of Mines car provides instruction from time to time.

Minor accidents such as bruising a finger or toe and getting dirt in the eye are unfortunately common in spite of every precaution. The use of goggles and hard-boiled hats is compulsory. There has not been a fatal accident underground at the Verde Central or a permanent disability accident more serious than the loss of a finger in more than five years.

All electric wires are carried in conduits through the underground workings. The drifts and crosscuts are kept as clean and free from débris as possible. Safety first is talked continually.

ADMINISTRATION ORGANIZATION

The following outline shows the organization of the staff at the Verde Central mine:

	Department head	Duties
Manager	Mine foreman	Mine Ore sorting plant Mine to mill railroad
	Mill foreman	Concentrator Tailings disposal Assay office
	Master mechanic	Hoist, air compressors, shops Pumps Electric maintainance Construction work
	Chief clerk	Accounting Contact with medical department Industrial compensation Sales
	Engineer (mining)	Surveying Bonus measurements Safety first Fire protection
	Storekeeper	Warehouse Freight and express Transportation (trucks) Purchasing

SUMMARY OF COST OF MINING BY SHRINKAGE METHOD, VERDE CENTRAL MINE, FROM JUNE 1, 1929 TO AUGUST 1, 1930. TONS HOISTED, 139,203[1]

1. Underground stoping costs per ton of ore hoisted

Mining	Labor	Supervision	Compressed air-drills and steel	Power[2]	Explosives	Timber	Other supplies	Total
Mining	$0.734	$0.073	$0.485	- -	$0.234	$0.135	- -	$1.661
Transportation underground	.449	.037	- -	.075	- -	.009	.010	.580
General underground expense	.065	.012	- -	.001	- -	- -	.033	.111
Surface expense applicable to underground operations	.020	.004	- -	- -	- -	- -	.009	.033
Total	$1.268	$0.126	$0.485	$0.076	$0.234	$0.144	$0.052	$2.385

2. Stoping costs in units of labor, power and supplies

	(Stoping)
Labor, man-hours per ton:	
Breaking (drilling and blasting)	0.72
Timbering	.33
Haulage and hoisting	.61
Supervision	.11
General	.10
Total labor underground	1.87
Average tons per man shift	4.28
Labor per cent of total cost	66.31 per cent[3]
Power and supplies:	
Explosives (40 per cent gelatin)	1.16 pounds per ton
Timber, Oregon fir	3.08 board feet per ton
Total power	16.44 kw. hrs. per ton
1. Air compression	8.86 kw. hrs. per ton
2. Hoisting	4.58 kw. hrs. per ton
3. Pumping	2.81 kw. hrs. per ton
4. Ventilation	.12 kw. hrs. per ton
5. Lighting	.07 kw. hrs. per ton

1 These costs are for stoping only, and exclude major prospecting and development work. They include, however, all development work done in an ore body after stoping has commenced, such as raising from stopes to levels above.

2 The master mechanic and some electricians are charged into "Power" before this cost is distributed, since their chief function is to insure a steady supply of power to all desired points. Therefore the figures shown in this column include some labor cost.

3 This percentage is based on pay-roll figures, and does not exactly check the labor costs shown above, for the reason explained in footnote 2.

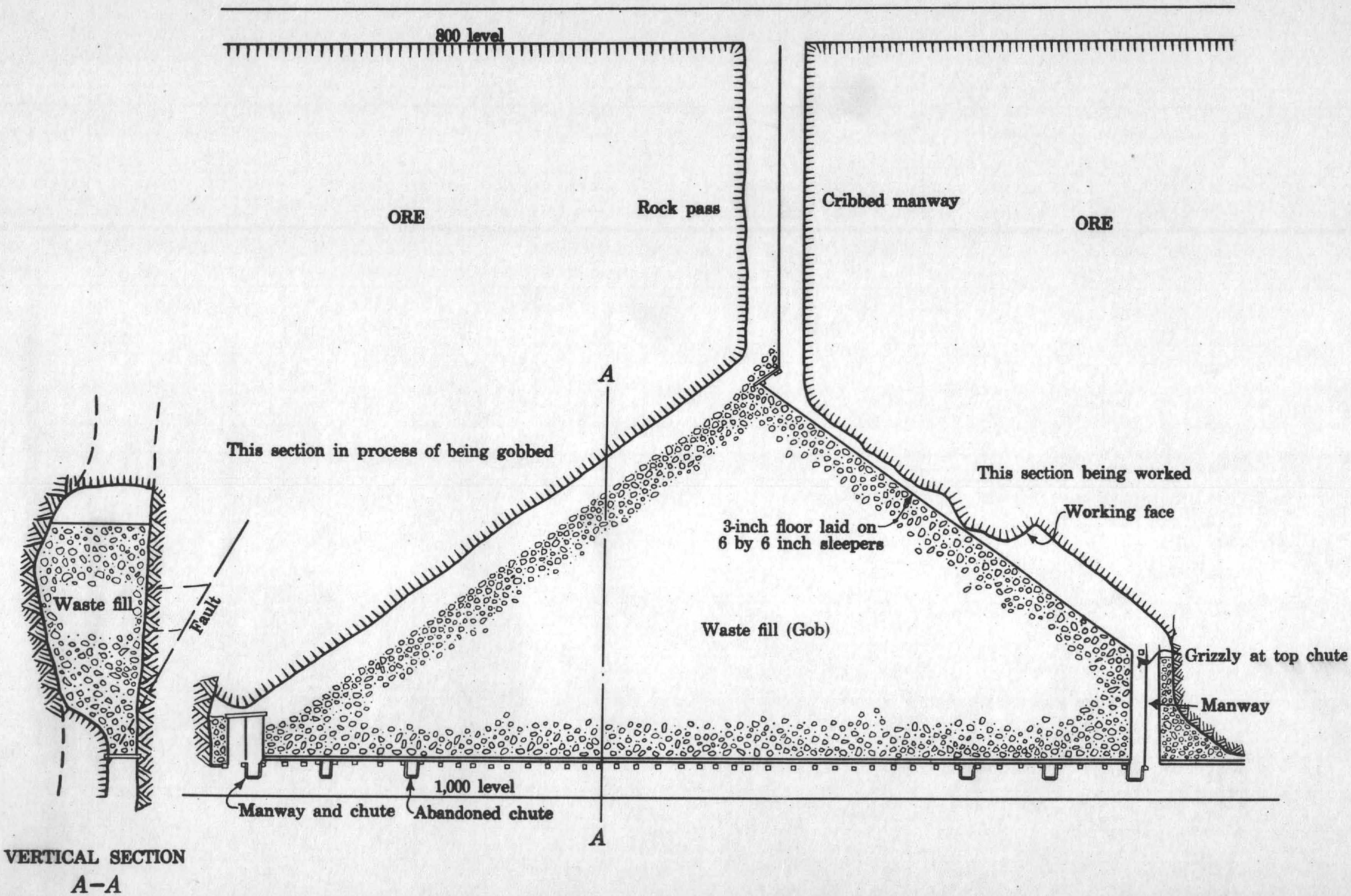

Figure 7.–Longitudinal section through 10-2 stope illustrating mining method

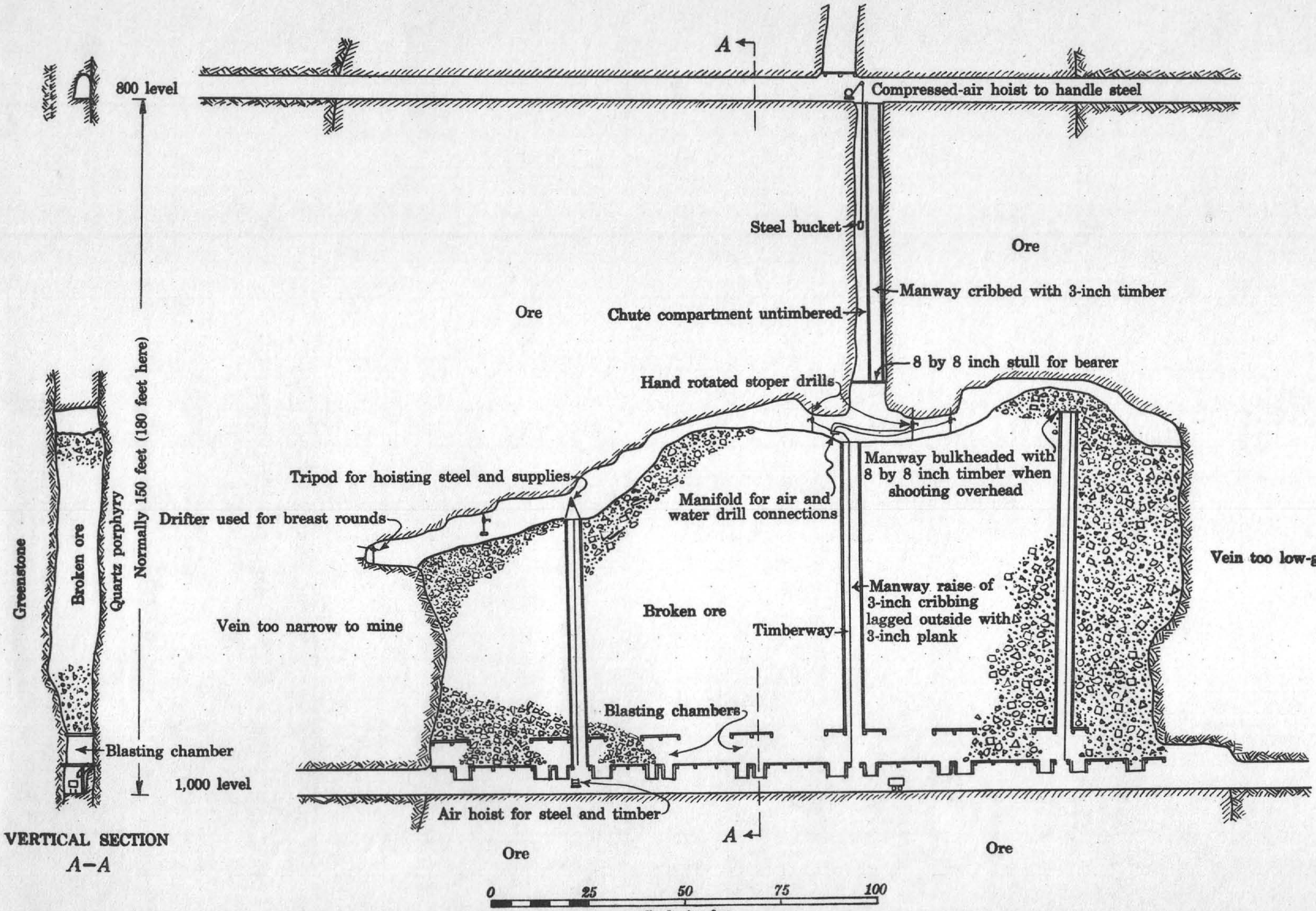

Figure 4.–Longitudinal section through 10-1 stope

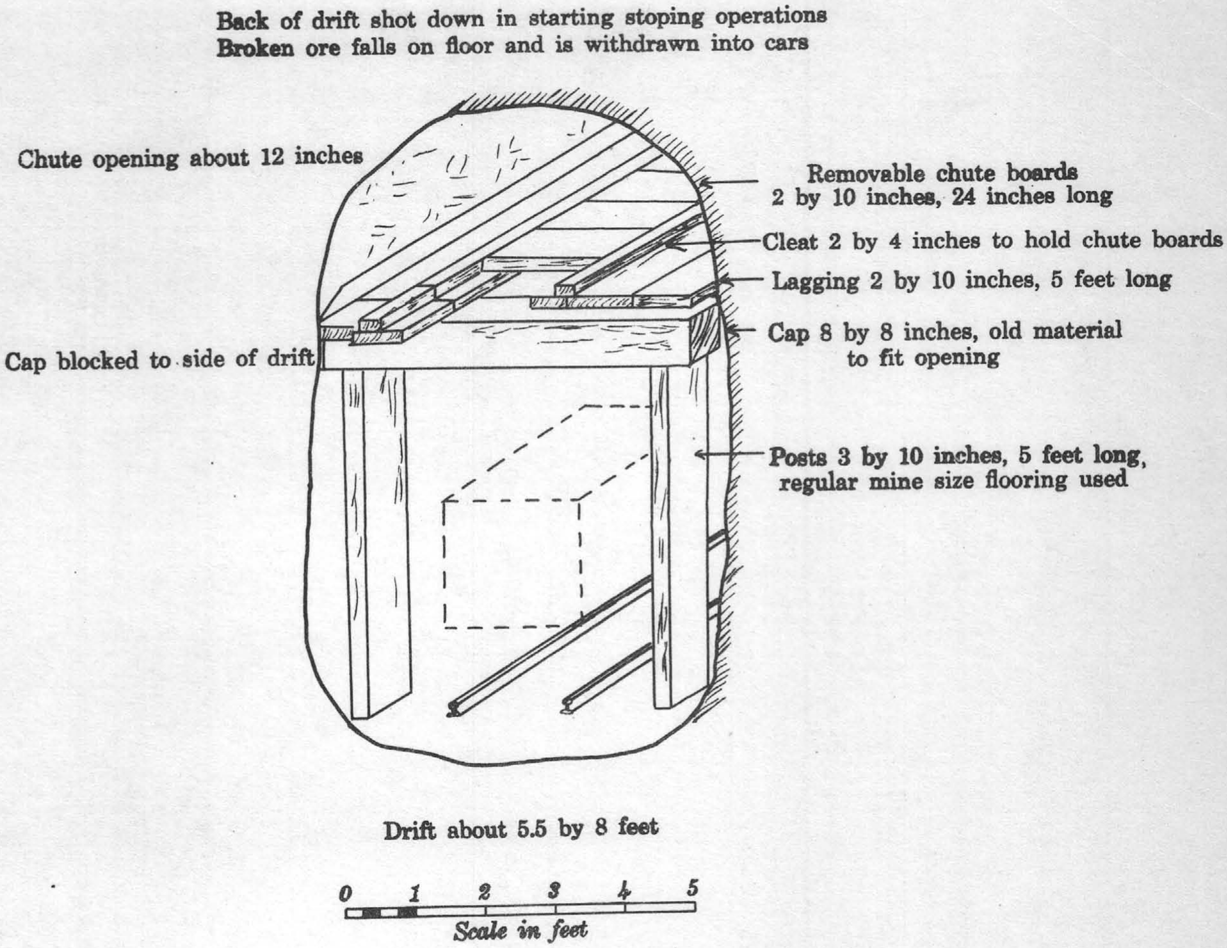

Figure 5.—"Cousin Jack" sets as used to obviate mucking

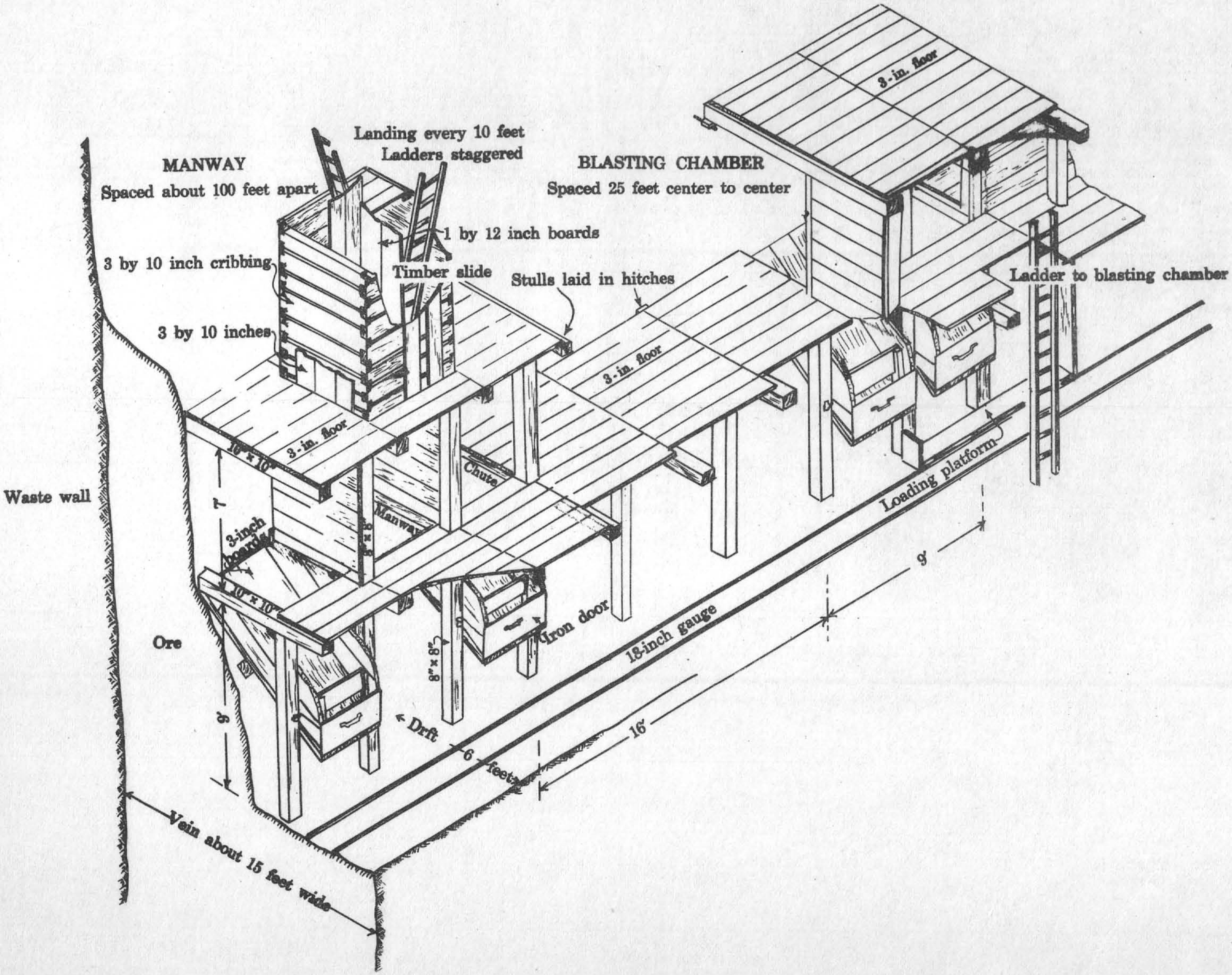

Figure 6.–Timbering at bottom of shrinkage stopes

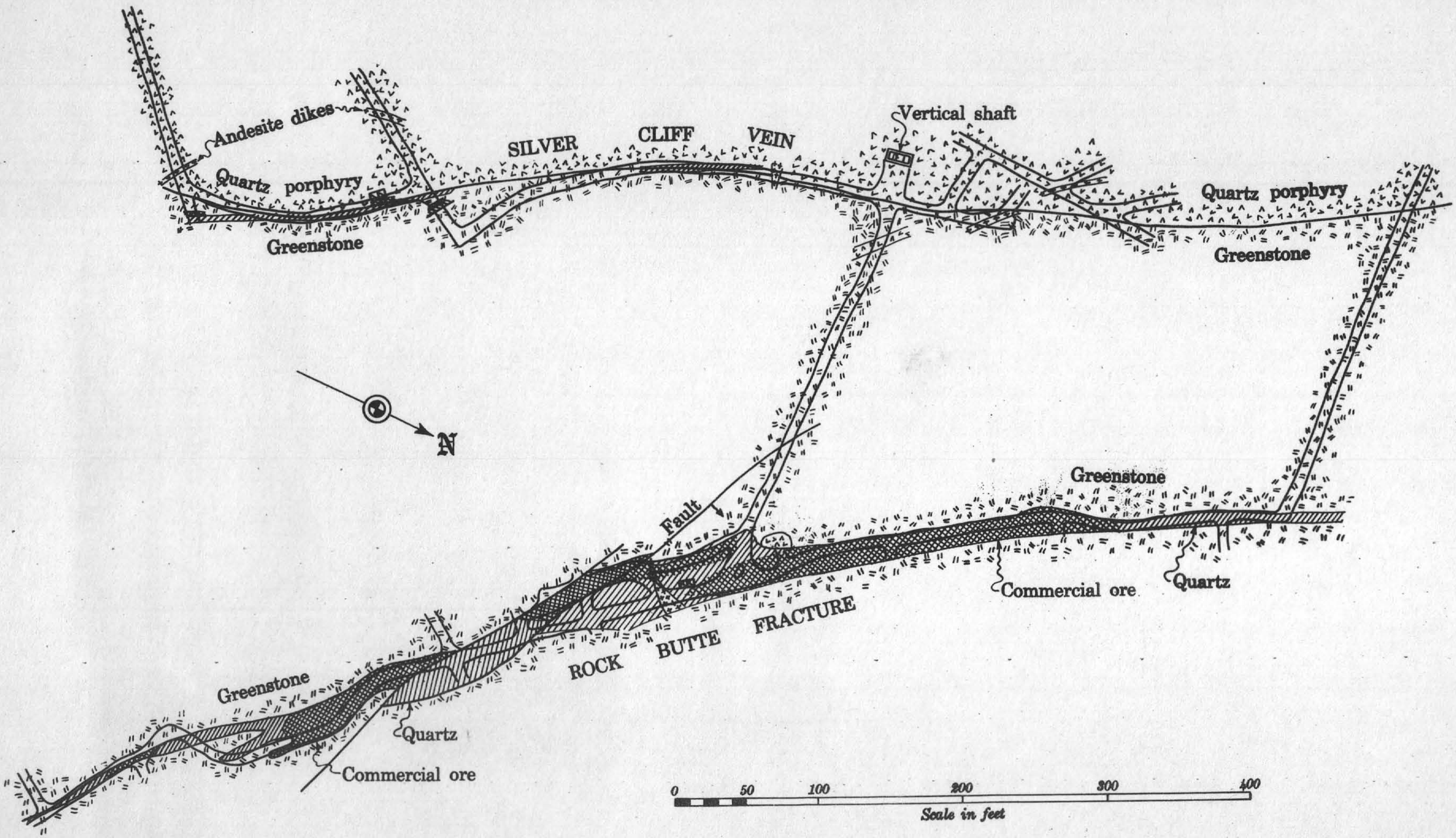

Figure 2.--Geologic map of 1,100-foot level

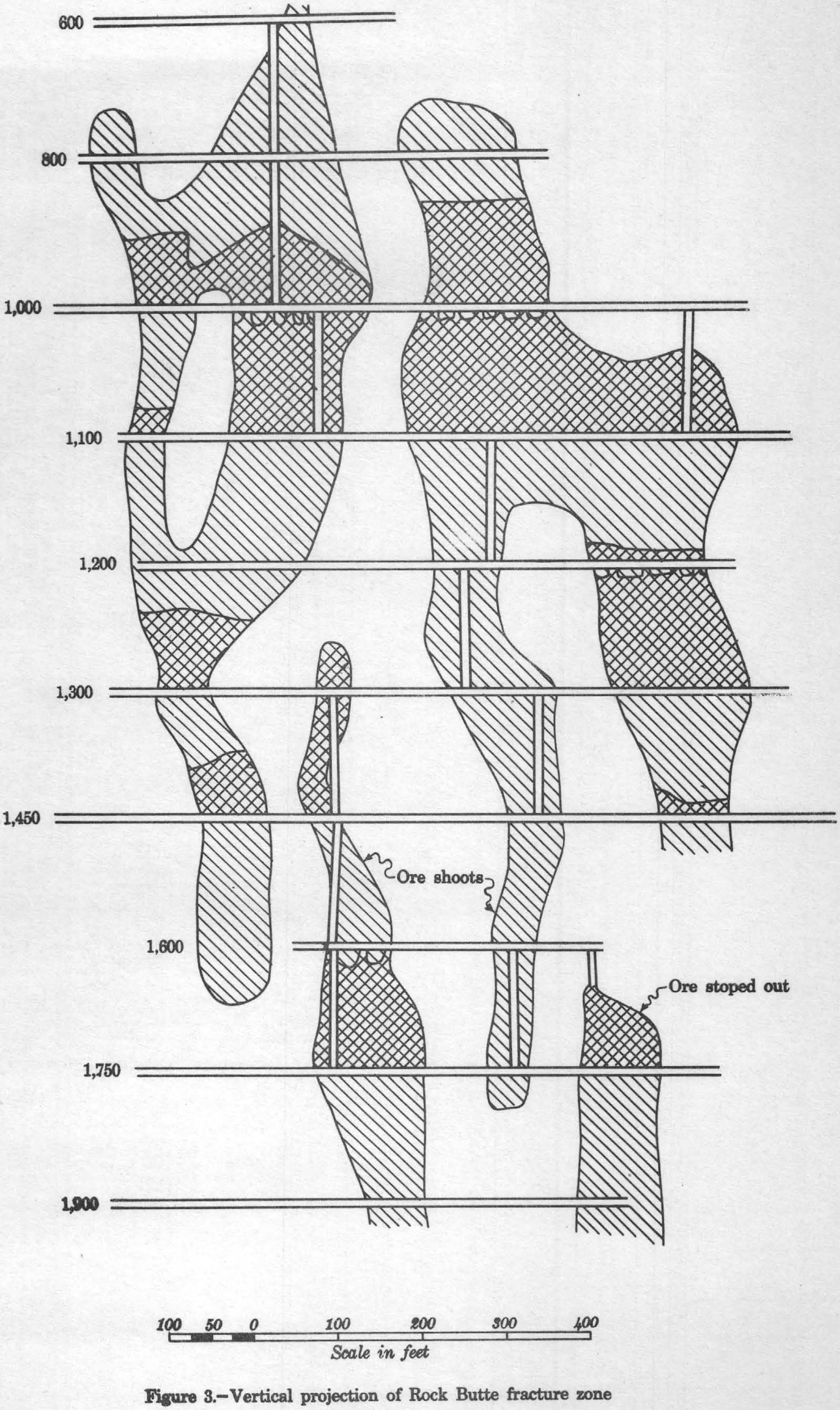

Figure 3.—Vertical projection of Rock Butte fracture zone

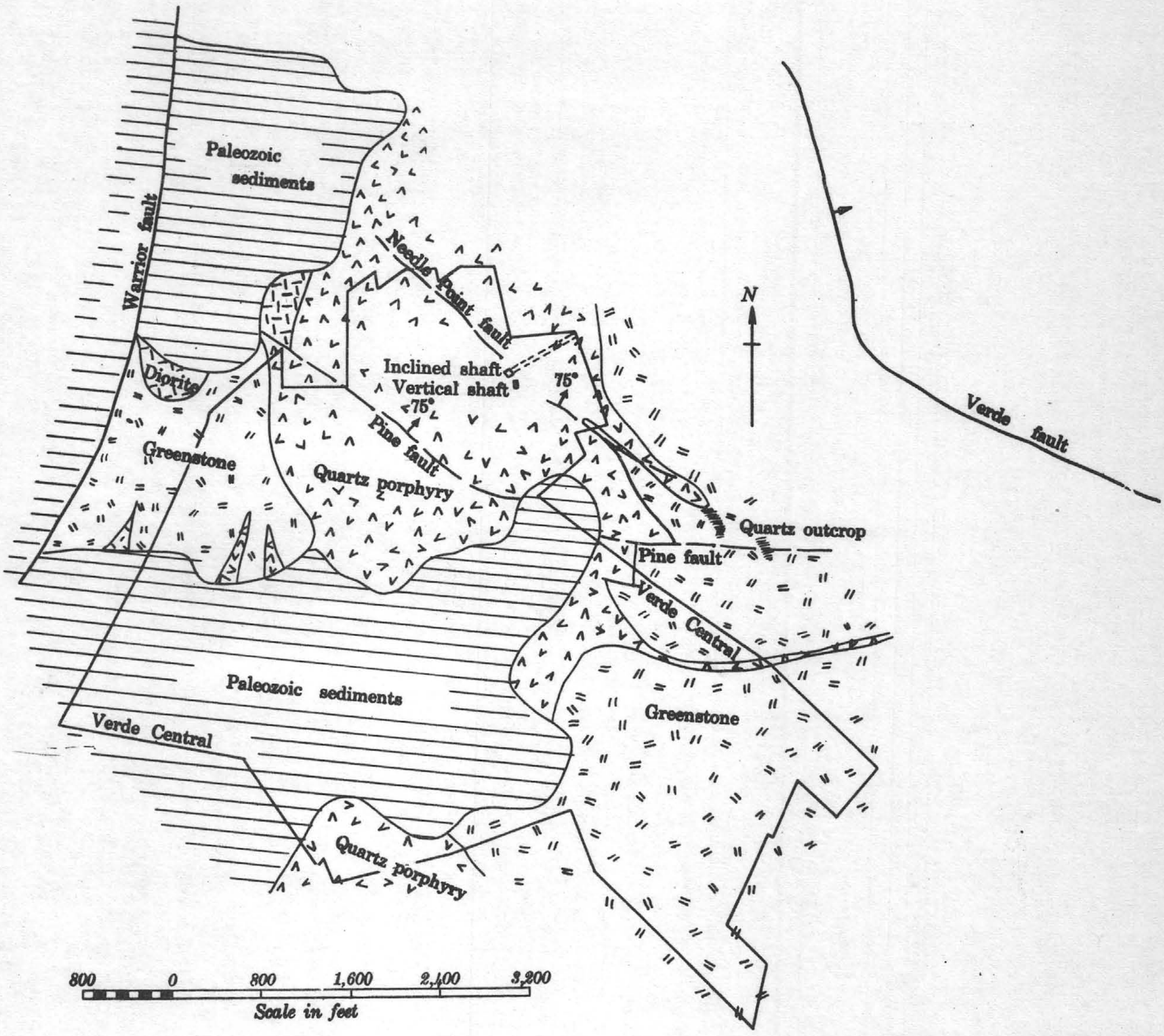

Figure 1.–Surface geologic map of Verde Central mine

September 30, 1926.

VERDE CENTRAL SAMPLES.

No. 1. Reject from samples in 1000' and 1400' level ore bodies.

	Au. Oz.	Ag. Oz.	Cu. %	Ins. %	Fe. %
No. 1	.02	.4	3.50	66.0	12.0
Concentrates	.07	1.3	12.68	10.4	35.6
Middlings	.03	.6	6.65	40.8	22.1
Tails	Tr.	.1	.28	85.2	4.9

No. 2 Reject from samples mostly from 1000' level stopes.

No. 2.	.01	.4	5.23	63.6	12.0
Concentrates	.03	1.7	14.58	26.0	25.2
Middlings	.01	.6	2.32	67.6	10.4
Tails	Tr.	.1	.26	82.2	4.2

No. 3.

No. 3	.02	1.2	10.92	51.2	16.6
Concentrates	.04	1.6	22.13	17.2	22.7
Middlings	.02	.7	5.12	65.2	11.4
Tails	Tr.	.3	.92	85.4	5.2

No. 4.

No. 4	.01	.5	4.34	60.6	12.4
Concentrates	.03	1.6	15.10	28.4	23.2
Middlings	.01	.6	5.36	61.2	12.2
Tails	Tr.	.1	.42	87.0	3.8

No. 5. From dump, - Ore Body and black schist from shaft and on 800 level.

No. 5	.01	1.0	10.40	27.6	21.6
Flot. Concentrates	.02	2.5	26.75	5.4	30.4
Middlings	.01	1.1	8.43	26.0	21.3
Tails	Tr.	.15	.58	43.8	14.8

Since purchased by the U. V. Copper Co.

September 26, 1925.

Colonel Robert M. Thompson,
16 East 43rd Street,
New York, N.Y.

Dear Colonel Thompson:

Re: VERDE CENTRAL.

I will write you a special letter on this matter, since I know you are much interested.

Yesterday afternoon in company with Mr. Williams, our Smelter Superintendent, I spent going over the Verde Central proposition with Mr. Staunton, Manager, and we carefully examined the dumps and took several samples which are now being assayed and tested to determine the suitability for concentration.

On the dump the Verde Central have accumulated practically 10,000 tons of ore which should average well in excess of 3% copper. It appears to me an excellent milling ore and altogether suitable for treatment in our concentrator.

The development of the Verde Central has been proceeding steadily; their deepest level is now being run at 1450 feet and they have some ore shoots fairly continuously from this point up to the 800 foot level. Their Manager estimates that they have developed close to 500,000 tons of ore averaging about 5% copper, and with gold and silver values about $1.00 per ton. The great bulk of this ore is highly

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CONSOLIDATED ARIZONA SMELTING COMPANY

FROM HUMBOLDT OFFICE. SUBJECT: DATE:

TO NEW YORK OFFICE.

Colonel Robert M. Thompson,
14 East 42nd Street,
New York, N.Y.

Dear Colonel Thompson:

Re: VERDE CENTRAL.

I will write you a special letter on this matter, since I know you are much interested.

Yesterday afternoon in company with Mr. Wilkins, our Smelter Superintendent, I spent going over the Verde Central proposition with Mr. Thornton, Manager, and we systematically examined the dumps and took several samples which are now being sampled and tested to determine the suitability for concentration.

On the dump the Verde Central have accumulated practically 10,000 tons of ore which should average well in excess of 2% copper. It appears to be an excellent milling ore and altogether suitable for treatment in our concentrator.

The development of the Verde Central has been proceeding steadily; their deepest level is now being run at 1000 feet and they have ore and are stopes fairly continuously from this point up to the 300 foot level. Their Manager estimates that they have developed close to 500,000 tons of ore averaging about 5% copper, and with gold and silver values about $1.00 per ton. The great bulk of this ore is highly siliceous and suitable for milling, although they have in places some higher grade basic material, but I should say that 80% or more of the run of mine product ought to be treated in a mill. They are at present mining 25 to 30 tons per day of ore from various faces in development and by about December 1st they expect to have completely opened up one stope of high grade material which they figure will run 7% to 8% copper and which they will probably consider mining and shipping immediately thereafter.

The transportation facilities from the mine are not good, and they will have to expend some money to either provide facilities with a proper truck road to handle the ore to Jerome or else construct an aerial tramway to the Jerome depot. The first arrangement will cost them $3,000 to $4,000, the second arrangement probably $30,000, but the cost of trucking will probably be 50¢ per ton, as against 15¢ for operating the rope-way.

Later on if their mine develops as they hope, and proves up a million tons or more of ore, they will probably run a tunnel to connect with the main haulage tunnel of the United Verde Extension Mine, and handle all their product out through this opening, but the expense involved in making this connection would be between $150,000 and $200,000.

I told Mr. Staunton that we were prepared to handle their ore immediately, and that if it would concentrate as I

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CONSOLIDATED ARIZONA SMELTING COMPANY

FROM HUMBOLDT OFFICE. SUBJECT: DATE:

TO NEW YORK OFFICE.

assumed with high recovery of value and a ratio of three to one, or better, we could offer them very favorable terms, which we discussed in a general way without getting down to definite figures, as I did not feel in a position to make him a positive offer until I had conferred with you, and also determined by experiment exactly how the ore would concentrate.

Mr. Staunton will submit this matter to his directors and it is probably that Mr.Campbell, the President, and some of the other directors will be out this way in late October or early November, and will then reach a decision in regard to ore shipments, and while there is nothing positive as yet, Staunton will pretty surely recommend that they begin the shipments as soon as the proper facilities can be provided, which I should say might be in December, if they decide to truck the ore, or by January or February if they build a rope-way. In this latter connection I am trying to sell them our old rope-way from DeSoto, which ought to answer their purposes very nicely. I should judge that they could commence shipments at the rate of approximately 100 tons of milling ore per day from the dump, and perhaps 50 tons of higher grade smelting ore from the stope mentioned above and other development work, and, needless to say, we will very much desire to secure this material.

In figuring with Mr. Staunton I assumed that we would secure the 50¢ freight rate from Clarkdale which has

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CONSOLIDATED ARIZONA SMELTING COMPANY

FROM HUMBOLDT OFFICE. **SUBJECT:** **DATE:**

Colonel Thompson, - 5. September 20, 1918.

TO NEW YORK OFFICE.

assayed with high recovery. Of [illegible] and a ratio of [illegible] to one, or better, we could offer them very favorable terms, which we discussed in a general way without getting down to definite figures, as I did not feel in a position to make him a positive offer until I had conferred with you, and also determined by experiment exactly how the ore would concentrate.

Mr. Staunton will submit this matter to his directors and it is probable that Mr. [illegible], the President, and some of the other directors will be out this way in late October or early November, and will then reach a decision in regard to our proposition, and while there is nothing positive as yet, Staunton will pretty nearly agree and that they would make the shipments as soon as the necessary facilities can be provided, which I should say might be in December, if they decide to truck the ore, or by January or February if they build a road-way. In this latter connection I am trying to sell them our old tram-way from [illegible], which might fit their purposes very nicely. I should judge that they could commence shipments at the rate of approximately 150 tons of milling ore per day from the dump, and perhaps 50 tons of higher grade smelting ore from the stope mentioned above and other development work, and, needless to say, we will very much desire to secure this material.

In figuring with Mr. Staunton I assumed that we could secure the low freight rate from [illegible] [illegible] been repeatedly promised by the Santa Fe, that our milling cost would be slightly in excess of $1.00 per ton, to which we would add a 50¢ profit, and the cost of smelting the concentrates would not exceed $2.50 per ton, to which we would add, as I have previously written you, between $1.00 and $2.00 for smelting profit. I gave him our deduction from the copper market as 3¢ per pound to cover converting, freight, refining and marketing, or as 2.75¢ per pound if they were willing to accept settlement three or four months after smelting at Humboldt, in which event we should not have to allow for interest on money advanced. This latter arrangement they would probably prefer.

To the best of my knowledge and belief we have no serious competition to meet in this matter. The United Verde certainly will not take their ore; the United Verde Extension cannot take the bulk of it because of its high silica content, and I feel satisfied that they will not attempt to handle the small proportion of direct smelting ore; Hayden could not compete against us by reason of freight differential, unless they wanted to handle this material at less than cost and for various reasons I do not think the Verde Central would do business with Hayden if it could possibly be avoided. It appears then to be merely a question of when the Verde Central will decide to start shipping ore rather than store it on the dump, and in this matter I feel they will be guided largely by the trend of the

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CONSOLIDATED ARIZONA SMELTING COMPANY

FROM HUMBOLDT OFFICE. SUBJECT: DATE:

TO NEW YORK OFFICE.

Colonel Thompson, - 2 September 2[illegible], 19[illegible]

been [illegible] prospects of the [illegible], that our milling [illegible] cost will be slightly in excess of $1.00 per ton, to which we would add a fair profit, and the cost of smelting the concentrates would not exceed $4.00 per ton, to which we would add, as I have previously written you, between $1.00 and $2.00 for smelting profit. I have not deducted in treating [illegible] copper market at 2[illegible] per pound to cover converting, freight, refining and selling, or [illegible] figures [illegible] would be [illegible] wishing to [illegible] smelting [illegible], [illegible] smelting [illegible] allow for interest on money advanced. This [illegible] they [illegible].

In the [illegible] and [illegible] we have no serious competition to meet in this section. The United Verde certainly will not [illegible] United Verde Extension cannot take the [illegible] of [illegible] high silica content, and I feel satisfied that they would not attempt to handle the small proportion of [illegible] smelting [illegible] which would not compete against us by reason of [illegible] freight differential, unless they wanted to handle the [illegible] at less than cost and for whatever reason I do not think the Verde Central would do business with them if it could possibly be avoided. It appears that to be merely a question of when the Verde Central will decide to start shipping ore rather than store it on the dump, and in this matter I feel they will be guided largely by the trend of the copper market and perhaps other considerations with which I am not familiar, but I feel very much encouraged over the result of my interview with Mr. Staunton and sincerely hope and rather expect that we shall close some definite arrangement by November of this year.

Once the Verde Central start active stoping operations, which will probably not be before the Spring of 1926 or possibly not before the Summer of that year, they can easily produce 200 to 300 tons of milling ore per day, and perhaps 50 tons of direct smelting ore. This, in addition to our other charge from Blue Bell and elsewhere, would keep us going on a very satisfactory and economical basis.

Yours very truly,

General Manager.

GMC-s

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CONSOLIDATED ARIZONA SMELTING COMPANY

FROM HUMBOLDT OFFICE. SUBJECT: DATE:

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