CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
416 W. Congress St., Suite 100
Tucson, Arizona 85701
520-770-3500
http://www.azgs.az.gov
inquiries@azgs.az.gov

The following file is part of the

James Doyle Sell Mining Collection

ACCESS STATEMENT

These digitized collections are accessible for purposes of education and research. We have indicated what we know about copyright and rights of privacy, publicity, or trademark. Due to the nature of archival collections, we are not always able to identify this information. We are eager to hear from any rights owners, so that we may obtain accurate information. Upon request, we will remove material from public view while we address a rights issue.

CONSTRAINTS STATEMENT

The Arizona Geological Survey does not claim to control all rights for all materials in its collection. These rights include, but are not limited to: copyright, privacy rights, and cultural protection rights. The User hereby assumes all responsibility for obtaining any rights to use the material in excess of "fair use."

The Survey makes no intellectual property claims to the products created by individual authors in the manuscript collections, except when the author deeded those rights to the Survey or when those authors were employed by the State of Arizona and created intellectual products as a function of their official duties. The Survey does maintain property rights to the physical and digital representations of the works.

QUALITY STATEMENT

The Arizona Geological Survey is not responsible for the accuracy of the records, information, or opinions that may be contained in the files. The Survey collects, catalogs, and archives data on mineral properties regardless of its views of the veracity or accuracy of those data.

J. H. C.

CALCULATION OF PIT LIMITS AND ORE RESERVES, COPPER CITIES MINING COMPANY

BY J. H. Gray and W. W. Simmons

INTRODUCTION

Copper Cities Mining Company operates an open pit mine in a disseminated copper deposit which is located in the Globe-Miami district, Gila County, Arizona.

Some attention was given the deposit early in the history of the district, but the first major exploration was begun in 1917 by the Louis d'Or Mining and Milling Company. This work, consisting of a shaft 360 feet deep and 12 drill holes, totaling nearly 9000 feet, was completed in 1922. Further exploratory drilling was done in 1929 and 1930. These efforts showed the presence of disseminated copper minerals, but the grade was too low to be considered ore at that time.

Miami Copper Company purchased the property in 1940 and organized a wholly-owned subsidiary, Copper Cities Mining Company, to operate the property. This paper is chiefly concerned with exploration by this company.

Systematic exploration by churn drilling was started in 1943 and completed in 1948 with the blocking out of 33,800,000 tons of low grade ore amenable to open-pit operations. Stripping was begun in November, 1950, and mill production was started in August, 1954.

ACKNOWLEDGEMENTS

The writers are indebted to many people for help of various kinds. We are particularly indebted to Dr. N. P. Peterson and Mr. E. N. Pennebaker. Their excellent work, Dr. Peterson's published paper

and Mr. Pennebaker's private reports on the Copper Cities ore body are the basis for much of the geological thought in the present work.

GENERAL GEOLOGY

Figure 1 is a generalized geologic map of the Copper Cities pit and the immediate area around it. The Lost Gulch quartz monzonite is the predominant rock in the pit. The other important rock in relationship to the ore is granite porphyry. In general, the quartz monzonite is the better grade material. A small amount of diabase is included within the pit limits.

STRUCTURE

The most important structures in relationship to the ore body are the Coronado and Drummond fault zones. The Coronado which lies near the west side of the pit, strikes north and dips steeply west. The Drummond, near the eastern pit limit, strikes northwest and dips about 60° northeast. Along the northern edge of the pit is the Sleeping Beauty fault which strikes northeast; its dip is unknown. The area bounded by these faults has been raised relative to the adjacent blocks. The ore body itself is intricately dissected by many minor fractures with no dominant pattern.

MINERALOGY

The principal hypogene minerals in the deposit are quartz, pyrite, chalcopyrite and molybdenite. Chalcocite is the predominant supergene sulfide mineral, and malachite, azurite and turquoise are the principal acid-soluble copper minerals in the ore body.

CHURN DRILLING

The base pattern for churn drilling was a 250 foot grid. From previous experience in the district, this was thought to be sufficiently close for accuracy of grade and tonnage calculations, but as a measure of insurance, some holes were drilled at intermediate points. Drilling at intermediate points was also used to more precisely define the pit limits on some sections. The base grid was oriented to make the sections at right angles to the supposed elongation of the ore body. As finally developed by the drilling, the right angle relationship did not hold exactly, but no serious error was introduced by this fact.

The churn drill holes were sampled at 5 foot intervals using the conventional Jones splitter. Each 5-foot sample was assayed for total copper and oxidized copper. Composite samples of each 50 feet were assayed for gold, silver and molybdenum. All samples were logged for rock type and other geological features.

Preceding and concurrent with the drilling, a geological map of the surface was made as a guide for the exploration. At the completion of the drilling, a map was prepared showing surface geology, topography and drill hole collars.

COMPILATION OF DATA FROM EXPLORATION PROGRAM

Using the plan map as a base, cross sections and longitudinal sections were made showing the assay data of drill holes. By inspection, it was obvious that correlation between drill holes on assay data alone did not show a coherent nor probably true relationship. By plotting rock types and other

geology on the sections, more reasonable ore outlines could be drawn.

On the basis of previous district experience, it was decided that a 45 foot bench height would give the best mining operations. By inspection and cut and try, the elevation of the bottom level of the pit was set at 3600 feet. Factors which influenced this selection were maximum ore production, working room, pit drainage, haulage and others, but to some extent the final figure was arbitrary and depended on the judgment of the planners. Using this base, the upper bench elevations were drawn on the sections and the average grade of the holes through each bench was plotted along the hole. These sections were the work sheets for the determination of the ultimate pit limits.

DETERMINATION OF ULTIMATE PIT LIMITS

The exploration indicated an ore body of small tonnage and low grade. It was improbable that mining would disclose enough additional ore, or that market conditions would change sufficiently, to justify enlarging the initially set pit limits. For these reasons, it was of prime importance that the pit as initially planned be the best economically for the life of the operation. This meant that the pit limits must be extended to the theoretical slope lines which would just meet a set of conditions to give an acceptable minimum profit.

From the exploration, it was known that the waste to be stripped was a relatively uniform cover. It was also evident that the copper mineralization was largely gradational which meant that the pit limits would be assay boundaries rather than some other geologic feature. For these reasons, it was believed that the pit limits could

be set by consideration of a line rather than by calculating a three-dimensional tonnage figure. This concept materially reduced the required calculations.

To locate the theoretical slope lines certain assumptions were necessary. These were grouped as the cost to concentrate one ton of ore, the costs per pound of copper, and the per cent copper extraction.

The cost to concentrate one ton of ore includes:

a. Cost per ton mined.
b. Cost per ton milled.
c. Miscellaneous costs per ton.

The costs per pound of copper include:

a. Smelting cost per pound of copper.
b. Miscellaneous costs per pound copper.
c. Minimum acceptable profit per pound of copper.

The gross value per pound of copper can be defined as the market value minus the costs per pound of copper.

The mine grade which will just satisfy the assumed costs per ton of ore and per pound of copper can be calculated by the following formulae.

A. $\dfrac{\text{Total cost to concentrate 1 ton ore}}{\text{Gross value/lb. Cu}}$ = Net lbs. Cu/ton ore required to give minimum acceptable profit.

B. $\dfrac{\text{Net lbs. Cu/ton ore}}{\text{\% Extraction}}$ = Gross lbs. Cu/ton ore required to give minimum acceptable profit.

C. $\dfrac{\text{Gross lbs. Cu/ton ore}}{2000}$ = % Cu required in mill heads to give minimum acceptable profit.

To illustrate the use of the above formulae, certain figures, not necessarily those used at Copper Cities, are shown as follows:

Cost to concentrate 1 ton of ore: $0.94

Cost/lb. Cu: $0.0575 = Cost/lb. Cu to put 1 lb. Cu in concentrate into a market product.

Market price/lb. Cu	$0.18000
Minus cost/lb. Cu	0.05751

$0.12249 = Gross value/lb. Cu at mimimum acceptable profit.

$$\frac{\$0.94 \text{ (cost to concentrate 1 ton)}}{0.12249 \text{ (Gross value/lb. Cu)}} = 7.674 = \text{Net lb. Cu per ton of ore required to give minimum acceptable profit.}$$

$$\frac{7.674 \text{ (Net lbs. Cu/ton ore}}{.87511 \text{ (\% extraction)}} = 8.769 = \text{Gross lbs. Cu/ton ore required to give minimum acceptable grade.}$$

$$\frac{8.769 \text{ (Gross lbs. Cu/ton)}}{2000} = .438\% \text{ Mine grade necessary to satisfy above assumptions.}$$

The mine grade of ore necessary to meet the above conditions plus the mining of 1 ton of waste is calculated by the same basic formula, but must include the additional cost of mining 1 ton of material, i. e.,

$$\frac{\dfrac{\$0.94 + \$0.20}{.12249}}{\dfrac{.87511}{2000}} = 10.635 \text{ lbs. Cu ton} = 0.532\% \text{ mine grade.}$$

The grade necessary to give the minimum acceptable profit with a different tonnage of waste removal is calculated similarly.

A table of waste to ore ratios for which a corresponding mine grade of ore will meet the assumed conditions including profit is presented in Table 1.

With the aid of this table, the determination of theoretical slope lines which fix the ultimate pit limit can more easily be made.

At Copper Cities, it was decided that a 45° backslope could be safely maintained. The initial step in locating a backslope line on any section was to arbitrarily draw a 45° line on the section.

This initial line was, of course, located near one extremity of the ore body and as near the correct position as judgment based on quick visual inspection permitted.

In the investigation of the slope lines, it was assumed that ore grade in any prospect drill hole could be projected to the mid point between 2 adjoining holes. This assumption was subject to modification by geological conditions. The investigation consisted of several steps. First the total length of the backslope line from the bottom of the lowest ore bench to point where the line intersected the surface was measured. Next, the integral lengths of lines along the slope through each ore bench was measured, and each length multiplied by its related grade of ore. The summation of these "Grade Lengths" divided by the summation of the integral length of slope line through the ore benches is the average grade along this section of the backslope line. The total length of backslope line minus the length in ore is the measured length of slope line in waste.

By reference to Table 1, a waste to ore ratio corresponding to the average grade as computed can be found. By multiplying this waste figure by the length of ore line, a theoretical length of waste line is obtained. If this theoretical length of waste line is greater than the length of line actually measured, the ultimate slope line lies in the direction away from the center of the pit. Conversely, if the theoretical length of waste line is less than the measured waste line, the ore along this slope will not pay for the waste and the ultimate pit limit lies in a direction toward the pit center. By sufficient trial and error, a slope line can be located along which the measured waste will just equal the theoretical waste that can be carried by the ore under the assumed conditions.

Figure 2 is a section of Copper Cities. On line A, the total length of the backslope line from pit bottom to intersection with surface is 545 feet. The summation of the integral length of line through each ore bench multiplied by its related grade of ore divided by the total length of line in ore is .674%, which is the average grade of that portion of the slope line in ore. The length of the slope line in waste is 290.40 feet. By reference to Table 1, the waste ore ratio corresponding to .674% is 2.5. By multiplying the waste figure by 254.60, the length of line in ore, we see that the slope line can be moved outward. By similar calculations on Line B, the waste-ore ratio was found to be 1:1, and the waste figure multiplied by the ore length exceeded the measured waste length and the slope line must be moved inward. On Line C, the theoretical waste line was 323.33 feet and the measured waste was 329.05 feet. It was felt that this was as close as the accuracy of the original data and assumptions permitted, and this line was used. The detailed calculations of the lines are given below as illustrative of the method.

CALCULATIONS: INVESTIGATION OF SLOPE LINES

LINE A

Measured line			Slope Line Thru Ore		Grade
Designation	Length Feet	Bench	Length	Grade	Length Units
Ore	255.0	3645	45.40	.55	25.02
Waste	290.40	3645	18.15	.47	8.53
	545.40	3690	63.65	.70	44.56
		3735	63.65	.58	36.92
		3825	63.65	.89	56.65
			254.60	.674	171.68

From Table 1: .674% ore grade will carry 2.5:1 waste to ore
∴ 254.6 (ft. ore) x 2.5 = 636.50 ft. of waste which 254.6 ft. of .674% ore will carry. Since this is greater than the 290.4 ft. of waste actually measured along this slope (A), line B was tried.

LINE B

Measured line Length Designation	Feet	Bench	Slope Line Thru Ore Length	Grade	Grade Length Units
Ore	88.65	3645	63.65	.47	29.92
Waste	456.35	3690	25.00	.70	17.50
	545.00		88.65	.535%	47.42

From Table 1: .535% ore grade will carry 1:1 waste to ore ∴ 88.65 ft. x 1.0 = 88.65 ft. of waste which 88.65 ft. of .535% ore will carry. Since this is less than the 456.35 ft. of waste measured along this slope line B, line C was tried.

LINE C

Measured line Length Designation	Feet	Bench	Slope Line Thru Ore Length	Grade	Grade Length Units
Ore	230.95	3645	63.65	.47	29.92
Waste	329.05	3690	63.65	.70	44.56
	560.00	3735	63.65	.58	36.92
		3825	40.00	* .50	20.00
			230.95	.569	131.40

* Assume .5% grade from midpoint to fault.

From Table 1: .569% ore grade will carry 1.4:1 waste to ore. ∴ 230.95 ft. of .569% ore will carry 323.33 ft. of waste. The slope line thru waste actually measures 329.05 ft. As this is as close as the accuracy of data, this line was used as theoretical slope line.

The back slope lines on all sections were calculated by a similar process.

The portion of the line near the Drummond fault illustrates a modification of the basic method based on geological conditions. Ore was figured to the fault rather than to the midpoint between holes.

After completion of the sections, the data was transferred to plan maps. The initial step was to pick a key level near the midpoint of the ore column. A composite plan map was constructed by mechanical development from the key level up slope to surface, and down slope to the pit bottom. Individual bench maps were then made by transfer of theoretical bench outlines from the composite map and plotting of prospect holes with average grade and geology as derived from the sections.

Polygonal areas of grade influence modified by the geology were constructed around each hole. By planimeter, the area of influence of each hole was determined. From this data, the average grade and the tonnage of ore and waste on each bench was computed. The total tonnage of ore and waste in the pit was obtained by addition of individual bench tonnages and average mine grade by the bench average grade weighted by bench tonnage.

Based on these calculations, the Copper Cities pit contains 33,800,000 tons of ore. The waste to be removed at the start of mining totaled 34,700,000 tons which is a 1.03 to 1 waste to ore ratio.

TABLE 1

ORE-WASTE RATIO AND ITS REQUIRED MINE GRADE

Ratio Waste to Ore	Mine Grade Ore	Ratio Waste to Ore	Mine Grade Ore	Ratio Waste to Ore	Mine Grade Ore
0.0:1	.438%	1.7:1	.597%	3.4:1	.756%
0.1:1	.447	1.8:1	.606	3.5:1	.765
0.2:1	.457	1.9:1	.616	3.6:1	.774
0.3:1	.466	2.0:1	.625	3.7:1	.784
0.4:1	.475	2.1:1	.634	3.8:1	.793
0.5:1	.485	2.2:1	.644	3.9:1	.803
0.6:1	.494	2.3:1	.653	4.0:1	.812
0.7:1	.504	2.4:1	.662	4.1:1	.821
0.8:1	.513	2.5:1	.672	4.2:1	.831
0.9:1	.522	2.6:1	.681	4.3:1	.840
1.0:1	.532	2.7:1	.690	4.4:1	.849
1.1:1	.541	2.8:1	.699	4.5:1	.858
1.2:1	.551	2.9:1	.709	4.6:1	.868
1.3:1	.560	3.0:1	.718	4.7:1	.877
1.4:1	.569	3.1:1	.727	4.8:1	.886
1.5:1	.578	3.2:1	.737	4.9:1	.896
1.6:1	.588	3.3:1	.746	5.0:1	.905

J. H. Courtright

PROGRESS REPORT ON COPPER CITIES MINING CO.

Copper Cities is the new open pit being developed by Miami Copper Company to take the place of Castle Dome when its ore is depleted about the end of 1953. The mine is located three miles north of the International Smelter. There is a close similarity in the geological features of Copper Cities and Castle Dome. As at Castle Dome, the ore body lies in a highly altered zone of coarse grained quartz monzonite porphyry. The topography is rugged as is typical of the district. But where the Castle Dome pit consisted in general of a side hill cut along the south flank of Porphyry Mountain, the Copper Cities pit lies in a basin on the south flank of Sleeping Beauty Mountain. There is a long, fairly continuous ridge on the east side, and a large hill on the west side from which three prominent ridges extend into the pit area.

The exploration drilling was first started by the Miami Copper Company, in August, 1943 and a total of twenty holes were drilled by December of that year when the operation was curtailed because of the wartime manpower shortage. Prospect drilling was again resumed in October, 1946, and completed in June, 1948. A total of 109 holes were drilled in and near the ore zone. The general drilling pattern was on a grid system with 250-foot spacing. Some intermediate holes were drilled around the fringes of the ore.

With the aid of the drill logs and sections constructed on each of the grid lines, mine plan maps were made up. From these maps there was found to be an estimated 33,800,000 tons of ore with average copper content of .71 % and 34,700,000 tons of waste--or a 1.03 to 1.00 waste to ore ratio.

The plan was formulated to move the Castle Dome plant to Copper Cities after the depletion of the Castle Dome ore body. The depletion date is expected to be near the end of 1953.

And the assumption was made that a year would be allowed for time to dismantle, transport, and re-erect the plant. From a study of the mine maps it was apparent that a minimum of 20,000,000 tons of waste would have to be stripped in order to be able to maintain mill tonnage by the time the mill was ready to operate.

It was also decided to do as much preliminary work in the shop site, crushers, and mill areas as could be done prior to the actual moving operation.

Thus to meet the tentative schedule, actual field operations were started in November, 1950. The first stop was to provide immediate access to the Copper Cities area. This consisted of widening existing roads that were needed for the preliminary stripping equipment. Also started immediately were temporary facilities for a powder magazine, a shop site, and mine water. While this was being done, a permanent main road from Highway 60-70 was started so that the heavier shovels, etc., could be hauled to the area as needed.

Since the pit area was lying in a basin, a large drainage, or coronation ditch was designed to drain water 2 ways from the high point at which the pit limit daylighted on the back slope. And by making a road on the outside of the ditch along its entire length, there would be a permanent access around the back side of the pit for a power line and the main mine water supply. Eventually the road, the power line, and water line will all make a loop around the mine. From this coronation ditch road, the upper mine benches were started. By January 1, 1951, the first 1½-yard diesel shovel was at work on the coronation ditch, digging from the east point of discharge. And a 2½-yard diesel shovel was busy on an access road to reach the drainage ditch from the other end.

By May, 1951, work on the coronation ditch was nearly finished and it was possible to move a 2½-yard diesel shovel onto the top mine bench at the 4230 elevation to start stripping.

From the 4230 bench this shovel moved to the next lower bench at elevation 4185, where it was still in operation June 9, 1951, when the first $4\frac{1}{2}$-yard Marion electrical shovel was put into operation.

Prior to this time all operations were carried out on a one-shift basis. However, as the shovel headings were enlarged to give larger operating areas and as more standby muck could be provided, the operation was changed first to 2 shifts and eventually to 3 shifts per day, 6 days per week by the latter part of July, 1951. This operating schedule was maintained to the date of this writing.

Mining practice at Copper Cities is patterned upon that at Castle Dome, with only minor changes to better suit local conditions. All benches are 45 feet high. All drilling and blasting is done on day shift. Primary blasting consists of shooting 10 to 15, 9" churn drill holes per shot in single rows with hole spacing 25 to 30 feet and distance between rows varying from 42 to 50 feet. Average depth of holes below grade is six feet. Secondary blasting, which has decreased considerably from the start of operations, when all ground was broken by the use of either jackhammer or wagon-drill holes, now consists of mud capping a few boulders and making preliminary toe cuts along the contour of new benches where the distance from churn-drill holes to daylight at grade exceeds 30 yards.

With the exception of the 3 top benches, which consisted in general of slicing off the tops of ridges and filling in the intervening canyons for access, all mine benches have been opened from both ends in order to balance the length of haul to the dumps.

To date all waste haul roads have been relatively free of adverse grades. Over 80% of the waste tonnage will have haul roads level or at the most plus 1.5% grade. There are approximately 6,400,000 tons of waste that will have to be hauled up grades with a maximum of 5 %.

The ore will be delivered to the primary crusher by dumping into an ore pocket such as the one at Castle Dome. There will be only one ore dump level, which will be at the 3800 elevation. This will mean that about 750,000 tons of ore from the top ore level at 4005 elevation will have to be hauled down 1800 feet of 5% grade, then down 3285 feet of 3.5% grade to the dump point. Each succeeding level will have a more favorable haul until the 3780 level is reached. From this point down to the bottom level at 3600 elevation, there will be an adverse grade to the dump point, with maximum grade planned to be plus 5%. Thus 18,200,000 tons of ore will be hauled down grade with maximum of minus 5%, and 15,600,000 tons will be hauled up grade with maximum of plus 5%.

The preliminary stripping schedule of 20,000,000 tons at Copper Cities was based on the amount of equipment which had become idle at Castle Dome. This surplus existed because in the earlier stages at Castle Dome the stripping ratio was 2.5 to 1.0, while by the time Copper Cities operations was started the ratio had dropped to 0.24 to 1.0. Consequently, one electric shovel and its accessory fleet of trucks was idle.

Loading equipment consists of one 4½-yard Marion electric shovel with a 5-yard dipper, operated 3 shifts per day, 6 days per week; two 2½-yard diesel shovels operated on day shift only 6 days per week, and one 1½-yard diesel shovel used as standby for the 2½-yard shovels, or as a dragline or auxiliary crane. One of the diesel shovels is employed in the mine nearly 100 % of the time, starting benches, making final bench cleanup, digging ramps, standby in case the electric shovel breaks down, etc.

The other diesel shovel is utilized on jobs outside the pit such as excavation and backfill in concentrator, crusher, and shop areas. There is one Caterpillar D-8 bulldozer assigned to work with each operating shovel crew.

Other Mine Equipment consists of:

Haulage Equipment

8 Knuckey Trucks, 30 ton capacity

4 Model F D Euclid Trucks, 15 ton capacity

Road Equipment

2-D-8 Caterpillar dozers

1-15-yard carryall

1-Le Tourneau rooter

1-Caterpillar-motor grader

Churn Drills

3-29T Bucyrus-Erie churn drills

3-22T Bucyrus-Erie churn drills

Pneumatic Drill Equipment

11-Gardner Denver S-55-D Jackhammers

2-Ingersoll Rand X-71 Wagon Drills

3-Gardner Denver, 365 cu. feet portable Air Compressors

Auxiliary Equipment

1-3/4-yard Hough Payloader

1 Portable conveyor and screen to make stemming

1-3-ton Reo Truck with 1,000 gal. water tank

1-3-ton Reo Truck with dump bed for stemming

1-3-ton Reo Truck for powder haul

3-Kohler light plants, 1500 watts

Customary service trucks and pick-ups

The following are some efficiency data from the start of operations to April 1, 1953:

Total tons mined in pit:	10,783,138
Average tons per electric shovel shift	6,291

Knuckey Trucks:

Average tons per shift:	935.6
Average trips per shift:	30.4
Average tons per trip:	30.94
Average round trip haul (miles):	1.03
Average ton miles per shift:	473.4

Drilling and Blasting:

Average feet per churn drill shift 29T:	104.3
Average feet per churn drill shift 22T:	66.8
Average feet per bit change - 29T:	101.3
Tons broken per lb. of powder:	4.54
Tons broken per foot of churn drill hole:	74.32

Also there has been 551,189 yards of excavation in the shop, mill, crushers, and tailing thickener areas. The operating efficiencies in these areas are not included with the above.

In conclusion, it should be mentioned that the stripping operation is well on schedule and the plant should be processing ore by the end of 1954.

Presented at Annual Meeting of
Arizona Section of A.I.M.E.
Bisbee, Arizona - May 29, 1953

J. H. Gray