THE
UNITED VERDE COPPER COMPANY

• • •

THE UNITED VERDE COPPER COMPANY

● ● ●

A series of articles describing the organization, operations, and activities of this company in the Jerome District of Arizona.

Reprinted from
THE MINING CONGRESS JOURNAL
April, 1930

SENATOR CLARK

The Pioneer

• • • • • •

A Biographical Sketch of WILLIAM A. CLARK

WILLIAM ANDREWS CLARK, ex-United States Senator, was born near Connellsville, Pa., January 8, 1839, son of John and Mary (Andrews) Clark. He was a descendant of a Huguenot family which had emigrated from France to Scotland to escape religious persecutions, later moving to the north of Ireland, and then settling in Pennsylvania soon after the close of the Revolutionary War.

His father was a farmer and his boyhood days were spent on the homestead, where he enjoyed the advantages of three months' winter school and nine months' of farm work. In 1856 his father moved to Van Buren County, Iowa, and there developed a large and productive farm, in which the son had his share of work to do. He taught school for several winter terms, then entered an academy at Birmingham, Iowa, later attending the Iowa Wesleyan University at Mt. Pleasant, where he studied in the academic and law departments. In 1859-1860 he taught school in Missouri, and from then on his life was entirely given over to the building up of the new empire in the west.

In 1862 he crossed the great plains, driving a team to South Park, Colo., and that winter worked in the quartz mines in Central City, gaining knowledge and experience for his future life in the mining industry. In 1863 news of the gold discoveries at Bannock, Mont., reached Colorado and he was among the first to start for this new El Dorado. After 65 days' travel with an ox-team, he arrived in Bannock, just in time to join a stampede to Horse Prairie Creek, where he secured a claim. After several years' experience in placer mining, he engaged in wholesale mercantile enterprises which he continued for about five years. During this time his travels took him to the mining camps of Virginia City, Blackfoot City, Helena and Elk Creek, to

W. A. Clark, Jr., Vice President

Chas. W. Clark, Chairman of the Board

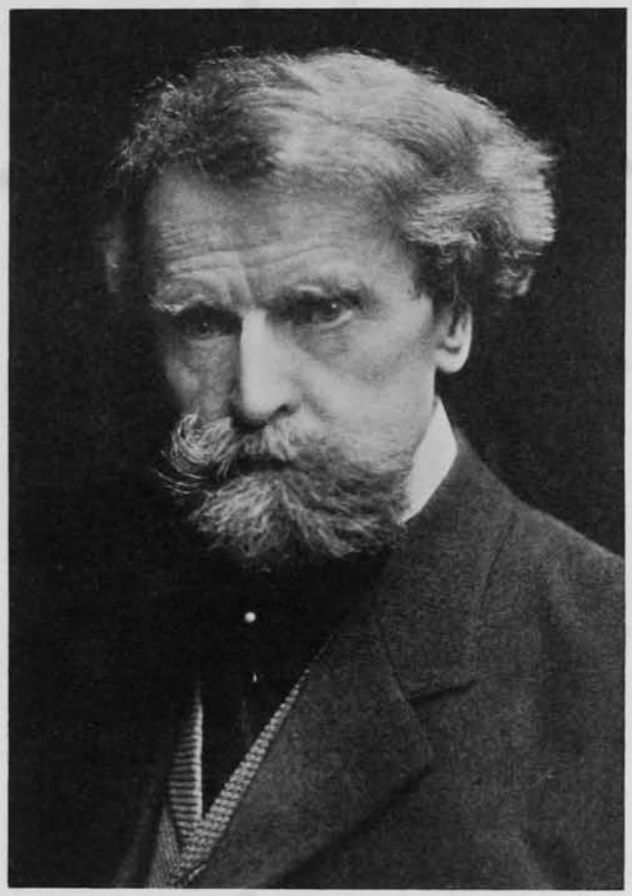
Copper Company

San Francisco and central California to Portland, Oreg. In addition he also successfully carried on the duties of a mail contractor on the star route between Missoula, Mont., and Walla Walla, Wash.

In his various undertakings he had been eminently successful and in 1868 he formed a partnership which engaged in the wholesale mercantile and banking business, locating at Helena, Mont., later moving to Deer Lodge, and, after buying out his partners, this resulted in the locating at Butte of the banking firm of W. A. Clark & Brother.

However, it was principally in the mining field that he devoted most of his time and in which he made his greatest success. In 1872 he was attracted to the quartz prospects of Butte and purchased, in whole or in part, the Colusa, Original, Mountain Chief, Gambetta and other mines, nearly all of which developed into rich producers. The "Old Dexter," the second stamp mill of Butte, was finished in 1876 through his financial aid, and the first smelter of consequence in Butte, the Colorado smelter, was erected by a company organized by him. In 1880 he organized the Moulton Company, which erected the Moulton Mill and developed the mine of the same name. In 1914 he constructed a large concentrating plant for the treatment of ores from the Elm Orlu mine, near Butte, which mine developed into one of the largest zinc and copper mines of that time. In the meantime he acquired and developed mining properties in other western states, including coal properties in Colorado, in the course of which he became interested in the United Verde mine in Arizona, and it was this latter mine which his genius developed into one of the best equipped mines in the world.

When acting as Commissioner for Montana to the exposition in New Orleans in 1885, he saw numerous specimens exhibited from a mine in Arizona called the United Verde mine. These specimens were rich in copper, gold and silver. Later on, when re-organizing the Orford Copper Company, to which company he had been selling the matte from his Butte smelters, he found on the books numerous shipments of matte from the United Verde mine, which at that time was operating two 50-ton furnaces. These mattes were extremely rich in gold and silver contents. With his usual keenness, he started to investigate and about the first of the year 1888, secured, through the assistance of some New York friends, particularly Mr. James A. Macdonald, who later became vice president of the company under Mr. Clark's reorganization and who died about a year ago, a second option on the United Verde mine which, in spite of the rich surface ores, had up to that time been unsuccessfully operated. His first visit to the United Verde was with his mine superintendent, Joseph L. Giroux, in March, 1888. Availing himself of the fact that the first option had been given up, he immediately made the first payment required under the option and proceeded to buy in as much of the outstanding stock as was possible. This stock was scattered all over the globe, and at the time of his death he and his family owned about 299,000 shares out of 300,000.

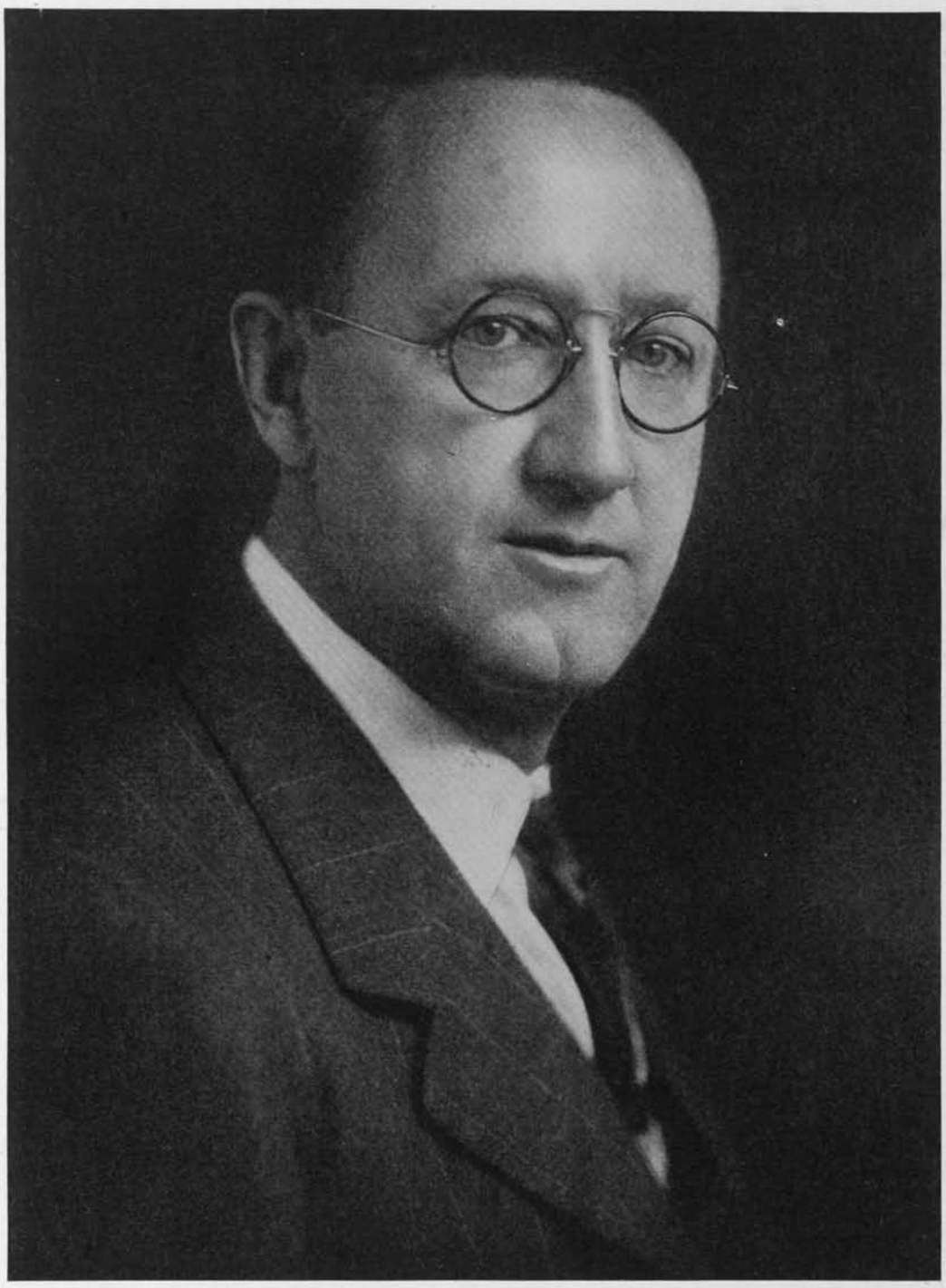


The Late Senator William A. Clark

And so, having decided that the mine held great possibilities, with his usual thoroughness and genius for organizing and putting his desires into active operation, the property was developed into one of the large copper producers in the United States.

In the foregoing biography there has been purposely omitted reference to Senator Clark's activities in the fields of public utilities, water power undertakings, construction and operation of railroads, ranching, manufacturing industries, etc., which in themselves are entitled to a separate recitation in order to depict the many and varied ramifications of a brilliant intellect, keen in its instantaneous grasp of values, combined with a force and vitality to carry his convictions to a successful termination.

Senator Clark died in March, 1925, and is survived by his widow, Mrs. Anna E. Clark, and two sons and three daughters —Charles W. Clark, William A. Clark, Jr., Mrs. Marius de Brabant, Mrs. Lewis R. Morris, and Mrs. William M. Gower.



Robert E. Tally

President of the United Verde Copper Company

RIGHT—MANAGING AND OPERATING STAFF OF UNITED VERDE SMELTER, VERDE TUNNEL & SMELTER RAILROAD COMPANY, AND UPPER VERDE PUBLIC UTILITIES COMPANY.

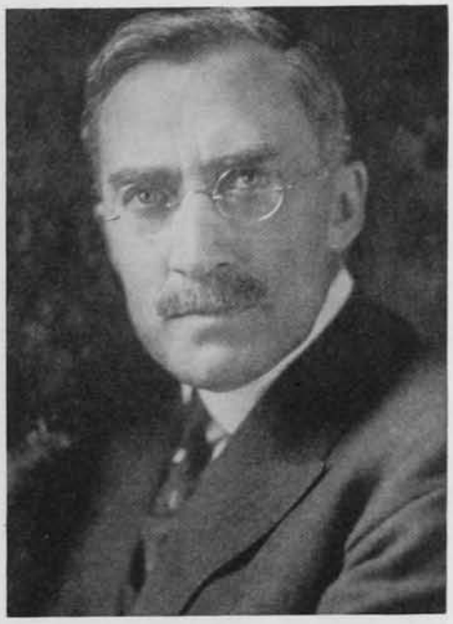
FRONT ROW, LEFT TO RIGHT—A. I. Greenwood, *Chief Electrician;* J. E. McLean, *General Superintendent of Railroad;* C. R. Kuzell, *Smelter Superintendent;* W. V. DeCamp, *General Manager;* Robt. E. Tally, *President;* Thos. Taylor, *General Smelter Superintendent;* Dave Hopkins, *Purchasing Agent;* F. O. Twitty, *Trainmaster of Railroad;* F. H. Jones, *Chief Bonus Engineer.*

CENTER ROW—F. X. Mooney, *General Foreman Reverberatory Department;* George Mieyr, *Master Mechanic;* Frank Avis, *Chief Power Engineer;* J. J. Williams, *General Smelter Foreman;* A. L. Reese, *Chief Chemist;* H. P. Hughes, *Auditor of Railroad;* J. R. Marston, *Metallurgist;* P. C. Keefe, *Crushing Plant Superintendent;* R. K. Duffey, *Chief Clerk;* P. C. Steinel, *Traffic Manager.*

BACK ROW—L. M. Barker, *Concentrator Superintendent;* H. V. Young, *Secretary to the President;* J. C. Harding, *Safety and Employment Engineer;* F. H. Parsons, *Assistant Smelter Superintendent;* J. E. Lanning, *Chief Mechanical Engineer;* D. L. Bouse, *General Storekeeper;* A. N. Jones, *Superintendent, Upper Verde Public Utilities Co.;* Dr. L. E. Walsh, *Physician and Surgeon;* O. C. Ralston, *Director of Research.*

John H. Hall, Jr., Secretary

H. H. St. Clair, Treasurer



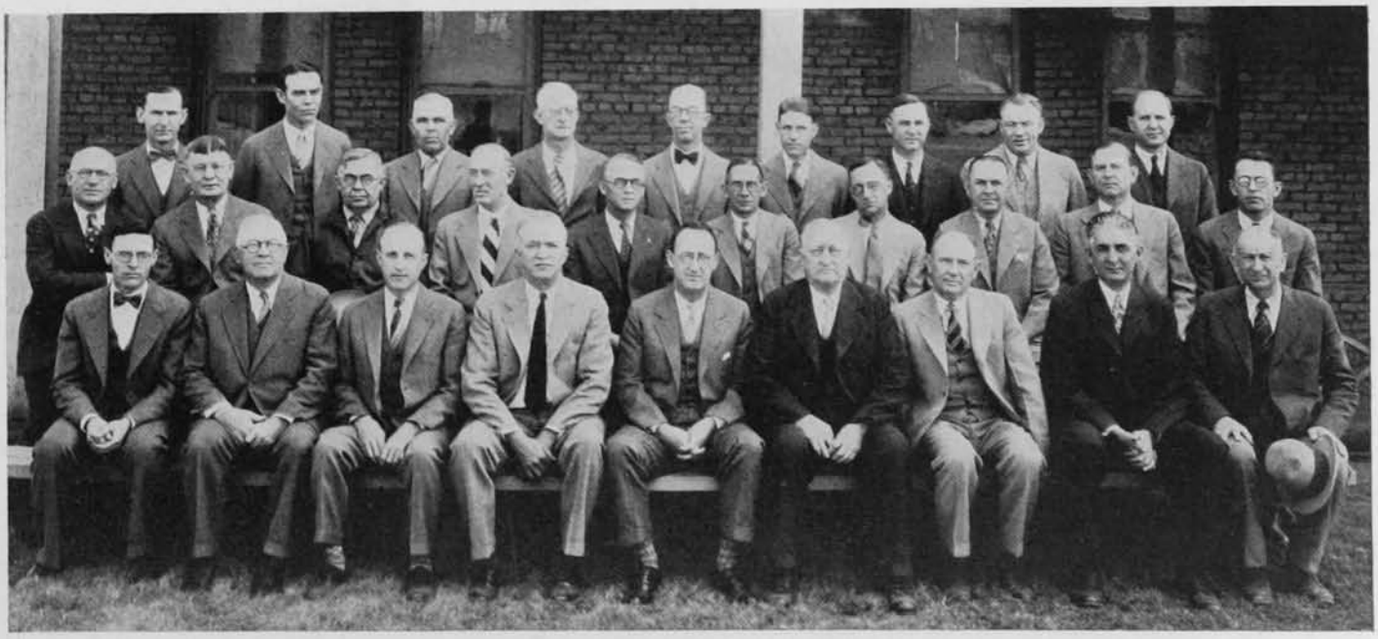
Wm. B. Gower, Comptroller

HISTORICAL SKETCH
of the
UNITED VERDE COPPER COMPANY

By Herbert V. Young
SECRETARY TO THE PRESIDENT

THOUGH the greater development of the United Verde Copper Company is inextricably bound up in the empire-building activities of Senator William A. Clark, the actual history of the property and its growth dates its beginning from 1876, the year in which the first claims were located.

The United Verde mine is situated on the eastern slope of the Black Hills, in Yavapai County, Ariz., overlooking the Verde Valley—a great gash in the landscape, at the bottom of which winds the Verde River. This valley was inhabited for prehistoric centuries by the aborigines; numerous remains of their dwellings are found in the cliffs and gorges and on the mountain tops. It was first settled by white men in 1865, when a group of pioneers emigrated from



United Verde Copper Company

W. V. De Camp, General Manager

Aerial View of Jerome, Arizona

Prescott, 40 miles distant, located several tracts of fertile bottom land near the river, and commenced farming. During the next 10 years there was a steady immigration of settlers, and despite frequent depredations of the Indians the community began to prosper.

One of the pioneers was M. A. Ruffner, who, though he owned land, found his main interest not in farming but in prospecting the surrounding hills. In 1876 he discovered an outcrop of ore which was recognized as copper bearing, and he located two claims, which he named the Eureka and Wade Hampton. He drove a tunnel on this outcrop, and this development confirmed his personal belief that the discovery was a rich one. However, it held little interest for the other settlers of the district, as none of them could see a future for copper on account of the transportation problem. The nearest railroad terminal was hundreds of miles distant, and the nearest originating point for water transportation was a port on the Colorado River.

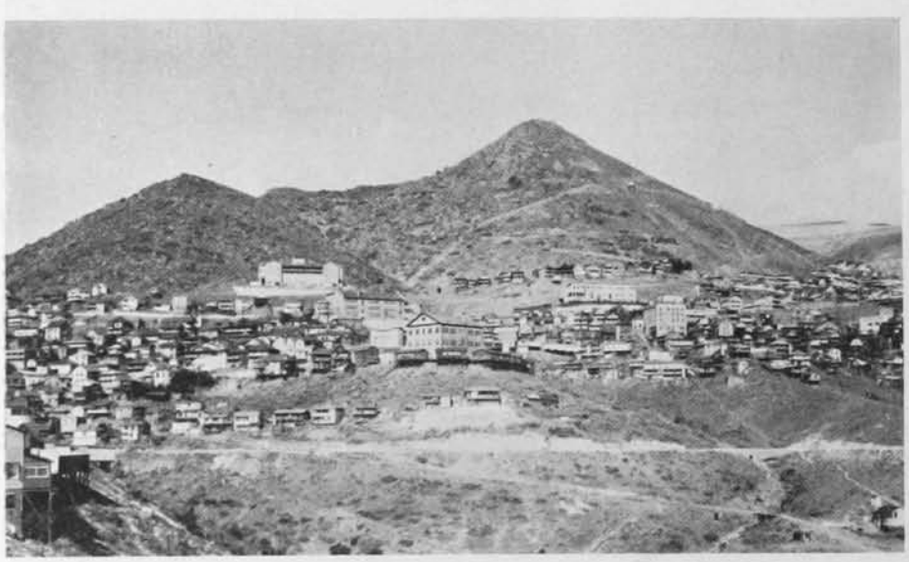
Herbert V. Young

Ruffner was unable financially to prosecute development of his claims in the manner he wished; he therefore interested two brothers, George and Angus McKinnon, to whom, in exchange for a grub-stake and their personal services, he transferred two-thirds interest in the property. The three men proceeded to sink a shaft to a depth of 45 feet, which resulted in favorable showings of good ore. Other prospectors were attracted by the developments, and soon much of the ground in the vicinity of the original locations was filed upon.

The McKinnons then began to talk of selling; they were afraid to sink their shaft further for fear they would pass through the ore into barren ground. While Ruffner had greater faith in the depth of the ore, he was willing that the majority should rule.

The capitol of Arizona was then at Prescott, and Hon. F. A. Tritle, then governor of the territory, became interested in the district. He enlisted the financial aid of James A. Macdonald and Eugene Jerome, of New York (the town of Jerome being named for the latter) and in 1882 the United Verde Copper Company was organized, with Macdonald as president and Jerome as secretary. The property of the new corporation included the Eureka and Wade Hampton claims and a number of the surrounding locations which had been acquired.

Plans were immediately laid for the construction of a small reduction plant.



Jerome, Arizona

United Verde Copper Company

Aerial View of the United Verde Workings, Jerome

The Santa Fe Railroad had extended its line through Arizona in 1882, and a wagon road was built from Ash Fork, a small station on the Santa Fe, to Jerome, covering a distance of about 60 miles, and over which supplies were freighted by mule- and ox-team. Two small water-jacket blast furnaces were set up, and production of matte and bullion was commenced in 1883.

Even under the great expense involved and the crude methods employed a profit was made during the time the rich copper, gold and silver bearing oxides near the surface were being extracted and treated. The hopes of the promoters ran high, and financial success seemed assured. Then, as the costs of mining increased and the ores became leaner in values, the profits ceased and losses began to mount. The plant was closed early in 1884.

Governor Tritle retained his faith in the property, and in 1887 secured a lease and resumed operations. This proved to be a losing venture.

The following year came Senator Clark, who picked the property for a winner after other mine operators and engineers had rendered adverse decisions. He secured control. He envisioned vast operations, and through the years his vision has come to fruition in a splendid reality.

NOTE: Detailed accounts of the growth of the United Verde Copper Company under the Clark regime will be found in other articles in this issue of THE MINING CONGRESS JOURNAL.

Thomas Taylor, General Smelter Supt.

THE BOSSING ORGANIZATION AT THE UNITED VERDE MINE

BACK ROW, LEFT TO RIGHT—Dr. A. C. Carlson, *Chief Surgeon;* M. J. O'Boyle, *Division Foreman;* M. A. Heckey, *Division Foreman;* E. M. J. Alenius, *Chief Engineer, Shovel Dept.;* M. G. Hansen, *Chief Geologist;* C. J. Thomas, *Mine Foreman;* W. H. Riddle *Master Mechanic;* E. W. Fredell, *Chief Electrician.*

CENTER ROW—Denny O'Neill, *Division Foreman;* Frost L. Benham, *Superintendent, Utilities Company;* Walter Mutz, *Division Foreman;* Paul Allsman, *Chief Chemist;* C. S. P. Gardner, *Chief Timekeeper;* H. V Kruse, *Chief Mechanical Engineer;* Thos. Dennison, *Fire Chief.*

FRONT ROW—O. A. Glaeser, *Safety and Ventilation Engineer;* J. C. Perkins, *Shovel Superintendent;* F. J. Cowley, *Planning Engineer;* J. R. Allen, *Shovel Foreman;* W. J. Flood, *Underground Construction Foreman;* C. E. Mills, *Assistant Mine Superintendent and Chief Engineer;* W. V. DeCamp, *General Manager;* T. W. Quayle, *Mine Superintendent.*

United Verde Copper Company

GEOLOGY and ORE DEPOSITS of the

I. Introduction

THE geology of the Jerome district has been previously described by Reber,[1] Lindgren,[2] Provot,[3] Finlay,[4] and others in considerable detail and thoroughness, yet leaving a few questions of major importance and details of the occurence of ore at the United Verde mine unanswered. It is the purpose of this paper to express the writer's view of these questions as well as to set forth the details of ore occurrence at this mine.

The United Verde mine lies about half a mile northwest of the town of Jerome, Yavapai County, north-central part of Arizona. Lying on the northeastern slope of the Black Hills Mountain Range at an elevation of 5,400 ft., the property covers about 2,137 acres (127 claims). The Jerome district covers an area 7 miles long and 3 miles wide and parallels the Verde Valley and River, the latter at an elevation of approximately 3,300 ft.

The character of the relief is mountains, due to considerable faulting and the general regional structure. The trend of the mountain range is northwest and merges with the plateau country on the northwest and southwest. The tops of the mountains (Woodchute and Mingus) within the range are more or less flat-topped, due to the presence of sedimentary rocks or lava flows. The main drainage system flows in a southeasterly direction, fed by tributaries normal to the scarp.

II. General Geology

The rocks of the Jerome district include igneous, sedimentary, and metamorphic, and range from pre-Cambrian to Tertiary in age. The ore deposit is of pre-Cambrian age and is of the massive sulphide-schist replacement type. Structurally the district is divided into two sections by the main or Verde fault, which bears in a northwesterly and southeasterly direction. Erosion of the uplifted scarp of the block west of the fault uncovered pre-Cambrian formations, while on the east side of the fault the Tertiary lavas are still exposed.

The oldest rocks of the district consist of a greenstone complex comparable to and synonymous with the Yavapai schists, intruded by a rhyolite quartz porphyry and augite diorite. Overlying this, but separated by a great unconformity, is the Tapeats sandstone (Cambrian), the Redwall limestone (Devonian and Mississippian), and the Supai sandstone (Permian). Basaltic lavas (Tertiary) cover the other formations and is followed by the deposition of impure limestones (Verde Lakebed formations) in the then existing Verde Valley. The areal distribution of these rocks are shown on *Figure 1.*

The younger rocks which have no significance as to ore occurrence will be but briefly described.

According to Reber, "The sea reached the Jerome district in the middle Cambrian time and deposited a thin blanket of beach sand and fine pebbles, which formed the Tapeats sandstone. Its thickness varies from nothing to 100 ft. and tends to fill the minor irregularities of the pre-Cambrian surface. The sandstone is of a maroon color, and highly ferruginous.

"Overlying the basal sandstone are from 300 to 500 ft. of limestone of Devonian age, from 300 to 500 ft. of limestone of Mississippian age, and from nothing to 500 ft. of red sandstone and shale of Permian age. The formation of

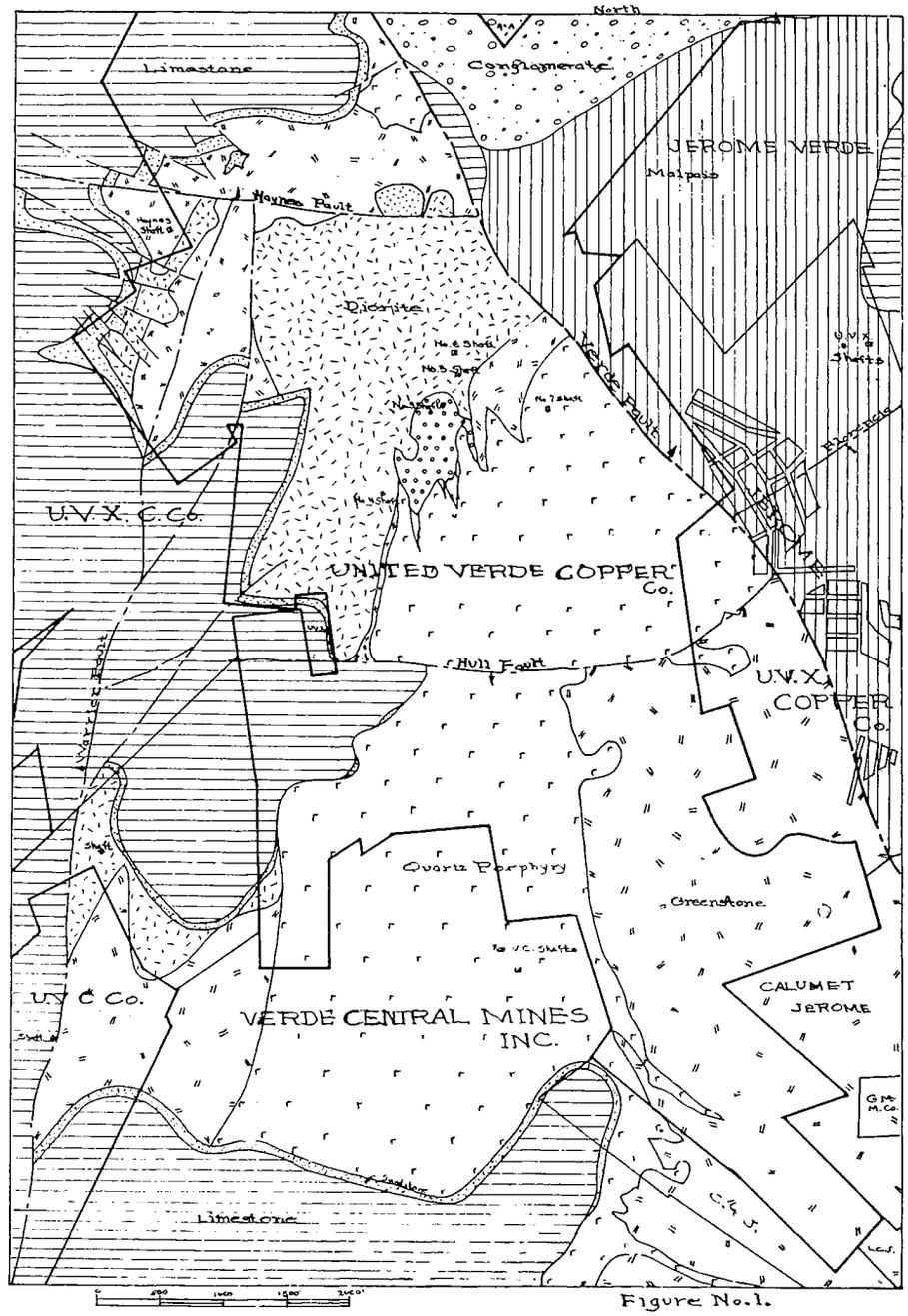


Figure No. 1.

[1] Reber, L. E., Jr., "Geology and Ore Deposits of Jerome District"; A. I. M. E., Trans., Vol. 66, pp. 3-26, 1922.

[2] Lindgren, W., "Ore Deposits of the Jerome and Bradshaw Mountains Quadrangles, Arizona"; U. S. G. S., Bull. 782, pp. 54-97, 1926.

[3] Provot, F. A., "Geological Reconnaisance of the Jerome District, Jerome, Ariz."; May 13, 1916.

[4] Finlay, J. R., "The Jerome District of Arizona"; E. M. J., Sept. 28 and Oct. 5, 1918, pp. 106, 557, 605.

UNITED VERDE MINE

By Mayer G. Hansen
CHIEF GEOLOGIST

the deposits of each of these periods, Cambrian, Devonian, Mississippi, and Permian, was preceded and followed by periods of uplift and erosion."

Damming of the Verde Valley by the Tertiary lava flows produced a lake of considerable size, possibly 35 miles long and 6 or 8 miles wide. Impure white calcareous sediments were deposited to a depth of over 1,200 ft., and named the Verde formation.

Igneous rock exposures are numerous and plainly discernible for study. Variations in texture, size of grain and general metamorphism, offsets this value and makes a correct interpretation of origin and classification a most difficult one.

A rhyolite quartz porphyry is the first intrusive rock in the Jerome vicinity and owes its origin to the Bradshaw granite batholith from which it is a differentiation product. It is typically exposed on Cleopatra Hill, from which it receives the name "Cleopatra Quartz Porphyry," though locally referred to as "quartz porphyry" or merely "porphyry."

The granite of the southern part of the Jerome quadrangle is definitely known to be a direct continuation of that of the Bradshaw Mountain region. Reber[1] notes that the quartz porphyry near the south end of the district is of a more granitic character than that near Jerome. A progressive change from the fine-grained granite of the Bradshaw Mountains to the typical rhyolite quartz porphyry may be observed.

The petrographic character varies considerably and embraces a diversified group of rocks. It is a white to gray rock whose essential minerals are quartz, orthoclase, and albite, and lesser amounts of plagioclase. The accessory minerals include biotite, augite, and pyrite. The analyses of three distinct types indicate its variable nature.

Type Quartz Porphry	% SiO_2	% Al_2O_3	% Fe	% CaO	% MgO
Normal porphyry.....	67.8	13.2	2.9	3.50	1.30
Silicious porphyry....	73.8	19.1	2.9	2.15	0.75
Metamorphosed por'ry	57.5	14.8	1.1	1.10	1.70

The texture is similarly variable from felsitic to granitic. Quartz is the characteristic phenocryst though occasionally the white feldspar is equally developed. The quartz phenocryst is normally about an eighth of an inch in diameter and round, though rudely hexagonal crystals are found. They constitute, roughly, about 10 percent of the rock.

Dynamic metamorphism has altered the porphyry mass in numerous places sufficiently to become a characteristic of the rock. The schistose structure has developed chlorite predominantly and sericite to a lesser extent. Gradations from fresh porphyry to chloritic schists indicate that between 40 and 60 percent of all the black schist was originally quartz porphyry. A large part of this percentage is now completely replaced by sulphides.

The United Verde Diorite is a massive, medium to coarse grained augite diorite. Jaggar and Palache[5] recognized two distinct ages of diorite in the Bradshaw quadrangle. The first varies from a quartz diorite to a gabbro and is believed to be intimately related to the Bradshaw granite; the other is a gray quartz diorite distinctly younger than the granite. In the United Verde mine at only one location was quartz diorite observed which showed a gradual transition to the normal fresh United Verde diorite through a zone of over 300 ft.

A chemical analysis of typical U. V. Diorite shows:

% SiO_2	% Al_2O_3	% Fe	% CaO	% MgO
48.8	19.8	6.8	4.05	0.7

The diorite is generally accepted as an intrusive rock and its longer axis has a general northerly trend. Within its mass it becomes quite altered due to shearing, and produces chlorite and kaolinized material. It is younger than the porphyry and is undoubtedly closely related to the sulphide mineralization. Pyrite crystals are found sparsely distributed in it. Its concave margin dips to the northwest and served as a structural feature in localizing the ore solutions. The margin is usually finer grained than the interior, often a characteristic of intrusive masses.

Both the diorite and porphyry are of pre-Cambrian age, though probably widely separated themselves in time of formation. Further igneous activity failed to make its appearance until late Tertiary time, when basaltic lavas emanating essentially from the San Francisco Mountain volcanic fields covered the Carboniferous sediments, in places to a depth of 700 ft. Undoubtedly the numerous basalt dikes found in the district assisted materially in covering the area. Subsequent faulting and erosion has removed much of this deposit so that only the higher pinnacles remain capped.

Included in the pre-Cambrian rocks are a number of metamorphic rocks of different types justly termed the pre-Cambrian complex. As a whole the group corresponds to the Yavapai schists of the Bradshaw Mountains. Several geologists have studied the district and have attempted to differentiate the various members of the group. As yet perfect accord has not been reached.

The writer believes that the greenstone complex may be divided into four major units; tuffs, fragmental greenstone (Deception Porphyry), flows, and bedded sediments.

A distinctly tuffaceous member is exposed on one of the lower levels of the United Verde mine which is pale green to gray in color. It is medium to coarse grained and contains rather large white feldspars. Fragments of quartz are found most of which are somewhat rounded due to pressure. The abundance of quartzes varies over a short distance in places becoming numerous and well represented, in others sparse and scattered. The groundmass is very fine grained and quite silicious, somewhat altered and schistose. Most of the alteration has been produced by pressure and temperature resulting in considerable kaolinization and sericitization. The bed strikes in a northerly direction with a steep dip to the west. Quartz porphyry has intruded it precluding the possibility of measuring its width. An analysis of this material contains:

% SiO_2	% Al_2O_3	% Fe	% CaO	% MgO
56.7	18.9	5.5	1.85	0.85

Andesitic tuffs are also recognized which take on the appearance of fine grained diorite with the exception that mineral grains are sharp and angular and apparantly volcanic in origin. It is dark green in color. Alteration by pressure produces chlorite. Its composition is:

% SiO_2	% Al_2O_3	% Fe	% CaO
42.4	16.1	8.6	8.3

Fragmental greenstone, named Deception porphyry by Finlay because of its exposure in Deception gulch, is well represented in many places and in large areas throughout the district. It is rhyolitic in composition and volcanic in origin. Exception has been taken to this due to an occasional brecciated condition indicating a flow rock. Others have called it Deception Quartz Porphyry and believe it to be an intrusive. Evidence of its intrusive character is decidedly lacking. Its contact with other green stones is smooth and straight except in those instances when the whole greenstone area was folded. Where it is in contact with the quartz porphyry the latter has cut into it with dikes and stringers, sometimes inclosing large lenses. It is characterized by triangular or irregular shaped quartz phenocrysts having sharp pointed edges or corners. The size of the quartzes is small to medium, averaging usually about a sixteenth of an inch in its longest dimension. It is grayish green to dark green in color and alters to a black chloritic schist on dynamic metamorphism. On the surface it weathers to a yellowish buff colored, platy schist. When subjected to pressure or heat it apparently readily alters to chlorite and consequently becomes a favorable rock for replacement by sulphide solutions. It frequently happens that sulphides are found on the contact of this rock with quartz porphyry.

Several beds of rhyolite flows are distinguishable varying from 8 ft. to over 200 ft. in thickness, the whole series having a thickness of over 400 ft. It is light

[5] Jaggar, T. A. Jr., and Charles Palache: U. S. G. S. Sur. Folio 126, "Bradshaw Mountains Folio," 1905.

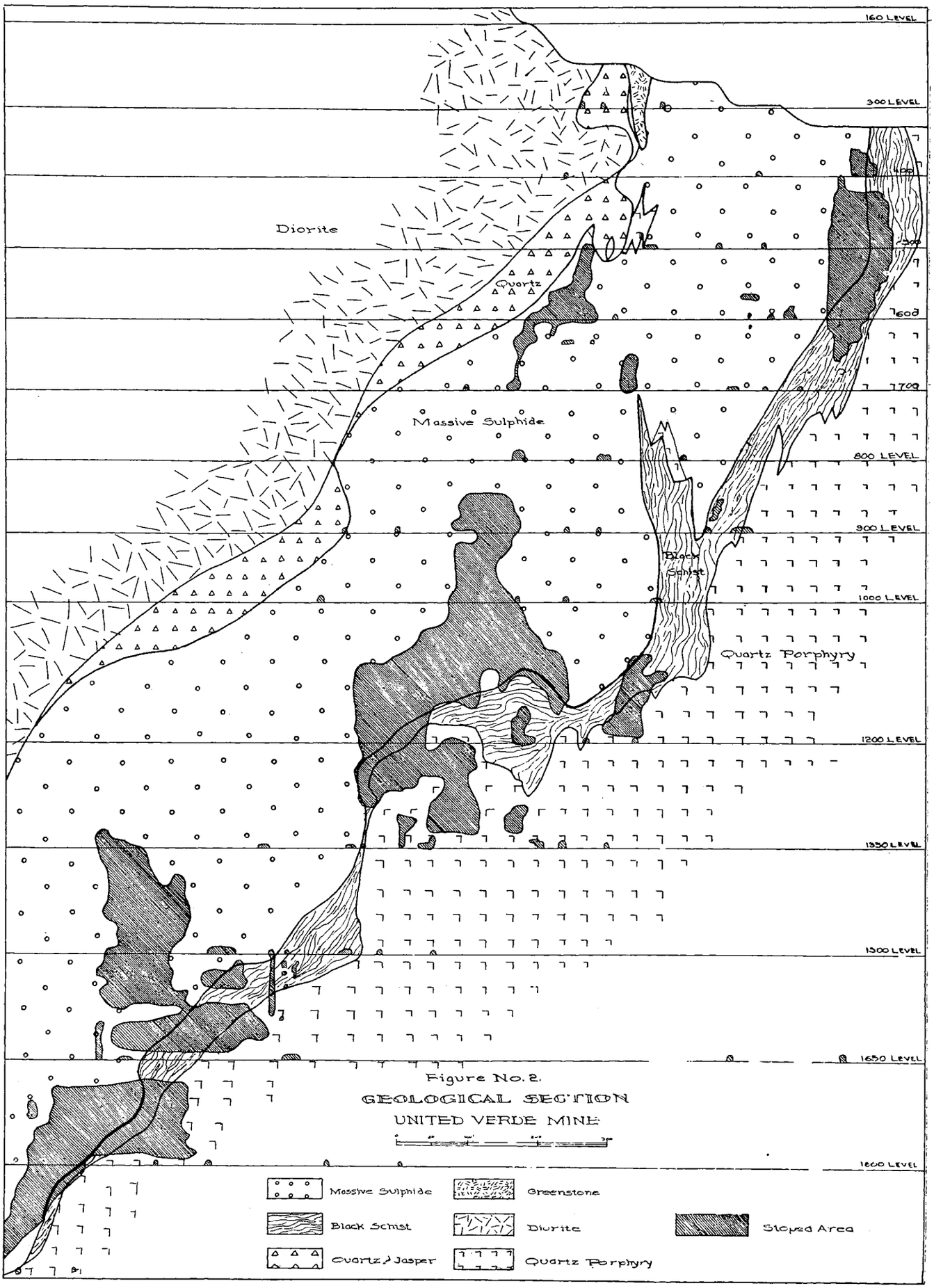


Figure No. 2.
GEOLOGICAL SECTION
UNITED VERDE MINE

United Verde Copper Company

colored to gray or pale green, having an abundance of white feldspars which are generally altered and kaolinized. The basal portion of each successive flow is marked by small round jasper pebbles ranging from microscopic to an eighth of an inch in size. The groundmass is very silicious. It is best exposed on the northeast side of the 2,400 level. Zones of brecciation further suggests flow structure. Where pressures have rendered the rock schistose the alteration usually produces sericite. The rocks as a whole are somewhat schistose, the direction of shearing being at an angle of about 30 degrees to the strike of the beds. In this area the beds bear in a northwesterly direction and dip to the northeast apparently very steeply. No sign of mineralization has been found in this series though more or less favorably located in proximity to sulphides. As a whole these flows are very uniform in mineralogical content and texture and run:

% SiO_2	% Al_2O_3	% Fe	% CaO	% MgO
68.0	16.5	3.2	2.2	0.6

Overlying the rocks described are a series of well exposed sedimentary rocks which are widespread throughout the district. Several varieties are observed, chiefly of which is the silicious phase which approaches a quartzite in appearance. It is light gray to pale green in color and very fine grained. The bedding is very pronounced and unmistakeable. A chemical analysis of the material gives:

% SiO_2	% Al_2O_3	% Fe	% CaO	% MgO
71.0	12.8	4.7	0.6	0.5

On the west side of the 2,400 level it has a strike to the northeast and dip to the northwest.

The next sediment of importance is the andesitic variety which is grayish green in color and fine to medium grained in texture. The bedding planes are less distinct and are represented mainly by differences in color. There is a marked absence of silica as compared with the first mentioned sediment. Doubt may be expressed as to whether or not this is a true sediment although the uniformity and continuity of the bedding is strong in support. On extreme alteration by surface weathering and by the heat produced by being included in the zone now on fire in the mine, this material becomes white and clayey though fairly hard and solid. The feldspars are the medium to basic types.

Other varieties have been noted as a 20-ft. bed of limestone or marble occurring on the 800 level in the vicinity of No. 4 shaft and also calcareous slate occurring in the same area. A coarse grained variety of conglomerate consisting of well rounded pebbles in a tuffaceous matrix, including numerous well preserved feldspars giving the appearance of a feldspar porphyry where the pebbles do not show was noted by Reber.[1]

III. Structure

Following the successive deposition of the various members of the greenstone complex came a period of intense metamorphism which compressed the rocks into close folds and tilted them steeply to the northwest. Because the regional shearing and the pre-mineral major shear zones were in a northerly and southerly direction it is inferred that the compressional stresses were from an approximately east-west direction.

Further shearing and schistosity in the greenstones occurred with the intrusion of the quartz porphyry which, of course, would be accompanied by intense heat and pressure. The later intrusion of the diorite, however, metamorphosed much of the greenstones to schists as well as limited zones of the porphyry.

Such intense metamorphism naturally caused a thinning and thickening of the greenstone beds so as to make exact figures on thickness of the initial flows and sediments nothing but a conjecture. On the other hand the structure can, in a general way, be deciphered and the rocks classified and the whole very useful economically.

The distribution of the rocks in the Jerome vicinity is shown in *Figure 1*, which also shows the faulting.

The United Verde sulphide mass was formed after the intrusion of the diorite and controlled structurally by it, see geologic section in *Figure 2*. It formed as a more or less cylindrical pipe, hypogene in origin, limited on the north and northwest sides by an impervious diorite hanging wall.

After its formation the region was further subjected to dynamic forces resulting in major faulting along some of the pre-existing shear zones.

One of these, known as the Main or Verde fault, moved shortly after the deposition of the main sulphide mass and resulted in the formation of the second mine of the district, the United Verde Extension.

It is estimated that the vertical dislocation on this fault was about 2,500 ft. in pre-Cambrian times. Its strike is approximately N. 43 degrees W. with a dip of 58 degrees to the east. The faulting was normal, throwing the upper portion of the original sulphide pipe down to the east, with practically no horizontal displacement.

Erosion followed which removed about 1,940 ft. of sulphide before it was covered by the Cambrian sandstone.

Further deformation failed to take place until late Tertiary after the deposition of the basaltic lavas, when the Verde Fault again moved. This time the vertical dislocation amounted to 1,360 ft., also, a horizontal displacement of 900 ft. to the south.

Displacement in Tertiary time is more readily discernable because of sedimentary horizon markers. The extensive faulting that occurred at this time is only partially shown on *Figure 1*, because of the limited area covered by the sketch. As shown, an almost parallel fault occurs several hundred feet west of the United Verde Mine which strikes in a northerly and southerly direction and has a dip to the west. The vertical dislocation is comparatively slight, possibly not over 200 ft. Not shown on the sketch, but a few hundred feet east of the United Verde Extension shafts another north-south fault occurs, called the Bessie Fault, which has a dip to the east. Its vertical dislocation is approximately 600 ft.

Cross faulting also occurred. One called the Haynes Fault, which lies about 1,500 ft. north of the United Verde mine, has a steep north dip of about 69 degrees. Its strike is approximately east and west, and has a vertical dislocation of about 500 ft. South of the mine at about the same distance from it another east-west fault, known as the Hull Fault occurs, with a 12 degree dip to the south. The amount of movement on this fault will not be estimated since there is a possibility of pre-Cambrian movement occurring on this plane with subsequent Tertiary movement.

Slightly north of the Hull Fault, but on the east side of the Verde Fault, lies the Florencia which also has a general easterly strike and with a 74 degree dip to the south. The vertical dislocation on this fault is believed to be about 175 ft.

Other faults of apparently minor consequence also occur, as shown on *Figure 1*. It is quite possible that pre-Cambrian faulting occurred in planes other than those mentioned. The lack of extensive workings in the fault areas and of definite horizon markers, preclude the possibility of definite decision.

IV. Orebodies

The amount of oxidation of the sulphide mass is very slight, not over 100 ft. in depth and is represented by the yellowish brown limonite chiefly and to lesser extents by the red variety and rarely by some platy hematite. Much quartz is present, some as secondary cementing material though mostly as residual quartz and jasper.

Similarly very little secondary enrichment exists. In sufficient quantities to be of commercial importance it will not be found in depths greater than 200 ft. below the gossan. Chalcocite and covellite are the secondary sulphide minerals. Over a widespread area copper carbonates are important, forming commercial orebodies of rather large size in the Malpais, conglomerate and limestones. Some cuprite and tenorite is also found but not in large quantities.

An explanation of the small amount of gossan and enrichment occurring in a sulphide pipe of this type, formed in pre-Cambrian time, lies chiefly in the action of erosion. As stated, a pre-Cambrian as well as Tertiary fault scarp existed which was especially subject to erosion and consequently followed enrichment very closely. It was also suggested by Winchell that possibly a high zinc content was unfavorable towards enrichment.

The form of the pipe in the upper portion of the mine is cylindrically lenticular, measuring about 750 ft. in the longer axis and 400 ft. in width. With depth the mass becomes tabular, measuring 1,100 ft. along the strike and with an average width of 275 ft. The average area of all levels of the sulphide mass is about 292,000 sq. ft., while the average mineralized black schist is 52,800 sq. ft.

As seen in *Figure 1*, the diorite bounds the sulphide mass on the north and west side, which condition continues throughout the mine and forms a hanging wall. The south side is the irregular footwall and shows all of the irregularities of typical replacement deposits.

The strike of the sulphide is approximately N 50 degrees E, with a dip of 53 degrees to the northwest. The pitch of the mass bears N 22 degrees W at an angle of 62 degrees.

The irregular interfingering of the sulphide with the schists and porphyry indicates the replacement type of ore deposition. Banding within the sulphide mass, due to unreplaced gangue material, is a further indication of replacement.

Gradations from fresh porphyry to chloritic schists indicate that between 40 and 60 percent of all the schist was

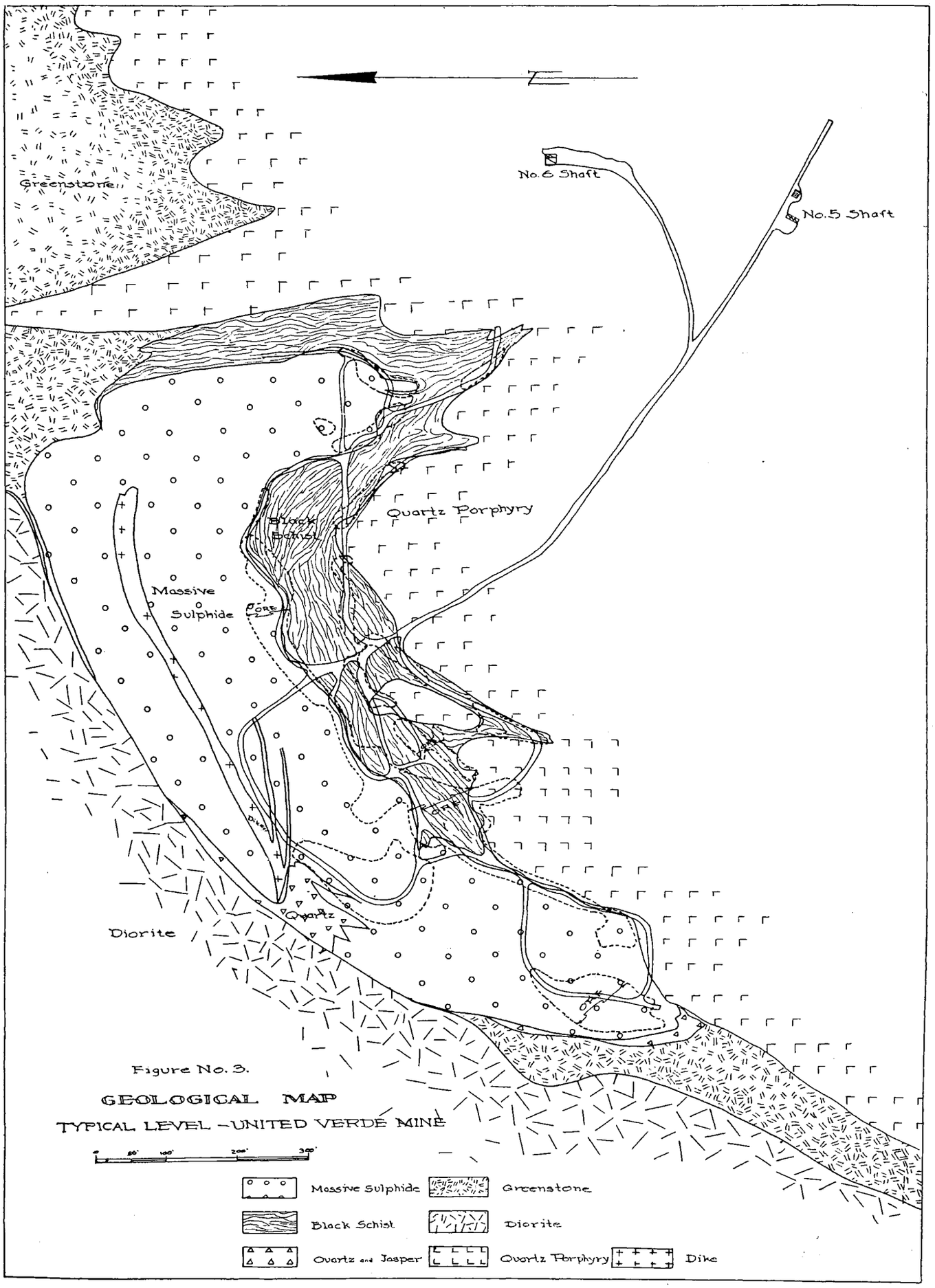


Figure No. 3.

GEOLOGICAL MAP

TYPICAL LEVEL—UNITED VERDE MINE

United Verde Copper Company

Class	% Cu	Oz. Au	Oz. Ag	% SiO_2	% Fe	% Al_2O_3	% S	% Zn	% CaO	SpGr
Silicious sulphide	4.80	0.02	1.76	37.2	21.4	3.7	21.3	1.9	0.64	3.30
Massive sulphide	5.52	0.02	1.48	11.8	31.4	3.5	32.6	2.4	1.80	4.05
Black schist	6.04	0.01	0.89	29.9	20.7	12.7	15.4	0.7	2.28	3.23
Quartz porphyry	4.11	0.01	0.89	40.1	15.7	12.5	10.8	0.9	1.05	3.10

originally quartz porphyry. Much of this schist is, of course, now replaced by sulphide although still recognizable. The material which has been replaced includes quartz porphyry, silicious bedded sediments and certain types of greenstone, all of which are occasionally found within the massive sulphide in an unreplaced condition.

Structural shear zones are occasionally found at a distance from the sulphide mass which contain inpregnations and disseminations of pyrite and chalcopyrite. When commercial, these orebodies are long and narrow, an example of which is found on the 500 level which measures 650 ft. by 30 ft.

The orebodies in general form on or near the sulphide-schist contact and where the mineralization is most intense it continues through the schist area into the quartz porphyry. *Figure 3*, of a typical level (showing the ore outline in dotted line) illustrates the irregularities in plan. As is shown the main ore body, following essentially the sulphide-schist contact, measures some 600 ft. in length and from zero to 150 ft. in width. Ores other than the main ore body are also found along the contact, within the heart of the sulphide mass, and within the schist and porphyry rocks. The vertical bounds of the ore are equally, if not more irregular, due to the fingers and pendants of ore which are frequently hidden from normal stoping operations by false walls because of slips, dikes, horses of waste and character of spotty mineralization.

Numerous slips of little consequence as to dislocation of ore occur both pre-mineral and post-mineral in age. These are more apparent in the schist and porphyry than in the sulphide, for more often than not the latter becomes brecciated over a larger area. Andesite dikes which cut the sulphide and other rocks are frequently offset a few feet though more often than not the displacement can be measured in inches.

Due to the character of the mineralization which is spotty, particularly in the schist and porphyry, horses of waste 15 to 30 ft. in diameter occur. The copper mineralization as has been suggested is most intense near the sulphide-schist contact decreasing in value as it approaches the center of the sulphide or schist areas. The copper mineralization along the schist-porphyry contacts occasionally becomes equally as strong, in which cases, the ores may extend from within the sulphide mass through the whole schist area and into the porphyry.

The dip and pitch of the ores vary considerably from level to level. The pitch of the ores has been as flat as 30 degrees, while the dip has been as low as 45 degrees. Similarly both have been vertical. It is natural to expect the pendants and offshoots of ore to occur more frequently on the flatter dips.

Underneath the diorite footwall, jasper and quartz masses of vast cross-section occur which grades into the pyritic masses. Occasionally ore bodies occur in the jasper areas accompanied by only a very little pyrite.

Four classes of ore are recognized of quite variable composition.

V. Mineralogy

The mineralogy will include only the ore minerals and close associates and not the general rock forming minerals. The following minerals are found, listed according to Dana's scheme of classification.

I. *Native Elements*:

Copper
Silver
Sulphur

II. *Sulphides, Selemides, Tellurides, Arsenides, Antimonides:*

Argentite
Arsenopyrite
Bornite
Chalcocite
Chalcopyrite
Covellite
Galena
Marmatite
Pyrite
Pyrrhotite
Sphalerite
Unknown

III. *Sulpho-Salts*:

Tetrahedrite (Freibergite)
Tennantite (?)

IV. *Haloids-Chlorides*:

Fluorite

V. *Oxides*:

Cuprite
Hematite
Jasper
Limonite
Magnetite
Quartz
Specularite
Tenorite

VI. *Oxygen Salts*:

Azurite
Calamine
Calcite
Chrysocalla
Dog-tooth spar
Dolomite
Epidote
Gypsum
Goslarite
Malachite
Sericite
Serpentine
Siderite

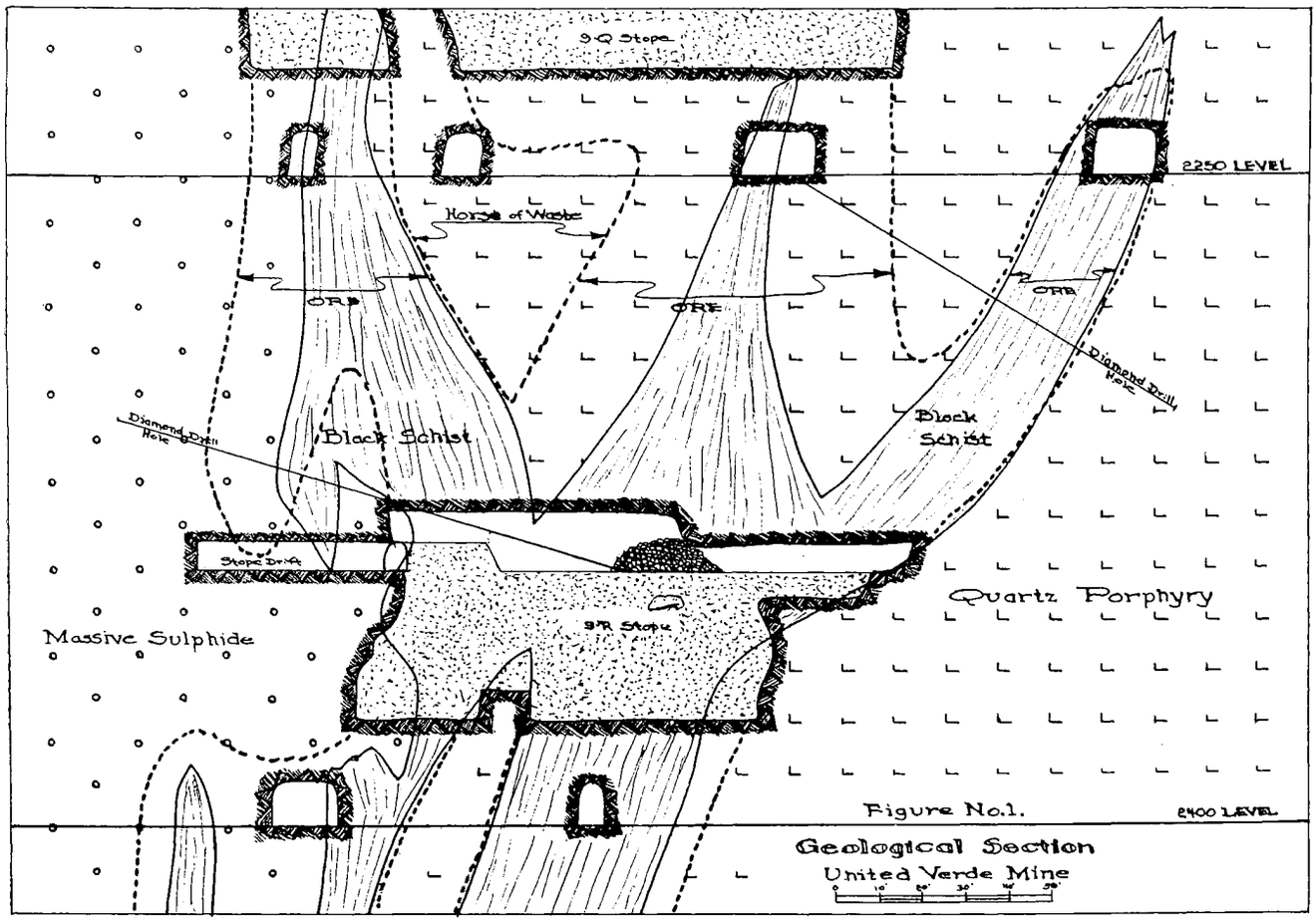
For the purposes of this paper, description of the individual minerals will not be given. Suffice it to say that chalcopyrite is the chief ore-forming mineral except in isolated places where copper carbonates have formed secondary ones.

The diorite on its footwall side is very undulating in vertical section. As the sulphide solutions came up within its concave margin they had a tendency to bulge more or less in conformity to the diorite wall.

The condition of the footwall schists and porphyry is such as to indicate the strength, intensity and extreme heat of solutions in attacking the minerals. The large amount of chloritization with lesser amounts of sericitization and kaolinization give evidence to this fact. The porphyry when in juxtaposition with sulphide is rarely fresh and unaltered for many feet from the sulphide contact.

The ore bearing solutions, given off from a magmatic reservoir below, would tend to travel upward through a permiable schistose channel to the site now occupied by the sulphide mineralization.

Compared to the relatively long interval of time between the intrusions of porphyry and diorite, the short interval of time between the intrusion of the diorite and the sulphide indicates that the latter owes its origin to the diorite, which is a magmatic differentiation product of the Bradshaw granite.



EXPLORATION

By Mayer G. Hansen
GEOLOGIST

EXPLORATION work may be divided into two classes, one covering prospecting in general, the other covering the main ore zone.

The main ore zone includes the sulphide area, the black schist area, and a portion of the porphyry immediately surrounding them. Within this zone lenses of ore occur from 25 sq. ft. up to 50,000 sq. ft. in area. The most favorable location for ore is on the massive sulphide-schist contact; secondly, the contact of the schist and porphyry; thirdly, within the sulphide area; fourth, within the schist, and lastly within the porphyry areas.

Within the main ore zone the geology and known ore areas are projected with sufficient accuracy to permit planning of the haulage and contact drifts. The latter is so named because it is driven on or near the sulphide-schist contact. On the last lift of new levels this drift was driven on the schist side on definite bearings, as compared with the old practice of actually following an irregular replacement contact. The advantages are obvious, in that no rounds are wasted, the rock is softer and the cost per foot is cheaper, straighter drifts are obtained, curves may be gradual rather than abrupt, and being in the footwall side the chutes are better spaced when the stope is silled out. Guidance of this drift is accomplished by means of occasional diamond drilling, a hole being drilled to determine the distance to the contact and the extent of ore in the hanging wall. A second hole is often drilled on the same bearing as that planned for the next chord of the drift and the latter governed accordingly. It also receives constant geological guidance. Upon completion of this drift the foot and hanging walls are thoroughly diamond drilled, delimiting the ore accurately, so that the footwall and hanging wall drifts may be driven, and the method of mining with its stope and pillar system planned.

When drill holes encounter isolated patches of ore they are further explored by drifts or crosscuts.

After stoping operations have commenced, the floor silled is mapped in detail, which usually gives an intimation of what may happen geologically between levels. Should it appear necessary, a light weight diamond drill is taken into the stope and up-holes drilled to further obtain intermediate level data. This data is important since the dip and pitch of the ores are so irregular that all offsets of ore, fingers, etc., must be determined in advance of stoping operations to permit the proper spacing of chutes. As mentioned under geology, numerous slips and dikes occur which but rarely limit the ore. Thus in the course of normal stoping operations when a stope has been bounded on a side for a few floors by a slip or dike, it is necessary to prospect beyond it with either a short stope drift or drill holes. This is done at regular floor intervals until it is assured that a false wall does not exist. Also, when a down hanging pendant of ore occurs, the exact bottom is determined and a stope drift driven under it so that the ore in the pendant can be carried simultaneously with the stope in order not to hold up operations when the pendant joins the stope. An example of this is shown in *Figure 1.* A generalized section and typical mine level is shown under the article on the geology and ore deposits.

Smaller areas cut by drifts are explored by raising to more or less determine their height. Then if the ore persists a diamond drill is set up in the raise to determine the area.

In the broader phases of prospecting long drifts or long diamond drill holes (usually flat) are used. The regional schistosity bears in a northerly and southerly direction, in which direction the long drift is driven, generally with a definite objective in mind to some geologic feature. From this drift crosscut diamond drill holes are driven to determine other structural features, as greenstone-porphyry contacts, shear zones with hard impervious hanging walls, schistose or shear zones within the porphyry or embayments in the diorite. Naturally if sulphide mineralization is encountered, further prospecting is carried on by detailed drilling or drifts and crosscuts.

Thus in prospecting or exploration work, diamond drilling is the chief method employed, followed by level drifts, crosscuts and raises. Five diamond drills are operated underground, drilling all sizes of holes from "ES" to "N". The total annual footage obtained is about 30,000 ft. As to drifting, raising, etc., the total footage is about 25,000 ft., or at the rate of 70 tons mined per foot of development.

DEVELOPMENT and MINING METHODS

By J. F. Cowley
PLANNING ENGINEER

T. W. Quayle
MINE SUPERINTENDENT

J. F. Cowley

T. W. Quayle

THE orebody of the United Verde Mine consists of irregular shaped lenses in the massive sulphides and wall rocks. They have an average dip of approximately 63 degrees, and vary widely in size and shape. Widths may be from 10 ft. to 180 ft., and lengths from 20 or 30 ft. to 800 ft., the complete orebody being crescent shaped. The hanging wall, in general is the non-copper bearing, extremely hard massive sulphide, and the footwall a tough quartz porphyry. Orebodies occur separately and in combinations of massive sulphide, black schist, and porphyry, the latter being the most common, having sulphide on the hanging wall side, grading into schist, and from schist to porphyry and to barren porphyry. These conditions preclude the possibility of accepting any single mining method and results in the use of many different methods. However, conditions as a rule favor horizontal cut and fill stoping, and approximately 60 percent of the total tonnage is now being mined by this method. The general development plan of the mine has been evolved with these general conditions in view.

Development and stoping is planned to insure a daily production of approximately 2,000 tons of copper ore until such time as the open pit is exhausted, when production will be increased underground. The level interval has been found to be most economical at 150 ft. A closer spacing leaves too high a percentage of ore in the level pillars above cut and fill stopes, which is mined by the comparatively high cost square set method, and an excessive development charge. A greater distance between levels results in excessive costs in repairs to ore chutes.

To insure the necessary tonnage, the vertical depth developed annually has been found to be one level, including all drifts, crosscuts, raises, and chute raises. Drifting per year runs from 12,000 to 20,000 ft.; raising from 4,000 to 6,000 ft. The total development averages about 25,000 ft. The tonnage blocked out is approximately 70 tons of ore per foot.

This program has been followed in recent years by sinking the main ore hoisting shaft in 600 ft. lifts, drifting to the location of the service shaft, and raising to connection from each level. Upon completion of the shaft work, the orebody on the upper two levels is developed, after which the remaining two levels are started. This method allows a period of two years in which to prepare the two levels for stoping.

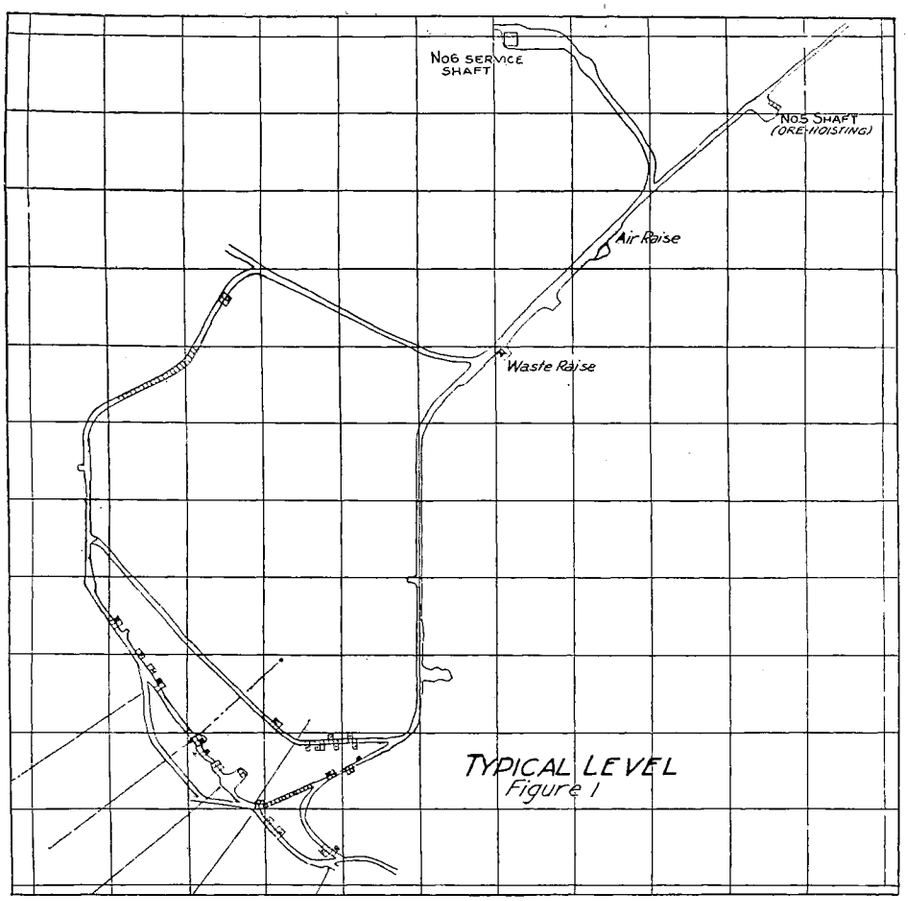
Outlines of the anticipated ore areas and formation contacts are prepared by the Geological Department by projections from levels above the occasional down hole diamond drilling. From these outlines the main crosscuts to the orebody are planned and crosscutting is started from the hoisting shaft. Some years ago it was customary to drive the main ore haulage crosscut north to a point near the center of the orebody and through the iron-schist contact. From there, contact drifts were driven along the strike to develop the orebody. The object of this method was to divide the haulage, both of ore from the stopes above and waste to the stopes below, which resulted in less congestion in the gangways.

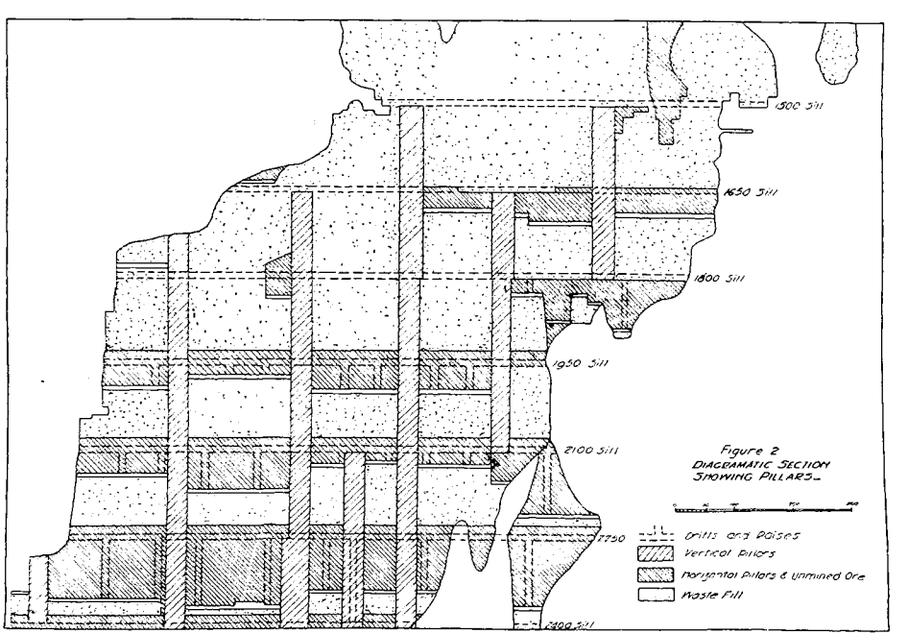
Recent practice is to drive the crosscut to a point near one end of the orebody, drift on the contact for its entire length and connect back to the crosscut at the nearest convenient point. *Figure 1.* This change in the development plan has several advantages, since it allows much better ventilation during preparatory stope work, such as driving raises, chute raises, and auxiliary drifts. It has the same advantage of a split in the haulage which is more flexible than formerly, and it allows greater speed and more efficiency in development, as we have two sides to work from in any given area. The general plan is to start work in the middle of the orebody and retreat to both ends. As soon as the contact drift has been completed, diamond drill stations are cut at desired intervals, and the limits of the orebody carefully outlined. The ore outlines and geology is mapped and a study made, aided by the use of glass models to determine the most economical and safest methods of mining the particular block of ore. Except in particular cases, the horizontal cut and fill method is first planned.

The next step is the establishment and location of vertical pillars. This, of course, depends to a great extent upon those already in place on the levels directly above. However, due to the rake of the orebody, one of the pillars on the level above is sometimes dropped, and occasionally at other points the orebody becomes so narrow that it is feasible to drop an entire pillar, the object being, of course, to leave as little ore as possible tied up in this pillar system as is consistent with good safety practice. A diagrammatic section showing the pillar system is shown in *Figure 2.*

Mining of these pillars must be done in regular sequence, starting with those left on the upper levels and working down. On each of the older levels the start is at the ends of the haulage drifts, and they are mined back to the center and the main crosscut. On the newer levels, with haulage from both ends, the first mining would start in the middle of the orebody and retreat to the ends. The footwall and hanging wall having been established by diamond drilling, auxiliary drifts to provide proper chute spacing are now driven at intervals of approximately 40 ft. This distance, however, varies with the irregular outlines of the orebody. Raises are driven next, first, one to each stope for ventilation while silling, and later, for both ventilation and admission of waste. Chute raises and manways are then driven, and the stope silled to the ore and pillar limits on the 4th floor, or 24 ft. above the level. While the stope outline is being determined by silling, additional raises may be driven to provide waste, the present practice being one waste raise for each stope up to 5,000 sq. ft. of area, and two waste raises for stopes of over 5,000 sq. ft. area. The maximum limit of stopes is held to approximately 12,000 sq. ft. Raises are also driven through vertical pillars for ventilation, and are used later in the mining of the area.

All stoping at the United Verde is followed by waste filling; all development plans must therefore include the system of raises throughout the mine from which this waste is obtained. One raise system extends directly from the open pit operations to the present 2,700-level, which is the lowest level on which stoping operations are carried on. There are two bulldozing chambers in this raise system, one at the 500-level with 24-in. spacing between grizzlies, and one on the 1,000-level with 16-in. spacing, the two lifts producing a marked crushing effect in the 500-ft. drop of material. The second raise system extends from the 2,700-level to the 1,000-level main haulage. Through this raise waste is transferred that can not conveniently be handled through No. 1 raise from the shovel pit. It is also connected with the main hoisting shaft, and all development waste is dumped here and used as fill where needed. On the 2,700-level the two raises have been brought together, and the lower extension will consist of but one raise. In planning





main crosscuts, these waste raises must be connected with the raises above at angles of not less than 65° inclination. The main ventilation raise extends vertically from the surface to the 1,000-level fan room, and from there to the main crosscut on each level. There are no definite limits set on the inclination in this case, but it is in close proximity to the two waste raises and is, therefore, about the same.

United Verde Copper Company

Drifts and Raises

Main crosscuts are driven 6 x 8 ft. in the clear, and sideswiped to 8 x 10 ft. for a distance of 140 ft. to include both waste raises to allow ore and waste trains to pass each other. These main crosscuts are usually driven with two machines on 3-in. double jack columns. The crew consists of two miners on one shift and two muckers on the other. Rounds are standard uppercut, of from 19 to 27 holes, and averaging about 5½ ft. in depth. *Figure 3* shows typical round.

The auxiliary haulage drifts are also 6 x 8 ft., but are generally easier drilling, and one miner drills the round with two muckers on the other shift.

Mechanical loading has been experimented with in the main haulage drifts, but has not been satisfactory in a 6 x 8 ft. section. The miners have difficulty in averaging more than 5 ft. per round without excessive powder cost, and as there is but the single heading on each level, two muckers shovel this amount easily at a much lower cost. However, in tunnel headings 9 x 11 ft. on the 1,000-level, they have proved very satisfactory. (See *Figure 4.*) It is planned to use shovels on development levels for mucking when cutting out raises and chute raises for timber. Scraping into loading docks has been discarded for the same reasons as shovels in 6 x 8 ft. drifts.

Track and pipe are installed in all headings by separate crews, the track crew consisting of a trackman and helper. Pipemen work singly.

All through raises are driven 6 x 11 ft. with standard cribbing, as shown in sketch. In the case of the main ventilation raise and in other special cases, raises are sideswiped to 8 x 15 ft. Where raises are used strictly for ventilation and waste transfer, if ground conditions permit, the cribbing is stripped out as soon as connection is made. This is also done in stope waste raises where there are two or more raises in the stope.

Raise rounds consist of from 12 to 32 holes, and all cut holes are pointed away from cribbing. (*Figure 5.*) The crew consists of two miners on one shift, who do all the drilling and timbering. The opposite shift pulls the muck after blasting and shoot any missed holes. Two shifts working in raises have never been efficient due to gas in the muck, missed holes, and time lost in pulling muck after blasting. Advance per round varies widely, due to the differences in ground which varies from soft black schist, where 1½ ft. per miner shift is reached, to hard massive sulphide with jasper, which runs as low as 0.2 ft. advance per miner shift.

Chute raises and manways are the last units in level development, and will be taken up in more detail in the different stoping methods.

Determination of Mining Method

After a stope is silled to ore limits, the general conditions are carefully inspected. If the back is in good shape, with no evidence of undue weight, and the stope is in the main orebody requiring vertical pillars to limit the size of stoping area, it is considered as horizontal cut and fill ground, and developed and mined as such. In the case of schist stopes, with heavy ground such as is found in the upper levels near the old fire stopes, flat square set methods are used. In cases of isolated orebodies in the hanging wall, with sulphide in both walls and of a size not requiring pillars, it is considered very carefully as a possible shrinkage stope. Walls must be fairly uniform, with no great possibility of offsetting. If one wall is weak, incline cut and fill methods would be considered. In cases of vertical and horizontal pillars, flat square set is most often used. However, experimental incline square set stopes have been successfully worked, their most serious drawback being inability to obtain satisfactory sorting. This method, however, can be

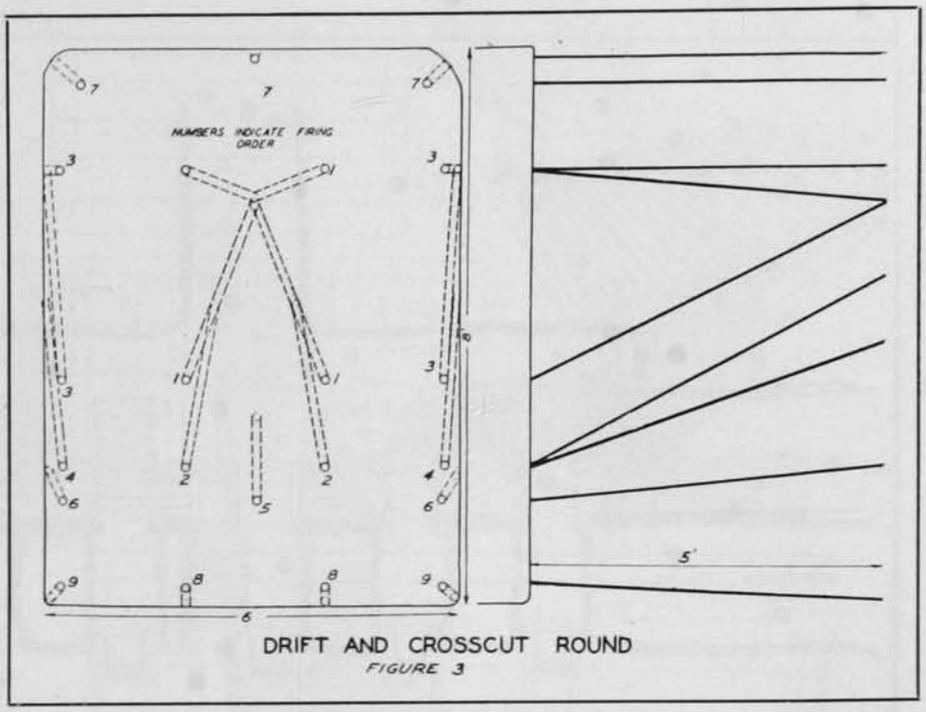
adopted in many cases in orebodies of uniform grade. Top slice stoping methods have been used in special cases.

Horizontal Cut and Fill Stoping

Horizontal cut and fill stoping is the most widely used method at the United Verde Mine. It is easily applicable to orebodies of different sizes, although in large areas vertical pillars of ore are left between two or more stopes for support. These pillars are removed by square set mining after cut and fill stopes are finished. The general advantages of this method over other methods for the type of orebodies encountered at the United Verde include practically 100 percent extraction, clean mining, convenience for hand sorting in stopes, low timber cost, high daily tonnage per square foot of area, low development cost, ease of ventilation and low fire hazard, and ease of wall prospecting. Of these advantages, clean mining is the most important. The orebody, which ordinarily has a uniform dip, has many irregularities or areas of offset ore in both walls. These areas are easily mined by the cut and fill method as the stope advances. Where geological conditions favor such offset ore, it is very simple to drive drifts for prospect work, and is also possible to do necessary diamond drilling. With proper supervision and use of cribbed bulkheads, a remarkably good safety record is obtained, and working conditions in general are excellent.

The size of stopes vary greatly—from 10 to 160 ft. between hanging wall and foot wall, and up to 200 ft. in length. The total area, however, is held to a maximum of approximately 12,000 sq. ft. by leaving vertical pillars which extend from footwall to hanging wall. These pillars have been 30, 35 and 40 ft. in width, but present practice standardized on 35 ft. (See *Figures 6 and 7.*)

Occasionally a limited area in a cut and fill stopes becomes heavy, due to a bad wall or seepage of water. This area may be square set for a few floors, then the sets dropped when the condition is remedied. In schist and porphyry stopes, the presence of water seepage is regarded as a dangerous condition, and is stopped as soon as possible. All ditches on the various levels are concreted and inspected frequently to keep them water tight.



Stope Development

One waste raise to be used for ventilation during silling operations is the first step in the layout of cut and fill stopes. With the ore outlines fairly well established, this raise is driven in the footwall of the stope, where the ground is weakest as a rule, and where it is planned to start the successive stope cuts. Waste raises are not driven at a flatter inclination than 65°, as that has been found to be the flattest angle on which the waste will run freely. When this raise has been started, it is often desirable to spot a chute and manway and two or three single chute raises, in order to begin silling the first floor as soon as possible. This can be done by using a small fan for ventilation and connecting with the raise as soon as it is holed through.

Chutes are spaced at 16½-ft. centers in sulphide ore, and 22-ft. centers in schist and porphyry. The massive sulphides cause much greater wear on lining and chute timber in general, which makes it advisable to carry more chutes to be given area than in the silica. Most hand sorting is done in these silica sections, and it is advisable to keep chute spacing greater. The spaces in which to pile sorted waste are more convenient and a greater percentage of the muck is handled

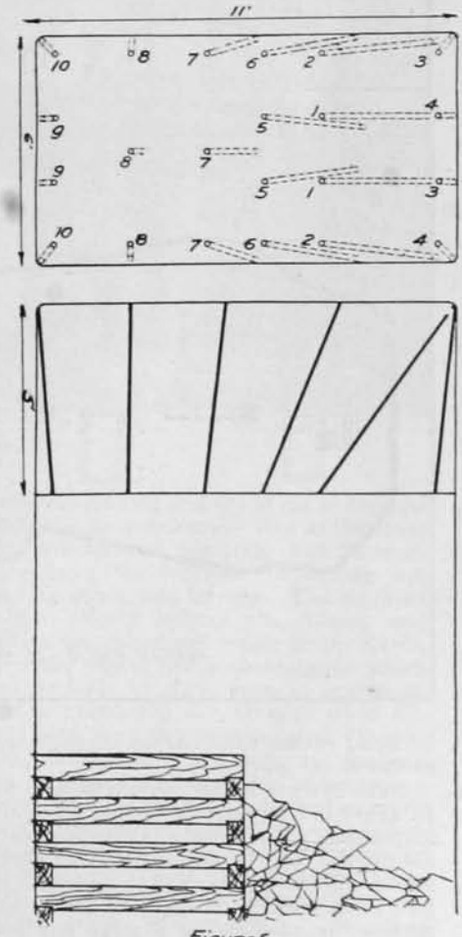


Figure 5
RAISE ROUND

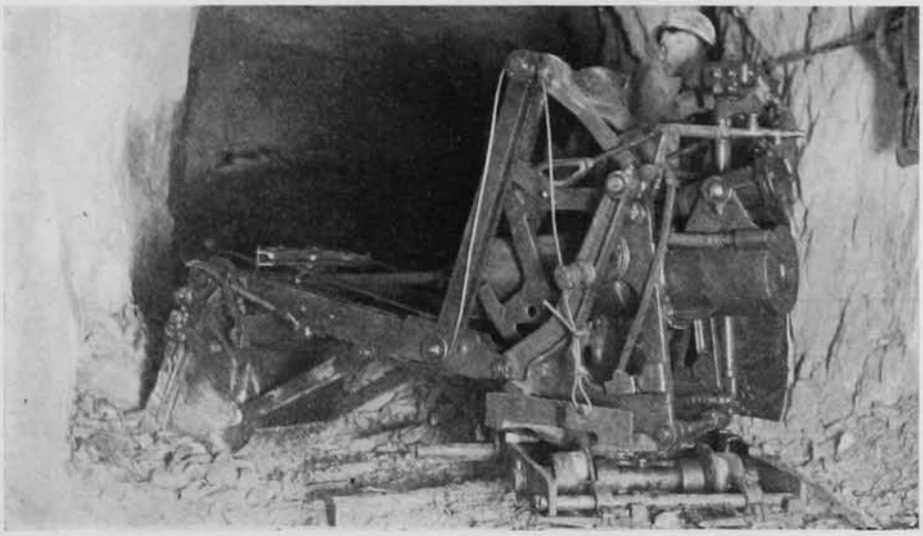
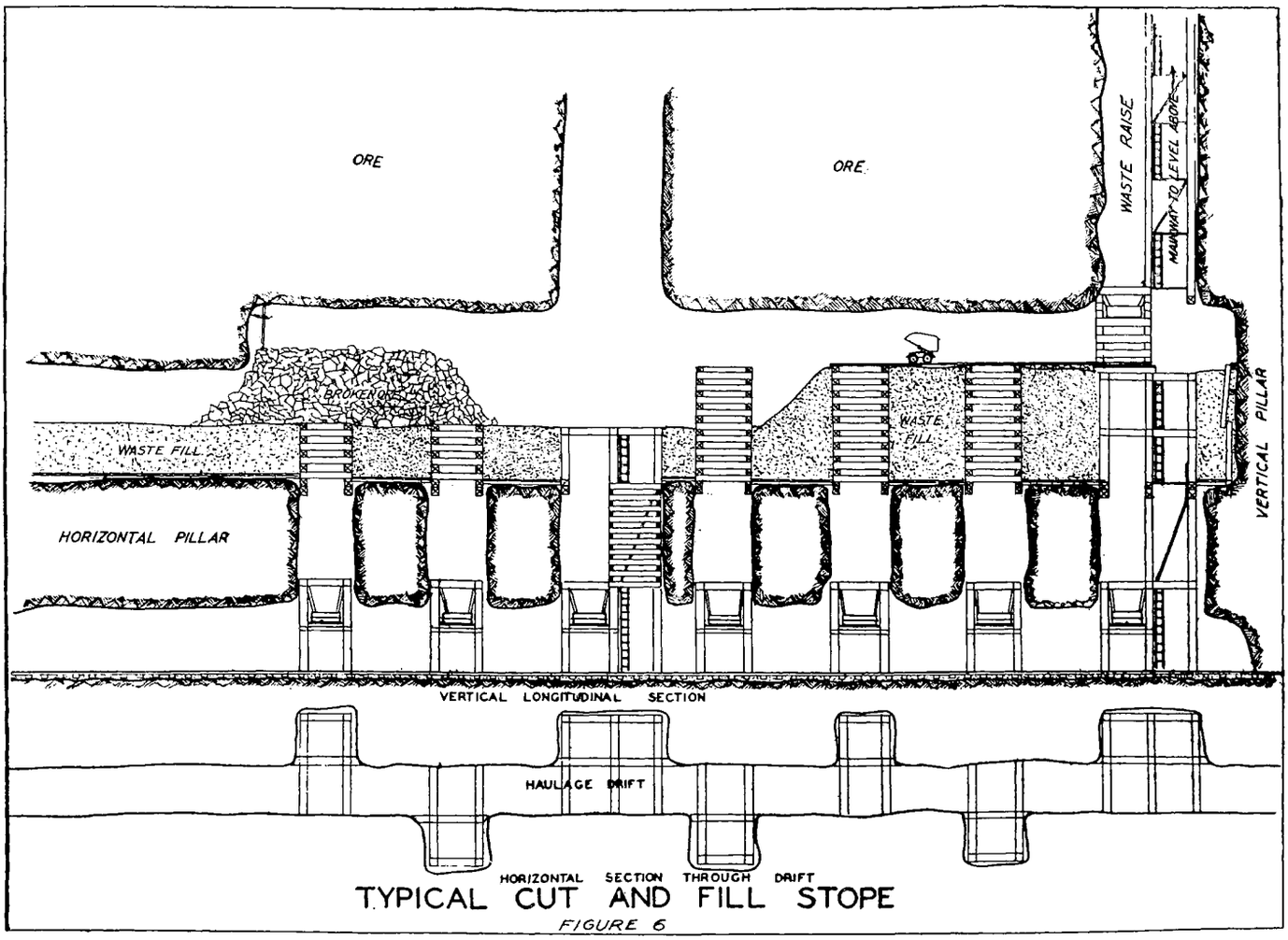


Figure 4

United Verde Copper Company



TYPICAL CUT AND FILL STOPE

FIGURE 6

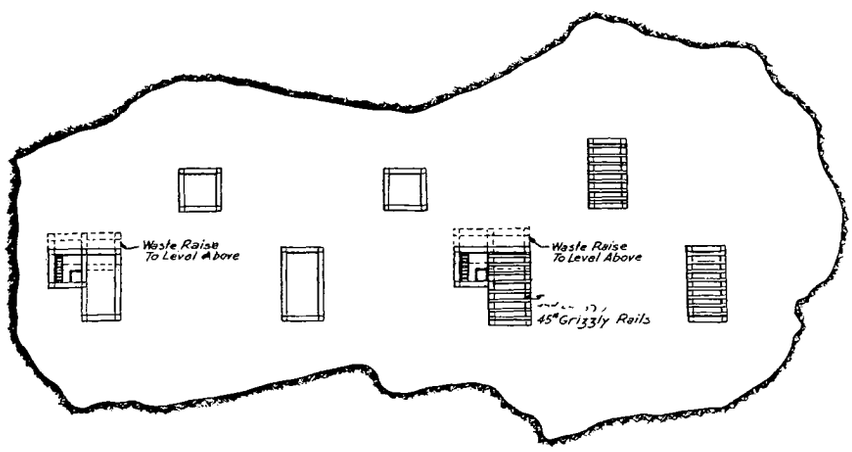
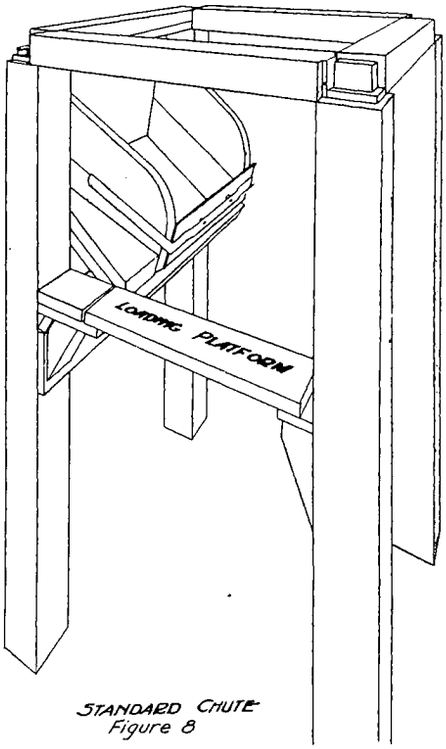


Figure 7
PLAN OF TYPICAL CUT and FILL STOPE SHOWING CHUTES.



STANDARD CHUTE
Figure 8

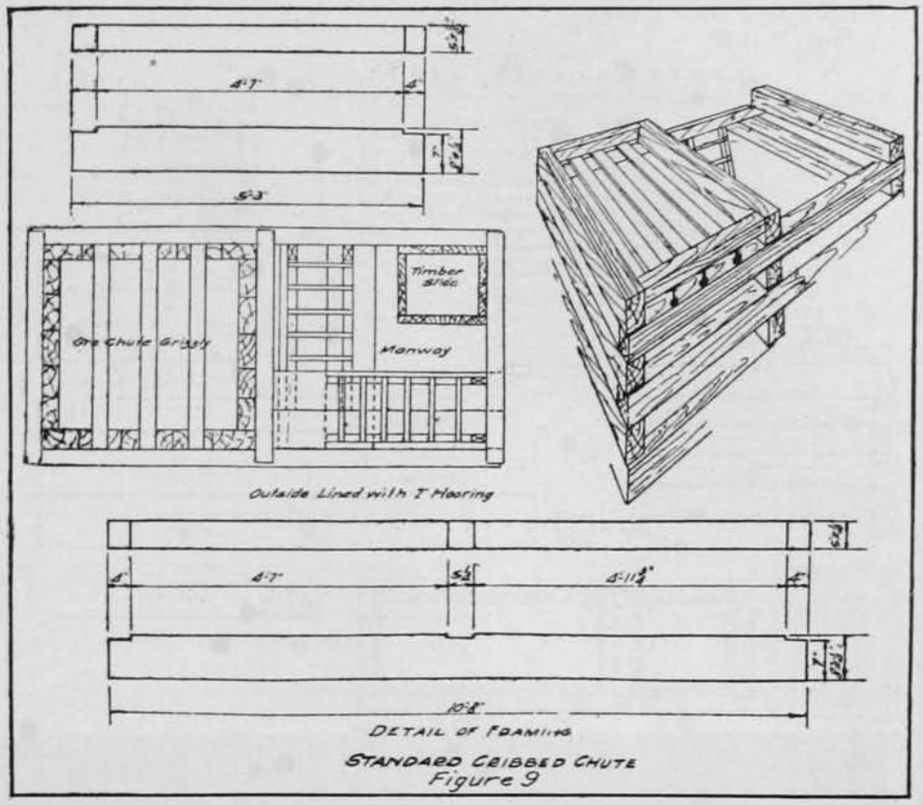
by hand rather than barred through the grizzlies. (*Figure 8.*)

Manways are spaced approximately 50 ft. apart in long narrow stopes. For wide stopes they are planned so that the greatest number will be open at a given time, and in different gangways if possible. It has been found advisable to have one manway in each end of a stope and one near the center. The manways under waste raises are dropped, due to accident hazard from rocks falling down the raise. Each manway has staggered ladders and a platform every 7 ft. (*Figure 9.*) The timber slide is lagged on all sides as a safety precaution, and is served by an Anaconda type air hoist. (*Figure 10.*) Drill steel, machines, miscellaneous tools, etc., are hoisted in specially built steel

United Verde Copper Company

skips, in which they can be securely fastened. (*Figure 11.*)

The first mining floor is a cut from 7 to 9 ft. in height, started 24 ft. above the track level. This rib serves as a support for gangways, which seldom need timber in such cases, and requires less floor pillar below the level. This is a decided advantage in handling timber and supplies when square set mining starts. When the sill cut has been finished, additional waste raises are driven if found necessary. Often, due to irregularity of outline, additional chute raises must be driven and, if this is impractical from the standpoint of gangways, plans are formed for offsetting chutes in the stope to accomplish the desired spacing. Double 12 x 12 stringers are laid over the chute raises to support stope chutes, and are lined up in the direction the chute may offset. As the stope advances, it is necessary to offset most footwall chutes to follow the dip of the orebody. A cribbed chute constructed of 10 x 10 in cribbing (*Figure 12*) has been designed for these offsets on the first few floors. The former special 6-post chute is still used above the 12th floor (*Figure 13*). The cribbed chute offers much more resistance, to blasting and costs about the same to construct. However, wing chutes, as described below (*Figure 14*), are impractical in conjunction with them, but can easily be used with the old style offset. These wing chutes are efficient in stopes requiring little sorting and where the ore breaks up well. Salvage of timber is about 90 percent and, if the wing timber from one floor is used in the main chute on the next floor, it seldom wears out. Where these are used, spacing of chutes in gangways can be increased to 22 or 27-ft. centers. Sixty pound rails, 5 ft. 6 in. long are placed on all chutes for grizzlies, tied in by special blocks which are framed at the 500 framing shed. Blocks are framed to give 11-in. spacing between rails. Chutes are 48 in.



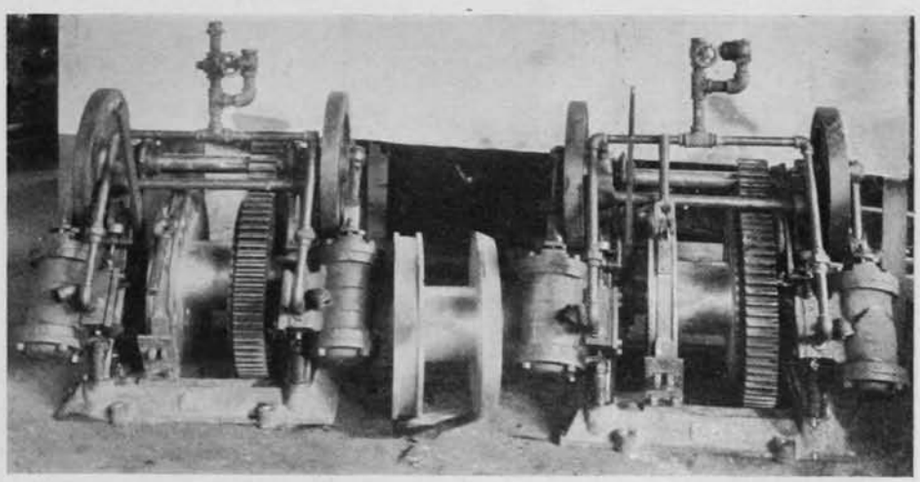


Figure 10

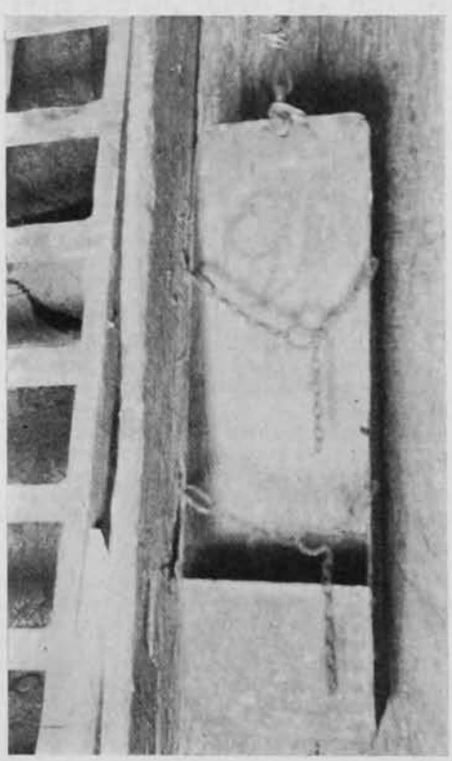


Figure 11

square inside measurement. Single chutes are raised with 5 x 8 in. cribbing, and all chutes lined with 4 in. native pine lagging (*Figure 15*). After chute stringers are placed, the floor is silled, using 4-in. x 12-in. x 11-ft. sills, spaced at 5½-ft. centers, and covered by double 2-in. x 12-in. x 10-ft. 8-in. lagging, breaking joints every 5½-ft. as shown in *Figure 16*. Sills are laid with care and no space allowed between them and the ground.

A 7-ft. cut of ore is now stoped down. This cut starts at a waste raise in what is considered the weakest ground, two rows of horizontal holes being drilled and blasted by each miner shift. In stopes having but one waste raise, the cut is started by drilling vertical holes around the raise while filling is still going on. After the waste pocket is removed, these holes are blasted and the stope is finished with flat hole drilling. One miner does all drilling and blasting, but receives help from the muckers in setting up, tearing down, and loading. The muckers follow closely behind the miner, and chutes are raised and waste filling starts. A waste pocket has been designed which can be built by three men in one shift, and is practically all salvaged when filling from the raise is completed (*Figure 17*). Just enough waste is run through the raise on which to build the pocket, as shown in *Figure 18*, eliminating as much hand mucking to level the cone as possible. Waste fill is spread using an 18 cu. ft. scoop nose car (*Figure 19*) and 8-ft. lengths of 18-in. gauge 12-lb. sectional track (*Figure 20*). Standard curves are furnished and all irregularities in stope outline are easily reached.

United Verde Copper Company

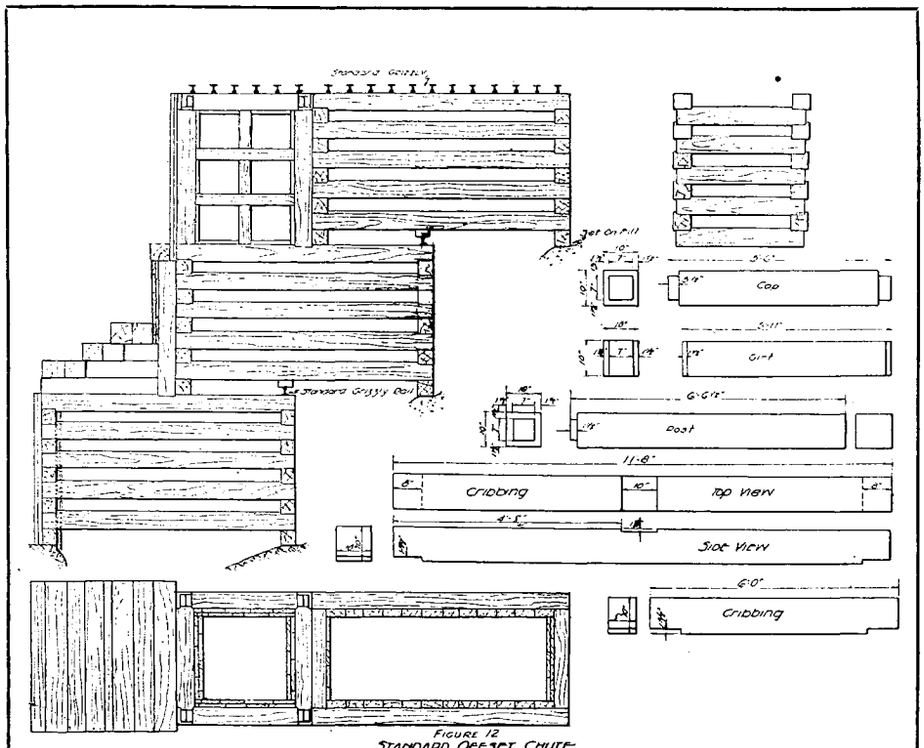
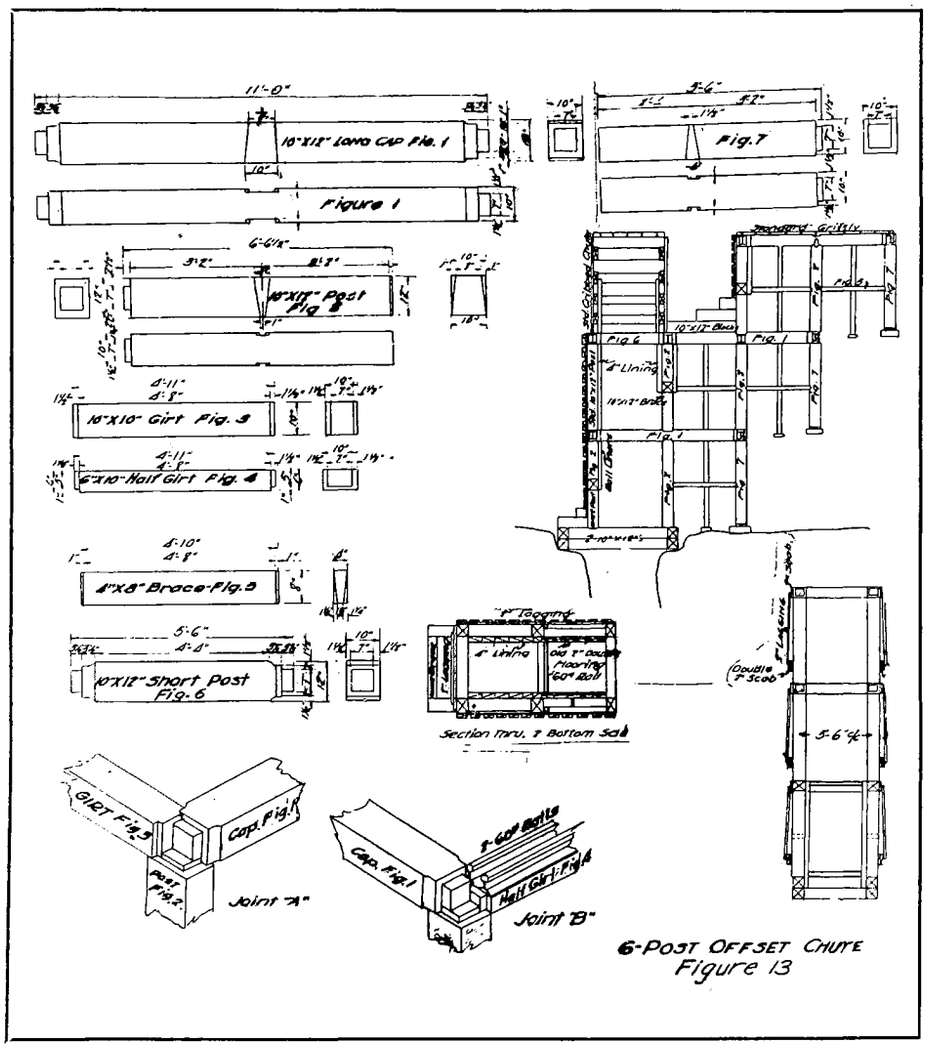


FIGURE 12
STANDARD OFFSET CHUTE



6-POST OFFSET CHUTE
Figure 13

United Verde Copper Company

One man trams from 60 to 100 cars of dry waste per shift. Along pillars a fence is erected to hold waste from ore. Six-inch by 8-inch posts are spaced at 3-ft. 7-in. centers, single lagged, are used. The posts are well blocked to the pillar wall and floored between fence and pillar each cut and the space filled with ore. When mining the pillar and breaking into this fence, only one floor of loose ore can be run out (*Figure 21*).

In blasting long holes a great many large boulders are found. To offset this, sectional loading of holes, using one stick of powder, one or two of tamping, and so on, connecting powder with Cordeau Bickford fuse, has been used with great success. The same charge of explosive which formerly was tamped in to the bottom of holes is distributed over the entire length. Each hole is fired separately, using standard fuse and 8-X caps. Each round blasted is sampled daily, and plainly marked by number, and shift bosses are furnished with copies of assay sheets on the same day. As most of the blasting is done on night shift and the sampling on day shift, the night shift bosses know the values before any ground is blasted. Occasionally rounds are drilled in low values unknowingly, but these are not blasted when assays prove the ground of no value.

Nearly all mucking is done by hand. Mechanical shovels and scrapers have not proved satisfactory due to the presence of many large boulders in the muck pile. The most satisfactory mechanical mucking is a plan where three drums are used on a scraping hoist, two tail ropes being used. This permits the operator to place his scraper in any spot within the two tail blocks. It also enables him to go around boulders which are too large to move and, after mucking on both sides of it to loosen it up, it is usually a simple matter to pull it out into the floor where it can be plugged. An appreciable saving in mucking has been accomplished, but one scraper will not produce the tonnage per shift that regular mucking crews would shovel, thus slowing down the routine of the whole stope. It is also difficult to keep the stope clean; hand sorting and waste filling are materially retarded due to the area necessary for operation. To date the saving in mucking costs has not been great enough to overcome the objections.

Hand mucking in an average stope of from 5,000 to 9,000 sq. ft. in area is by a crew of about six muckers and one pluggerman on each shift. All boulders must be broken to pass the 11-in. grizzly openings, and those too large to be broken with hammers are rolled back from the face of the muck pile and plugged with jackhammers, blasting being done at lunch time and quitting time on each shift.

Each cut of waste is floored with 2-in. x 12-in. x 5-ft. 4-in. lagging, which makes excellent shovelling conditions. After the muck pile has advanced a short distance, this floor is pulled up and all waste in the ore is hand sorted and piled to be covered as waste fill advances. Bulkheads of 8-in. x 10-in. x 6-ft. are thrown up as waste advances; 14-ft. bulkheads are raised ahead of the waste when needed.

In stopes where there are two or more waste raises, a continuous production is maintained, a new cut being started from one raise while filling of the previous cut is progressing from the second raise. In planning waste raises, the first one driven being on the footwall, it is advisable to

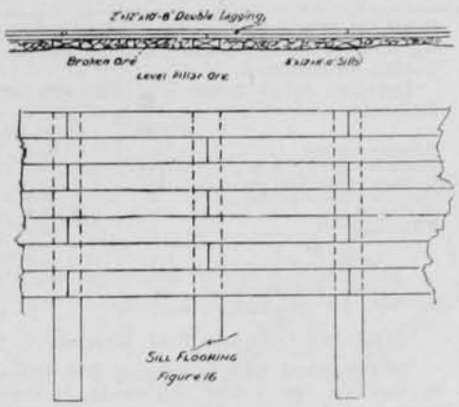


Figure 16

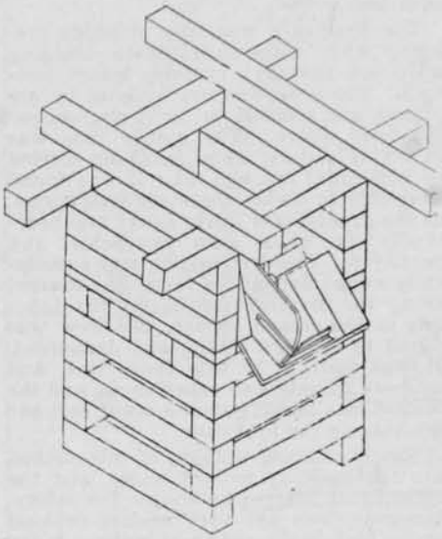
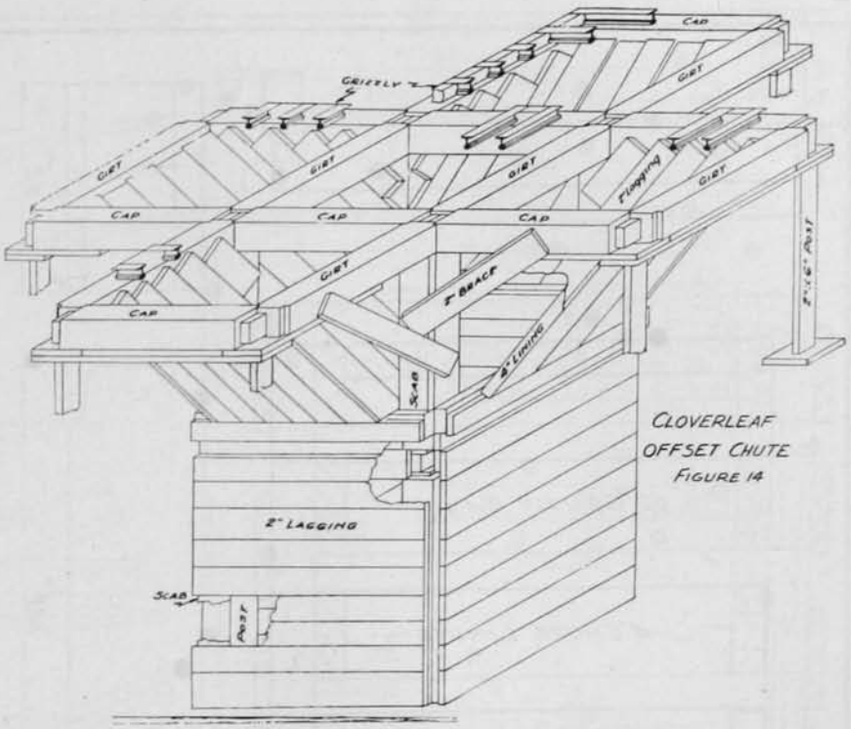


Figure 17



Cloverleaf Offset Chute
Figure 14

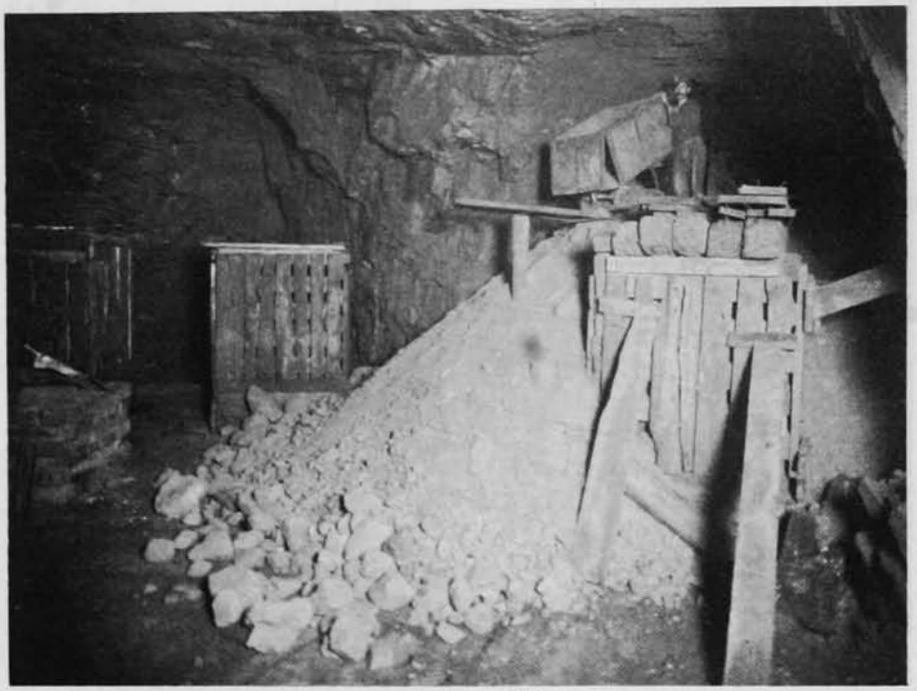


Figure 18

have a second one near the center of the stope and, if a third is desired, it is located in the opposite end. Filling from the first raise can be carried up to the second, and the waste pocket constructed without free dumping from the level above. Waste can be trammed much cheaper than levelling the cone and cleaning down the raise.

When stopes are mined out to within two or three floors of the level above, it is necessary to change the method to square set and fill. In narrow stopes with good walls, it is possible to mine cut and fill to the level occasionally, but as a rule two floors below are square set. With three floors above the level to sills, a total of five floors of each stope must be square set, or approximately 25 percent of the total.

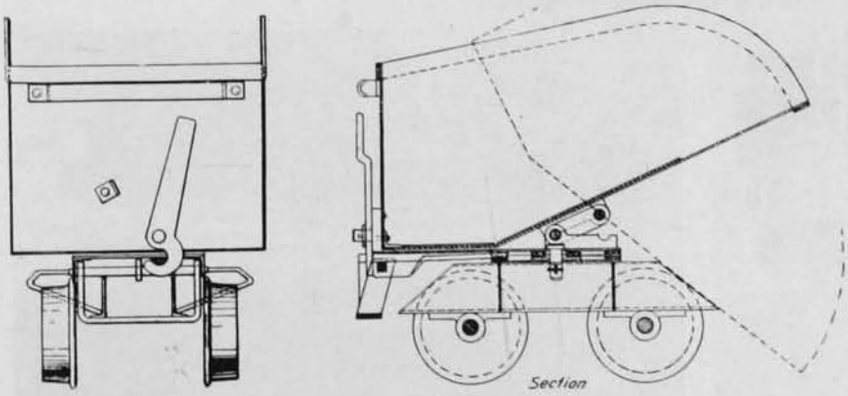


Figure 19
18 Cu Ft. Scoop Body Car

Miners drill from six to eight 6-ft. holes in hard sulphide and up to twelve 9-ft. holes per shift in schist. Muckers shovel from 16 to 25 tons per shift, averaging about 20. Waste spreaders average 70 cars per shift. One timber crew, consisting of a timberman and helper, will raise all new timber, including waste pockets, for a stope producing 150 tons per day. Repair work, in addition to new timber, often uses the same number of shifts monthly.

All stope operations are paid for either on bonus or contract system. Mining,

United Verde Copper Company

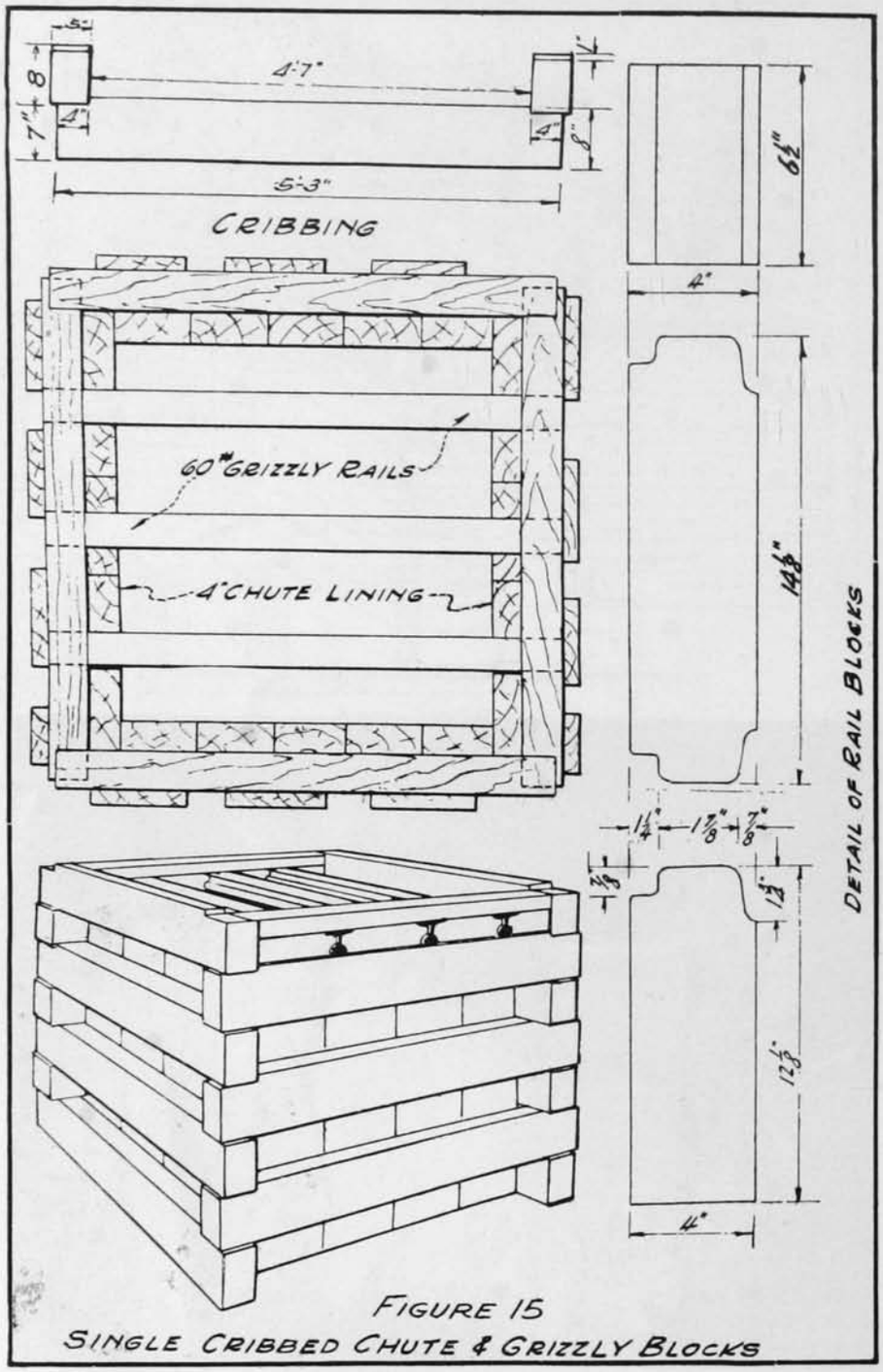


FIGURE 15
SINGLE CRIBBED CHUTE & GRIZZLY BLOCKS

mucking, waste filling, and repair timbering are on bonus standards, with all new timber on contract.

Detailed labor costs for 1929 are as follows:

	Percent	Cost, ton
Miners—drilling	14	$0.12
Pluggers—blockholing	14	0.12
Carmen—shoveling	34	0.29
Carmen—waste filling	8	0.07
Timber crews	24	0.20
Miscellaneous	6	0.05
Total	100	$0.85

Explosives, 0.6 lbs. 50 percent per ton.
Timber, 6.95 bd. ft. per ton.
Tons per man-shift in stope, 8.

Incline Cut and Fill Stoping

This method has been used but little in the last few years. However, it was found to be applicable to two different ore areas, and was used successfully in each case.

The first case was in a sulphide orebody, with massive sulphide hanging wall, but having a crumbly schist footwall. The area was not bounded by ore pillars, and pinched out as it approached the level above. The second case was in a wide orebody which had been started as horizontal cut and fill. It was found advisable to divide the stope lengthwise, as the ground was fairly heavy and practically the whole area was schist and porphyry. The first section was successfully mined flat, and a fence line carried along the dividing point. At the fence line in the second section, the brow was found to be very slabby and dangerous. It was inclined to this fence line, and no more trouble was experienced, and the section was mined out at a lower cost per ton than in the first case.

The chief disadvantages of this method are the inability to sort waste, and the difficulty in placing bulkheads for safety. However, men are very seldom exposed under high backs and these bulkheads are not as necessary as in flat cut and fill stopes. Sampling is more difficult, as the face is usually choked with muck after blasting.

The development of incline cut and fill stopes is much cheaper than that of flat stopes. Chute raises are needed only at the foot of the incline, and one waste raise located at the weakest side of the footwall is sufficient. In starting the stope, it is advisable to have one or two chutes widely spaced to handle muck from silling, but these are dropped as the incline is established. One manway and chutes spaced 16½-ft. centers are driven. The waste raise is driven in the footwall and silling started. The first cut is similar to flat stoping. Then the incline is established by mining 10-ft. slices from the waste raise, carrying the

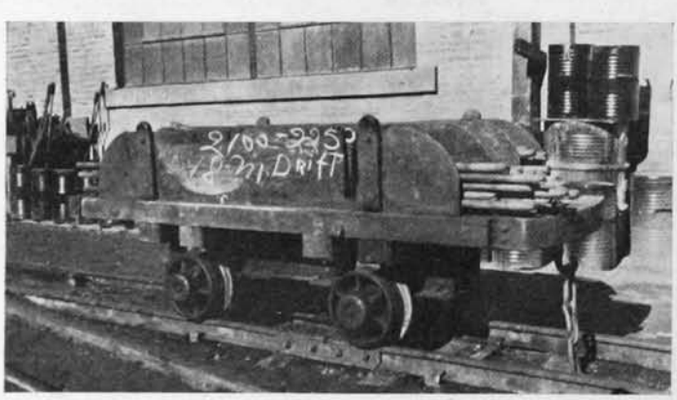


Figure 26. Standard Steel Truck

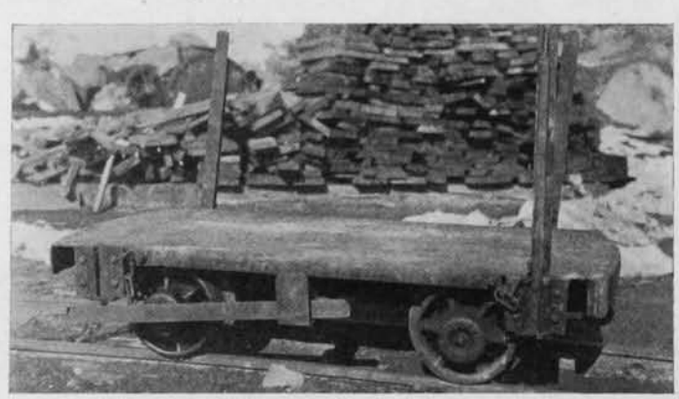


Fig. 28. Standard Timber Truck

United Verde Copper Company

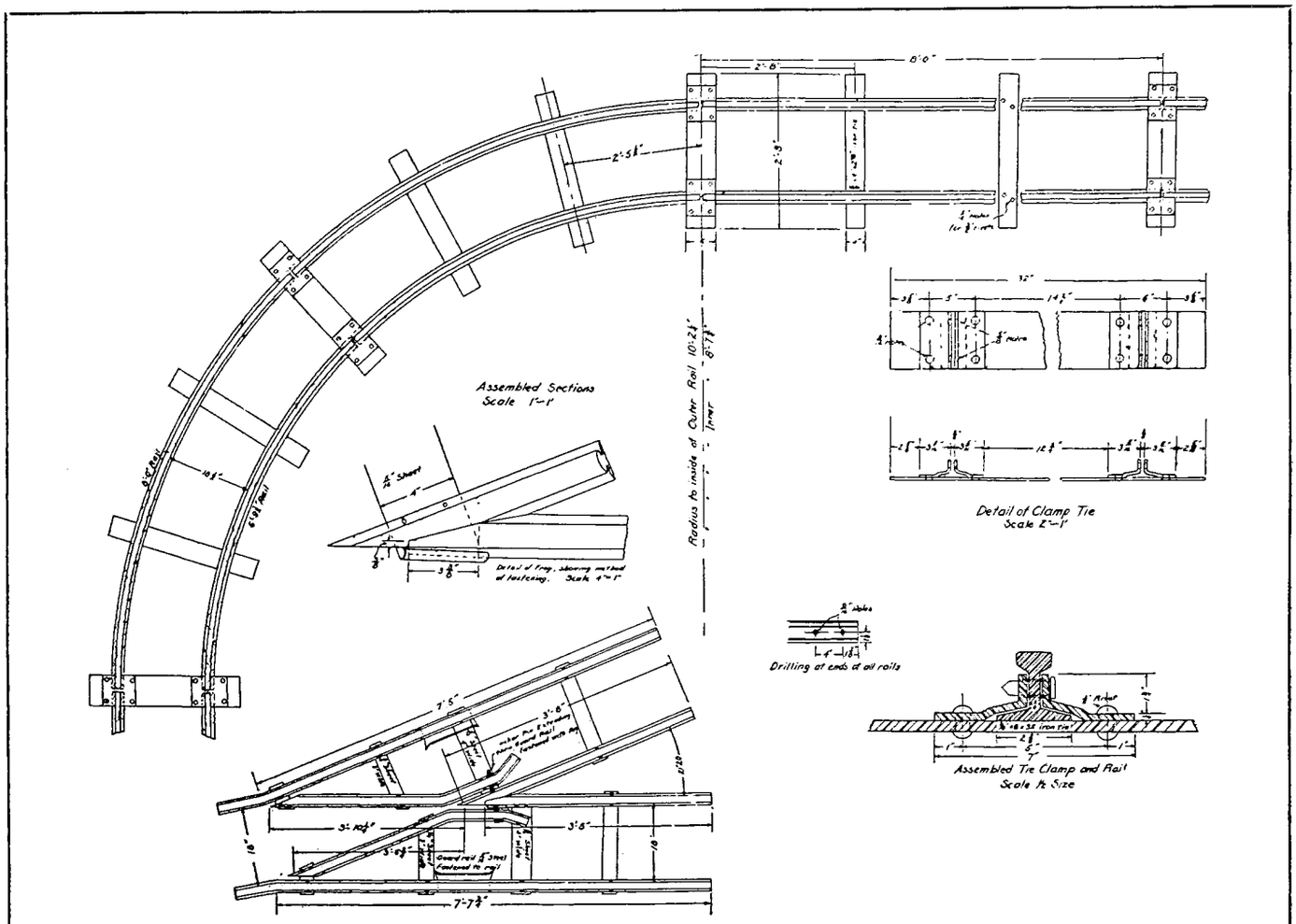
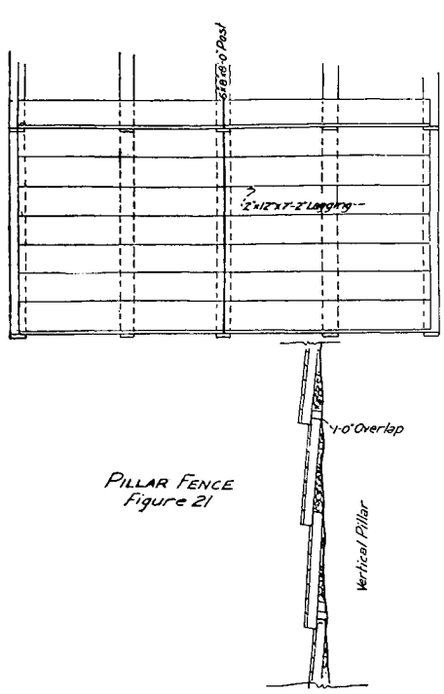


Figure 20. Sectional Stope Track



PILLAR FENCE
Figure 21

back at an angle of 40° with the floor. Ten-ft. cuts of ore are taken, measured normal to the floor, and as each cut is mined and shoveled, flooring is put in as shown in *Figure 22*, and kept about two sections ahead of the waste and 3 ft. from the back of the stope. The floor is established on a 40° slope, but after the cut is blasted it is found to be about 36° or 37°, due to the greater settlement of waste at the head of the incline.

When the incline is established to the hanging wall, a blasting chamber is timbered, using a single row of standard square-set timber and butt blocks 3½ ft. to the wall. This necessitates mining a slice 19 ft. above the grizzlies, carried flat to the point where it meets the incline. After the blasting chamber is completed, mining on the incline cut is started from the bottom and the face carried through to the crest, the section under the waste raise being carried ahead of the remainder and enough muck being drawn to keep the stope from choking up and to give room for the machine set ups.

After the cut has passed the waste raise on one side of the stope, the muck is pulled and the floor cleaned down and salvaged. The chutes and blasting chamber are raised 14 ft. above the grizzlies, and scabbed on the incline side, and the waste poured in from the level above. As the waste reaches the top of the square sets, flooring is started and the operation continues until the entire stope is filled. In wide stopes, mining can be made continuous; in narrow stopes this is not possible, the limit for continuous mining of 10-ft. cuts being about 80 ft. in width. *Figure 23* shows a section of a typical incline cut and fill stope.

In raising timber, it is possible to salvage the square sets in the blasting chamber and install them again as a blasting chamber between chutes after the fill is finished. By leaving them and using wing chutes, a solid line of grizzlies is established in the blasting chamber, and the muck can be handled cheaply. The cost of salvaging and extra handling of ore, balanced against new timber, is practically the same at the present time. Different labor or timber costs would probably swing the method from one to the other.

Detailed labor costs for this method in 1928 are as follows:

	Percent	Cost, ton
Miners	27	$0.19
Pluggers and carmen	34	0.24
Wastemen	8½	0.06
Timbermen	22	0.16
Miscellaneous	8½	0.06
Total	100	$0.71
Tons per man-shift (stope labor)		9.0
Pounds explosives per ton		0.55
Board feet timber per ton		5.9

OVERHAND SQUARE SET AND FILL STOPING

This method of stoping as used at the United Verde differs very little from that commonly used at other mines. With the exception of areas in the vicinity of old workings and fire stopes, it is used

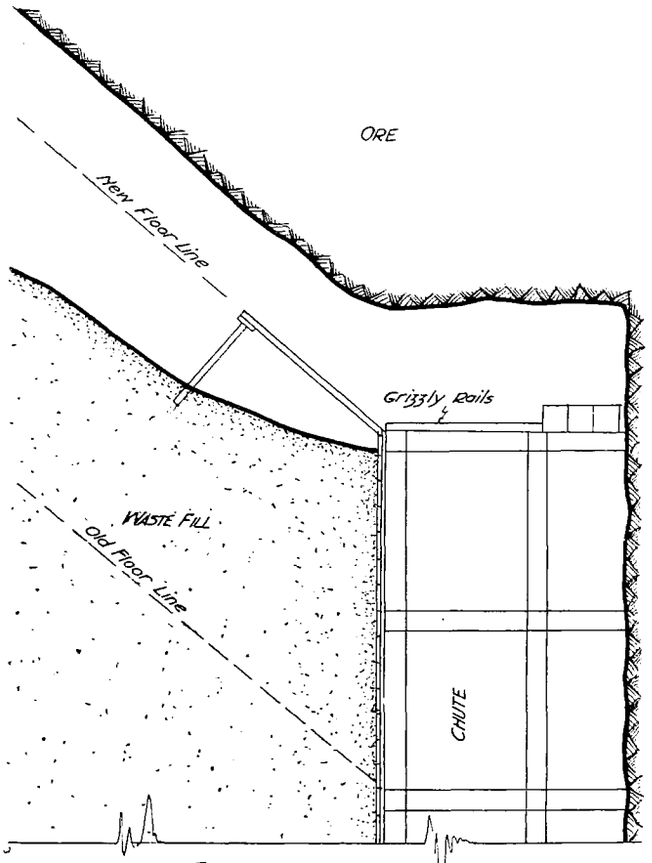


Figure 22
METHOD OF FLOORING INCLINE CUT & FILL STOPE

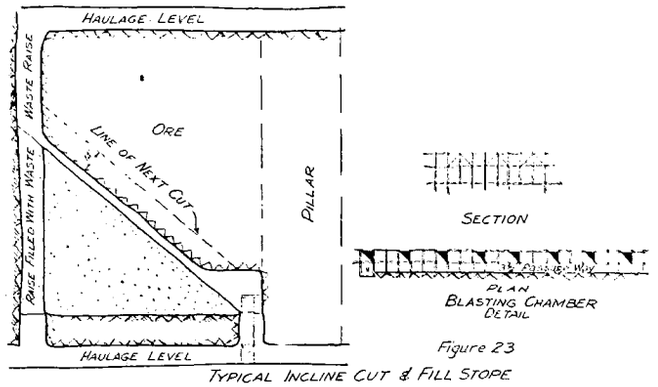
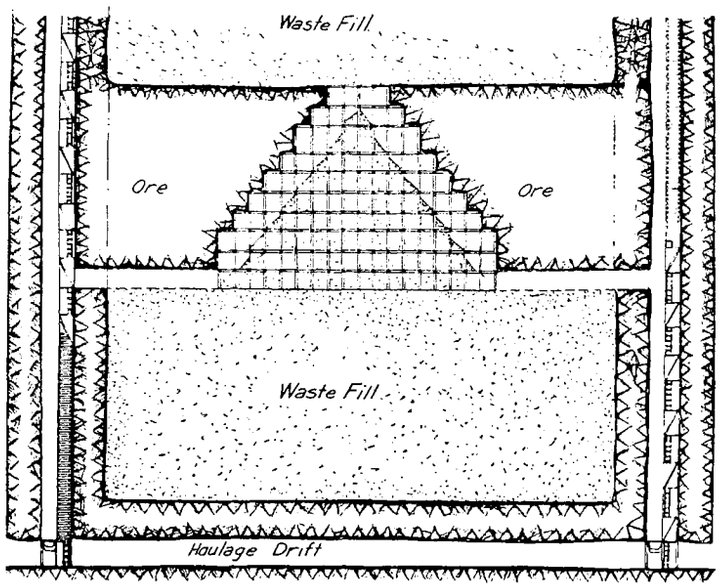


Figure 23
TYPICAL INCLINE CUT & FILL STOPE



Longitudinal Vertical Section
Figure 26

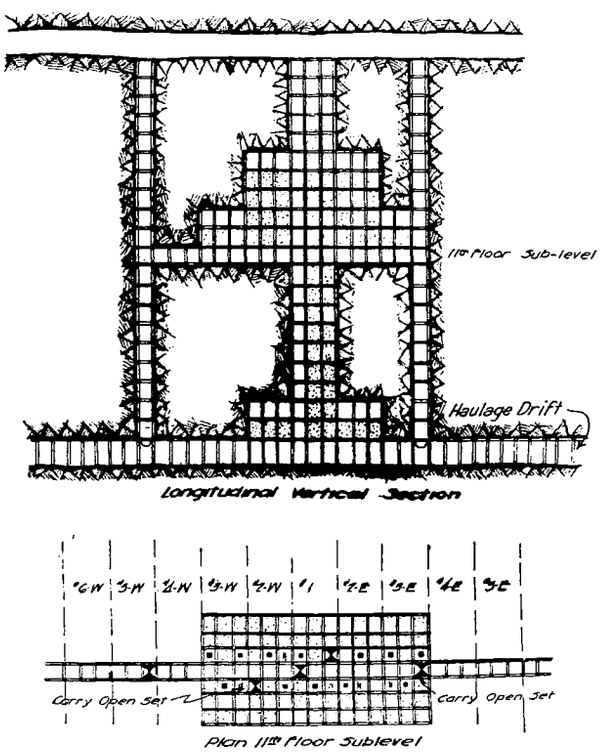
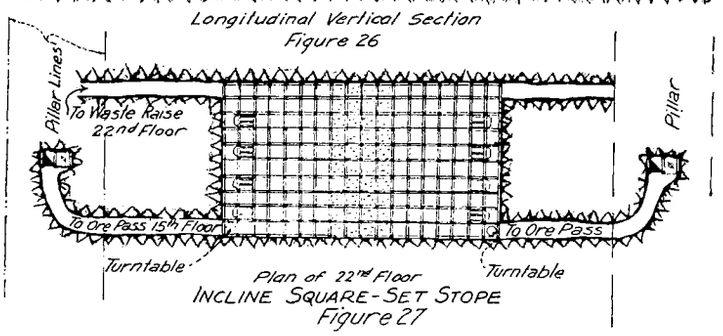


Figure 24
SQUARE-SET METHOD OF MINING
— VERTICAL PILLAR —



INCLINE SQUARE-SET STOPE
Figure 27

United Verde Copper Company

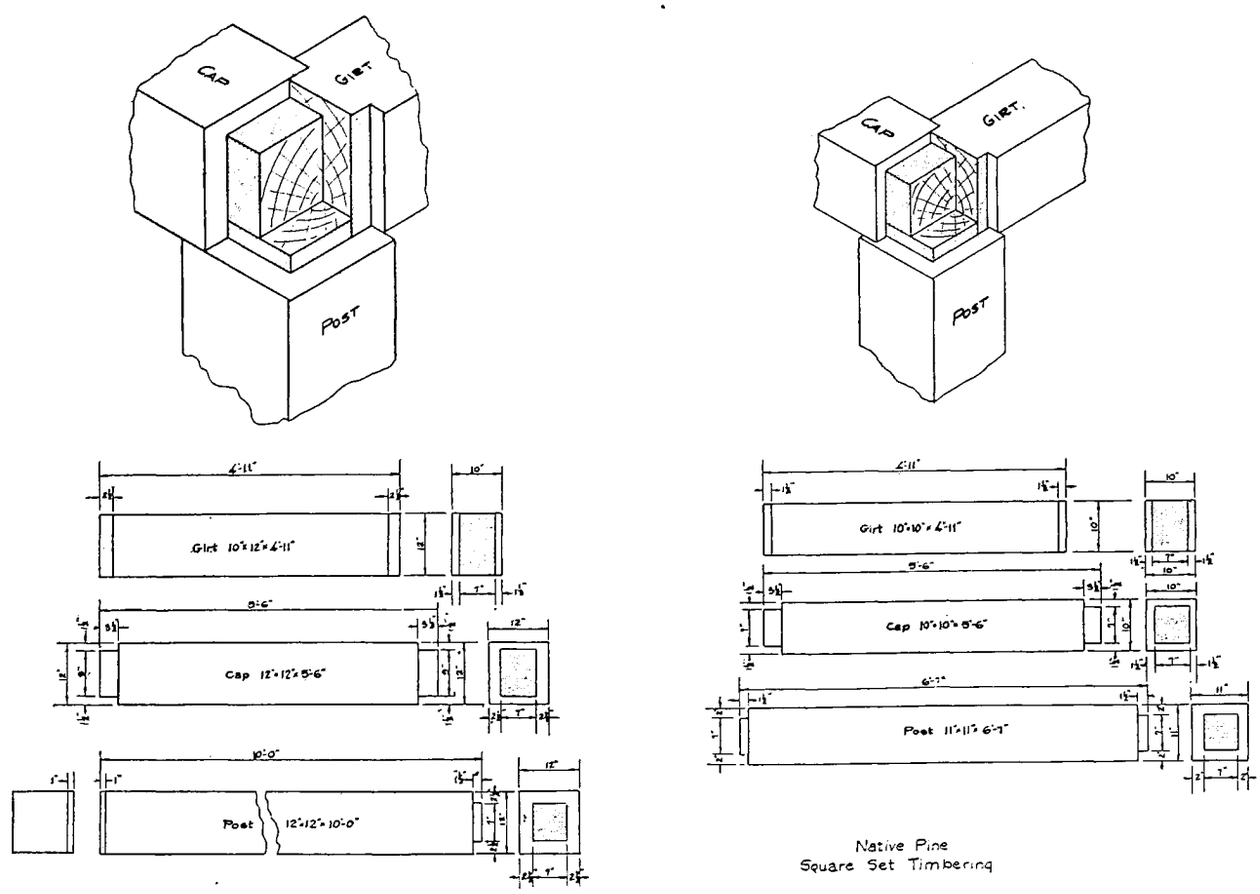


Figure 25

Gangway timbering—Square set stopes

chiefly in the extraction of the pillars left in horizontal cut and fill mining. These pillars may be either vertical or horizontal, and are usually quite heavy due to the waste pressure from the stopes that have been mined.

Areas to be square set are generally split into small sections that can be mined and filled rapidly. Ore outlines are definite, being projected from mined out floors of adjacent stopes. In cases of horizontal or level pillars, chutes from the cut and fill stopes are carried through the level pillars, a mat of old flooring is laid, followed by 2 in. by 12 in. by 11 ft. sills. This mat allows the sets to settle evenly when the stope begins to take weight. In the case of vertical pillars, the sills are put down in the same manner as in horizontal cut and fill stopes.

Vertical pillars may be divided vertically and gangways of standard square sets run on the first and eleventh floors, and silling on the second and twelfth floors. The sections of the lower half are carried slightly behind those of the upper half, as shown in *Figure 24*. Filling of vertical pillars is easily done by dumping directly into the waste raise on the level above. Filling of level pillars above the haulage level is accomplished by running a timbered drift directly under the sills of the stope above and connecting with the vertical pillar raise.

The miner does all his own timbering, receiving help from the mucker or pluggerman in setting up the machine and raising timbers. The average tons per man (direct labor in stopes only) is 4.7; the pounds of powder per ton is 0.6; and the board feet of timber per ton is 12.52. The timber consists of cap butting sets of dimensions as shown in *Figure 25*.

Detailed labor costs for this method are as follows:

	Percent	Cost, ton
Miners	24	$0.34
Pluggers and carmen	45	0.66
Wastemen	8½	0.12
Timbermen	19	0.27
Miscellaneous	3½	0.05
Total	100	$1.44

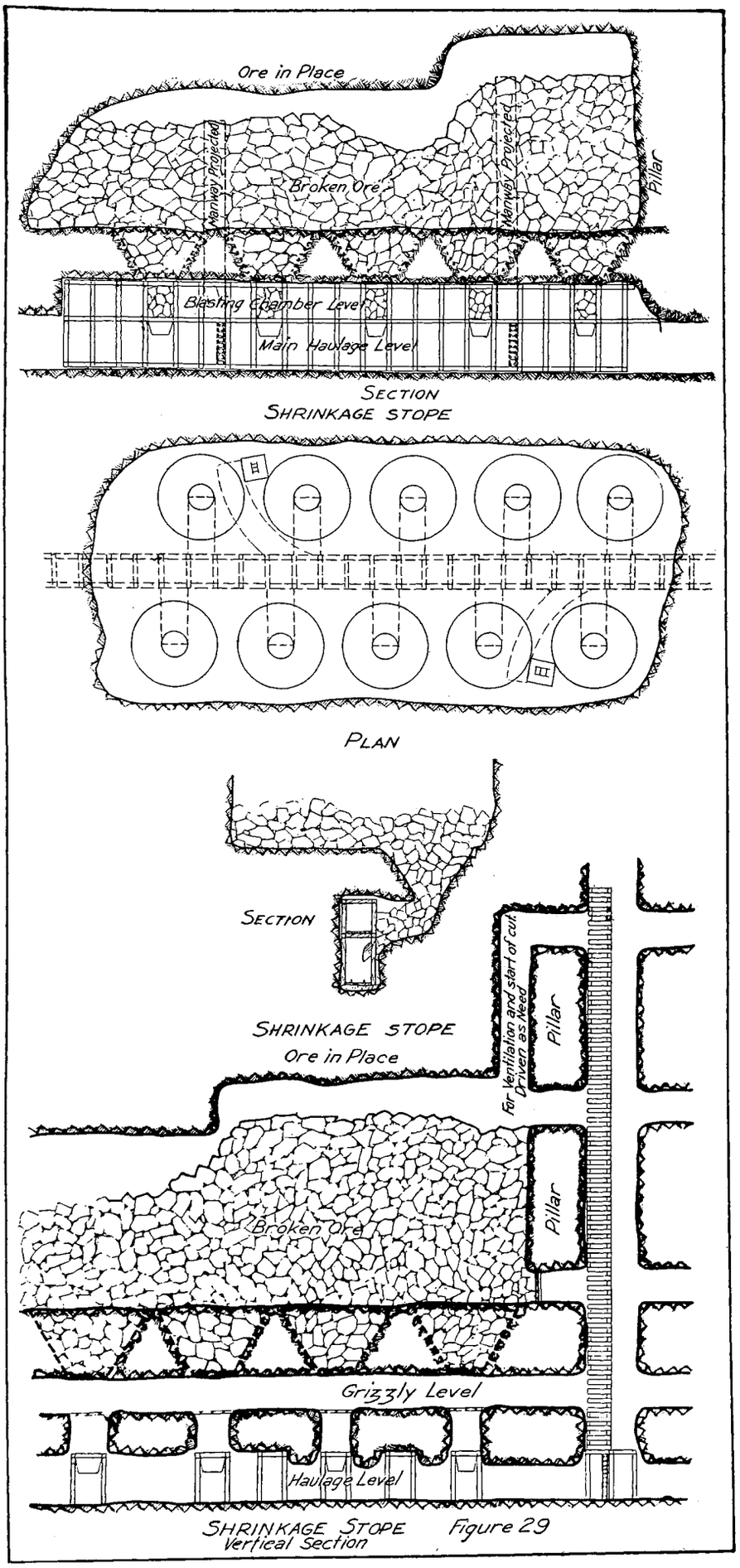
Incline Square Set Stoping

Incline square set is a method of overhand stoping with square set timbering and filling, but with the working floors carried inclined, as shown in *Figures 26* and *27*. It is applicable to heavy ground that must be closely timbered for support. A single or double cut may be taken, depending upon conditions. Hand mucking is practically eliminated and rock handling limited to barring through grizzlies and tramming. When used in extraction of horizontal cut and fill pillars, the old chutes are available and hand tramming eliminated. The greatest disadvantage of this method is the inability to easily sort waste rock from the ore. Open sets can be left between chutes for this purpose, but ore passes through grizzlies very readily and much waste is included.

Incline square set is still in the experimental stages, three stopes having been mined very successfully by this method. As yet all inclines have been used in the extraction of level pillars, the direction of the incline being along the contact of the orebody. It is planned to introduce this method in vertical pillars, however, and have the direction of the incline follow the dip of the orebody.

The preliminary work of establishing incline square set differs very little from that of flat square set. The first section is taken up as a flat stope, the width from footwall to hanging wall, and the length on the first floor being such that on each succeeding floor two sets in width can be dropped, one on each side, and the final floor under the sills of the stope above consisting of two lines of sets, thus forming two incline square set sections. Chutes are carried up through the fill until the incline is established, then they are dropped and filled with waste. The two sections are then mined as separate stopes and retreat from each other as the successive cuts are mined.

Where chutes are not available, development consists of a footwall drift from the ore pass, into which ore can be trammed from each cut, and a hanging



United Verde Copper Company

wall drift under the sills of the stope above, from which waste fill can be obtained. These drifts are shown in *Figure 27*. Sections up to 12 sets wide have been carried successfully in fairly good ground. In heavy ground it is advisable to carry three or four set sections. The length of the stope in most cases depends upon vertical pillars and the thickness of the level pillar.

Chutes are spaced in alternate sets, and grizzlies carried two sets above the tramming level until the full length of the stope is reached, to allow storage. Both grizzly men and trammers can work continuously without the usual delay when no storage is available.

On completion of the mining cut, the flooring is salvaged and raised one or two sets, according to the slice to be mined. Detail of flooring is shown in *Figure 28*, and has been found to be very satisfactory. The flooring is placed before waste filling starts. A timber crew consisting of a timberman and helper averages 12 sets per 8-hour shift, including salvaging old floor and installing the new.

Waste filling is exceptionally efficient as compared with flat square set, since there is but one turn from the hanging wall drift and very little track moving. The last floor under the sills of the stope above is back filled by hand shovelling.

The crew consists of one miner, who does all drilling and standing of square sets, with help from the mucker to set up and raise timber, one mucker and one pluggerman. When the cut is finished, a timber crew raises chutes and puts down flooring, and one or two waste trammers work on filling. In sections more than six sets wide, mining, timbering, and filling is continuous; in the smaller sections this is not possible, due to the difficulty of keeping ore and fill separated.

Detailed labor costs for this method are as follows:

	Percent	Cost, ton
Miners	35	$0.46
Pluggers and carmen	41	0.54
Wastemen	7½	0.11
Timbermen	13½	0.18
Miscellaneous	3	0.04
Total	100	$1.33
Average tons per man (stope labor)		5.0
Pounds powder per ton		0.6
Board feet timber per ton		11.0

Shrinkage Stoping

Ore areas in general are not applicable to shrinkage stoping. The footwall is either schist or porphyry, which will not stand well; ore continuity in the walls is erratic; offsetting is common; and no waste sorting is possible. Areas in the sulphide hanging wall are located occasionally and meet all the requirements, and are stoped by the shrinkage method.

In the development of a shrinkage stope, drifts are run on the ore haulage level at 50-ft. centers, if more than one is necessary. Chutes are spaced 25-ft. centers. Twenty feet above the haulage level a blasting chamber drift is driven which connects to all chutes. From the blasting chamber drift, draw raises to the stope are driven, and the silling of the initial cut started. These draw raises are spaced 25-ft. and 32-ft. centers, as shown in *Figure 29*. Grizzlies are 11 ft. by 6 ft., using 11-ft., 90-lb. rails with 11-in. openings. The capacity under each grizzly is from 15 to 20 tons, allowing continuous mucking in the blasting chamber with no delays from motor haulage. Manways are carried in each end of the stope, cribbed through the muck if there are no vertical pillars. Where vertical

pillars are laid out, the manways and service compartments are placed in the center of each pillar and connections driven to the stope as required. *Figure 29* shows the general plan of such a stope.

Ore is blasted in 7 or 10-ft. horizontal slices, using 10 to 12-ft. holes and blasting heavy. Blockholing is done in the stopes by pluggermen, who also act as helpers to the miner in set-ups and loading, etc. Pluggermen in the blasting chamber also do what blockholing is necessary to run the ore through the 11-in. grizzlies.

A level pillar of 30 to 35 ft. is left under stopes above. The back is carefully trimmed and barred, and the stope pulled empty of muck. In case of good backs and walls, the pillar above the blasting chamber is removed by underhand stoping and scraper mucking. The blasting chamber pillar is mined from one vertical pillar line, retreating to the opposite pillar, using the blasting chamber drift to work from. There is very little exposure of men under the high backs until all mining and mucking is finished. Then timber crews go into the stope and lay sills as in cut and fill mining, and waste is poured into the stope from the level above. As the waste nears the pillars, a fence is erected 10 or 12 ft. vertically, and filling continues until the waste nears the top of the fence, when another section is raised, and so on until the entire stope is filled to within 7 ft. of the back. The level pillar is then removed by one of the square set methods.

Detailed labor costs for this method are as follows:

	Percent	Cost, ton
Mining	41½	$0.19
Pluggers	41½	0.19
Timbermen	4	0.02
Miscellaneous	13	0.06
Total	100	$0.46

Rock Drills

The inventory of rock drills for underground operations totals 180 Leyners, 167 stopes, and 81 jackhammers. Although this list includes numerous types and manufacturers, the following machines are standard equipment:

Drifts and stopes	Chicago Pneumatic CP-6
Raises in soft ground	Ingersoll-Rand CCW-11
Raises in hard ground	Waugh 39 & 77-H special, with other machines now being tried
Plugging & bulldozing	Sullivan DP-33 and Waugh 11
Sinking	Waugh 37

The average repair cost per drill shift is $0.41 for jackhammers, $0.92 for Leyners, $0.92 for stopers in soft ground and $2.50 for self-rotated stopers in hard ground. The average daily number of machine shifts in underground mining operations is 140. All machines are sent to the 500-level drill shop for repairs. After repairing a machine, it is tested and sent back to the same level from which it came. Each machine bears a serial number and a close account is maintained on each machine, so that it is possible to tell the exact cost of repairs for any particular drill.

Practically all drills are equipped with the Waugh LO-3 line oiler, a short length of hose connecting the oiler to the drill. Water tubes are made in the drill repair shop by means of two dies of simple design and operated by a small pneumatic gun. The tubing is heated in a small forge for drawing down the ends, whereas the buttons are formed cold.

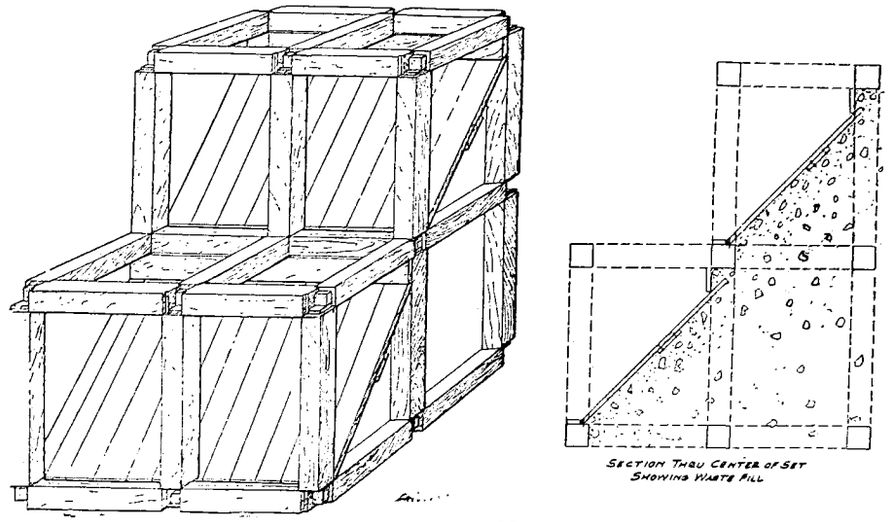


Figure 28
Incline Square Set—Showing Method of Flooring

Drill Steel

Two sizes of drill steel are used, ⅞-in. quarter octagon for jackhammers and stopers in soft ground, and 1¼-in. hollow round for all heavy type drills and self-rotating stopers. It is planned to replace the ⅞-in. quarter octagon section with 1-in. steel for use with the harder hitting stopers. The double taper cross bit, with 5° and 14° taper is standard for all steel. Starters are 30 in. long, with 1⅞-in. gauge. Changes are in multiples of 10 in. with 1/16-in. gauge change. Column bars and arms are made from 3-in. extra heavy pipe.

Approximately 85 tons of steel are used annually, or 0.223 lbs. of steel per ton of ore mined. About 36,000 pieces of steel are kept in circulation.

Steel Sharpening

The dull steel from the mine is brought out to the steel sharpening shop on standard steel cars during day shift, and distributed in racks according to lengths, each sharpener handling only two or three lengths of the same size steel. During the day the nippers fill their orders for the mine from revolving steel racks or excess storage racks outside the building, and this steel is delivered to the various levels on afternoon shift.

The dull steel is heated in an oil fired furnace, which is built in the mine shops and equipped with Gilbert & Barker 1½-in. single jet burner. Oil consumption is 3 gals. per hour of 24° Baumé semi-Diesel oil. The air is furnished at 1.25 lbs. pressure by a General Electric high speed centrifugal blower.

The bits are forged at 1,900° F. in Ingersoll-Rand sharpeners, and tempered by Gilman CE-21 heat treating machines. Bits are quenched at 1,400° F. and range in hardness from 550 to 600 in Brinell scale. Shanks are forged in a No. 8 Waugh sharpener at a temperature of 2,100° F., and hand tempered at 1,600° F. in Houghton No. 2 quenching oil. The hardness of shanks is maintained at 375 to 400 Brinell hardness.

All furnaces are equipped with thermocouples and indicating pyrometers. The equipment is checked daily by a precision potentiometer. An improved America Model Brinell hardness testing machine is used in testing hardness of bits, shanks, dollies, and forging blocks.

The capacity of the shop is 2,500 pieces of steel per 8-hour shift. Approximately 1,800 to 2,000 pieces are handled daily by 24 men. The breakage averages 2 percent on 1¼-in. hollow round and 8 percent on ⅞-in. quarter octagon. The cost per bit sharpened is $0.13, which represents a cost of $0.06 per ton of ore mined.

Hose

Two sizes of hose are used, viz, ½-in. for water and 1-in. for air. All hose is provided with a standard lugged coupling and spud which fits all types of machines and 1-in. pipe lines. All damaged hose is brought to the surface for repairs, and mended by means of a specially designed facilitator, which cuts hose, inserts nipples, wraps the joint with wire, and tests the hose, all by means of compressed air. A mechanical device forms the wire loops for wrapping. The nipples are made from scrap pipe, tapped at the ends and recessed in the center.

The average life and monthly maintenance costs of the two sizes are given below:

Size	Life	Total cost, mo., 50-ft. length
½ in.	16 mos.	$1.24
1 in.	26 mos.	1.26

Handling Explosives

Two strengths of explosives are used in underground mining, 50 percent gelatin in massive sulphides and porphyry and 35 percent gelatin in schist and for plugging boulders. All stick powder comes in red wrappers, which aids in locating missed holes.

Explosives are stored in a 20 by 50-ft. magazine located 3,000 ft. from the portal of Hopewell Tunnel. Powder is delivered to the shaft in a special insulated powder car, and distributed to the various levels on grave-yard shift. Sufficient powder for one day's operation is stored in small magazines on each level. These magazines are located conveniently with respect to the tool rooms to enable the powdermen on each level to look after the tool room also.

Fuse

A smooth, black finished, cotton countered safety fuse is received in spools of 3,000 ft., and cut to proper lengths by a mechanical (*Continued on page* 39)

United Verde Copper Company

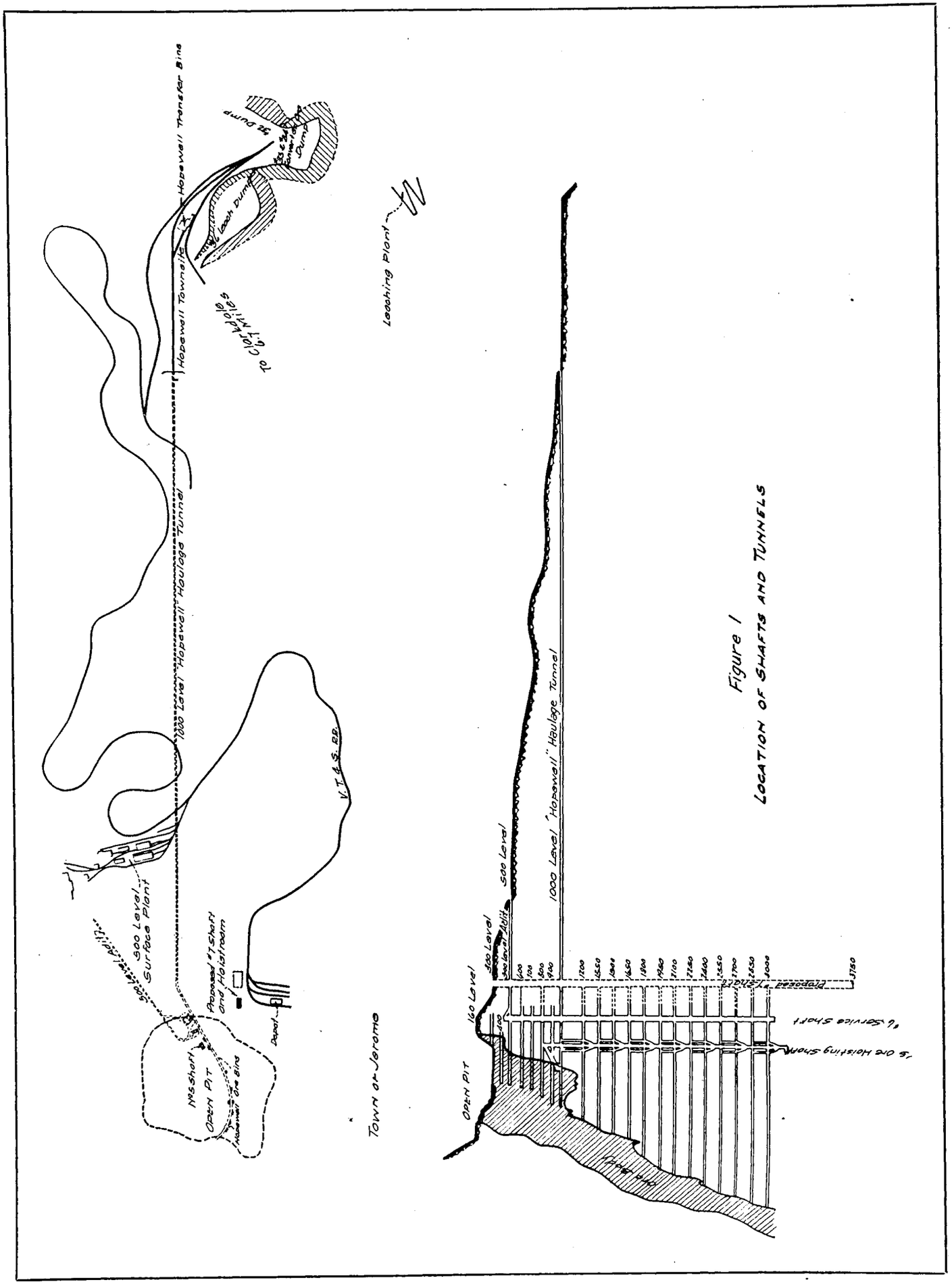


Figure 1
LOCATION OF SHAFTS AND TUNNELS

United Verde Copper Company

Shaft Practice and Hoisting Methods at the United Verde

By C. E. Mills
ASSISTANT MINE SUPERINTENDENT

THE general arrangement of shafts and tunnels at the United Verde mine is shown in *Figure 1.* Two shafts serve the mine, No. 5 handling all ore from below the 1,000-ft. level, and No. 6 handling all men and supplies. There are two main tunnels. All supplies are handled through the 500-ft. level adit to the collar of No. 6 shaft, whereas all ore is transported through Hopewell haulage tunnel on the 1,000-ft. level.

No. 5 Shaft:

No. 5 shaft is used exclusively for handling ore. It extends from the 800-ft. level to the 3,000-ft. level, and is now being sunk an additional 600 ft. The shaft is of reinforced concrete, with two 5 by 5-ft. hoisting compartments, and a 5 by 5-ft. manway and pipe compartment, which is also used in sinking operations.

No. 5 hoist room is located on the 1,000-ft. level. It is 47 by 81 ft. in section by 22 ft. in height, and is lined with reinforced concrete. A specially designed 20-ton crane facilitates the handling of hoist room equipment and repairs.

No. 6 Shaft:

No. 6 shaft is used entirely for service, and extends from the 400-ft. level to the 3,000-ft. level. Above the 1,950-ft. level it is concreted solid, and below with 2½-ft. rings on 6 ft. centers. The cage compartment is 8 by 13 ft., the manway 4 ft. by 9 ft. 4 in., and the counterbalance compartment 4 ft. by 3 ft. A sub-level at the collar of the shaft on the 500-ft. level permits loading and unloading of both decks at one time when handling the shift. This shaft is limited to its present depth by the hoisting equipment.

No. 6 hoist room is located on the 500-ft. level. It is 44 by 45 ft. in section and 26 ft. in height. Opening off of one side is a motor generator room 20 by 32 ft. by 13 ft. in height. Both rooms are lined with concrete, as is the cableway which extends from the hoist room to the head frame at the 400-ft. level, at an angle of 42 degrees.

Tunnels:

No. 6 shaft is connected with the mine surface plant by the 500-ft. tunnel. It is 1,600 ft. in length, 1,200 ft. being timbered with 10 by 10 in. Oregon pine. The sets are 8 by 9 ft. in the clear, and spaced on 5.5 ft. centers. All timber and lagging has been partially covered with diamond mesh wire and gunited for fire protection. A standard gauge 60-lb. track with intermediate rail for 18-in. gauge permits the delivery of steel and timber on specially designed trucks to No. 6 shaft without rehandling at the collar or on the stations below. The shift is handled to and from the collar in standard gauge enclosed cars, each car seating 26 men.

Hopewell Tunnel, on the 1,000-ft. level, is the main ore haulage level, and connects the underground and shovel-pit ore bins with the outside transfer and storage bins. (Refer to *Figure 2.*) From these bins the mill and direct smelting ore is shipped to the Clarkdale smelter, over the Verde Tunnel and Smelter Railroad, a distance of 6.7 miles. Excess waste from shovel operations and low grade leach ore is transferred to Western dump cars and placed on dumps for recovery of the copper content by leaching operations. Low grade silicious ore is transferred through the Hopewell bins and placed on reserve dumps for future smelting operations.

The tunnel is 10 by 13 ft. in cross section in the untimbered sections; 2,700 ft. is timbered with 10 by 10 in. Oregon pine sets at 5.5 ft. centers. The sets are 9 ft. high, with a 10-ft. cap and posts battered to give a width of 12 ft. at the sill.

No. 5 Shaft Construction Methods:

No. 5 shaft was completed to the 1,950-ft. level in September, 1918. Successive lifts have been to the 2,400-ft. level, 3,000-ft. level, and the shaft is now being sunk to the 3,600-ft. level, with an additional 150 ft. in depth to take care of the loading pockets and spill pocket below the 3,600-ft. level.

Figure 3 shows the general method of sinking No. 5 shaft. A 15-ft. pentice is left below the bottom level by sinking on the manway side and widening out below. The shaft is timbered with 8 by 10-in. Oregon pine shaft sets so designed as to permit concreting by the use of 2-in. plank panels which set in between the sets as forms for the concrete. Three and one-half by 5½-in. O. P. guides are carried in the manway side for the skip or crosshead.

The crew consists of three shaftmen, with one leader, top man and hoistman. A 32-hole round is ordinarily used in fresh porphyry, and the cut placed to take advantage of the ground. Electric delay detonators (1 to 10) are used and shot with 440-volt a. c. current. The following data is on the 600-ft. section sunk below the 2,400-ft. level:

Drilling speed—inches per minute	10
Number of holes	32 to 35
Advance per round—feet	4.5
Advance per shift	1.0
Powder used—sticks 1⅛ x 8 in. 50%	260
Powder cost per foot of advance	$5.20
Labor—mining and timbering	$31.85

Former sinking practice was to use a 20-cu. ft. sinking bucket suspended below a wooden crosshead. The hoist used was a 75 hp. Vulcan, with a rope pull of 5,000 lbs. at a rope speed of 500 ft. per minute. The collar of the shaft was provided with the usual flop doors to prevent spill from going down the shaft. The topman would dump the bucket into a mine car and transfer it to the shaft loading pocket.

Upon completion of the sinking operations, the bucket was replaced by a 38-cu. ft. self dumping skip, and the muck from cutting stations and skip loading pockets was drawn by gravity through a chute direct into the skip through a gap in the guide. The skip was provided with removable rollers and when hoisting, muck was dumped by means of a curved guide into a dog hole leading down into the main loading pocket.

In sinking No. 5 shaft below the 3,000-ft. level, the bucket has been discarded in favor of the skip as described above. The guide shoes have been extended to permit the skip being lowered to the bottom of the shaft. The muck is shoveled into an 18-cu. ft. pan which is raised by means of a small air hoist and dumped into the skip. The Vulcan 75 hp. sinking hoist (*Figure 4*) had insufficient power to handle the skip fully loaded, and was replaced by a Llewellyn 125-hp. dragline hoist with forward drum removed. The hoist is also provided with an auxiliary brake operated by compressed air, this extra brake having been

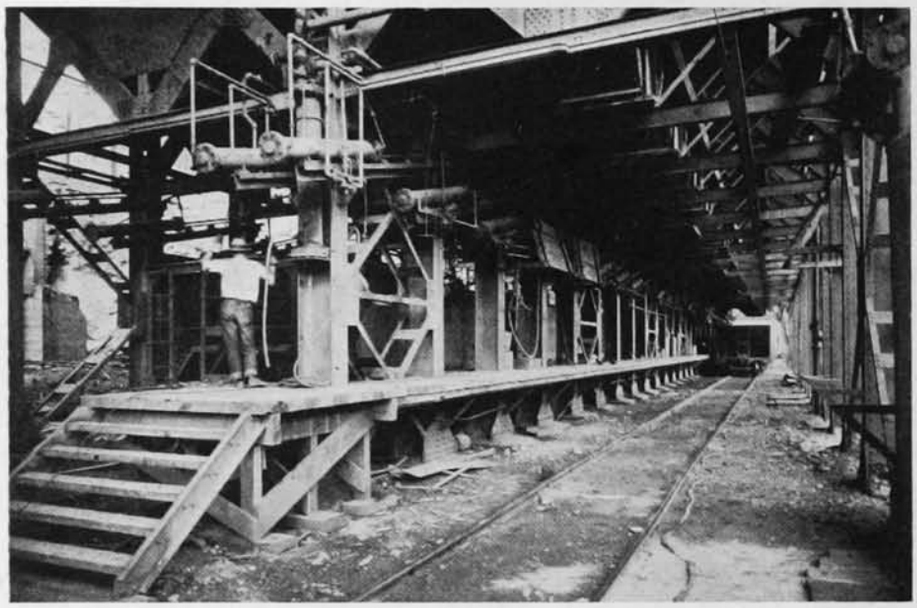


Figure 2. Outside Transfer and Storage Bins, Hopewell Tunnel

installed in the mine skips as an added safety measure. The cable is ⅞-in. 6 by 19 special plough steel.

Concreting No. 5 Shaft:

Several methods of concreting have been tried in this shaft. The first section from the 800-ft. level to the 1,950-ft. level was concreted by means of steel forms 5 ft. high, made up of 3/16-in. plate, backed with 2½ by 2 by 5/16-in. angles and split along the center line of the shaft. Concreting started at each level and progressed upward to the level above. This section of the shaft was raised, so that no shaft timbers were required. The second section, from the 1,950-ft. level to the 2,550-ft. level, was sunk and timbered with 8 by 8-in. shaft timbers. The shaft timbers were removed as concreting advanced, the walls being formed of 1-in. sheathing on 3 by 3-in. studs. The third section, from the 2,400-ft. level to the 3,150-ft. level, was timbered with 8 by 10-in. special shaft sets on 6-ft. 8-in. centers, with the outside face of the timbers set to finished concrete lines. Two-inch plank panels, backed by 3 by 3 by ⅜-in. angles, were set between the shaft timbers, and held in position with angle irons spiked to the wall plates. Wooden boxes were placed above the wall plates where they passed through the curtain walls to permit reclaiming the shaft sets after concreting. The guide bolt sockets were lined with plumb lines, and a uniform vertical spacing maintained by strap iron spacers which remained in the concrete. This latter method of forming the shaft proved the most economical and fastest method used.

A permanent concrete mixing plant is located at No. 6 shaft below the 500-ft. level station. Standard gauge mine cars deliver sand and gravel to the shaft and dump into bins beneath the track. Small measuring pockets of the proper size to give a 1:3:5 mix discharge into a 9-cu. ft. mixer, which delivers the batch to a 5-in. standard pipe in the manway compartment.

Two methods have been used in transferring the mixed concrete over to No. 5 shaft. For the section from the 1,950-ft. level to the 2,550-ft. level, the concrete was transferred in a box type mine car holding two batches and provided with a 6-in. pipe discharge to empty the batch into a hopper at the upper end of a 4-in. concrete column in No. 5 shaft.

In concreting the shaft below the 2,400-ft. level, the concrete was transferred from No. 6 shaft to No. 5 shaft on the 2,400-ft. level through a horizontal 6-in. pipe line a distance of 460 ft. by means of compressed air. The receiving hopper or gun was built in the local shops. The hopper and the top was equipped with a quick operating conical plug valve. Compressed air was admitted above the batch by a 1-in. air connection, and into the tee on the bottom discharge by a 2-in. air line. With air pressure at 90 lbs., two batches were shot at one time, and concrete placed at the rate of 8 cu. yds. per hour.

Considerable segregation occurred in dropping the mixed concrete a vertical distance of 1,900 ft. and through the 460 ft. of horizontal line to No. 5 shaft, so that it was necessary to install a remixing box in the No. 5 shaft column before placing the concrete in the shaft forms. This scheme worked out quite well, although it was necessary to keep close watch on the shaft crew to prevent them from diluting the mixed concrete with raw water at the compressed air gun.

Twenty-one men made up the three shaft crews; 18 shaft miners, 2 mixed men, and 1 man at the receiver. The total cost per cubic yard of concrete placed was $22.50.

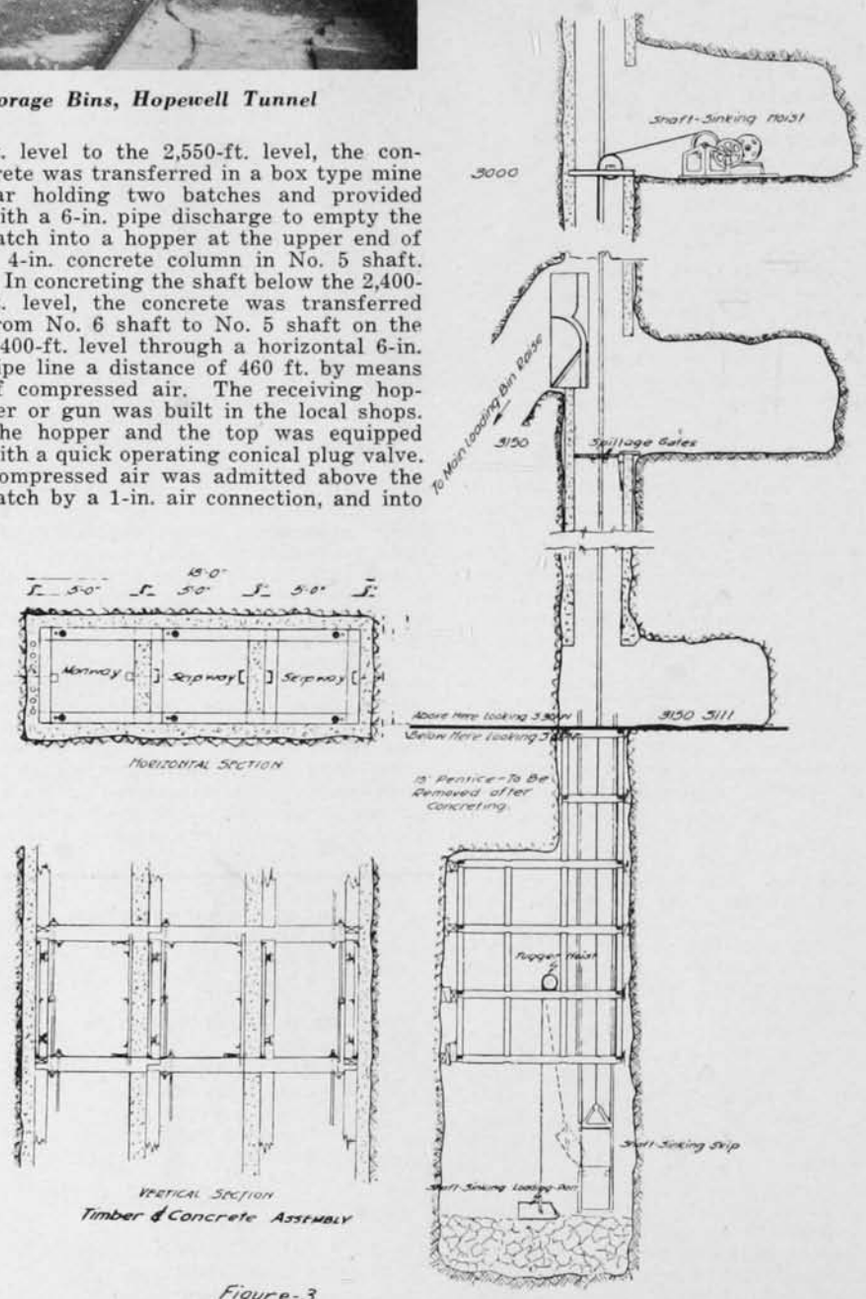


Figure-3
SHAFT-SINKING LAYOUT #5 SHAFT.

United Verde Copper Company

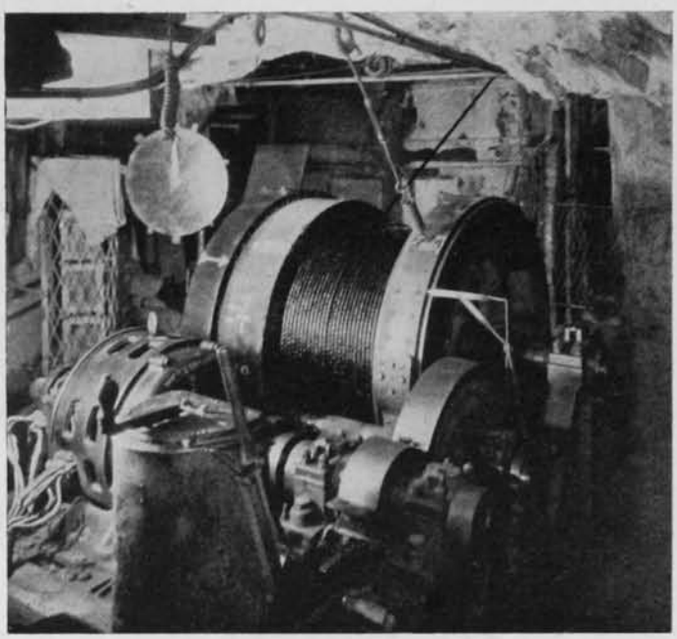


Figure 4. No. 5 Shaft-Sinking Hoist

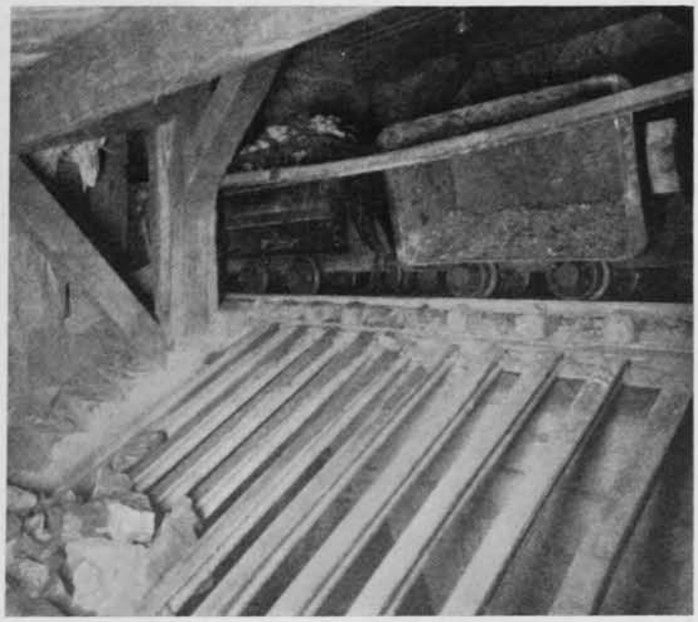


Figure 6. No. 5 Shaft Ore Pocket Grizzley

Loading Pockets:

The level interval above the 1,000-ft. level is 100 ft. One 200-ft. interval below the 1,000-ft. level proved this vertical distance excessive because of high chute repairs in the stopes and added raise development costs. Below the 1,200-ft. level the interval is 150 ft. Skip loading pockets are constructed at 300-ft. intervals, or below every alternate level.

Loading pockets, *Figure 5*, are of 500 tons capacity, and arranged according to general practice in the mines of the Southwest. The upper gate is of the undercut plate type, while the cartridge gate is of the undercut arc type. Both gates are operated by compressed air cylinders. The bottom of the pocket forms a chute to catch spill and fines from the loading pocket above, and a small hinged gate permits this muck to be drawn direct into the skip when cleaning the pockets. The air lines leading into the pockets are provided with a valve locked with a padlock, so that no one but the skip loaders can operate the gates.

The pockets are located 75 ft. below the sill, and are connected to the level by raises 8 by 11 ft. in section, which provide adequate storage. The pockets are in duplicate, one on each side of the shaft, to take care of the two ore classifications, namely, mill and direct smelting.

Figure 6 shows the grizzlies above each pocket. The grizzly bars are inverted 90-lb. rails, annealed and faced with a 1 by 5-in. wearing plate. The inclined rails are spaced to give a 9-in. opening, and the lower horizontal rails 12 by 20-in. openings. This construction permits an entire train to be dumped without stopping to bulldoze boulders. Men working on the grizzlies are required to wear safety belts attached to a sliding ring on a cable strung between two eye bolts at either end of the grizzly chamber. Intermediate levels without loading pockets are equipped with the same type of grizzly pockets, but the raises lead to the level below, and the ore is passed through a small wing raise to the skip pockets.

The cost of a single loading pocket as shown, together with 75 ft. of raise to the level and complete with grizzly is as follows:

Loading pocket	
Excavation and concrete work	$5,498
Steel work, gates, and machine work	7,858
Grizzly excavation and steel work	1,594
Total	$14,950

Spill:

The massive sulphide ore at the United Verde is very heavy and abrasive, and necessitates rugged construction on all ore handling equipment. The spill from loading pockets will average 6 tons on a daily tonnage of 2,000 tons. This amount of spill increases the wear on guides and cuts into the shaft walls. During nine years of operation, the 10-in. curtain walls were worn in places to 5 or 6 in., and exposed the reinforcing. After considering several methods of rebuilding these walls, guniting was resorted to and afforded a speedy and satisfactory method of repair with little or no delay to hoisting operations.

A heavy hinged bulkhead or deflector has been constructed in the hoisting compartments at the 1,950-ft. level to divert the spill to the ore pockets. This deflector is electrically operated by remote control from the engineer's platform, and a set of pilot lights indicate the open and closed position. The portion of spill which reaches the bottom of the shaft is drawn through standard chutes into an 18-cu. ft. mine car and hoisted through the manway compartment to the level above.

Although it is not desirable to have any obstructions in the main hoisting compartments, the question of spill is so serious that in the future it is planned to equip all loading pockets with deflectors directly below the skip position to divert the spill into small storage pockets. These deflectors will be operated by the skip loaders, and a system of pilot lights on the hoist operator's platform will indicate the open position of these doors.

Hoisting:

No. 5 hoist, *Figure 7*, is an Allis-Chalmers, double drum, single reduction, geared type, driven by a 650-hp. d. c. motor at 500 volts, and a normal speed of 300 r.p.m. Power is supplied to the hoist motor by a 600-kw. flywheel motor generator set, with 20-ton flywheel. The drums are cylindrical, 10 ft. in diameter, with a 5-ft. smooth face and will hold 2,500 ft. of 1⅜-in. extra plow steel, 6 by 19 hoisting rope in two layers. Each drum has a double disc multiple arm friction clutch. The brakes are of the conventional parallel motion, post type, released by oil pressure and set by a dead weight. The safety features include an overspeed and overwinding device, in addition to limit switches in the shaft.

The skips, which operate in balance, are of 112-cu. ft. capacity, holding 7.8 tons of iron or 6.5 tons of schist ore. They are of rugged construction and weigh 7 tons. The rope speed is 900 ft. per minute. The average loading time is 7 seconds.

The ore is hoisted to the 800-ft. level unloading pocket, where it is dumped through an electrically operated selector to one of three ore bin raises, depending on the class of ore or waste being handled. This selector is controlled from the hoistman's platform, and colored pilot lights indicate the position of the selector.

These ore bin raises lead to the 1,000-ft. level ore bins, which have a capacity of 1,000 tons each. The design and construction of these bins is shown in *Figure 8*. The ore is drawn through these bins into 40-ton standard gauge cars and transported to the outside bins through Hopewell Tunnel.

No. 6 Shaft Construction:

No. 6 shaft, *Figure 9*, was raised in full section (14 ft. 4 in. by 14 ft. 4 in.) from level to level, using a double cribbed manway. The walls were checked by the engineers every second round and in contract work the contractors were penalized 10 cents per cu. ft. for all overbreakage.

United Verde Copper Company

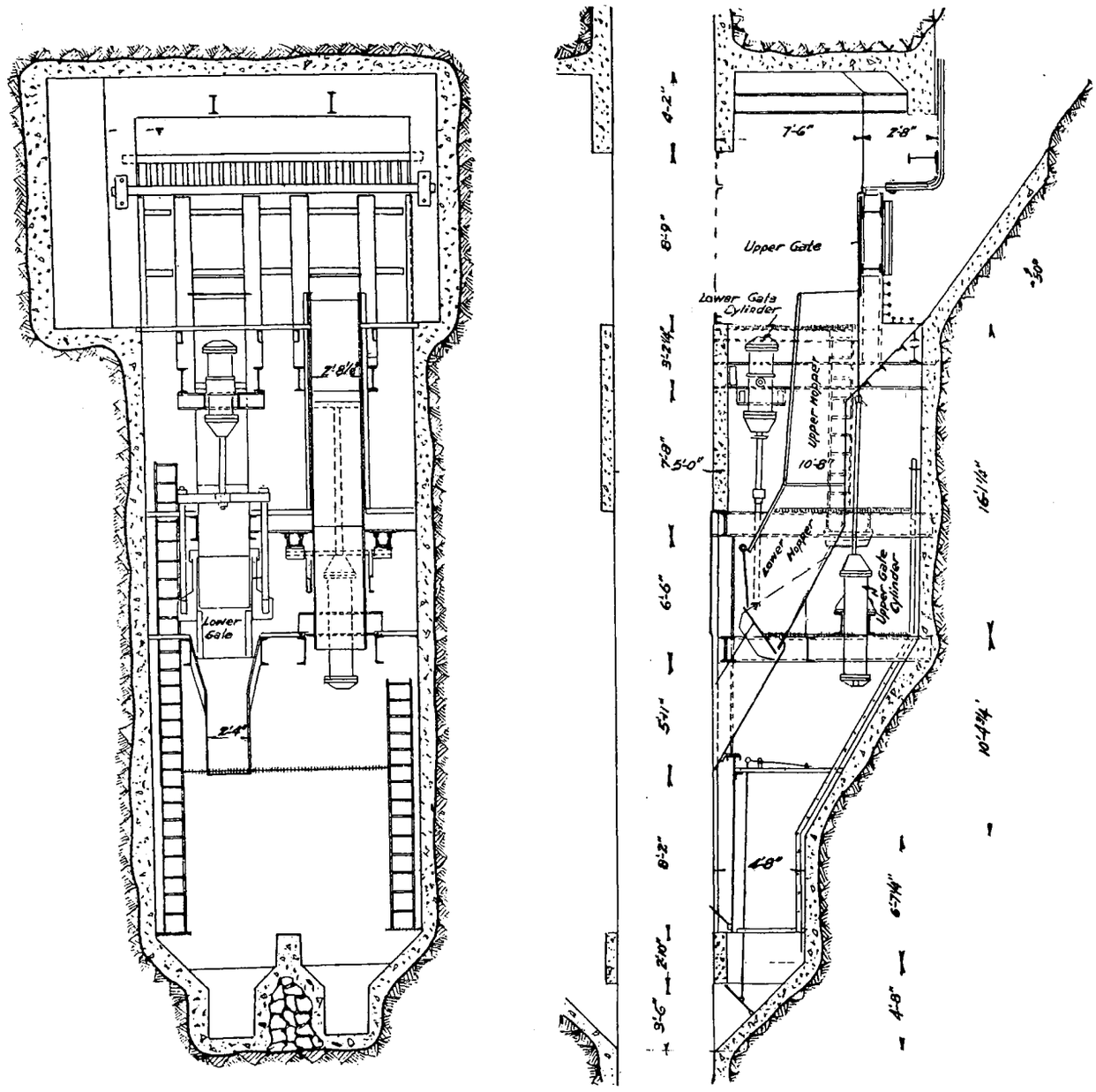


Figure 5
— No. 5 Shaft Skip-Loading Pocket —

The shaft above the 1,950-ft. level was concreted solid, using square sets and wooden forms. The muck was drawn down 20 to 50 ft., depending on the ground. Sills, square sets, and forms were then placed, and concrete poured, using the mixing plant already described.

The section of shaft below the 1,950-ft. level was in good ground, requiring little support, and the concrete placed in 2½-ft. rings at 6-ft. centers. (Refer to *Figure 10.*) The chief difficulty in developing the ring method of concreting was the problem of hanging forms and sealing the forms to the irregular ground line. This was solved by suspending a sill frame of 4 by 10-in. material from the steel form above by means of adjustable hanging rods with brackets to support the sills. The sills are leveled, lined, and sealed to the wall by 1-in. ship lap laid at right angles to the rock line and scribed to fit the irregular walls. If the walls are over 2 ft. from the form, it is necessary to use additional bracing below the sheathing.

The reinforcing steel consists of six ¾-in. bars, with ⅜-in. vertical rods spot welded to make an easily handled unit. The reinforcing is next placed on the sills and overlapping corners wired together. The eight sections of steel forms are then lowered from the top or fifth ring above, and held in place on the sill by pins. The corners fasten together with gusset plates and the centers are supported by angle braces. The ring is then checked for alignment, bolts set for steel work by templates, and the ring poured.

As each ring is finished, the muck is drawn down, the walls trimmed, the cribbing stripped, and the cycle repeated. Eighteen men, working on three shifts, will pour a ring every day, except at stations which require special form work. After each level is concreted, the steel work and ladders are placed and the guides installed prior to removing the pentice.

Data on raising the lower portion of the shaft is as follows:

Average depth of hole	feet..	5
Average number of holes		44
Total footage drilled per round	feet..	220
Average advance per round	do....	4
Average advance per shift	do....	1½

The following tabulation shows the comparative cost between the ring and solid concrete method, also the detailed costs on the ring method per foot of shaft:

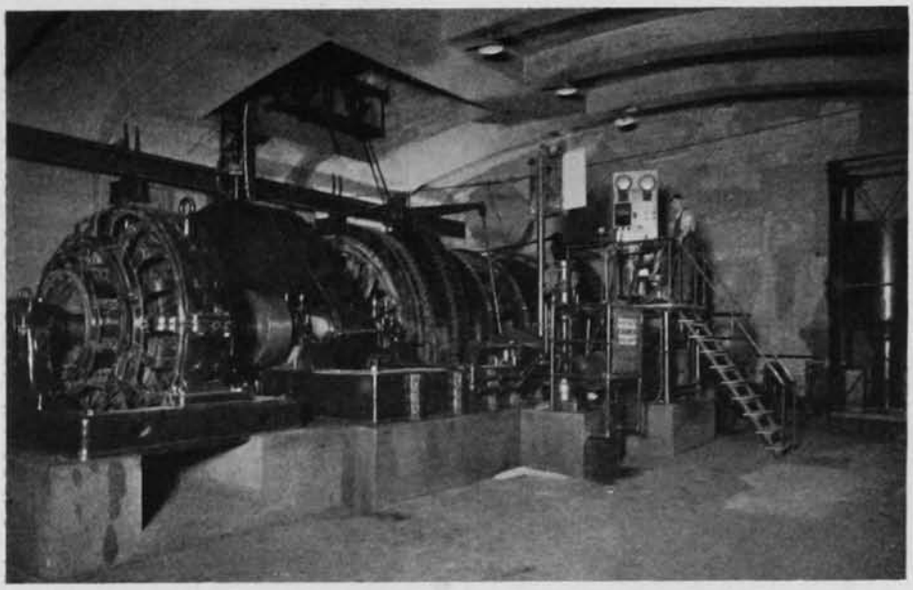


Figure 7. No. 5 Shaft Ore Hoist

Method	Feet	Exc'v'n, ft.	Concrete, ft.	Total, ft.	Cost, cu. yd. concrete
Solid 1:3:5 mix	1,550	$68.17	$146.84	$215.01	$58.80
Ring 1:2:4 mix	450	77.11	50.11	127.22	61.30

DETAILED COST PER FOOT OF SHAFT BY RING METHOD

	Excavation	Concreting
Labor	$52.07	$16.60
Shops	0.26	5.90
Supplies	7.68	26.51
Engineers	.....	1.05
Explosives	8.47	.....
Air	2.08	.....
Repairs	6.55	.....
Miscellaneous	.....	0.05
Total	$77.11	$50.11

The shaft stations have three 18-in. gauge tracks at 42-in. centers. On the upper levels, the trucks or cars are run on to the cage over turn sheets, whereas the lower stations are equipped with a special track layout with kick switches. Where the stations are at all slabby or heavy, the ground is supported with concrete and steel sets. The station construction and type of shaft gates is shown plainly in *Figure 11*.

The section of shaft from the 2,400-ft. level to the 3,000-ft. level has recently been completed at the following unit costs:

	Per cu. ft.
Shaft raising	$43.40
Trimming and drawing muck	5.30
Concreting	45.55
Steel work, ladders, guides, miscl. equip.	28.75
Total	$123.00

AVERAGE COST OF STATIONS

	Per cu. ft.
Excavation	$1,673
Concrete and steel sets	1,873
General equipment, including track, gates, etc	1,044
Total cost per station	$4,590

No. 6 Hoist:

No. 6 service (*Figure 12*) hoist operates a double decked cage in balance with a counterweight at a rope speed of 800 ft. per minute. The acceleration is 6 seconds and retardation 5 seconds. The hoist is of Nordberg manufacture, and has a single drum 12-ft. in diameter, with a 6-ft. spiral grooved face. A reel for ½ by 5½-in. flat rope for the counterweight is rigidly coupled to the drum. The brakes are of the usual parallel motion post type, set by a dead weight and released by oil pressure. In addition to the main brake, an auxiliary brake has been installed on the motor drive shaft pinion as an added safety measure.

The hoist is driven through a flexible coupling and herringbone gear reduction unit by a 500-hp. slip ring induction motor, at a normal speed of 350 r.p.m. The hoist is equipped with the usual safety devices and limit switches in the shaft.

The cage (*Figure 13*) is of double deck design, 6 ft. 9 in. by 12 ft. 7 in. in section, and will accommodate 50 men to the deck, or four standard service trucks. The cage is provided with inward swinging gates with offset hinges to prevent the possibility of gates swinging into the shaft walls. The tracks are equipped with a rack alongside of each rail, with a wheel stop of convenient size with handle and the bottom lugged to engage the rack and hold the cars or trucks in place on the cage.

The pull bell system is used in the shaft for signalling the hoist operator. Each station is provided with a push button, which operates an annunciator on the engineer's platform, indicating the level from which the call comes. A return flash is given by a return signal lamp above each call bell, all lamps being connected in parallel. The annunciator system is used on both the call buttons and cage signals as a matter of safety.

This type of hoist driven with an alternating current motor has the advantage of greater economy than the Ward Leonard control with generator set for the power unit. The flat cable reel, with

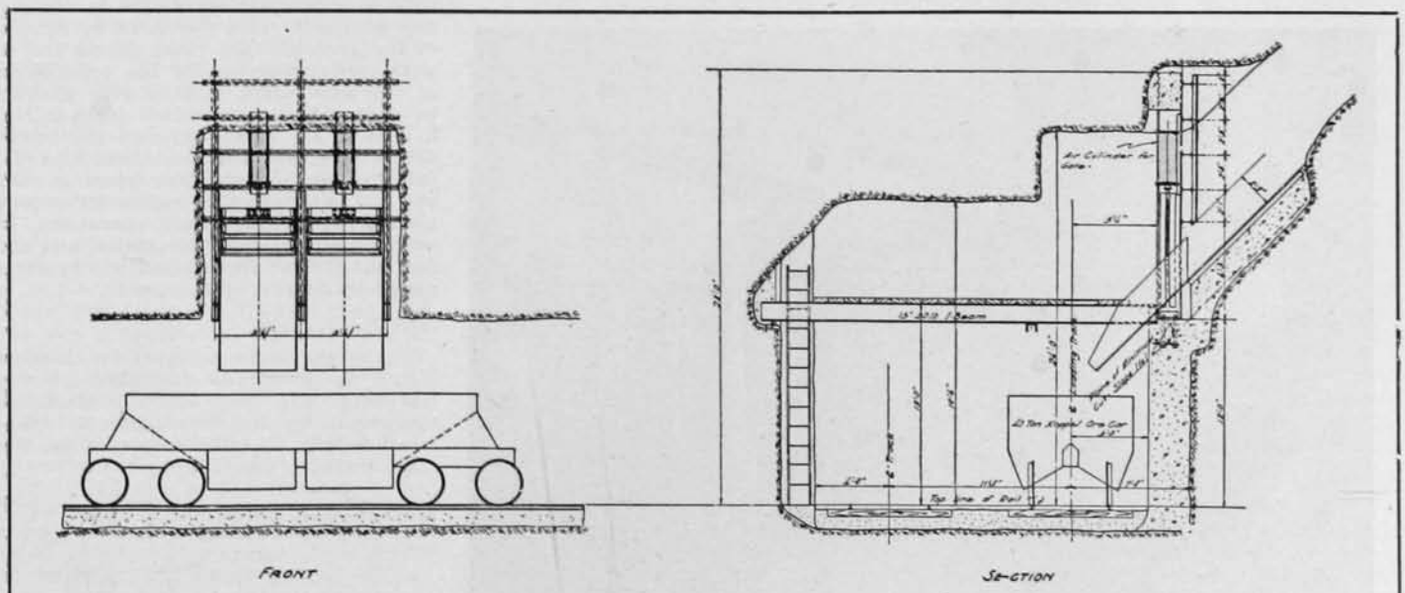


Figure 8. Hopewell Tunnel Loading Bins

United Verde Copper Company

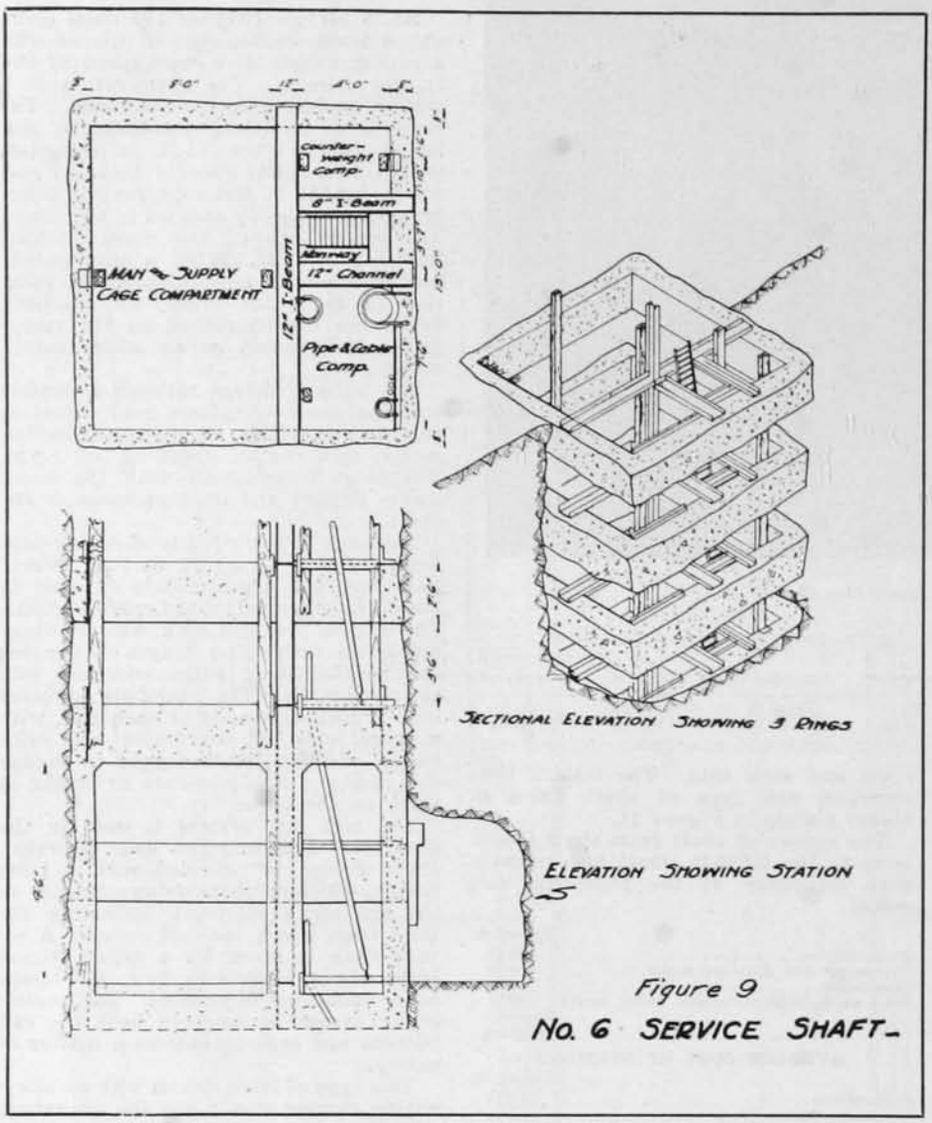


Figure 9
No. 6 SERVICE SHAFT

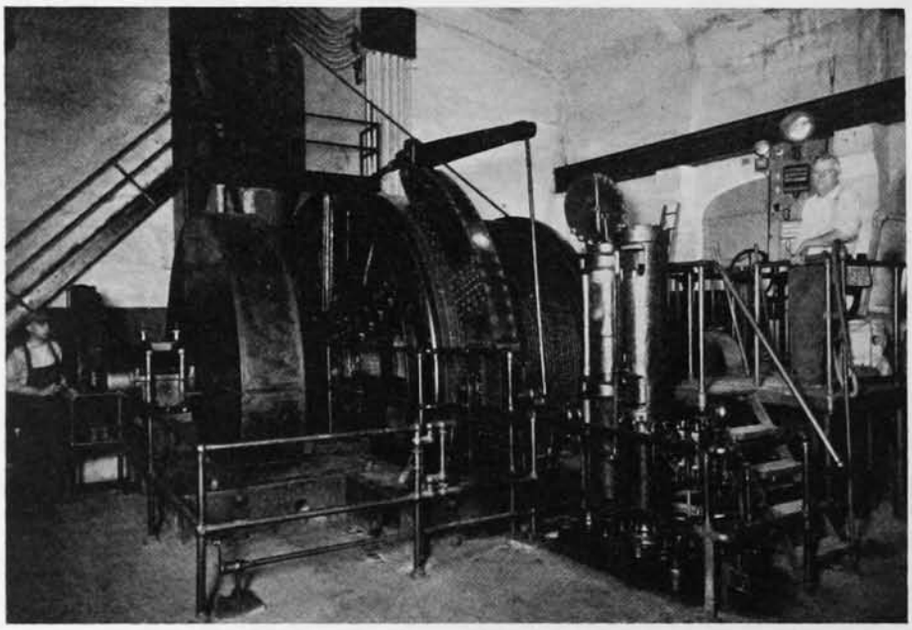


Figure 12. No. 6 Shaft Hoist

United Verde Copper Company

its varying lever arm, tends to balance the weight of suspended cables in the shaft. The great disadvantage is the comparatively short life of the flat cable, with its attendant high replacement costs.

Practice at the United Verde indicates that a large single decked cage is very advantageous in handling men and material. The efficiency is, however, limited to shallow depths with the present restricted speed of 800 ft. per minute. This limitation in depth is probably between 2,000 ft. and 2,500 ft.

New Shaft Work

Prior to 1918, the mine was served by two timbered shafts. No. 3 extended from the surface to the 1,950-ft. level, and No. 4 from the 1,000-ft. level. Both shafts were equipped with cages and steam hoists with flat cables. These shafts had insufficient capacity for more than 1,500 tons daily, and were inside the proposed shovel pit limits.

No. 5 shaft was planned in 1915, and No. 6 in 1917, and the locations were determined by existing mine openings with little attention to the physical characteristics of the ore body. Although No. 5 shaft is in the porphyry footwall, the hoist room is on a main iron-schist contact, and a small amount of movement has been noticeable during recent years.

Recent mining at greater depth, and the acquisition of adjoining ground with ore extensions in a direction not anticipated at the time No. 6 shaft was located, will eventually involve this shaft also. No. 6 shaft is limited in depth to the 3,000-ft. level by its hoisting equipment, so that either an offset shaft below the 3,000-ft. level was required or a new through shaft to surface.

A new combined service and ore shaft was tentatively planned in 1928, and crosscuts started towad the new No. 7 shaft location in October, 1929. This program will require approximately four years to complete and, because of the settlement of the present No. 5 hoist room, it was also decided to equip No. 5 shaft with new hoisting equipment to obviate any possibility of a shutdown by failure of the present hoist room during the 4-year period required for the completion of the new shaft, and to give greater hoisting capacity and added depth to No. 5 shaft. An abrupt ground movement above the 1,200-ft. level on June 29, 1929, justified the new shaft program as laid out and, although this movement caused no interference to shaft operations, it emphasized the need for action and the new No. 5 shaft installation will be completed by August of this year.

New No. 5 Hoist:

The mechanical equipment for the new No. 5 hoist will be furnished by the Nordberg Mfg. Co., and the electrical equipment by the Westinghouse Electrical & Mfg. Co. The following are the main features (see *Figure 15*):

Rope speed.....feet per min...		2,500
Vertical hoisting depth.....feet..		4,250
Size drums, with 7-ft. face.....feet..		12
Loads:		
Skip .....Pounds..	14,000	
Ore .....do....	15,600	
Cable, 4,280 ft. 1⅞-in....do....	23,800	
Total.....Pounds..		53,400
Hoisting capacity.....Tons per day..		4,500
Total weight of hoist.....Pounds..		430,000
Total weight of elec. equipment....do....		411,500

The hoist will be equipped with all modern safety devices, including the Lilly D controller, automatic accelerator and mechanical retardation. The hoist will be direct connected to a 2,700-hp., 550-volt, 66-r.p.m., shunt wound d. c. motor. Forced ventilation will be resorted to to keep the motor size down sufficiently to allow it to pass through a 7-ft. by 9-ft. opening. The Ward Leonard control system will be used, with the Ilgner flywheel system to stabilize voltage.

The flywheel set will be driven by a 2,800-hp., 660-volt, 3-phase, 60-cycle, 10-pole, wound rotor induction motor. Two 1,250-kw., 550-volt, 680-580-r.p.m., shunt wound, commutating pole, d. c. generators will be mounted on the same shaft. The flywheel will be 120 in. in diameter, and will weigh 85,000 lbs. A 45-kw., 250-volt, shunt wound, commutating pole exciter will be used for the field coils and control apparatus.

The size of all equipment is limited to a cross section of 7 ft. by 9 ft. to permit transportation through the main haulage tunnel on the 1,000-ft. level. This size necessitated two generators for the d. c. set, because the armature for a single generator is of such size as to require building up on the job.

Hoist Room:

The hoist and generator set rooms will be of the same cross sections. The combined rooms will be 41 ft. wide by 124 ft. long, and 25 ft. in height, the ceiling having a radius of 20½ ft., with a spring line 4½ ft. above the floor. The room is located 275 ft. from the center of the present shaft on the footwall side. The cableways are separate raises driven at an inclination of 38 degrees to meet the present sheave room. No changes will be required in shaft equipment or unloading pockets.

The hoist room will be supported with 5 ft. reinforced concrete ribs at 9 ft. centers. The excavation involves the removal of 5,300 cu. yds. of rock and the concrete work, including foundations, 1,200 cu. yds. The semi-circular section of the room precludes the use of an electric crane for installing and maintaining the hoist. A special hand operated bridge with two hand hoists of 10-ton capacity each will be used for installing the equipment. The heavier pieces will have to be handled by jacks and clocking.

No. 7 Shaft:

In selecting a location for a new shaft at the United Verde, several requirements were necessary.

(1) A surface installation was desirable to eliminate the time loss in transferring men through the 500-ft. tunnel, as the 8-hour regulation is from daylight to daylight. It was estimated that this time loss in transfer of men amounts to $45,000 yearly. Because of the rugged surface topography, the number of possible surface locations were limited.

(2) The question of ground subsidence from mining operations is of primary importance, and the shaft had to be located far enough in the footwall to be free from future ground movements. Experience at the United Verde mine shows that extensive mining operations in depth and laterally will cause ground movements, even though all openings are tightly filled with waste.

(3) The third consideration is the

Figure 10. Ring Construction in No. 6 Shaft

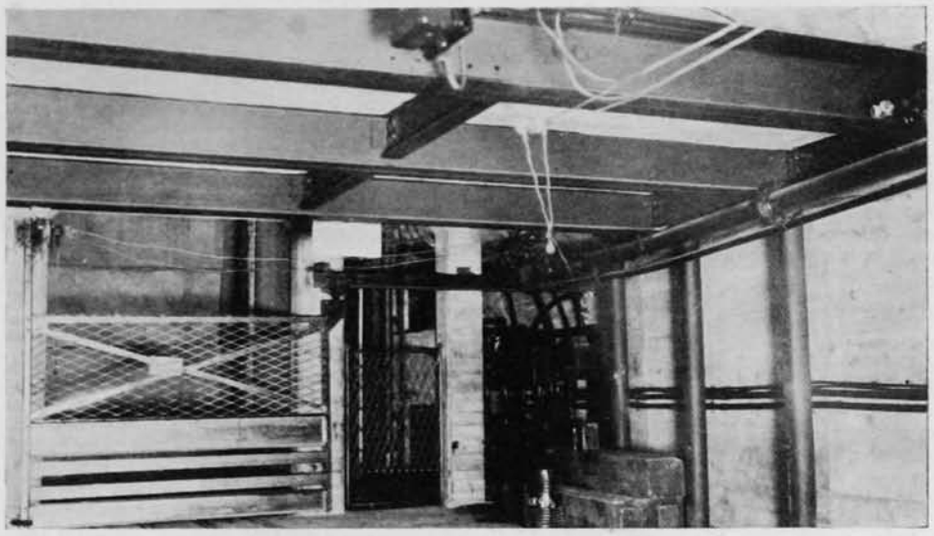


Figure 11. Typical No. 6 Shaft Station

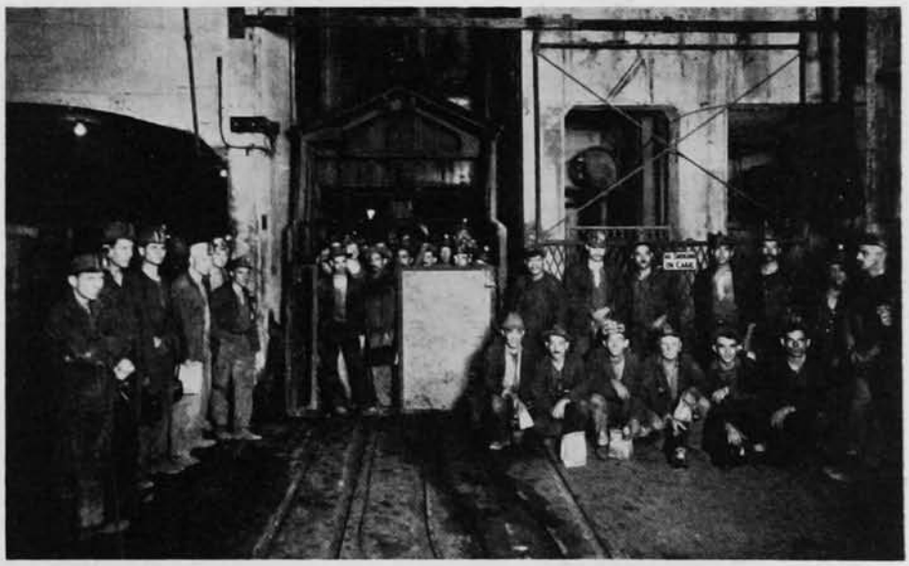


Figure 13. No. 6 Shaft Lowering Shift

United Verde Copper Company

character of the ground, as it is desirable to have the shaft in ground which requires very little support and does not slack upon exposure. The quartz porphyry footwall at the United Verde complies with this stipulation, and the site as selected will have very little heavy or blocky ground excepting near the surface.

No. 7 shaft as now planned, *Figure 14*, will have two 6 by 6-ft. hoisting compartments, two 5-ft. 6-in. by 12-ft. 10-in. service compartments, and one 4-ft. by 12-ft. 10-in. manway and pipe compartment. The walls of the hoisting compartments will be of solid reinforced concrete, 10 in. in thickness. The rest of the shaft will be of ring construction similar to No. 6 shaft. The rings will be spaced at 6-ft. centers, and the height of the ring will vary, depending on ground conditions.

Connections are being driven on all levels, excepting the 700 and 900-ft. levels. A pilot raise, 6 by 11 ft. in section, will be driven from level to level, and the shaft enlarged to full section by underhand mining, and timbered with temporary 8 by 10-in. shaft timbers. As a 150-ft. section is enlarged to full size, concreting will be started and carried upward as the next section directly below is being mined. The timbers will be reclaimed as concreting progresses and used over again.

The development program of extending the present mine headings to the new shaft location involves 14,800 ft. of 6 by 8-ft. headings, 1,600 ft. of 10 by 12-ft. haulage tunnel, in addition to the necessary loading pockets, ore bins, and auxiliary equipment.

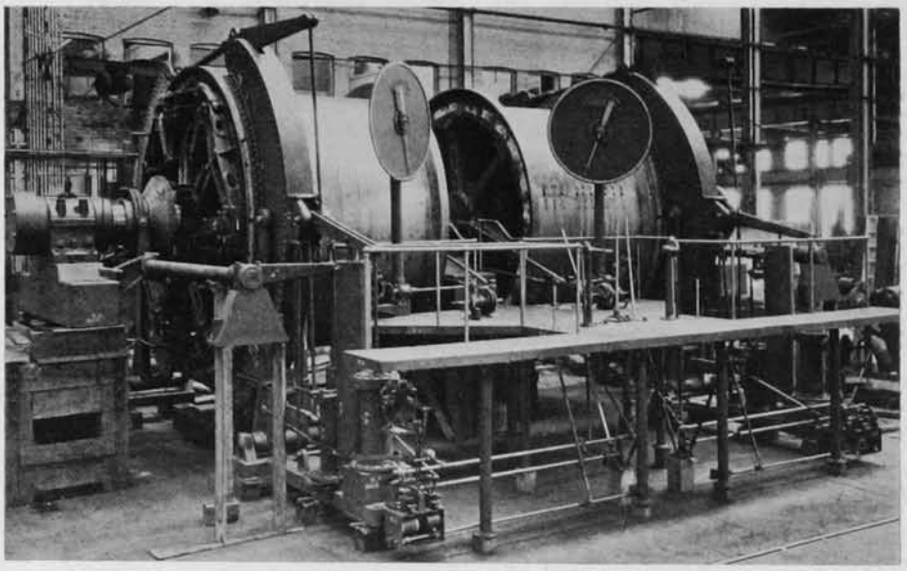
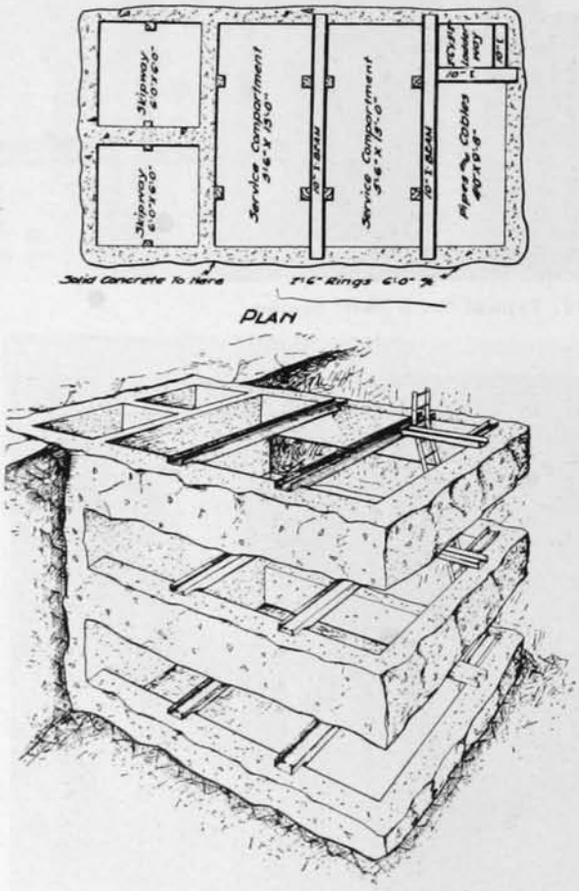


Figure 15. Double-drum Electric Hoist built by Nordberg Mfg. Co., said to be the largest ever built for underground installation and service

Hoist	Cable (Lang Lay)	Dead loads, tons	Safety factor	Avg. length service	Avg. tons hoisted
No. 5 shaft	1⅝-in. 6 x 19 Blue Center steel..	18.3	4.6	20 mos.	620,000
No. 6 shaft	1¾-in. 6 x 19 Blue Center steel..	22.8	5.0	2 yrs.	
No. 6 shaft	½ x 5½ 12-strand 4 x 7 plow steel with single lacing.....	17.0	7.4	7 mos.	



PERSPECTIVE
Figure 14
PROPOSED NO 7 SHAFT.

United Verde Copper Company

Cable Practice

The above tabulation shows the main hoisting cables at the United Verde.

Cable fastenings are inspected daily and the cable once a week. The broken wires are counted by means of a two-piece steel ring that fits over the cable and is held by the inspector. The cables are resocketed at six-month intervals and cut back on the drum end once a year. Vesuvius cable dressing applied at intervals of four to five weeks gives satisfactory penetration and keeps the cables well lubricated.

The recommendations as given in Bulletin 75, Rules and Regulations for Metal Mines, are adhered to for the ore hoist and flat cable on No. 6 shaft, but the round cable on No. 6 service hoist is changed at two-year intervals even though the cable is apparently in good condition. This is done to eliminate any possibility of breaking a cable with a cage loaded with 100 men.

Although both drums are grooved spirally and the cable wound in two layers only, recent practice is tending toward concentric or parallel grooving which affects a smoother wind on the second and third layers. Future hoists will be equipped with parallel grooved drums.

Guide Practice

Guides as first installed in No. 5 ore hoisting shaft were 4 by 8-in. clear Oregon pine, fastened with ⅞-in. bolts and shimmed with oak shims. The life was approximately eight years, with an additional 1½ years' service by repairing with wedge-shaped oak strips 3 in. wide and fastened to the worn edge of the guide with 2-in. wood screws.

It is to be noted that practically all wear is on one edge of the guide, due to the twist in the cable. Recent guide practice is to use 3⅝ by 7-in. O. P. guides, with a 1 by 3⅝-in. clear white oak wearing strip on one edge to take this wear.

The lower part of No. 5 shaft is equipped with 8-in. 21.5-lb. ship channel guides with backs out. Plates ¾ by 5 by 8 in. are welded to the flanges for fastening to the guide bolt sockets. These steel guides cost $2 per foot in place, or about double the cost of O. P. guides, but they have given very good service and are justified.

The design of guide bolt sockets is very important in concrete shafts. With the ring method on service shafts, the guide fastenings need be nothing more than a heavy angle with slotted holes. In shafts with solid walls, it is essential that the guide bolt sockets be provided with hand holes of sufficient size to permit a wrench being used on the nuts.

It has been found that in spite of all care used in spacing and aligning guide sockets in shaft construction, variations will occur which cause delays in making the guides fit. On new shaft work a steel wire is stretched between levels and the bolt holes measured from center line both horizontally and vertically. The work is done by the engineer and drawings made showing the position and spacing of all holes, for each guide. The guides are numbered before being sent into the mine. This additional work of detailing each guide effects an appreciable saving in the cost of installation.

Underground Haulage and Power Distribution at United Verde

By E. W. Fredell
CHIEF ELECTRICIAN
MINE DEPARTMENT

THE electrical distribution system at the United Verde mine has been laid out with four definite objects in view: *First*, safety; *second*, reliability; *third*, flexibility; and *fourth*, economy.

Power is received from the smelter, at Clarkdale, at 45,000 volts, over two transmission lines. Either of these transmission lines are of sufficient capacity to carry the whole plant load in times of emergency, although both are normally in operation.

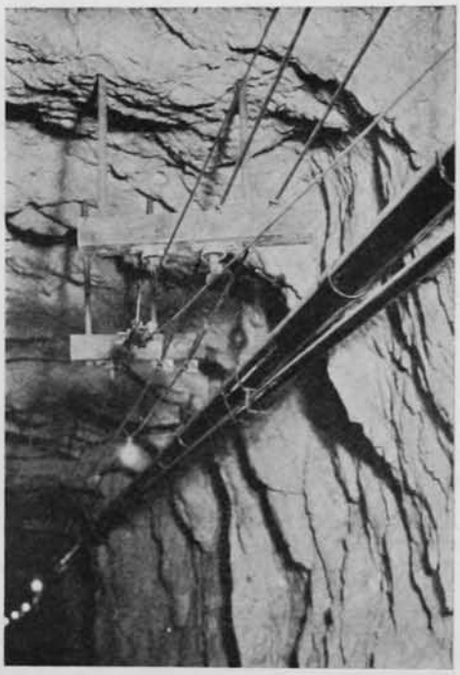
The main sub-station is located on the 500-ft. level, where transformers of 6,000 kva. capacity are installed. Here all power is transformed to 2,300 volts for distribution to the various load centers, and all outgoing lines, with one exception, are at this voltage.

Power for shop use is distributed at 440 volts, due to the multiplicity of small motors on the various shop tools.

For the open pit operations, located about three-quarters of a mile from the sub-station, two 2,300-volt circuits are carried to a substation and transformed to 440 volts. This voltage was chosen for transmission to the pit proper due to the peculiar type of pit operations. Extreme flexibility is required, and pit transmission lines have very temporary locations, so it was felt that the lower voltage would be less hazardous to life and give fewer equipment failures. All transmission lines in the pit are submarine type cable to junction boxes, while trailer cables on the shovels are of a rubber belted type, 400 ft. in length. Since the size of the shovels is such that it was not practical to mount a cable reel on them, the trailer cable must be coiled up on the ground, and is therefore subject to considerable abuse.

Three 2,300-volt circuits are carried underground for hoisting, pumping, ventilation, and haulage purposes. These circuits are carried in submarine cable direct from the substation, thus minimizing lightning trouble or any trouble to the lines from outside sources. The man and material hoist is carried in one circuit, haulage on another, and the ore hoist and pumping load on another. At the hoist room on the 500-ft. level, these circuits are so arranged that they can be interconnected, thus insuring as far as possible continuity of service.

In the shafts themselves, submarine type cables are used exclusively for lighting, power, signals, and telephone services. As the lighting circuit is used for signalling purposes in case of fire and for time signals for blasting, the main switch is placed in the hoist room on the 500-ft. level. This circuit is 2,300 volts, with a transformer on each level.

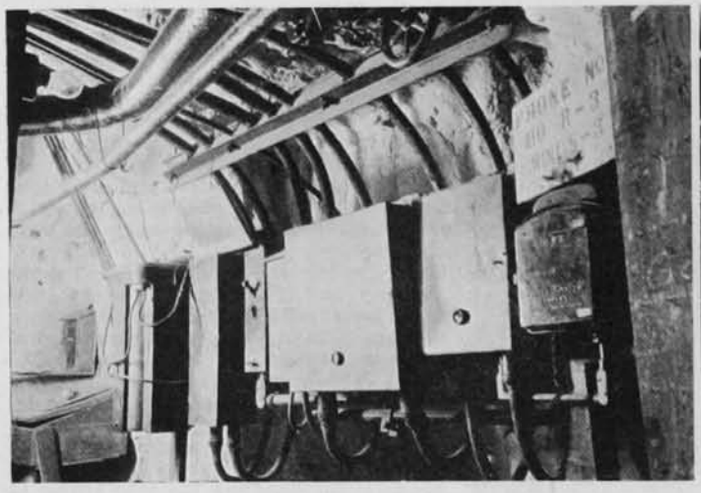


Trolley Construction

On the various levels, lighting and fan circuits are usually carried on open wiring, with either conduit or hose around chutes. The fire hazard from alternating-current circuits not being very great, greater latitude is allowed in them than with direct current. In all timbered sections it is now becoming standard practice at this mine to use either cable or conduit for all direct-current circuits.

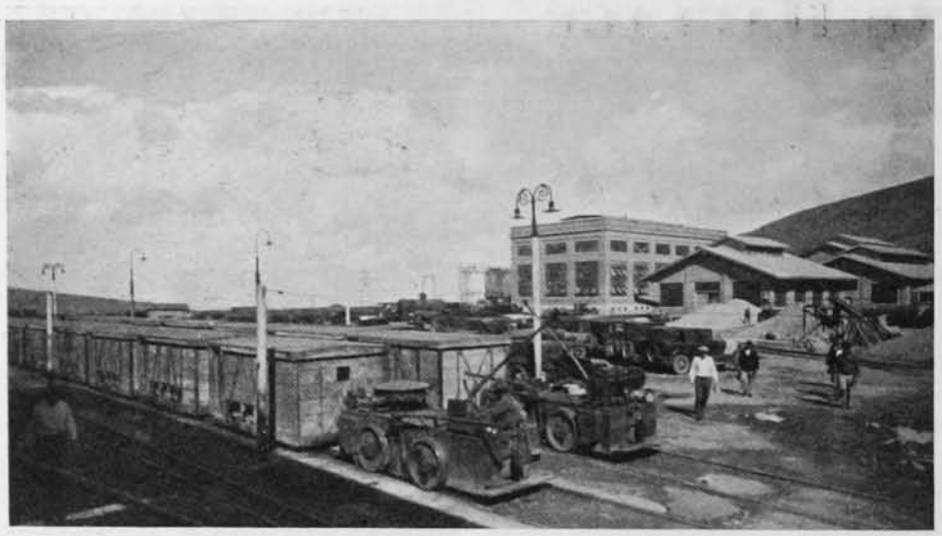
Wherever possible, full magnetic control has been installed on both direct and alternating current equipment. This increases safety protection, both to the men and the equipment.

For ventilation purposes 2,300 volts is used on the main fan on the 1,000-ft. level, while 250-volt direct-current is used on the small fans for local ventilation scattered throughout the mine.

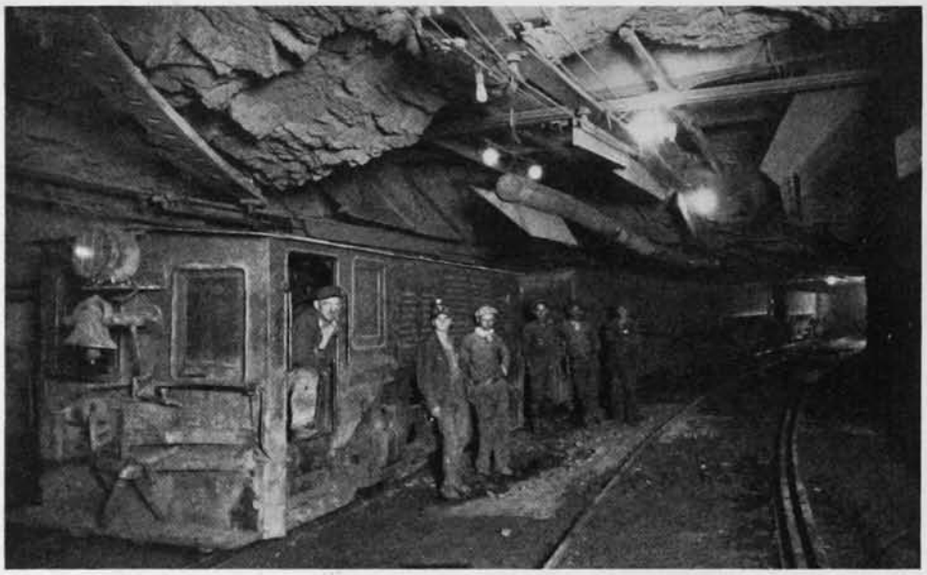


Junction Boxes at Shaft Stations

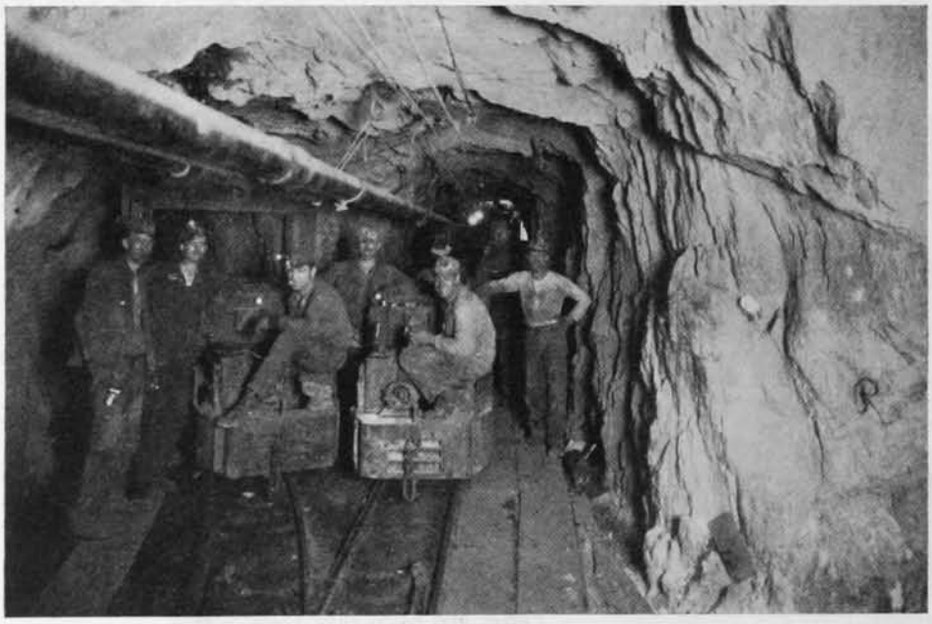
Power and Light Line Construction



Mine Surface Plant, Showing Cars for Transporting Men Into Mine



Hopewell Tunnel Ore Train



Ore Gathering Trains

United Verde Copper Company

Electric blasting is used quite extensively, and stray currents constitute a hazard to life and property. Due to this fact, and also to the fact that the rock does not offer a positive ground, an elaborate ground system has been installed. On both the 500-ft. and 1,000-ft. levels, two ground plates have been buried in creek bottoms, and all track, water lines, air lines, etc., tied to this positive ground, as well as one side of the haulage circuit. This ground wire is carried throughout the mine, and connections made to the various pipe lines at each shaft station and as far back as feasible on each level.

The haulage substation on the 500-ft. level consists of two 500-kilowatt synchronous motor generator sets, one of which is in continuous service and one spare. The trolley voltage is 250 volts, and is also used for battery charging and small fans for local ventilation. Due to the length of the Hopewell haulage and the heavy loads handled at the end of it, a 150-kilowatt full automatic motor generator set has been installed as a booster. This set operates from a time clock and is full automatic in all other respects.

Haulage

The underground haulage system of a mine of the size of the United Verde Copper Company is quite extensive, and has the following functions to perform:

1. Handling of ore from the shafts to surface.
2. Handling of men from surface to shafts and return.
3. Gathering of ore and distribution of waste on individual levels.
4. Distribution of materials on individual levels.
5. Return of empty trucks and dull steel to surface.
6. Gathering of material from shops and warehouse, and the handling to the supply shaft.

(1) *Hopewell Haulage:* All ore from the mine to the smelter is handled through the Hopewell Tunnel, 7,200 ft. in length, and located on the 1,000-ft. level, to the Hopewell ore bins. Ore from the pit operations is passed down to the tunnel level through raises, while ore from the mine is hoisted to bins on this level.

The track gauge in this tunnel is standard for American railway practice, and 75-lb. rails are used. The grade is from 0.25 to 0.50 percent in favor of the load. Rolling stock consists of 25-ton electric locomotives and 40-ton bottom dump ore cars. Nine cars constitute a train. Four 25-ton, 250-volt, direct current trolley locomotives are in use in this service. One locomotive is always held in reserve. The trains are equipped with electro-pneumatic control and full air throughout. Five hundred and forty tons is the trailing load when going out with ore, and 180 tons coming up grade with the empties.

The trolley wire used is of four-naught capacity, and is sectionalized every 1,300 ft. Three feeder taps are made in each section. Trolley life has averaged five years in the past.

(2) *Handling of Men:* On the 500-ft-level, the man and material shaft is located 2,100 ft. from the portal of the tunnel. This tunnel has a standard gauge and an 18-in. gauge track. The standard gauge is for handling the man trains at shift times, and the 18-in. gauge is for handling material to the mine. Two

10-ton, standard gauge, trolley locomotives are used in this service. The handling of men in this manner eliminates considerable confusion and lost time at the beginning and end of the shift. No trolley wire is run in the yard beyond the travel of the man trains.

(3) *Ore Gathering and Waste Distribution:* The mine was originally equipped with trolley throughout, but with the development of the battery locomotive less trolley has been used from year to year. At present only five of the older levels of the mine have trolley haulage, the remainder being handled by battery locomotives.

All tracks underground are laid with 50-lb. rail, with a track gauge of 18 in., and a grade in favor of the load of from 0.25 to 0.50 percent. Ore cars are equipped with roller bearings, and are of the rocker bottom type, 24 cu. ft. capacity. All trucks used for handling timber and steel are also equipped with roller bearings.

Trucks are loaded with steel and various materials at the shops on the 500-ft. level surface, and are landed at the various levels with no re-handling.

The size of the locomotives has been standardized at 5 tons, since this is the maximum size that can be obtained which will fit within our limiting dimensions.

Ore trains on the individual levels are made up of sixteen 24-cu. ft. capacity cars, making a trailing load of 43 tons, with the grade in favor of the load. Two hundred and twenty-five tons, over an average haul of 1,200 ft., is considered a shift's work.

Waste trains consist of 12 cars, with a trailing load of 32 tons and the grade against the load. The total tonnage per day per motor depends largely upon local conditions. The average haul for waste is 900 ft.

The United Verde has decided upon battery locomotive haulage for ore gathering and waste distribution, and are gradually eliminating trolley except for special cases. Trolley locomotives, however, will probably always be used and maintained in long drifts having no timber and for extremely heavy haulage conditions. The chief factors that have decided the fate of the trolley motor as against the battery locomotive are as follows:

1. Elimination of life hazard from trolley.
2. Elimination of fire hazard from trolley.
3. Cutting down of stray currents.
4. Elimination of rail bonding.
5. Cutting down size of drifts.
6. Speeding up loading cars at chutes.
7. Flexibility.
8. Lower maintenance costs.

The disadvantages of battery locomotives, as they have been found at this property, are as follows:

1. Limited capacity upon one battery charge.
2. Higher initial cost.

Both lead and alkaline types of batteries are in use. Under our conditions we have obtained an average life of 7½ years for alkaline batteries and 3½ years for lead batteries.

Twenty-four battery locomotives and 45 sets of batteries are in service at present. Nine 5-ton trolley locomotives are also in use.

Trolley wires are maintained at 7 ft. 2 in. from the rail throughout the mine. This is boxed in all main traveled drifts, and left bare where travel is not so heavy.

All track bonding is done with an electric welder, and bonds of the type that fit in the web of the rail are used. These bonds are long enough that fish plates can be replaced without injuring the bond.

Track is maintained in the best possible condition, since it has been found that poor track cuts production by a surprising amount.

On the surface and around the shops, a battery locomotive is used for gathering supplies, as the elimination of trolley in the yard permits the use of a locomotive crane for heavy lifting of all sorts.

The following tabulation gives a distribution of power as used in the various operations on a basis of kilowatt-hours per ton of ore produced:

Air compression	3.230
Hoisting	0.855
Ventilation	1.374
Mine haulage	0.475
Hopewell haulage	0.419
Shovel operations	0.382
Pumping	0.229
Shops and miscellaneous operations	0.554
Total kw.h. per ton ore produced	7.518

DEVELOPMENT AND MINING METHODS

(*From page 27*)

cutter which cuts eight pieces at a time, and is operated by a small air cylinder. A foot operated cap crimper is used, and one man can cut and crimp 600 fuse per hour. Six-X caps are used for plugging and stoping in soft ground, and 8-X caps elsewhere. Standard length primers are 4 and 5 ft. for blockholing and 7 and 9 ft. for drifts, raises, and stoping. The primers are delivered to the fuse magazines on each level in wooden lined carbide cans with hinged covers. Electric blasting is used in all sinking operations and development headings off of shafts or winzes.

Pumping and Drainage

The United Verde mine is comparatively dry, the volume of water pumped averaging 180,000 gallons daily. The surface waters from the upper part of the mine is delivered to the 1,000-level through 2 13/16-in diamond drill holes, where it is diverted to the ditch in Hopewell Tunnel and used in leaching operations at Hopewell.

Mine water below the 1,000-level is handled by two electrically driven Aldrich 6½-in. by 12-in. quintuplex plunger pumps. These pumps are rated at 500 gals. per minute against a 1,000-ft. head. Double acting, plunger type air pumps are used as auxiliaries. The pump column is a lead lined, 8-in. standard wrought iron pipe, with extra heavy C. I. flanges. It is supported in the shaft by 1 by 4-in. strap iron clamps spaced at 30-ft. intervals.

Open Pit at

By E. M. J. Alenius

CHIEF ENGINEER, STEAM SHOVEL DEPT.

DURING the past year operations at the United Verde lower pit reached the maximum stage. Since the beginning of the lower pit operations in March, 1925, when the first small electric shovel was put into service, operations have been steadily expanded. Additional equipment was acquired, so that in 1929 four shovels were operated by eight crews, three crews working on day shift, three on afternoon shift, and two on the night shift.

Lower Pit Production

Production of the lower pit to date is tabulated as follows:

Year	Cu. yds. ore	Cu. yds. stripping	Total
1925.....	57,507	154,667	212,174
1926.....	157,594	124,472	282,066
1927.....	230,006	138,591	368,597
1928.....	276,057	307,651	583,708
1929.....	433,847	465,919	899,766
Total....	1,155,011	1,191,300	2,346,311

Development of Surface Operations

On account of the serious mine fire which persisted throughout the upper levels of the United Verde mine, surface operations were deemed to be the only method by which the high grade ore on the upper levels could be recovered. Accordingly the stripping operations were started in 1918. By October, 1927, the major stripping operations above the 160 mine level were completed. The major stripping operations involved the removal of 8,077,451 cu. yds. of material, of which 803,354 cu. yds. were classed as ore.

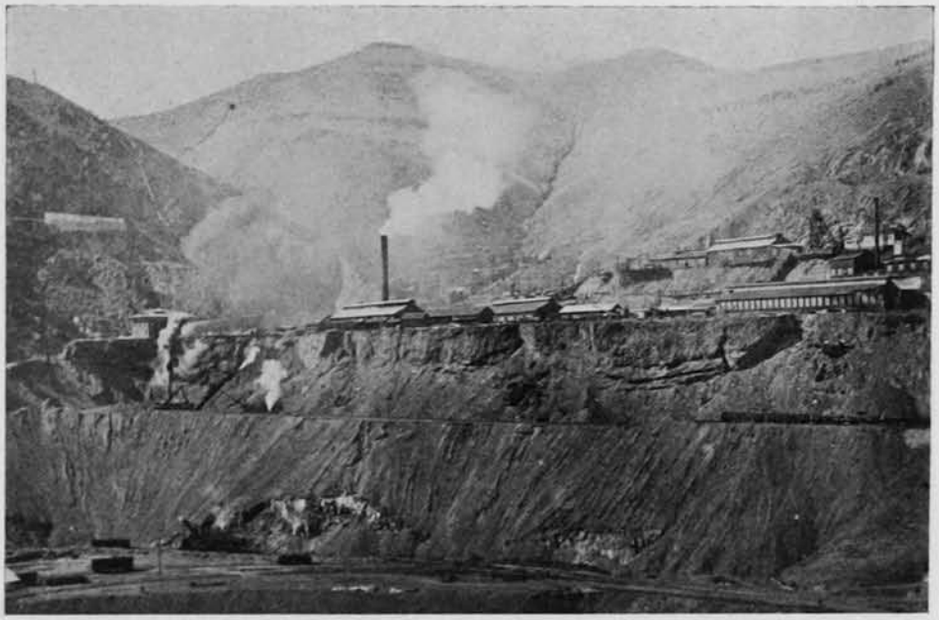
The lower pit is that part of the surface operations extending below the level of the 160 mine level. The 160 mine level is roughly the dividing line between the zone of exidation and the zone of secondary enrichment. Above the 160 level the oxidized material and waste were removed on eight benches varying from 50 to 110 ft. in height. The benches were connected by a series of switchbacks on the adjacent hillside. An 8-cu.-yd. full revolving and two 4-cu.-yd. railroad type steam shovels were used for this work. The material was loaded into 25-cu.-yd. dump cars hauled by switcher type steam locomotives over standard gauge track. The orebody was cut into on the 160 level, and below that level it is necessary to descend to the working benches. When plans for the open pit were first prepared it was intended to remove the material below the 160 level by mill-holing. The plans considered exposing the entire area of the orebody on the 300-ft. level. Above that level, material would have to be removed outside of the ore zone to provide a safe working slope. Below the 300 level ore would be left in the pit walls to be later removed by underground methods.

As soon as the orebody was exposed on the 160 level, several glory holes were started, the ore when broken falling into raises, occasionally transferred on intermediate levels, and dropped into the orebins above the main mine haulage tunnel on the 1,000-ft. mine level. The material was transferred to bins at the mouth of the Hopewell haulage tunnel for shipment to the smelter or for disposal on the various dumps.

It soon became apparent that the glory hole method of removing material below the 160 level was not practical. It was impossible to drive the necessary raises in the desired locations because of the

General View of Shovel Pit. Bottom of Pit 30 ft. Below 300 Level

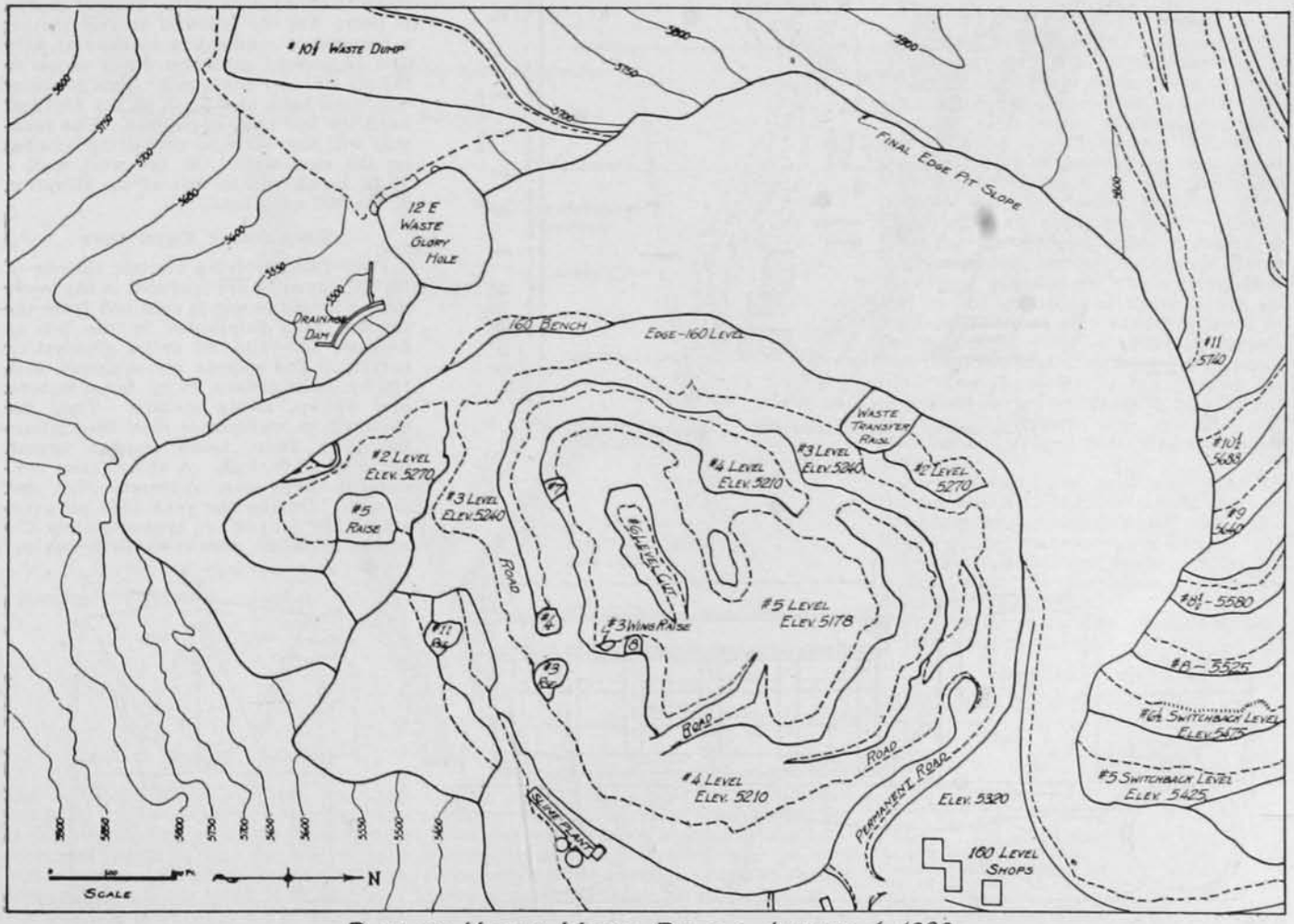
MINING
United Verde



Mine Surface Plant on 160 Level at Beginning of Shovel Operations

underground fire and the old mine stopes. As the United Verde orebody is not uniform in character, but consists of schist suitable for milling, sulphides for direct smelting, siliceous sulphides and quartz for converter ore, it was not only impossible to separate the various classes of ore from each other but also from the low grade, leach and waste materials. A definite production was impossible. Furthermore, gloryholing was extremely hazardous because of having to work on loose slopes and upon some ground which was on fire or through which gas and smoke were issuing. For these reasons gloryholing was abandoned.

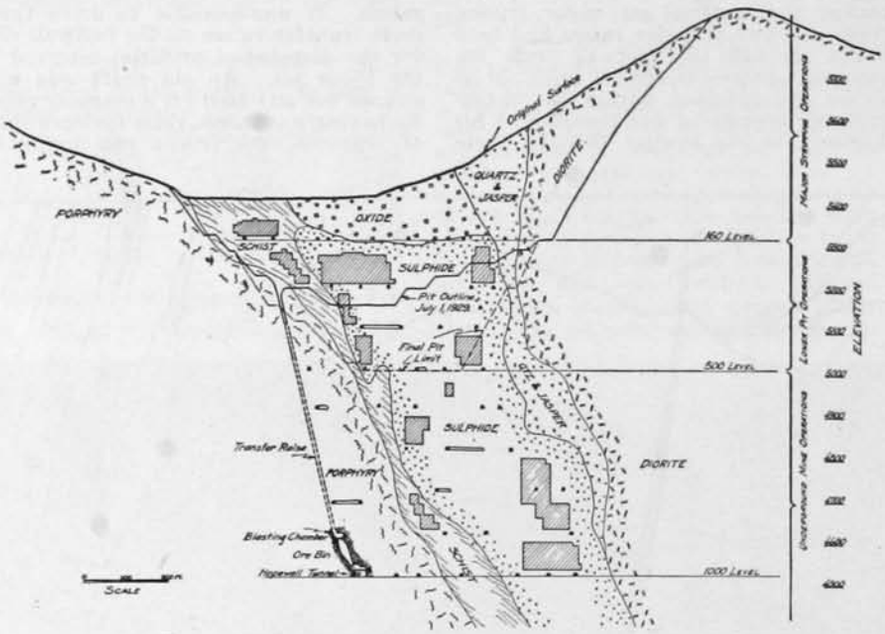
A plan was then devised by Mr. W. V. De Camp, assistant general manager of the United Verde Copper Company, and J. C. Perkins, shovel superintendent, by which the difficulties encountered in gloryholing were eliminated by the use of small shovels working on benches and loading the material on motor trucks. Previously two transfer raises had been driven through the footwall from the mine haulage level to the 160 level. Most of the ore obtained during the major stripping operations was transferred for shipment to the smelter through these raises. It was possible to drive three more transfer raises on the footwall side for the disposal of material removed in the lower pit. An old shaft was also cleaned out and used for a transfer raise. By having a separate raise for each class of material, the trucks can easily be



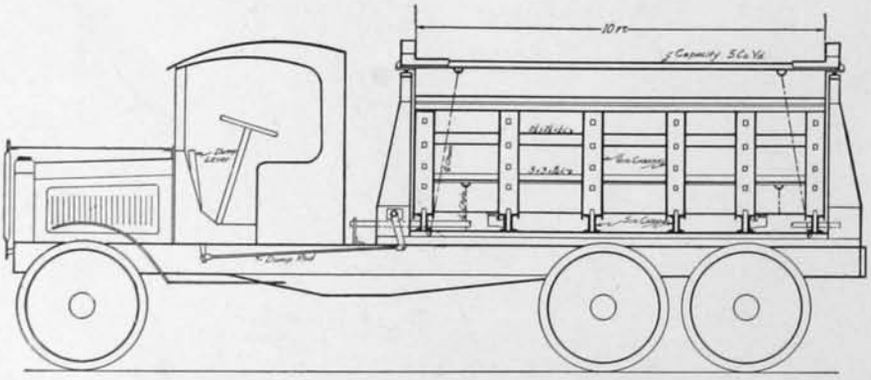
Plan of United Verde Pit ~ January 1, 1930.

United Verde Copper Company

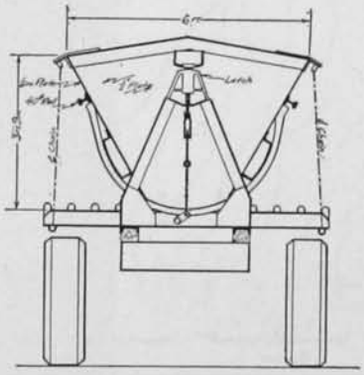
Battery of Motor Trucks Serving Electric Shovel in Pit Operations



TYPICAL SECTION OF UNITED VERDE PIT.



SIDE VIEW



BACK VIEW

TRUCK WITH UNITED VERDE STANDARD DUMP BODY.

routed to any raise and the various materials properly disposed of as encountered. It is necessary for the various materials to be sorted during digging, but by careful blasting and close supervision this is easily possible. Small 1¾-yd. shovels were installed on account of being able to sort the material better and to move more easily. A definite production of the various ores is possible, as the shovels can be quickly moved on the benches or between benches, and the trucks routed from any working place to any transfer raise. Furthermore, it is not necessary for the men to work on the hot ground or beneath high banks except below the pit walls.

Plan of Lower Pit

At present it is planned to carry the present pit operations from the 160 level, elevation 5325, to the 500 level, elevation 5015. The pit is divided into 10 benches varying in height from 25 to 33 ft. The pit walls are planned on a ½ to 1 slope in the firmer material. A 1 to 1 slope has been found to be necessary in the softer material on the east side of the pit. The major stripping operations were carried back far enough from the edge of the lower pit so that a bench could be maintained beneath the high banks. This bench, varying in width from 20 to 50 ft., serves as a protection for the lower pit, as small amounts of material sloughing off the high bank are caught on the bench. At present a portion of the bench is filled up. An attempt will be made to clean it off as soon as the stormy season is over. On the footwall side of the pit a permanent roadway is planned on a 10 and 12 percent grade by which access to the pit will be maintained. This roadway will wind back and forth on the footwall until the 500 level is reached. The roadway will also serve as protecting benches on the east wall. On the west wall a 20-ft. bench will be left at the elevation of the 300 mine level.

Mechanical Equipment

Four full revolving electric shovels of 1¾-yd. capacity are operated in the lower pit. Electric power is obtained from the smelter and distributed in the pit on 3-phase, 440-volts, 60 cycles alternating current. The shovels are equipped with 100 hp. hoist motors, 40 hp. boom motors, and 60 hp. swing motors. They are mounted on manganese steel caterpillars having a 36-in. tread and an overall width of 13 ft. 6 in. A shovel crew consists of three men, engineer, oiler, and pitman. During the year 1929 an average of 390 cu. yds., or approximately 870 tons of material, were removed per shovel

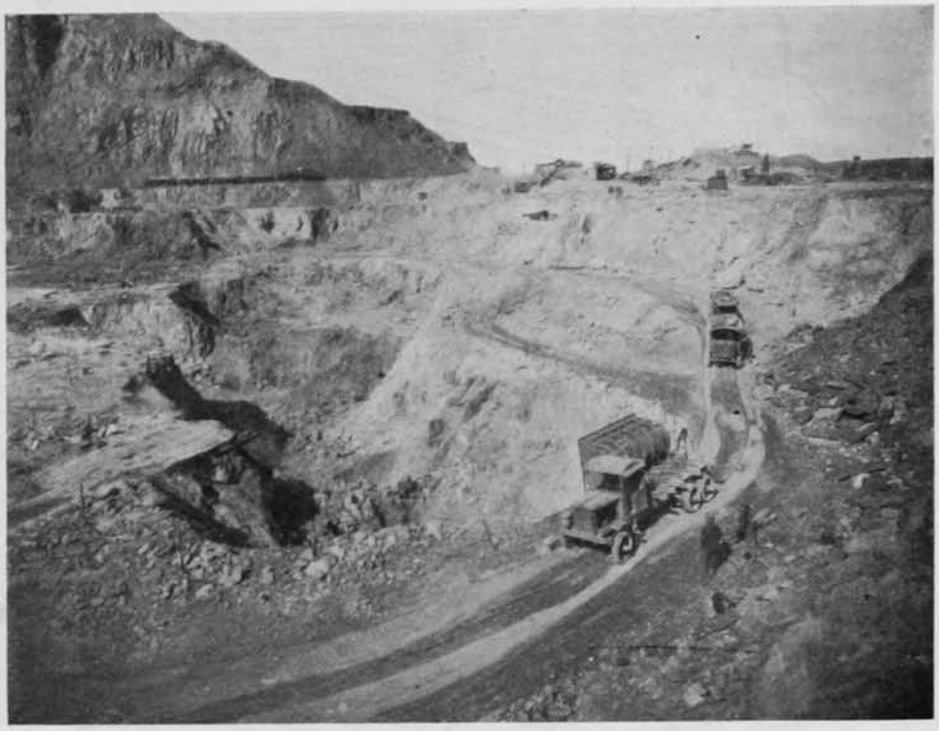
shift. Power consumption amounted to 0.966 kw. hr. per cu. yd. loaded.

Eleven 4-cyl. and 6-cyl. 6-wheel, 4-wheel rear drive trucks, and six 6-cyl. caterpillar tractor trucks are used for tramming the material from the shovels to the transfer raises. These trucks and tractor trucks are equipped with a standard gravity side dump body, designed and built in the United Verde shops. The loads are dumped when the truck is tilted and the latches released. The body has a capacity of 5 cu. yds. water level, or 3.5 cu. yds. solid material, or from 7 to 10.5 tons, depending on the character of the material. Both types of trucks are especially reenforced and equipped with compound transmissions. The tractor trucks are used in pulling loads uphill on steep grades. They are also valuable for keeping operations going in bad weather. One hundred and thirty-one cu. yds., or approximately 290 tons of material, were handled per truck shift during 1929. The average length of haul increased during the past year on account of the removal of material below the pit wall on the opposite side from the transfer raises. Gasoline consumption amounted to 1 gal. for 7.65 cu. yds. trammed; 2.97 truck shifts were operated for each shovel shift.

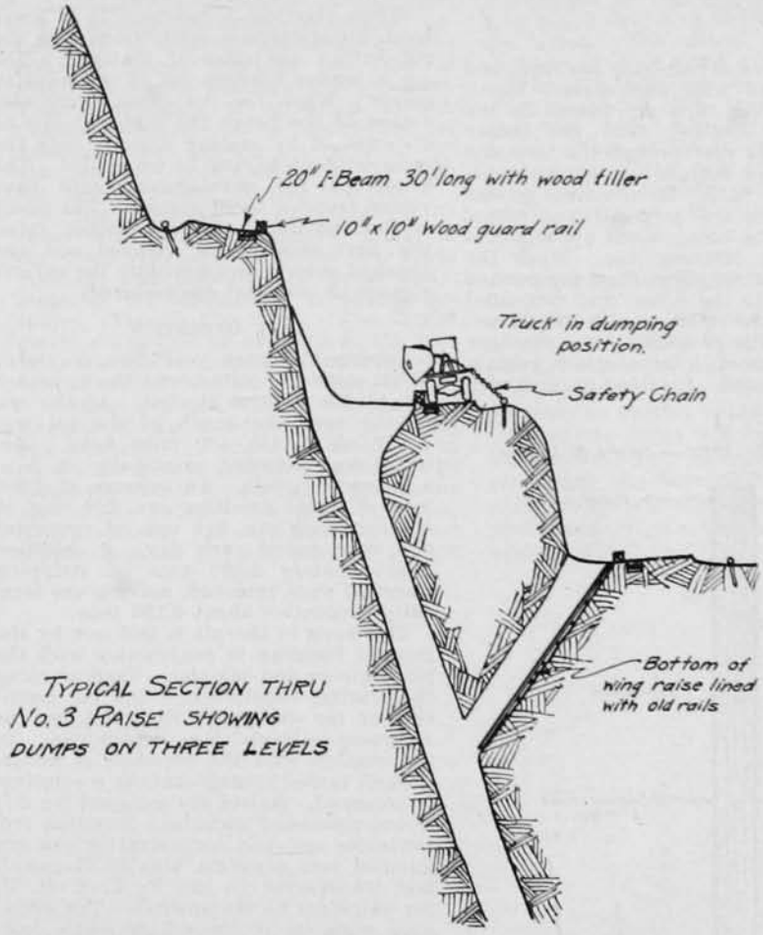
Several light trucks are used around the pit. A 1-ton truck and a 2-ton truck are used for handling powder, supplies, and drilling material. A 3½-ton truck equipped with an acetylene burning outfit is used by the bull gang for transporting tools and repair parts. For heavy material, such as drill steel, churn drill bits, and casing, a 5-ton truck is available.

A gasoline-powered crane is maintained for assisting in repair work and for handling heavy material. It is equipped with a half cu. yd. clam shell bucket for cleaning up spillover and doing small excavating work. A road grader is used for keeping the roads in shape in the pit and around the mine plant.

Miscellaneous shops are maintained on the 160 level adjacent to the pit for the minor repair work. The larger repair work is handled in the main mine shops on the 500 level surface.



Trucking Ore to Transfer Raise



TYPICAL SECTION THRU No. 3 RAISE SHOWING DUMPS ON THREE LEVELS

Drilling and Blasting

Drilling and blasting in the lower pit is governed entirely by the character of the ground. Care must be taken to break the various materials clean so as to avoid dilution. The depth, spacing, and location of holes are dependent entirely on the knowledge of geology and ore conditions. Three types of air drills are used as well as electric traction churn drills. The air drills are chiefly used in breaking the ground along contacts, in bank trimming, and in inaccessible places. Wherever possible in uniform material and in hot ground, the churn drills are used, as the cost of breaking ground by churn drill blasting is about 16 percent less than the cost of machine drill methods. The character of the ground has presented varied problems in drilling and blasting. However, the necessity for blasting hot ground presented the major difficulties.

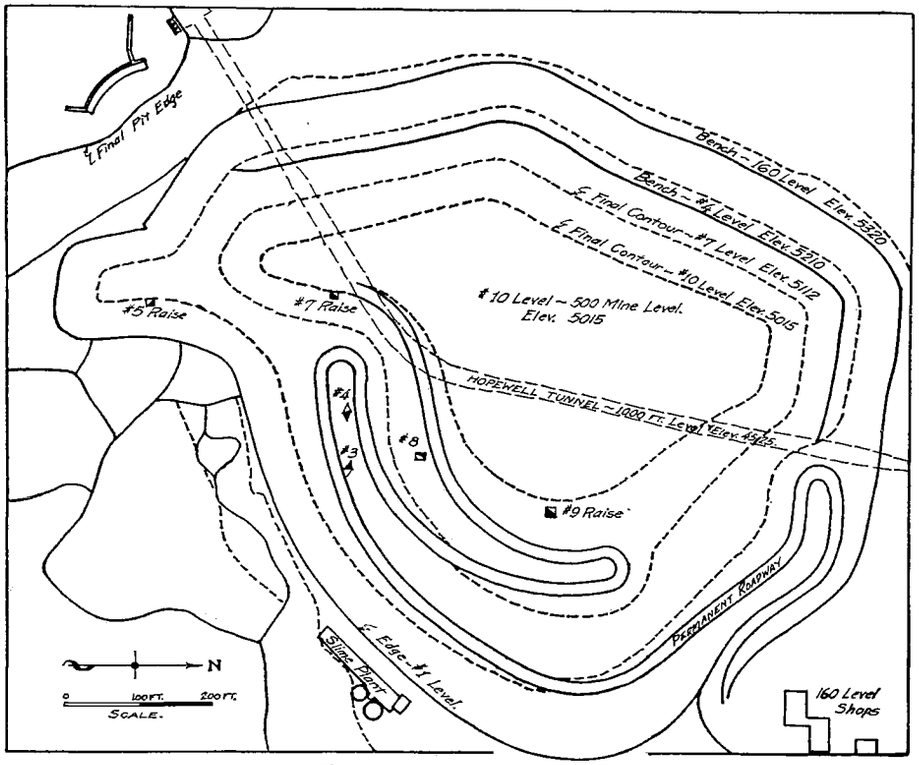
Jackhammers are generally used for the machine drill holes, except in the very hard sulphide material, which required a heavy drifter mounted on a tripod. Toe holes, slope holes, and vertical holes are drilled with the jackhammer, and toe holes with the drifter.

Miscellaneous data on machine drilling is tabulated as follows:

	Jackhammer	Drifter
Depth of hole....	10-20 ft.	10-20 ft.
Drilling crew....	One man	Two men
Powder charge..	100-200 lbs.	100-250 lbs.
Kind of powder used	35 & 50% gelatin	35 & 50% gel'n
Steel used.....	⅞-in. quarter oct.	1¼ - in. hollow round
Gauge of steel, starter	2¼-in.	2¾-in.
Gauge of steel, finish	19/16-in.	1⅝-in.
Length starter...	30-in.	30-in.
Length finisher..	20-ft.	20-ft.
No. of changes..	12	12

Four electric churn drills are maintained in the pit. A churn drill crew

United Verde Copper Company



FINAL PLAN – UNITED VERDE PIT.

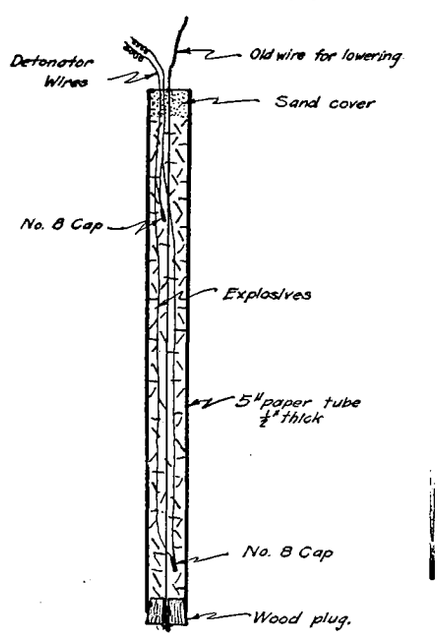
consists of a driller and helper. Six-in. holes are drilled and cased in loose ground if necessary. Holes are spotted close to the edge of the bank, and are drilled about 5 ft. deeper than the height of the bank. The spacing of the holes is varied. In hot ground the holes are placed 4 or 5 ft. apart, as it is necessary to reduce the charge in a hot hole. Drilling speeds vary from ¾ to 8 ft. per hour. During the past year 66,292 ft. of hole were drilled during 2,625 drill shifts, the average footage being 25.25 ft. per shift. Cost of drilling in 1929 was $1.08 per foot.

Blasting Hot Ground

While some hot ground was encountered during the major stripping operations, the seriousness of the situation was not fully realized until work progressed on the first and second levels of the lower pit. Not all of the material in the pit had been affected by the fire, the greater part of the ground broken being cold.

At first the temperature of the hot holes was measured by thermometers. Later a potentiometer with an iron-constanstan thermocouple was acquired. The highest temperature measured in a hole was 780° F. A temperature of 120° F. or lower is considered safe for loading and tamping. The usual methods are also used if the hole can be cooled so that it will remain below 120° F. during loading and tamping. Holes are cooled by running water into them, or placing sand in the bottom of the chamber. In case sand is used the hole must be chambered sufficiently to hold both sand and explosive.

The torpedo method was finally devised for blasting holes that could not be cooled. For small machine drill holes a cardboard tube about 1¼ in. outside diameter and 1 in. inside diameter from 4 to 8 ft. in length is used. In the larger churn drill holes a 4-in. inner diameter paper tube with ½-in. walls is used. A wooden plug is driven into one end, and the tube filled with explosives. No. 8 electric blasting caps are placed in the tube during loading. For the larger tubes a wire is run through the tube and fastened to the bottom for use in lowering into the hole. In shooting around the hot holes, the torpedos are placed adjacent to the holes, wired up, and connected to the blasting line. When the blasting signal is given they are pushed or lowered into the holes, and detonated as soon as the men are in the clear. The explosive is protected from the heat during the short interval they remain in the hot ground. Blasting ground with torpedoes is inefficient as a sufficient amount of powder can not be used, nor can the charge be chambered and tamped. The churn drill holes are preferred in hot ground as a larger amount of explosive can be used, and the men operating the drill do not have to remain constantly on the hot gassy ground.



SECTION OF TORPEDO FOR BLASTING HOT CHURN DRILL HOLES.

Churn drill holes are blasted with 35 and 50 percent gelatin and quarry special No. 1. In hot ground 35 percent gelatin is generally used and is obtained in large sticks 4-in. in diameter and 8-in. long. The large torpedoes contain between 75 and 100 pounds of explosives. Cold holes are chambered. The charge varies from 100 to 500 pounds and is tamped with sand. Seldom more than four holes are shot at a time.

All bank shots are shot by electricity, No. 8 electric caps being used, and detonated from the main 440-volt line. Blasting switches and lead wire reels are located in convenient positions in the pit. Blasting is done at noon, 4 p. m., 8 p. m., and midnight, in order that there will be a minimum delay and so that the underground men will be in the clear.

Considerable secondary blasting is necessary in the harder material. All boulders are blockholed and shot with fire fuse. The cost of the secondary blasting is about 2 cents per ton material broken.

Development of Working Bench

When starting a new cut to a lower level, an attempt is made to confine the cut within one class of material. The cut is headed towards one of the transfer raises. When the cut reaches the elevation of the bench the material can be disposed of by casting directly into the raise without having to be hauled. The grade of the development cuts have varied from 10 to 20 percent. The bench is then advanced toward another raise, the first raise being cleaned out and changed over to accommodate the various classes of material encountered.

Pit Operation

During the past year Nos. 1 and 2 levels were completed, and the approach cut to No. 6 level started. At the end of the year the depth of the pit was 40 ft. above the 400 mine level. Material was removed principally on Nos. 3, 4, and 5 levels. An average of 2,500 tons of direct smelting ore, 260 tons of concentrating ore, 510 tons of converter ore were mined each day. In addition approximately 2,860 tons of stripping material were removed, making the total daily production about 6,130 tons.

The work in the pit is laid out by the general foreman in conjunction with the pit engineer and sampler. Each morning the smelter requirements and the condition of the dumps at Hopewell for the stripping material are determined. In cooperation with the foreman of underground tunnel transportation, a schedule is arranged. Raises are assigned for different classes of material. Smelting ore, converter ore, and concentrating ore are dumped into separate bins at Hopewell and transferred to the V. T. S. R. R. for shipment to the smelter. The stripping material is classed as leach, low-grade converter, and waste. Separate dumps are maintained for their disposal.

Classification and Disposal of Material

Smelting ore is composed chiefly of sulphide and siliceous sulphide material above 1 percent copper. Burnt schist and

low silica content stope fill above 2 percent copper are mixed with sulphides when necessary. The average copper content of the smelting ore for 1929 was 5.3 percent copper. Converter ore is highly siliceous material above 60 percent silica and has a value of $4 or more in copper, gold, and silver. The mineralized schist and porphyry on the south and east sides of the ore body containing above 1.0 percent copper are shipped for milling. Care must be taken to mine the milling ore clean without contamination by burnt ores or oxidized stope fill.

It is necessary for the pit department to control the mixture of the basic and siliceous smelting ores so that when the ore arrives at the smelter the silica content will be about 23 percent. This is not always possible, but a fair degree of accuracy has been maintained.

Some low-grade material between 0.7 and 2.0 percent copper not suitable for the smelter is run out on the leach dump at Hopewell. Mine water is run through the dump to take the copper into solution. The solution is later collected and the copper deposited by scrap iron. Low-grade siliceous material below $4 in value is stored on another dump. A waste transfer raise connects the underground workings with the pit. Barren material is dumped in the raise when needed. Excess waste is placed on the waste dump at Hopewell.

A sampler is maintained on day shift for sampling the banks as cleaned up. Generally by a knowledge of the geology and the assays obtained the class of material can be accurately determined. This information is imparted to the powder foreman, who is responsible for breaking the ground properly. The sampler and foremen supervise the loading and tramming of the material.

A truck dumper is stationed at the raise, who fastens the safety chain to the truck, keeps a record of the loads dumped, and inspects the dump body. Some of the material is crushed when falling through the raise. The larger pieces are caught on grizzlies on the 900-ft. level, where they are blockholed before dropping into the ore bins.

Occasionally the raises will become clogged by the coarser rock, or packed and cemented by the finer material, if allowed to remain above the grizzlies. To eliminate this the material is drawn out as fast as put in. No piece is loaded which will not pass through the shovel dipper, and an attempt is made to mix the finer with the coarser. If the larger rocks pile up on the grizzlies, dumping is stopped until they can be blasted. The raises are accessible through doors on the various mine levels, so the material can be worked down if hung up.

Fire Problem

During the past year fire conditions in the pit improved. While some burning stopes were encountered on the fourth level, not as much of the burning ore was encountered as on the upper levels. In 1926 the fire in some areas in the pit was so severe that the material could not be loaded. In other places where loading was possible, the sulphide dust would explode when being dug or when dumped into a raise. Gas masks, respirators, and double crews were tried without success. In 1927 slimes from the mill at Clarkdale were shipped to the pit and distributed on the surface and pumped into churn drill holes extending into the fire areas. The slimes solved the fire problem, and in 1928 a slime mill was constructed adjacent to the pit. The plant consisted of a jaw crusher, rolls, and ball mill, together with storage tanks. The mill was operated in the early part of the past year and proved invaluable in putting out a serious fire which occurred in the underground workings.

Future Lower Pit Operations

If the pit walls stand up it may be necessary to carry the operations below the 500 level, probably to the 700 level. The production from the pit will steadily decrease as greater depth is attained, for the working areas will be limited. During the past year the stripping yardage remaining has been decreased considerably, which will also decrease the total production. It is estimated that an average of 40,000 cu. yds. will be removed monthly. At this rate it will require four years more before the pit is completed to the 500 level.

Costs and Data of Lower Pit Operations

A summary of direct operating costs based on total yardage for the year 1929 is tabulated as follows:

	Labor	Comp. air drills & steel *	Power	Explosives	Repairs †	Supplies ‡	Total
Shovel operation	$0.054		$0.012		$0.113	$0.002	$0.181
Drilling and blasting	0.151	$0.085	0.002	$0.073		0.029	0.340
Truck tramming	0.074				0.169	0.030	0.273
Total	$0.279	$0.085	$0.014	$0.073	$0.282	$0.061	$0.794

* Includes repairs to drills and sharpening steel and drill bits.
† Includes labor, supplies, and power used in repairing.
‡ Includes gasoline and oil used by trucks.

These direct costs are charged against stripping and mining in proportion to the tonnages. In addition to the direct cost per ton of ore, a charge is made for each ton of ore when mined to pay for the removal of the stripping material.

The direct costs for stripping and mining for the year 1929 are tabulated as follows:

	Cu. yd.
Stripping 465,919 cu. yds.:	
Shovel operation	$0.1620
Drilling and blasting	0.3497
Truck tramming	0.2444
Dump expense	0.0728
Miscellaneous	0.0086
Total	$0.8375
Mining 433,847 cu. yds., 1,055,684 tons:	
Shovel operation	$0.2025
Drilling and blasting	0.3281
Truck tramming	0.3033
Total ($0.3428 per ton.)	$0.8339

The cost of passing through raises and tunnel transportation is not included in the above costs.

General data on all operations for the year 1929 is as follows:

Total 8-hour shovel shifts	2,277
Total 8-hour truck shifts	6,757
Pounds explosives per cu. yd.	0.436
Total man-shifts	69,730
Cu. yds. per man-shift	13.69
Tons per man-shift	29.66
Labor cost per cu. yd.	$0.4212
Operating supplies and explosives	0.1249
Repair supplies per cu. yd.	0.1827
Total supplies per cu. yd.	0.3076
Total repair cost per cu. yd.	0.3390

United Verde Copper Company

Surface Plant, 500 Level

MINE SURFACE

at the

THE mine surface plant was built new in its entirety in 1919-20 when the open pit work was started, which necessitated abandoning the old plant. The mine was at that time quite well developed, so it was possible to determine accurately the plant requirements. The result was a well-planned and well-built plant sufficient for all probable needs of the mine.

The plant is located on what is known as the 500 level It has standard gauge rail connection wit hthe V. T. & S. Railroad, and standard and narrow-gauge connection to the main shaft of the mine through a tunnel 1,500 ft. long.

All buildings are of steel frame construction, with brick and steel sash enclosures. Good light and ventilation have been provided for by almost solid steel sash walls and glazed monitors on all roofs. All buildings are steam heated from a central heating plant. The principal buildings and equipment are as follows:

Air Compressor Plant:

Operating floor area, 8,000 sq. ft.

1 6,800-cu. ft. compressor, 1,200-hp. motor, direct connected

1 3,000-cu. ft. compressor, 600-hp. motor, belt drive.

1 1,500-cu. ft. compressor, 300 hp. motor, belt drive.

1 Aftercooler, 360 cu. ft. capacity, 1,300 sq. ft cooling surface.

4 Receivers, 1,540 cu. ft. capacity.

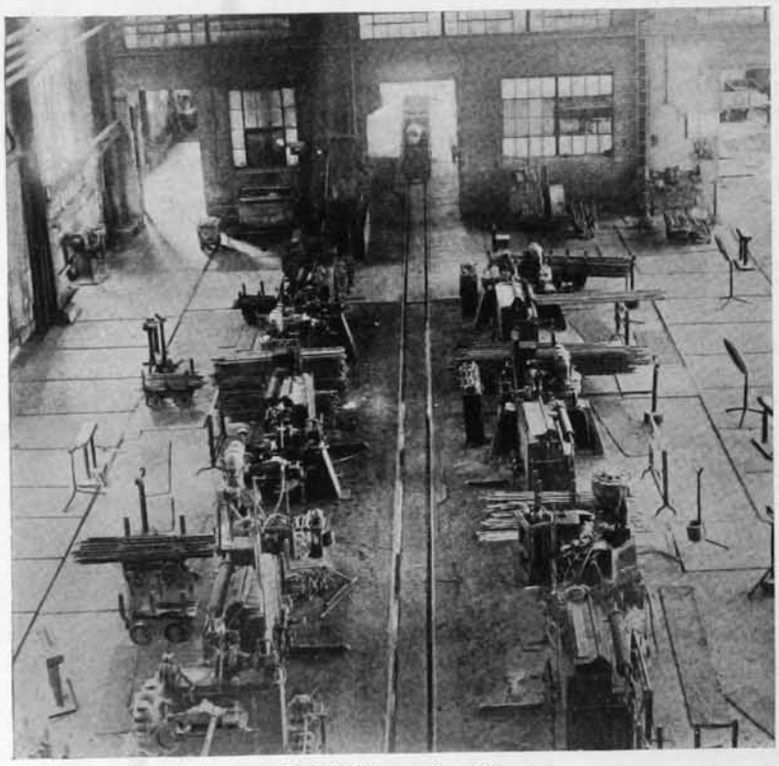
3 Circulating pumps, 1,000 g. p. m. each.

1 Atmospheric cooling tower, 15,000 cu. ft. volume.

1 Lime and soda water treater, 1,500 gals. per hr.

1 20-ton crane, 62-ft span.

This plant furnishes 100 lbs. air to the mine, open pit and surface plant.



Drill Sharpening Shop

Gilman C-21 Heat Treating Machine

PLANT

United Verde Mine

By H. V. Kruse
CHIEF ENGINEER,
MINE DEPARTMENT

Another View of 500 Level Surface Plant

Machine Shop:

Floor area, 14,600 sq. ft.
1 25-ton rane, 75-ft. span.
1 5-ton crane, 20-ft. span.
1 300-ton locomotive jack.
1 90-in. driving wheel lathe.
1 400-ton wheel press.
7 Lathes of various sizes.
1 Planer
2 Shapers.
1 Boring mill.
1 Slotter.
4 Drill presses.
1 Milling machine.
1 Tire press.

This shop repairs everything from 300-ton locomotives to small hand tools, and manufactures considerable equipment.

Boiler Shop:

4,800 sq. ft. of floor surface.
1 12-ft. bending roll.
1 12-ft. flanging clamp.
1 36-in. punch and shear.
1 48-in. punch and shear.
1 48-in. cold saw.
Complete arc welding and oxy-acetylene equipment.

Blacksmith Shop:

2,700 sq. ft. of floor space.
2 1,100-lb. steam hammers.
1 2,500-lb. steam hammer.
4 Oil furnaces.
5 Fires.

Drill Sharpening Shop:

8,500 sq. ft. of floor space.
7 Drill sharpening machines.
7 Oil furnaces.
2 Automatic heat treating machines.

Change House:

20,000 sq. ft. of floor space.
The change house has a capacity of 1,500 men. Each man has an individual locker and an overhead clothes dry hanger. The building is heated by means of steam heated air forced up through openings in the floor under the lockers. A 25,000-cu. ft. per minute fan furnishes the air; this gives four changes per hour, which keeps the air in good condition and assures rapi ddrying of the clothes.

Timber Handling Plant:

All timber is unloaded from cars and stacked by a locomotive crane. Timbers are framed at the yard and loaded on special mine cars handled by storage battery locomotives from the yard to the tunnel portal yard, where they are picked up by trolley locomotives and delivered to the shaft. This yard handles about three million board-feet of lumber annually. About one million feet annually are treated with zinc chloride. The plant for this purpose consists of a single retort, 5½ ft. in diameter and 32 ft. long, with all the necessary pressure and vacuum pumps, tanks, etc. The average absorption of zinc chloride is 0.644 lbs. per cu. ft. of timber. The cost to date has been $8.20 per 1,000 ft. b. m.

Miscellaneous Buildegs:

Mine Shop, 6.500 sq. ft. floor space; houses pipe shop, roustabout shop, paint shop, diamond drill repair shop, and drill repair shop.

Electric Shop, 4,700 sq. ft. floor space.

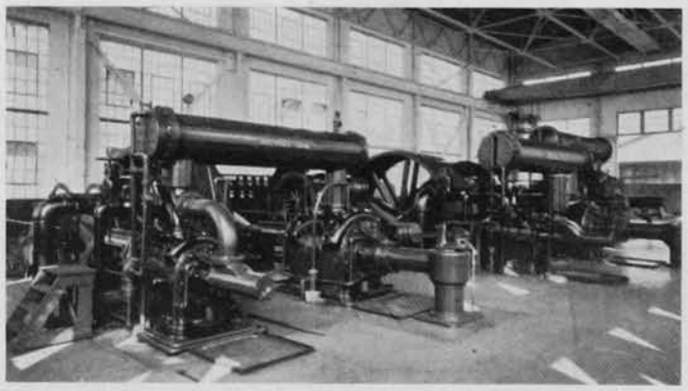
Cable Repair Shop, 1,600 sq. ft. floor space.

Warehouse, 14,000 sq. ft. floor space.

Heating Plant, 3,000 sq. ft. floor space.

Carpenter Shop, 3,000 sq. ft. floor space.

Framing Shop, 1,900 sq. ft. floor space.



Compressor, 500 Level Plant

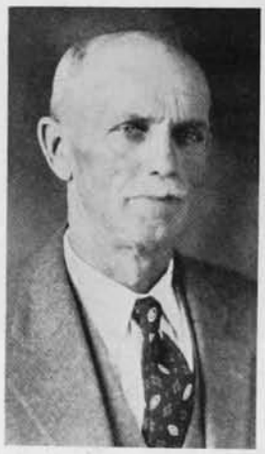
Interior of 500 Level Machine Shop

United Verde Copper Company

VENTILATION and PROTECTIVE

By O. A. Glaeser
SAFETY AND VENTILATION ENGINEER

O. A. Glaeser



Thos. Dennison

G. W. Mitchell

THE United Verde Mine is, perhaps, a pioneer in the Western Hemisphere in the mechanical ventilation of metal mines. Mechanical ventilation became absolutely necessary in the early days, some 35 years ago, because of mine fires which caused the shutting down of a large portion of the mine at that time and which are still burning in isolated places. Large sums of money were spent on ventilation in an attempt to work the "fire stopes," and they were worked successfully for a number of years, though at an extremely slow rate and at high cost.[1] Mining in this fire country was stopped and the area sealed off when steam shovel operations were started in 1918.

For ventilation purposes the mine is divided into two parts. The upper levels from the surface to the 700 level and the lower levels from the 700 to the 3,000 level.[2]

For some years after the ventilation system of the first country had been abandoned, the upper levels depended almost entirely for their ventilation upon natural air currents and return air from the lower levels. This was found adequate until the schist ore body was developed. The remoteness of the main ventilating system made a new fan installation necessary. A No. 9½ American Blower double inlet, non-reversible, backward curved blade fan was chosen, capable of producing 75,000 cu. ft. per minute at a pressure of 3½ in. of water. It is direct connected to a 100-hp. motor. It was placed on the 500 level adjacent to the schist ore body with the Dillon Tunnel, a long cross cut to the surface, as an intake, and connected to distribution raises to the 700. Regulator doors were placed in this raise on the 500, 600, and 700. No return airway was required as the country to be mined extended only from the 700 to the 160 level and the air finds its way back to the surface through the stope raises. There was a marked decrease in temperature and humidity of the air in this area after the installation of this fan.

The present system below the 1,000 level was started about 1916. On account of increased depth the original fan became inadequate and in 1927 the present fan was installed. The present equipment, located on the 1,000, consists of a double inlet, reversible type, forward curved blade Jeffery fan. The rotor is 4 ft. 6 in. by 9 ft. and is direct connected to a G. E. 400-hp., 2,200-volt, 280-r. p. m., 3-phase, 60-cycle, slip ring motor. A Westinghouse type CW wound rotor induction motor, of 350-hp., 2,200-volt, 350-r. p. m., 3-phase, 60-cycle, is belt connected as an auxiliary motor. It can be cut in at a moment's notice by means of a clutch on the drive shaft. *Figure 1* shows this installation. Both motor controls are on one switch board. A push button cuts the power from one to the other. Each motor has three speeds, operated by a push button type control with resistance banks in the secondary circuit. This fan is capable of forcing 297,000 c. f. m. into the mine at a pressure of 3.6 in. It has a dual electric feed; one line down No. 6 Shaft and one down from the surface through a churn drill hole to Hopewell Tunnel on the 1,000. With the three speeds on the belt drive, volumes range from 187,000 c. f. m. on low speed to 222,000 c. f. m. on high speed and with the direct drive, from 182,000 c. f. m. on low speed to 297,000 c. f. m. on high. In the hot summer months high speed on the direct drive is used and in winter second speed on the belt drive. A regular schedule is followed 24 hours of the day. In winter the fan is run at second speed belt drive on the two regular shifts, day and afternoon. Between shifts and at lunch time it is operated at low speed to slow down the air currents during blasting time. Most of the heavy blasting in massive sulfide must be done on the afternoon shift on account of the dangerous gases generated. In order to clear the mine of these gases quickly the fan is run at high speed for two hours at the beginning of the graveyard shift. It is then reduced to low speed to save power, as very few men work on this shift. The fan is attended on all three shifts by fan watchmen who, in addition to running the fan, must fire-patrol the 1,000 level.

The intake raise, rectangular in section, was especially driven for the purpose from the 1,000 level to the surface. It is 980 ft. deep with an effective cross section of 169 sq. ft. Only one connection to the mine above the 1,000 was made, through which work was carried on to the surface. This level is sealed from the raise by two iron doors set in concrete, which are kept locked so that there is no possibility of smoke or gas entering. The walls of the raise are supported by concrete rings 3 ft. wide spaced 4 ft. apart. The saving of the rings over solid concrete is estimated at 40 percent. This saving was thought to offset the increased resistance of the rings at the present range of air velocities. For perfect control of intake air a multiple leaf damper has been installed on the pressure side of the fan.

The main delivery air shaft (No. 3) is connected to the fan by an inclined raise between the 1,000 and 1,200 levels and has an effective cross section of 80 sq. ft. No. 3 Shaft was originally a hoisting shaft to the 1,950 level. Below the 1,950 it consists of a series of connected raises to the 3,000 level. The shaft is in good ground requiring no support and, like the intake shaft, it has been stripped of all combustible material.

At the junction of each level and No. 3 air shaft a reinforced concrete bulkhead is built which contains an iron door to regulate the volumes taken off. The latest type is shown in *Figure 2*. It consists of a 6-in. channel frame and a boiler plate door with a feed screw by which the door can be set at any required opening. It can be closed practically air tight and held so by lugs and wedges welded to the frame. Table I gives the main fan air distribution.

The air on each level is guided to the working places by means of doors made of 2-in. special tongue-and-grooved lum-

[1] Tech. Paper No. 117, A. I. M. E., Mine-Fire Methods Employed by the U. V. Co., 1916, R. E. Tally.

[2] Tech. Paper No. 199, Ventilation at the United Verde Mine; A. I. M. E., 1929; O. A. Glaeser.

MEASURES against MINE FIRES

Thomas Dennison
MINE FIRE FOREMAN

and G. W. Mitchell
ASSISTANT SAFETY AND VENTILATION ENGINEER

ber, well cleated together and hung on a 6-in. by 8-in. frame set in concrete. Each door is provided with a latch and counter weight to keep it closed. Doors are placed in critical places and may be left open or closed depending on requirements. There are about 500 of these doors in the mine. Air is conducted on the levels, as in most mines, through the haulageways from the delivery air shaft to the stopes. As air follows the path of least resistance it is natural that the stopes nearest the delivery air shaft should get a large volume and that those farthest should get very little. The volume to each stope is best controlled at the outlet from the stope to the level above. This is done by means of various types of iron, screen and wood raise coverings.

TABLE I—MAIN FAN AIR DISTRIBUTION

Level		Volume
800 (ventilated by return air)		
900 (ventilated by return air)		
1,000	Cu. ft. per min.	24,000
1,200	do.	11,000
1,350	do.	9,500
1,500	do.	8,500
1,650	do.	16,500
1,800	do.	23,000
1,950	do.	15,500
2,100	do.	16,500
2,250	do.	31,500
2,400	do.	40,000
2,550	do.	18,000
2,700	do.	10,000
2,850	do.	7,500
3,000	do.	8,500
Total	Cu. ft. per min.	240,000

With the exception of the downcast intake airways, all raises and shafts must be upcast. The deadly gases [3] generated by blasting in massive sulfide make it imperative that blasting start on the top levels and progress downward; and that all air currents be upcast. A downcast raise would carry these gases down to the level below and catch men who are waiting their turn to blast.

Prior to 1928 no provision had been made for the direct return of vitiated air. Air leaving the bottom levels fouled the fresh air entering stopes in the middle section of the mine. Aside from eliminating sulfurous fumes resulting from blasting in massive sulfide, powder smoke and dust from blasting in other classes of rock and eliminating air of high humidity, a return system was considered necessary to reduce the underground resistance to the passage of air. In 1928 a return system was put in use consisting of 600 ft. of new raise, old raises that had been enlarged and an old shaft (No. 4) to the surface which had all been stripped of timber. The effective cross section of this return is 120 sq. ft. Gathering levels were established

[3] Tech. Paper No. 276, A. I. M. E., Protective Measures Against Gas Hazard at the U. V. Mine; O. A. Glaeser, 1929.

[4] Tech. Paper No. 276, A. I. M. E., Protective Measures Against Gas Hazard at the U. V. Mine; O. A. Glaeser, 1929.

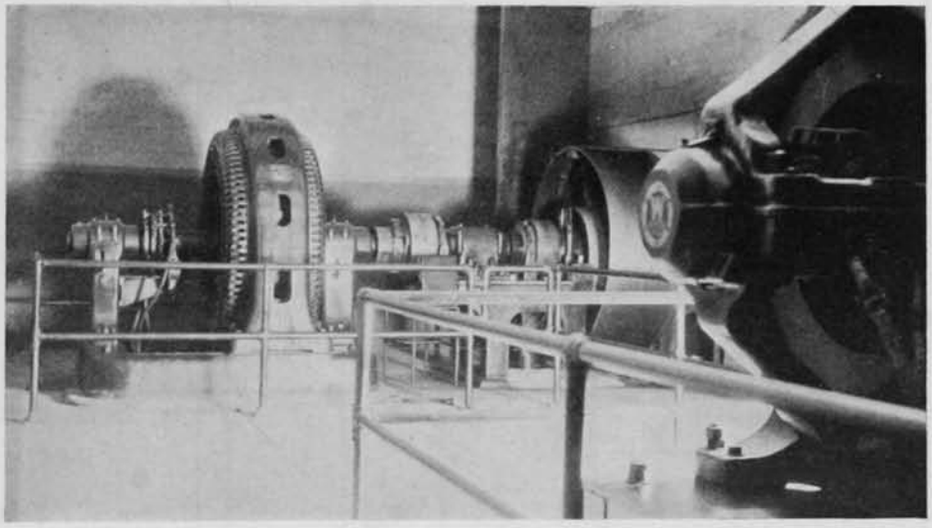


Figure 1. 1,000-level Fan Motor Installation

on the 2,100, 1,500, 1,350 and 1,200 levels. Practically all of the air entering below the 2,100 is taken into the return on the 2,100 allowing nothing but fresh air to enter on the 1,950. Vitiated air from the 1,950, 1,800 and 1,650 levels enters the return on the 1,500. This return system has aided greatly in removing quickly noxious gases and foul air from the mine.

In order to stop ascending air from passing from the 2,100 to the 1,950 and to stop the possible ascent or descent of fire in this horizon a fire break was established on the 1,950 level. About 30 ft. of the 2,100 level stopes below the 1,950 sill were left as a horizontal pillar. All the 2,100 raises on the 1,950 level were sealed with reinforced concrete collars and fireproof iron doors which are kept closed. The end stopes on the 2,100 have been carried through but an effective seal has been made by filling all chutes and manways. Fire breaks formerly existed on the 1,650 and 1,200 levels but they are now being destroyed in the process of mining the horizontal level pillars.

A recent survey showed that approximately 20 percent of the air from the main system was being used to ventilate old workings. At the present rate of mining the main fan equipment would become inadequate in a few years if nothing were done to save this air. A program of bulkheading these old areas is now in progress. As far as possible bulkheads are being placed not only to prevent loss of air but to isolate country where there is danger of fire. Bulkheads are of three types. Solid reinforced concrete bulkheads which vary from 8 in. to 3 ft. in thickness, hollow concrete block bulkheads 8 in. or 1 ft. thick and solid concrete bulkheads containing an iron door and frame. Fire stops are of the first type, air stops of the second and fire and air stops, where entry into the old country is required for inspection, of the third type.

The service shaft (No. 6) and the ore hoisting shaft (No. 5) are wholly underground. Both the shafts and their stations are of fireproof construction. They are cut off from the delivery air shaft by air locks to prevent the short circuit of fresh air. Leakage air to No. 5 shaft is utilized on the 1,000 level to help ventilate Hopewell Tunnel. Leakage to No. 6 shaft is utilized to ventilate old workings on the 500, 600 and 700 levels.

In the cross-cuts between the delivery air shaft and the stopes below the 1,000 level "fire doors" have been placed. They are painted red and stenciled in English and Spanish:

"In case of fire close this door after all men have passed through to No. 6 shaft station."

This door cut off fresh air from the stopes and forces more leakage air to No. 5 and No. 6 shafts and stations so that these stations and shafts will be in fresh air under all conceivable fire conditions. A door on the pressure side of the fan on the 1,000 level No. 6 shaft station can be opened to put more fresh air pressure on No. 6 shaft or if there should be smoke in No. 6 shaft (which is not likely) it can be swept clear in a moment from any level by opening the doors between No. 6 and No. 3 air shaft. As both the intake, distribution air shaft and No. 5 and No. 6 hoisting shafts are absolutely fire proof, it is not conceivable that a fire disaster can occur. If men should be caught in the stoping area they may find refuge in "gas chambers," several of which are provided on each level.[4]

No systematic effort was made prior to 1927 to ventilate dead ends. Several fans were provided for this purpose but little attention was paid to auxiliary ventilation and its efficiency was low. In 1928 it was decided that the type of fan which had been in use was too cumber-

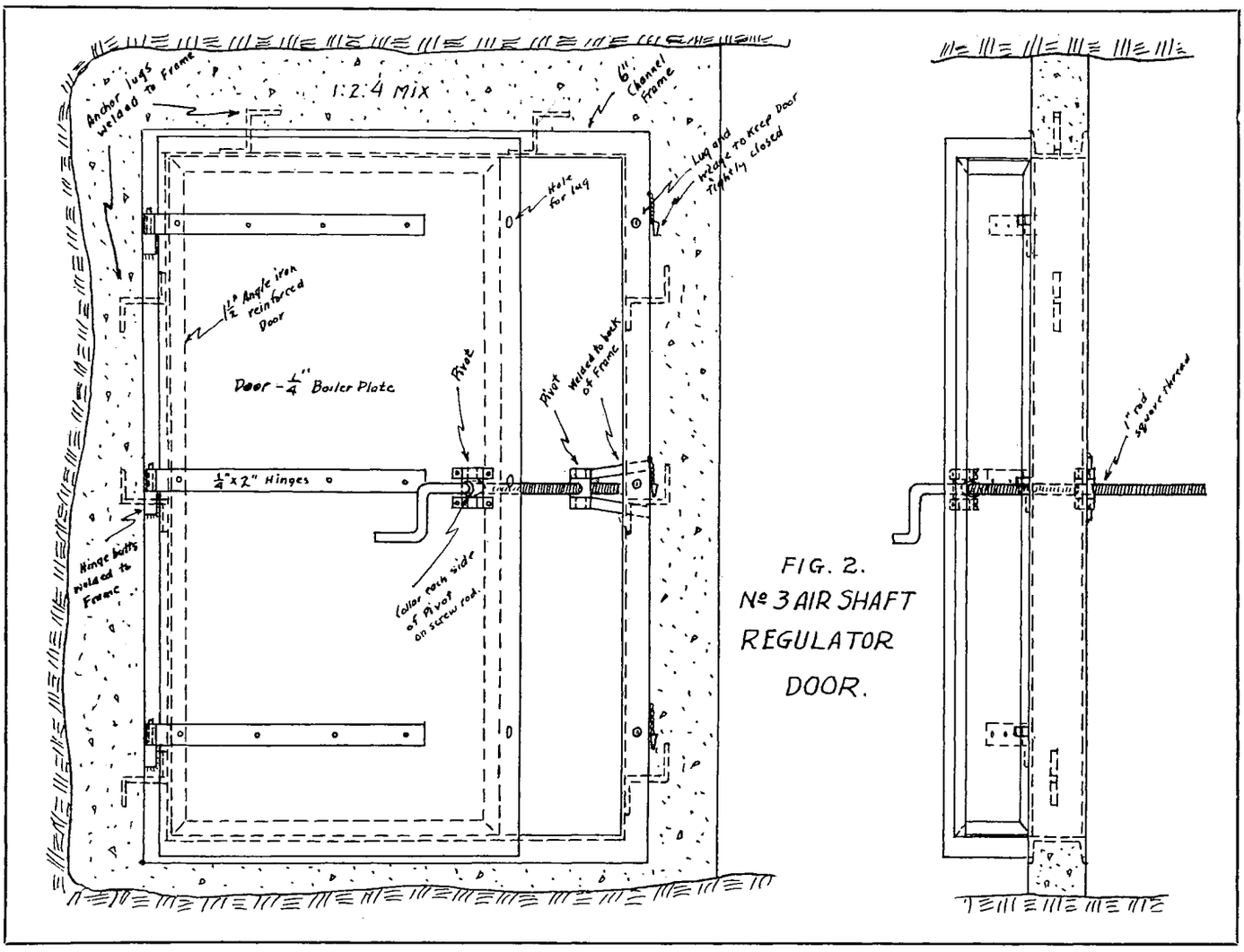


FIG. 2. Nº 3 AIR SHAFT REGULATOR DOOR.

some and would not produce large enough volumes on high resistance lines. A new high speed, light weight, backward curved blade, centrifugal fan was tried out and found to be satisfactory. For temporary ventilation lines or for ventilating short drifts and raises a 12-in. compressed air injector type of blower is being used and found highly satisfactory. It is of very light weight, can be placed in a few minutes and is economical in the use of compressed air. It is used in conjunction with a 12-in. flexible tubing. Six-in. injectors of this type and 6 in. galvanized pipe are used in many gassy raises and have proved satisfactory. At the present time there are 35 headings which are being ventilated by 15 fans and 20 injectors. Ten new high speed fans have been ordered and will replace some of the injectors now in use. There has been a gradual change from galvanized iron pipe to flexible tubing for air ducts. Sixteen in. tubing is used with a "blasting end" of about 150 ft. of 12-in. tubing for convenience in removing before blasting. It is estimated that $3,000 was saved in 1929 in labor and material by the use of flexible tubing. About 3,000 ft. of metal pipe are still in use.

A problem in auxiliary ventilation presented itself in 1928 which is worthy of special note. A remote part of the 2,100 level, a gathering level, had to be supplied with fresh air. (See *Figure 3.*) The return air course, 16 ft. wide, was divided by a wall and fresh air overcast to a footwall drift which conveyed the fresh air to its destination. The walls of the by-pass and overcast were constructed of 6 by 8 stulls lagged with 1-in. boards. The boards were covered with expanded metal lath which in turn was covered with two coats of gunite. The fresh air course is under positive pressure so that no return air can enter. This proved to be an excellent solution of the problem.

No attempt to condition intake air has been made to date but it is believed that better results would be obtained in the summer when intake air temperatures are as high as 100 degrees Fahrenheit if it were cooled. Some experimenting has been done with water sprays with unsatisfactory results but will be continued in the future. With the exception of Hopewell Tunnel, through which all ore is hauled, dust has not been troublesome. In this tunnel a large volume of air is used to sweep away the dust resulting from loading shovel pit and mine ore. Sprays are used to settle dust on level grizzlies and skip pockets.

The ventilation department is under the supervision of the Safety and Ventilation Engineer. He has an assistant and a ventilation jigger boss who spend most of their time on ventilation. The construction crew consists of six timbermen and six helpers who build doors, bulkheads and do other construction work.

Table II gives costs of ventilation from the year 1923 to 1929:

TABLE II—VENTILATION COSTS, 1923-1929

Year	Total cost	Cost, ton
1923	$47,785.06	$0.060
1924	90,145.46	0.085
1925	102,834.93	0.107
1926	137,317.17	0.173
1927	115,099.73	0.160
1928	93,735.84	0.120
1929	106,728.27	0.147

These costs seem high but include all work chargeable to ventilation. In 1929 they include the extension of No. 3 air shaft from the 2,400 to the 3,000 level and all construction work resulting from the opening of four new levels. The cost for 1926-7-8 are high because of the extensive improvement carried on during that period.

A set of 30 scale maps of the entire mine are kept in the safety and ventilation engineer's office. They are kept up to date and new doors, bulkheads, direction of flow of air and quantities are entered on them. They are on tracing cloth so any new development can be readily traced off from the mine engineer's maps. These maps have proved themselves invaluable in emergencies. Temperatures, humidities and volumes are kept on a card file.

Good ventilation reflects itself in the health of the miners. This can be shown to a limited extent by the number of gas cases reported by the safety department.

United Verde Copper Company

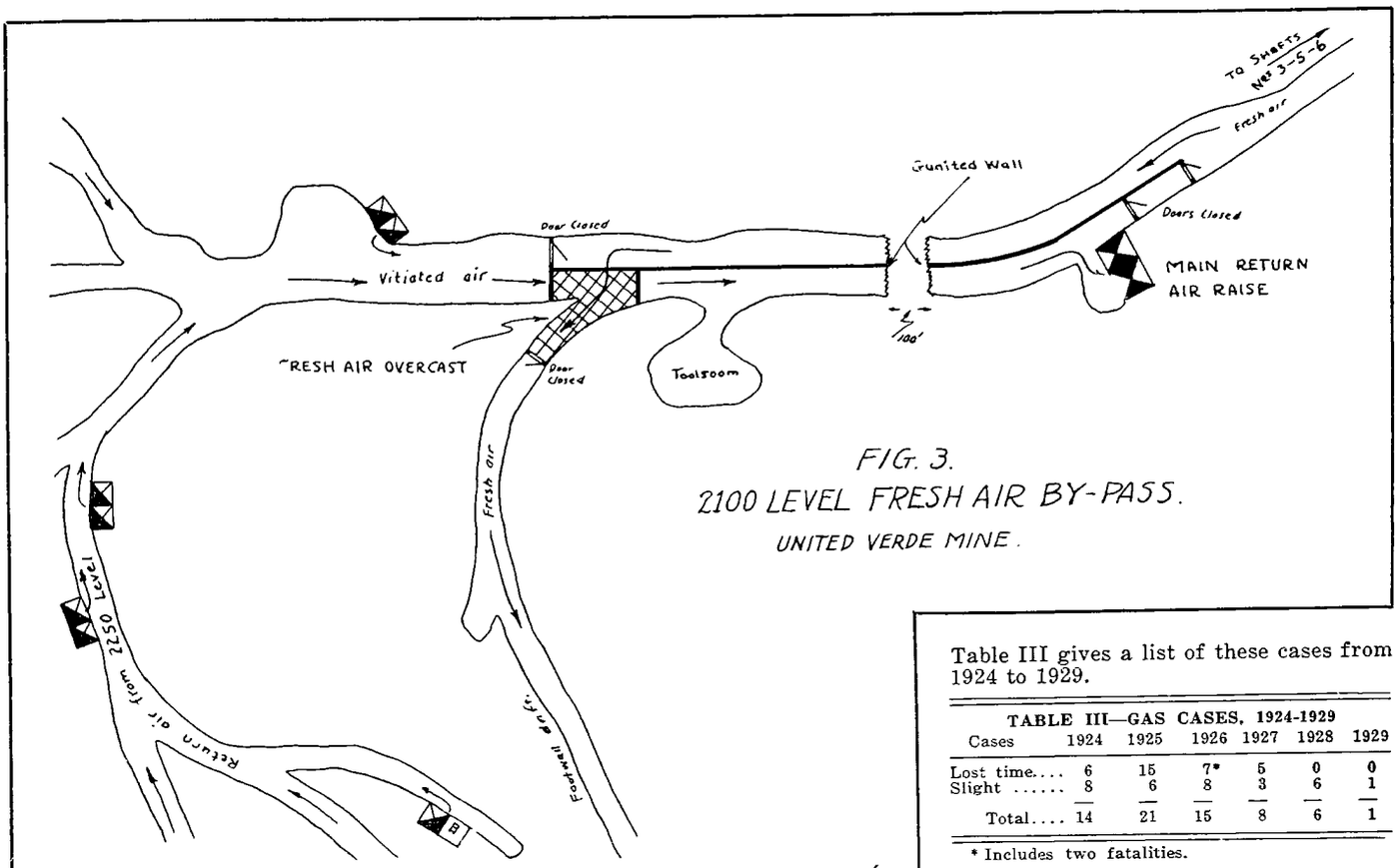


FIG. 3.
2100 LEVEL FRESH AIR BY-PASS.
UNITED VERDE MINE.

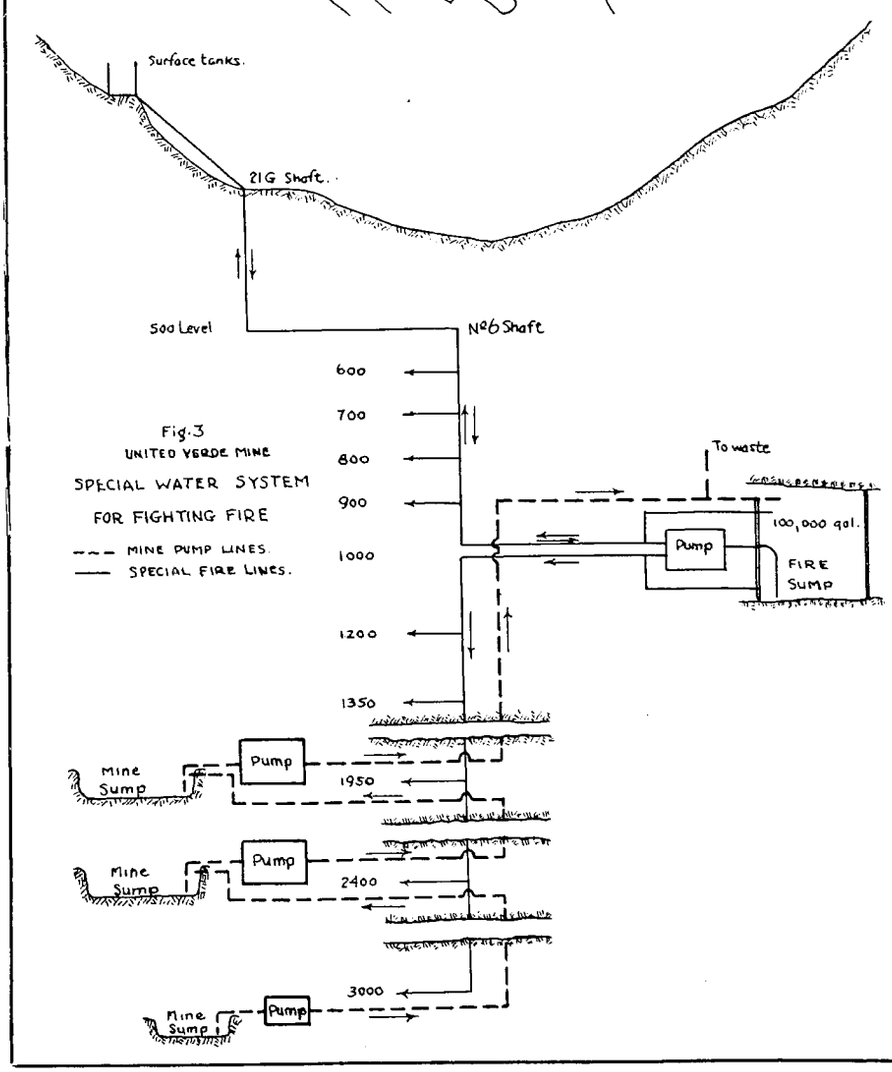


Fig. 3
UNITED VERDE MINE
SPECIAL WATER SYSTEM
FOR FIGHTING FIRE
--- MINE PUMP LINES.
—— SPECIAL FIRE LINES.

Table III gives a list of these cases from 1924 to 1929.

TABLE III—GAS CASES, 1924-1929

Cases	1924	1925	1926	1927	1928	1929
Lost time....	6	15	7*	5	0	0
Slight	8	6	8	3	6	1
Total....	14	21	15	8	6	1

* Includes two fatalities.

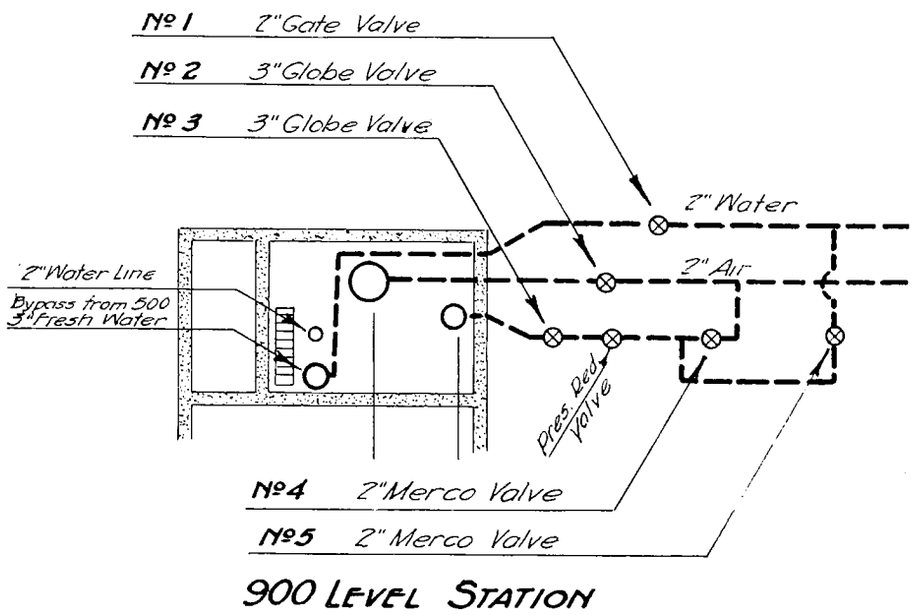
Mine fires have proven to be costly obstacles to ore extraction within limited areas at the United Verde. In recent years a concerted effort has been made, first, to do everything possible to prevent underground fires, and second, to be so equipped that a fire will not get out of control. The mine fire work has been organized to accomplish this, and its activities may be classified as follows:

1. Inspection.
2. Fire patrol.
3. Fire control measures.
4. Fire signals and warnings.
5. Water and slime system.
6. All other fire fighting equipment.

A fire foreman makes regular inspection tours of the mine. These inspections include good housekeeping, proper maintenance and distribution of some 3,000 ft. of special high pressure 1-in. hose, and adequate water lines and connections.

The fire patrol consists of seven full time underground watchmen who have a definite group of levels to patrol, and three fan watchmen who have also one level to patrol. Patrol stations are located at certain important points to which the watchmen must go a specified number of times each shift. Here they leave a signed card giving date, hour and condition. These cards are collected several times each week. The watchman must be familiar with the location of hose, water connections and control valves. They are also acquainted with the manipulation of fire extinguishers.

Fire control measures include fire doors, bulkheads and firebreaks. These are so closely allied with the ventilation that they have been covered in that part of this article. Iron doors hung in a concrete frame are placed only where it is felt that a wooden door would fail because of its combustible construction. Thus adjacent to timbered areas iron doors would be used where control is con-



900 LEVEL STATION

NORMAL OPERATION:
Valves Open—No. 1 Valves Closed—No. 3, No. 4, No. 5

TO PLACE FRESH WATER IN AIR LINES 900 LEVEL:
First Operation—Close Valve No. 2
Second Operation—Open Valve No. 4 and Valve No. 5

TO PLACE FIRE WATER IN WATER LINES 900 LEVEL:
First Operation—Close Valve No. 1
Second Operation—Open Valve No. 3 and Valve No. 5

TO PLACE FIRE WATER IN AIR LINES 900 LEVEL:
First Operation—Close Valve No. 2
Second Operation—Open Valve No. 3 and Valve No. 4

Figure 4. Fire System Valve Diagram—900 Level

sidered necessary in case of fire. All ventilation is arranged with two purposes in view: To furnish an adequate supply of fresh air and to give maximum protection and control in case of fire.

It is felt that even good protection does not necessarily prevent disasters and that therefore an efficient warning signal system must be provided. In this mine the electric lights have been used for this purpose. Nine flashes given three times is the fire signal. It is now planned also to use the stench method of warning in combination with the light signals in order to obtain a more general distribution of the warning.

Fire drills are held at regular intervals of three months. All men who happen to be in the mine at such time must go to the shaft station. All fire control doors are set as in case of a real fire, and each shiftboss must report all his men "out" before anyone is permitted to return to work. A check on the time required to get all the men to the shaft station from the time the signal is given ranges from six to eight minutes. The fan is also periodically reversed to see that this equipment is in proper working order and to keep certain men acquainted with the method of doing this.

A special 5-in. water line has been installed for fire fighting purposes (see *Figure 3*). As will be seen this line is connected to the mine drainage pumping system and to a special sump and pump on the 1,000-ft. level. This system is so arranged that water can be used underground from the tanks on the surface or pumped from the 1,000-ft. level to any other level in the mine or to the surface tank if desired. Arrangements for the utilization of the mine drainage water was considered because of the occasional water shortage experienced in the district. The 5-in. line is connected to the air and water pipes on all levels so that large volumes of water are obtainable on any level by the manipulation of certain valves. In order to avoid confusion which might result from setting the wrong valves, framed blue prints of all pipe connections are hung on each station. The valves are numbered and marked to correspond on the blue prints. Instructions for the manipulation of the valves for any kind of service are also shown on the prints. (See *Figure 4*).

Slimes were extensively used on a mine fire early in 1929. They proved so effective that it was decided to use them in any major fire. With this in view a special line will be placed in No. 6 shaft to connect with the slime line already installed. No connections will be made on the levels as but little time will be required to place a lateral when needed. The fine grinding plant which is used to make slimes was built several years ago when it was attempted to fill the old fire stopes with slimes.

All other fire fighting equipment is of a portable nature. Eighty 2½-gallon Foamite fire extinguishers have been placed at various points in the mine. Carbontetrachloride extinguishers are located wherever electrical machinery is used. Brattice cloth is kept at several stations in the mine, and 100 concrete blocks for the construction of bulkheads are kept on each level. The listed equipment is mounted on mine trucks and is kept ready for immediate use:

1 High pressure oxygen pump with two oxygen tanks.
2 Low pressure fans.
2 Foamite generators with hose and 160 pounds of mixture.
2 Telephones with 1,000 ft. of wire.
2 Tool trucks, containing:
  Tools.
  Pipe fittings.
  Extension light cords.
  Hose.
  Brattice cloth.
  Life lines.

In addition to this equipment, 15 two-hour oxygen breathing apparatus and 3 one-half-hour oxygen breathing apparatus together with 24 "All Service" gas masks are kept ready for service at all times. Forty trained helmet men receive additional training once every two months.

Past experiences have demonstrated the necessity of all the above. It may seem somewhat elaborate, but it must be borne in mind that the fire fighting equipment and the trained personnel that mans it is the only "fire insurance" the underground workings of a mining property have, and as such should be as complete and well trained as it is possible to make such insurance.

United Verde Copper Company

An Ore Train between Hopewell and Clarkdale

*By J. E. McLean **

Development of Transportation to the United Verde

When the mining claims which formed the nucleus of the present United Verde Copper Company holdings were first located in 1876, the nearest railroad station was at Abilene, Kans., at that time the terminal of the Atchison, Topeka & Santa Fe Railroad Company, and the nearest point from which water transportation could originate was at Ehrenburg, on the Colorado River.

It is hardly cause for surprise, therefore, that the belief was generally expressed that the claims were of little value. The Santa Fe, however, pushed its line westward to the coast, and it was completed through Arizona in 1882. It was then possible to build a wagon road from Ash Fork to Jerome, a distance of about 60 miles, over which freighting was conducted by mule team. While this was a vast improvement over the pack mule, the costs were necessarily high, the rate for transportation one way being about $20 per ton. However, by this method a small smelting plant was transported to Jerome, but on account of the primitive type of this plant and the excessive transportation charges both by mule team and by rail to and from eastern points, operation of the plant was conducted at a loss.

In September, 1887, the first railroad reached Prescott, with Selikman on the Santa Fe as its junction point. This road was known as the Bullock road, and was a poorly constructed, narrow-gauge line. However, it reduced the distance by rail to about one-third, and a wagon road was built from Jerome to Prescott. Freighting was being done over this route when Senator Clark took control of the United Verde in 1888.

The development work which Senator Clark carried on during the next five years was so favorable that when the railroad from Ash Fork to Phoenix was commenced in 1893, the Senator immediately made plans for the construction of a narrow gauge road about 26 miles in length connecting this line with Jerome.

Grading for the narrow gauge railroad was started in June, 1894, and the road was completed and the first train handling commercial freight operated over it into Jerome January 24, 1895.

This road was called United Verde and Pacific Railway and had a maximum gradient of 3 percent in both directions and a maximum curvature of 24 degrees. While it provided transportation at a greatly reduced cost over former methods, the costs were still high due to severe operating conditions and the necessity of transferring from standard to narrow gauge equipment at Jerome Junction, the junction point with the Santa Fe's Ash Fork to Phoenix branch.

When expanding operations made it necessary to build a larger smelting plant, a site was chosen on the Verde River about 5 miles from Jerome. This new location of the plant not only involved the construction of a railroad to handle ore from the mine to the smelter but also the construction of a modern standard gauge road into the Verde District. In October, 1911, the Santa Fe started grading for the Verde Valley Railway and in March, 1913, the road was completed from Drake, its junction point with the Ash Fork-Phoenix Branch, into Clarkdale, the new smelter town, a distance of 38 miles.

August 10, 1912, the Verde Tunnel and Smelter Railroad Company was organized and incorporated for the purpose of constructing and operating a standard gauge line from Hopewell, the location of the transfer bins where ore from the mine is transferred from mine cars to railroad cars, to the new smelting plant in Clarkdale, a distance of approximately six miles. Construction was started January, 1913, and the road completed February, 1915.

The Jerome yards and other terminal facilities of the United Verde and Pacific narrow gauge were located immediately over a large ore body, and when in 1917, it was decided to mine this ore by the open-pit method it became a question of changing the location of the narrow-gauge road from a point about 3 miles from Jerome to a new terminal location on the 300-ft. or townsite level in Jerome, or extending the Verde Tunnel and Smelter standard gauge to the same point. After a careful study of both plans it was decided to extend the standard-gauge line and grading was started January, 1918, and the extension completed and opened for operation January, 1920, at which time the narrow gauge road was abandoned.

The Verde Tunnel and Smelter Railroad has a continuous gradient of 4 percent over its entire length of approximately 11 miles, and a maximum curvature of 24 degrees. In the 11 miles of main line there are 31 24-degree curves, 12 ranging from 20 degrees to 22 degrees, 10 ranging from 14 degrees to 18 degrees, 12 ranging from 8 degrees to 12 degrees, and two switch backs, one located at Clarkdale and one at Hopewell. The longest tangent is 870 ft.

The original line from Clarkdale to Hopewell was laid with 75-lb. rail; but, in preparation for increased traffic and heavier equipment, this was replaced in 1919-1920 with 90-lb. steel. The extension from Hopewell to Jerome was laid with steel of the same weight.

The entire line is slag-ballasted, and guard rails have been installed on all the curves.

The Verde Tunnel and Smelter Rail-

* General Superintendent, Verde Tunnel & Smelter Railroad Company.

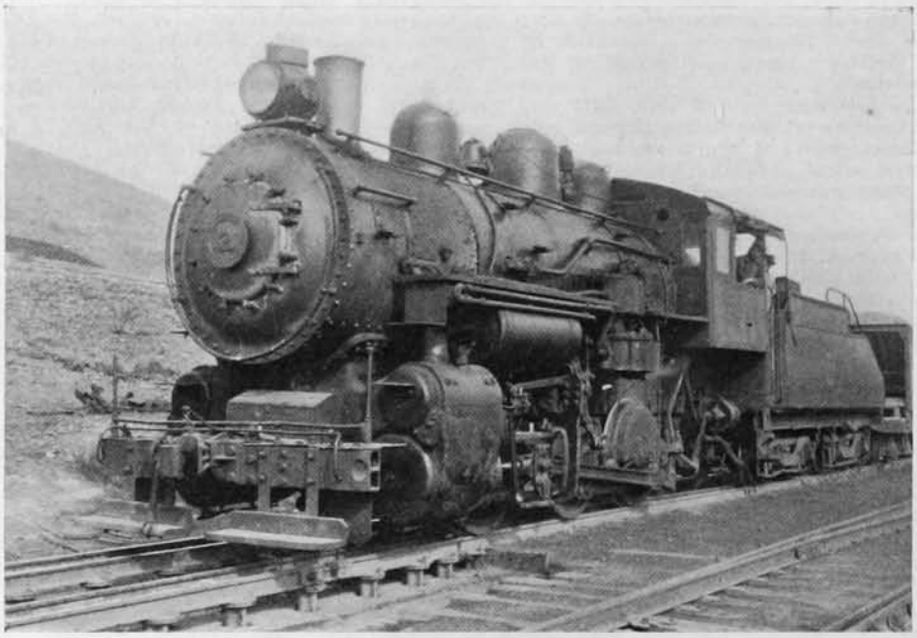
Mallet, compound type engine, type 2-6-6-2. Total weight, engine and tender, 611,000 lbs.

road started to operate in 1915 between Clarkdale and Hopewell with two 0-6-0 type locomotives built by the American Locomotive Company, 40 60-ton capacity steel ore cars and five 50-ton capacity steel flat cars. It was thought at that time this equipment would be adequate to take care of all requirements for a number of years. However, the constantly increasing production of the mine, the construction of new units for the smelting plant at Clarkdale, the new shaft, power house, shops and other facilities on the 500-ft. level in Jerome, and also the abandonment of the narrow gauge line, made it necessary to purchase more equipment from time to time as required. In August, 1915, another 0-6-0 type locomotive was purchased and May, 1916, the fourth locomotive of the same type was placed in service. In April, 1918, three more 50-ton capacity steel flat cars were purchased and in December, 1919, 10 25-yd. Western Wheeled Scraper dump cars, required for construction work, were placed in service. This additional equipment took care of all requirements until the narrow gauge line was abandoned in 1920, when all tonnage formerly hauled by this road had to be routed over the Verde Tunnel and Smelter Railroad.

About this time ore was being mined in the open pit and while the greater portion of this ore was passed through the mine and out to the transfer bins at Hopewell, a certain percentage had to be handled over the railroad direct to Clarkdale. With the completion of the Verde Tunnel and Smelter Railroad into Jerome the tonnage to be hauled had increased so much it could not be handled economically with the 0-6-0 type locomotives and it was necessary to replace them for line haul with larger and more powerful units. After a careful study of conditions, during which several different type simple locomotives were considered and discarded on account of the length of wheel base which even with lateral motion front and rear driver boxes was still too great to operate over the 24-degree curves, the American Locomotive Company recommended compound Mallets of the 2-6-6-2 type with a 10-ft. 4-in. rigid wheel base and two such locomotives were purchased and placed in service March, 1920. These locomotives have demonstrated their adaptability to the operating conditions on this line through 10 years of very satisfactory service.

An accompanying photograph shows an ore train between Hopewell and Clarkdale with one of the 2-6-6-2 Mallet type locomotives.

The capacity of the 0-6-0 type locomotive on the ascending 4 percent grade is 220 tons or 10 empty ore cars and prior to placing the Mallets in service, in order to handle the tonnage it was necessary to use two locomotives on the last two trips each day, the last trip with 20 empties which were set out at



At Right — Six-wheel switching engine, type 0-6-0. Total weight, engine and tender, 262,500 lbs.

United Verde Copper Company

Hopewell and loaded at night. The first two trips were made with one locomotive handling 10 empties each trip to Hopewell and 20 loads from Hopewell to Clarkdale. The capacity of the Mallets on the ascending grade is 600 tons or about 27 empty ore cars. Since these locomotives were placed in service the tonnage from all sources has been handled with one locomotive in an eight-hour day and all switching with one of the 0-6-0 type. The principal dimensions of the 0-6-0 and the 2-6-6-2 type locomotives are given in connection with the accompanying illustrations.

In 1922, the United Verde Extension opened up a lime pit near the Verde Tunnel and Smelter Railroad right-of-way and the lime rock from this pit was hauled to Clarkdale over this road for several months in V. T. & S. ore cars and, due to delays in transit and unloading at the smelter in Clemenceau, there would frequently be from 10 to 15 ore cars in this service, causing a shortage of ore cars on the V. T. & S. and, in April, 1923, 15 50-ton capacity steel hopper bottom cars were purchased for the Extension lime rock service and in February, 1925, on account of the further increase in tonnage, 15 75-ton capacity steel ore cars were purchased and put in service on the ore run between Hopewell and Clarkdale.

All the cars now in service, with the exception of the 10 Western dumps, were built by the Pressed Steel Car Company. The first 40 ore cars have been in the severest kind of service for over 15 years and to date have required no repairs other than ordinary running repairs.

SUBSIDENCE AND GROUND MOVEMENT IN COPPER MINES OF THE WEST

There are few phenomena connected with mining that are as inadequately understood as subsidence and ground movement, the United States Bureau of Mines points out. A vast amount of data have been collected relating to the failure of rock overlying mine workings, the action observed often being more apparent than real, and erroneous conclusions have been drawn.

With small workings the collapse of the walls and rock and consequent filling of the excavation does not reach to the surface unless the cover is slight; with more extensive workings there is not only a breaking down of the roof that extends to the surface, but a movement of the wall rock to close the excavation.

The controlling factors in ground movement are the lines of weakness that exist in rock masses, which are: Bedding planes, contacts between various formations, faults, and the natural system of jointing existing in all rocks. Failure follows existing lines of weakness and does not break across solid formations except in the occasional breaking up of rock masses, that, too, usually being on joints.

Joints or slip planes exist in all rocks and are symmetrically arranged, the system in any locality being readily obtained by observations on the strike and dip. They indicate character and extent of movement to be expected, except where other more prominent lines of weakness occur as outlined above. Combination of existing weaknesses in rock enclosing ore bodies furnish the key to the action resulting in subsidence and ground movement.

Field work on these problems is now in progress at the Southwest Experiment Station of the United States Bureau of Mines, Department of Commerce, Tucson, Ariz.

LABORATORY METHOD FOR DETERMINATION OF AMENABILITY OF AN ORE TO TABLING

A new method to determine the amenability of various ores to gravity concentration has been tried recently at the Mississippi Valley Experiment Station of the United States Bureau of Mines, at Rolla, Mo., in cooperation with the Missouri School of Mines and Metallurgy.

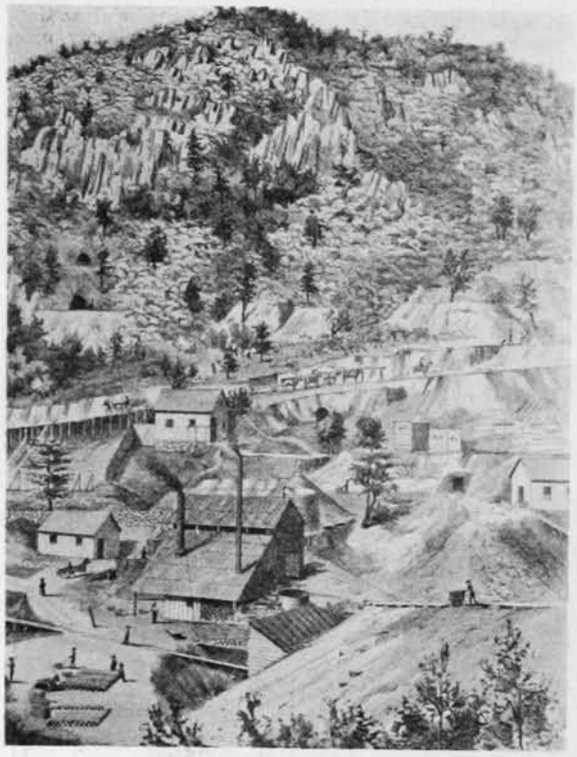
Tests have been made on a number of ores in which the ore was first sized, each screen product then being placed in a small glass tube constriction-plate classifier and the amount of incoming water varied until the grains were just in teetor. The water was then shut off and the products siphoned off. This permits the removal of any number of cuts or products which can be grouped as desired. If the ore is not completely unlocked at the size classified, a middling cut can be siphoned off, crushed to the size necessary for liberation, combined with the finer material and re-screened. If the ore is unlocked at the size classified a clean concentrate and a clean tailing can be obtained directly.

The method has been found to give results that are indicative of the results that can be obtained by close classification and tailing. The results are more easily obtained, permitting the use of smaller samples and the securing of clean concentrates and tailings, when no locked particles are present, without the necessity of re-treating middlings that are always present when tables are used.

One of United Verde Copper Company's divisions which has the distinction of having worked the longest without a lost-time accident. There are 176 men in the crew. The bosses are standing in the front row near the center. From left to right they are: Robert Kelly, John Blazina, John Rice, John Eddy, shift bosses, and Walter Mutz, division foreman.

United Verde Copper Company

Historical Growth of the UNITED VERDE



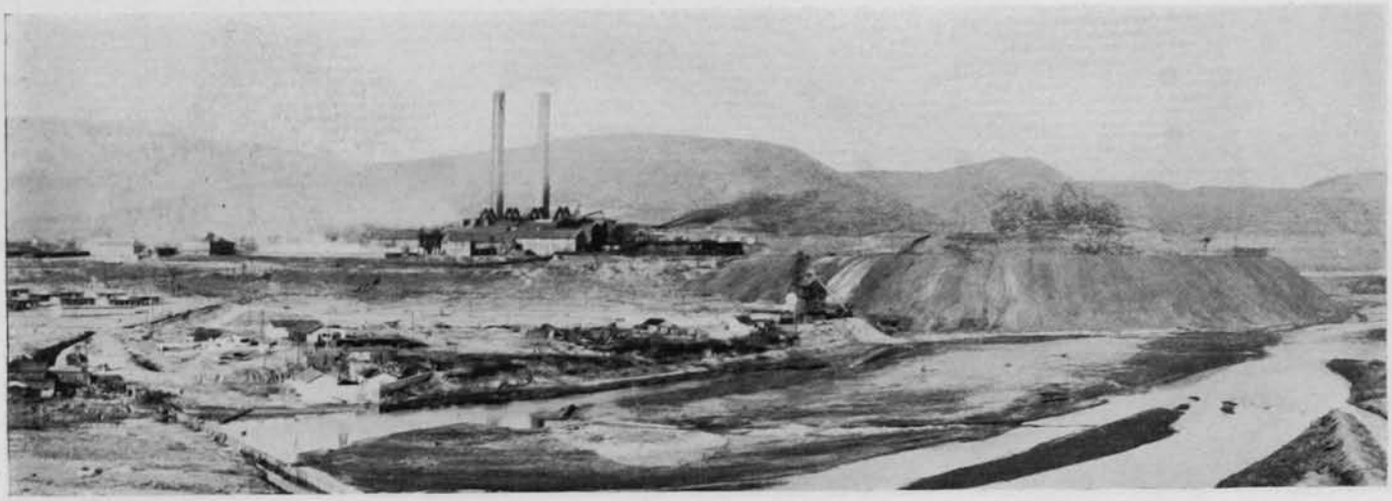
Old lithograph showing Jerome Smelter in 1884

***By* John E. Lanning**
CHIEF MECHANICAL ENGINEER

UNITED VERDE COPPER COMPANY ores have been smelted for over 45 years, and the accompanying pictures and data give an idea of the progress and development of the various smelters, from an anual capacity in 1888 of probably fifteen thousand tons to approximately one and three quarter million tons in 1929.

As an interesting item of comparison, the accompanying facsimile of a proposal under date of Oct. 3, 1883, on the equipment "necessary to duplicate your

Below, Clarkdale in 1914



Clarkdale as it appears today

SMELTING PLANT
at Clarkdale, Arizona

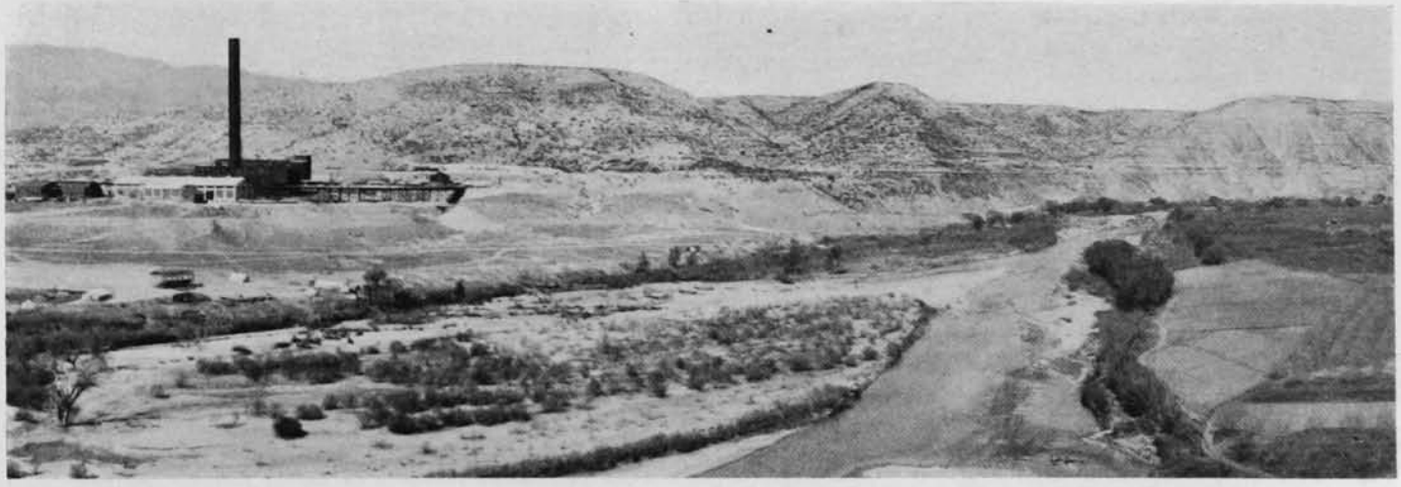
present plant," addressed to Mr. E. M. Jerome, at that time secretary of the United Verde Copper Company, gives concrete evidence of the cost and size of the plant, in that all the equipment could be bought for $8,000.00.

The scope of this article does not permit a detailed description of the various plants as operated at Jerome up to 1915. At that time the four blast furnaces and four converters were handling approximately 400,000 tons of ore annually, and the demand for increased capacity and facilities for treating lower grade ores, which could not be secured at the Jerome site on account of topographic limitations and the caving of the ground due to mining operations immediately below, had made imperative the construction of a new plant at a more favorable site.

As an example of the difficulties of operation, the power house, which was located on badly slipping ground, had to be rebuilt as occasion demanded, and the equipment eventually was distributed in four separate plants. One blowing engine in particular was operated for a time with the cylinders 5 ft. lower than the shaft and fully 9 ft. lower than the original elevation.

Difficulties of transportation into the district via the U. V. & Pacific Railway, a narrow gage line over the mountains requiring transfer of all shipments at its connection with the Santa Fe, Phoenix and Prescott Railway, at Jerome Junction, added to the cost of operation, but the development of new ideas, methods and improved equipment facilities permitted a steadily increasing production.

The accompanying photograph of some of the earlier type of equipment now in the Clarkdale Smelter Yard brings forcibly to mind the development of smelter technique in the period covered by the life of the United Verde. The 50-in. blast furnace, of cylindrical water jacket type, is one of two operating in 1888, when Senator W. A. Clark first visited Jerome. It was followed by 48-in.



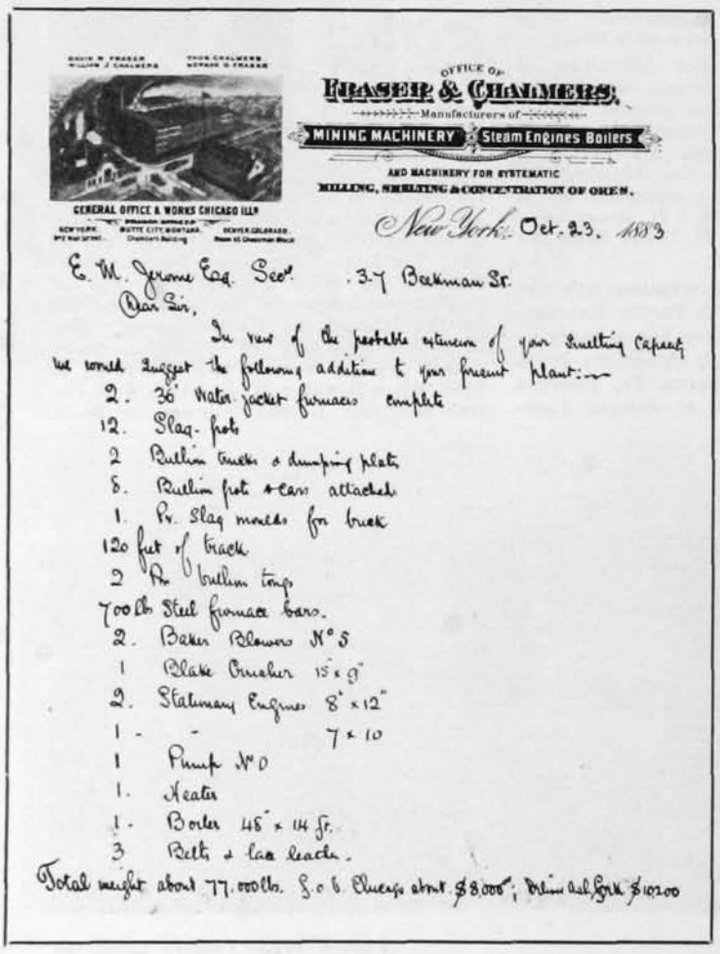
United Verde Copper Company

by 120-in. furnaces, which were later converted to 48-in. by 240-in. In the early days slag was trammed by hand in the small ladle shown. The electric locomotive in the picture was the first to be used for slag haulage in copper smelting works, succeeding Old Dobbin, who must have had his troubles on the slag dump, and in turn replaced by the modern 25-ton enclosed cab locomotive, hauling four 20-ton ladles of molten slag.

The story of United Verde Smelters would be incomplete without mention of Thos. Taylor, at present general smelter superintendent, to whom no small credit is due for his untiring efforts toward their success. Mr. Taylor, or "Tom," as he is known wherever copper is smelted, during the 33 years he has been in the organization, has progressed from furnace foreman to his present position, and has been directly responsible for many of the improvements and refinements now generally in use throughout the industry.

The accompanying graph shows actual tonnage of ore treated by both the Jerome and Clarkdale Smelters from 1904, when the district was 20 years old, to 1929, a length of life seldom exceeded, and with an ascending curve that shows no indication of an immediate drop.

Preliminary investigation of possible sites for the proposed Clarkdale Smelter was undertaken in 1910, and after considering all possibilities the present location was fixed as lending itself most satisfactorily to all aspects of the problem in hand. Adequate water supply, both for potable and industrial uses; good drainage; ample sand and gravel deposits in the adjacent Verde River for construction purposes; a fairly large and satisfactory deposit of clay suitable for building brick; easy access to the tracks of the Verde Valley Railway as then proposed; sufficient area at a satisfactory elevation for slag disposal, and for tail-



OFFICE OF
FRASER & CHALMERS,
Manufacturers of
MINING MACHINERY Steam Engines Boilers
AND MACHINERY FOR SYSTEMATIC
MILLING, SMELTING & CONCENTRATION OF ORES.

GENERAL OFFICE & WORKS CHICAGO ILL.

New York Oct. 23. 1883

E. M. Jerome Esq. Sec'y. 37 Beekman St.

Dear Sir,

In view of the probable extension of your smelting capacity we would suggest the following additions to your present plant:—

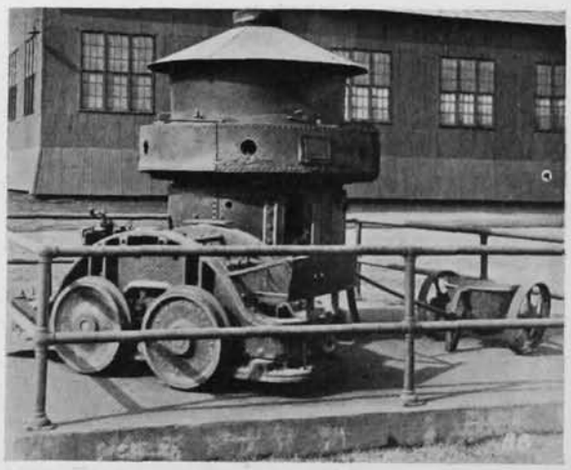
2. 36" Water jacket furnaces complete
12. Slag-pots
2 Bullion moulds & dumping plate
8. Bullion pots & cars attached
1. Pr. Slag moulds for brick
120 feet of track
2 Pr. bullion tongs
700 lb Steel furnace bars.
2. Baker Blowers No 5
1 Blake Crusher 15" x 9"
2. Stationary Engines 8" x 12"
1. " 7 x 10
1 Pump No 0
1. Heater
1. Boiler 48" x 14 ft.
3 Belts & lac leaches.

Total weight about 77.000 lbs. f.o.b. Chicago about $8.000; Deliv. Ash Fork $10200

The above includes everything necessary to duplicate your present plant in double exclusive of supplies of which you would probably require fire brick and extra piping to the extent of about $1000= delivered in Ash Fork, and the machinery would be as perfect and give as good satisfaction as that already supplied you.

We will be glad to call upon you at any time to give you any further information required.

Yours truly
Fraser & Chalmers
Walter McDermott mgr.
R



Old equipment formerly in use at Jerome and now on display at Clarkdale Smelter yard

Jerome, Arizona, in 1914

ings disposal for the milling plant which was already visualized; a satisfactory townsite adjacent to the plant, capable of expansion to the necessary future requirements; all these features were given due consideration, and in 1911 the first engineering work was undertaken. Ground was broken in 1912, and the first furnace blown in on May 26, 1915.

The rated capacity of the plant as designed and built was 4,500,000 pounds of copper per month with 5 percent ore, and the accompanying plot of the plant as at present, outlining the original buildings in hatched areas, and the various extensions with their dates of completion, shows the progressive change and the expansion to its present capacity. It also shows the commendable foresight and good judgment used in planning for future growth for it will be noted that every increase in the size and capacity of the various units has been accomplished on ground originally available, and no major changes in arrangement have been necessary to accommodate this growth.

The plant is served by 13.2 miles of standard gage railway trackage inside of the smelter yard, of which 3.9 miles are owned by the V. T. & S. Ry., a subsidiary, and 9.3 miles owned and maintained by the United Verde Copper Company, with approximately 2 miles additional on the slag dump and to the gravel plant. Main haulage is by steam locomotives; charge cars, calcine cars and slag cars, are moved by electric trolley locomotives.

The total footage of conveying equipment is as follows:

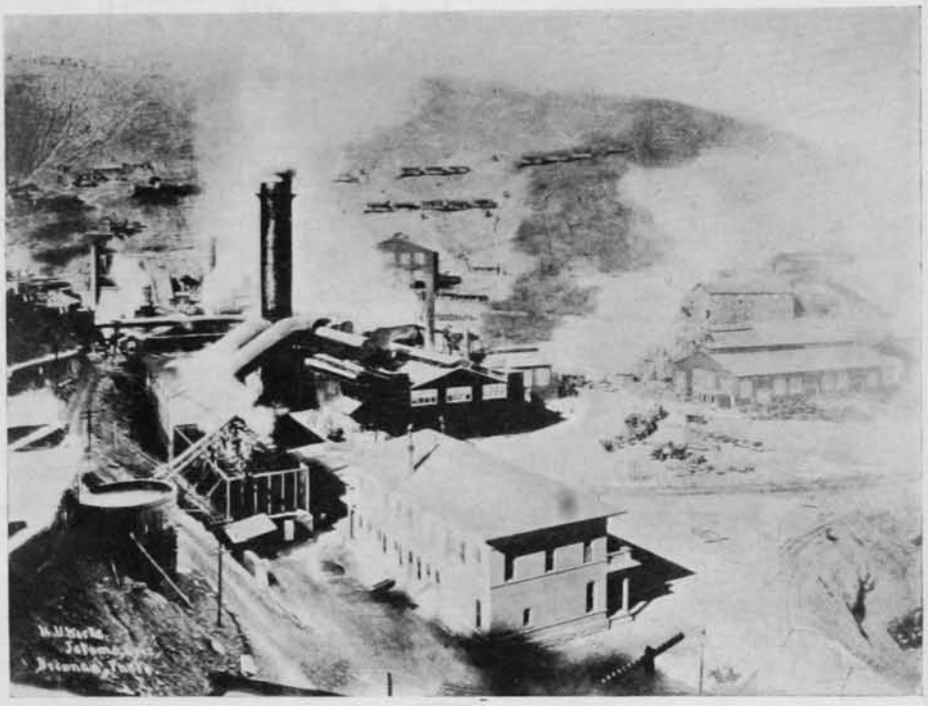
8,655 lin. ft. of belt conveyors—20 to 36 in. wide.

1,100 lin. ft. of screw conveyors—14 in. in diam.

660 lin. ft. of belt elevators—14 to 24 in. wide.

The total area of the smelter yard is 196 acres, entirely fenced, and in addition the slag dump covers an area of 32 acres and the tailings disposal 65 acres.

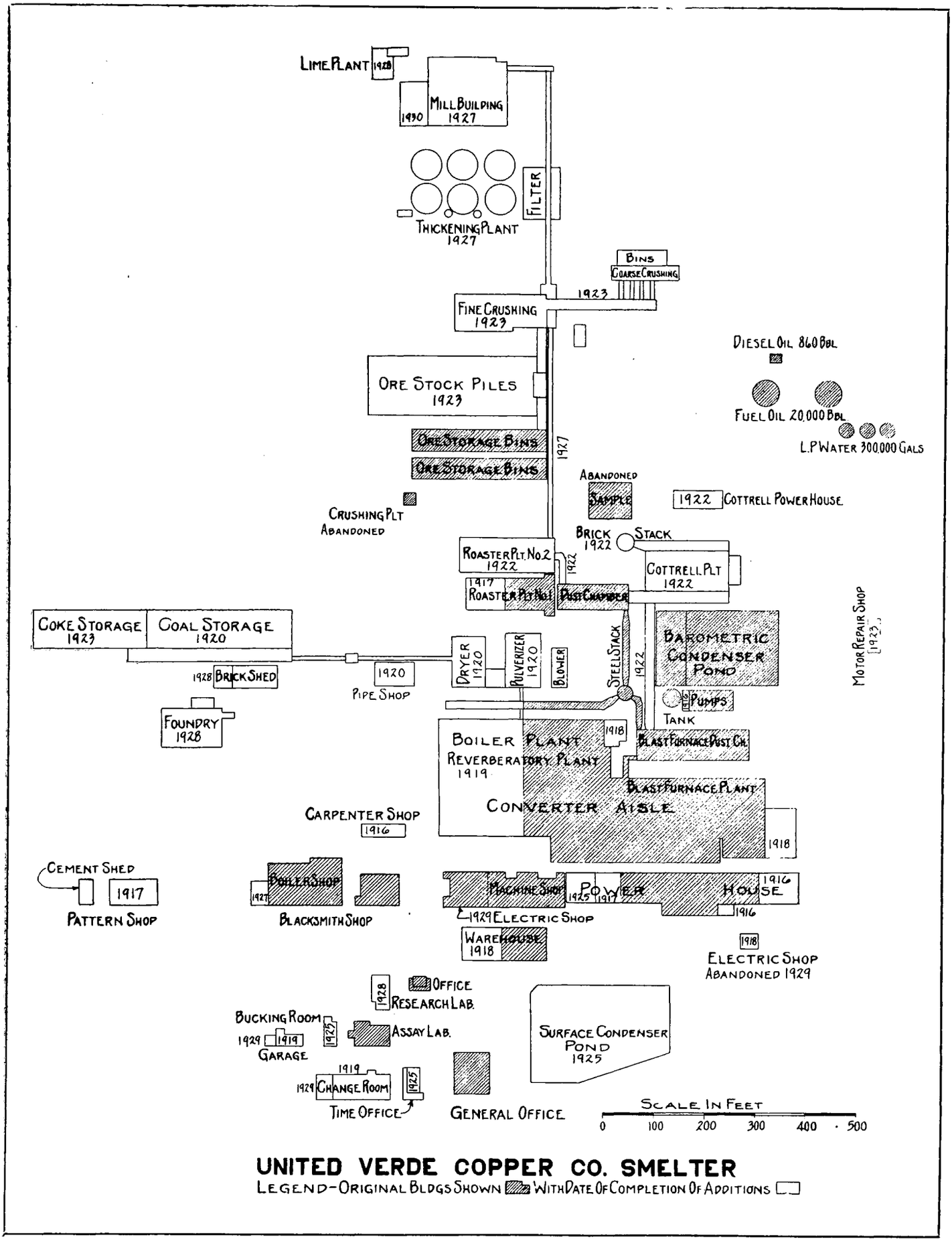
As originally designed, all crushing was at the small crushing and sampling plant shown, later abandoned when the smelter crushing plant was built in 1923, and ore was delivered to the smelter receiving bins by 60-ton bottom-dump cars via the V. T. & S. Ry. The west, or receiving bin, is 41 ft. 6 in. by 270 ft. in plan, and has a capacity of 12,000 tons. This bin is divided into various compart-



Jerome plant about 1900

Inspecting proposed site of Clarkdale Smelter in 1910
Left to right—Senator W. A. Clark, T. C. Roberts, C. H. Repath, C. V. Hopkins, Tom Taylor, and Robt. E. Tally

United Verde Copper Company



UNITED VERDE COPPER CO. SMELTER

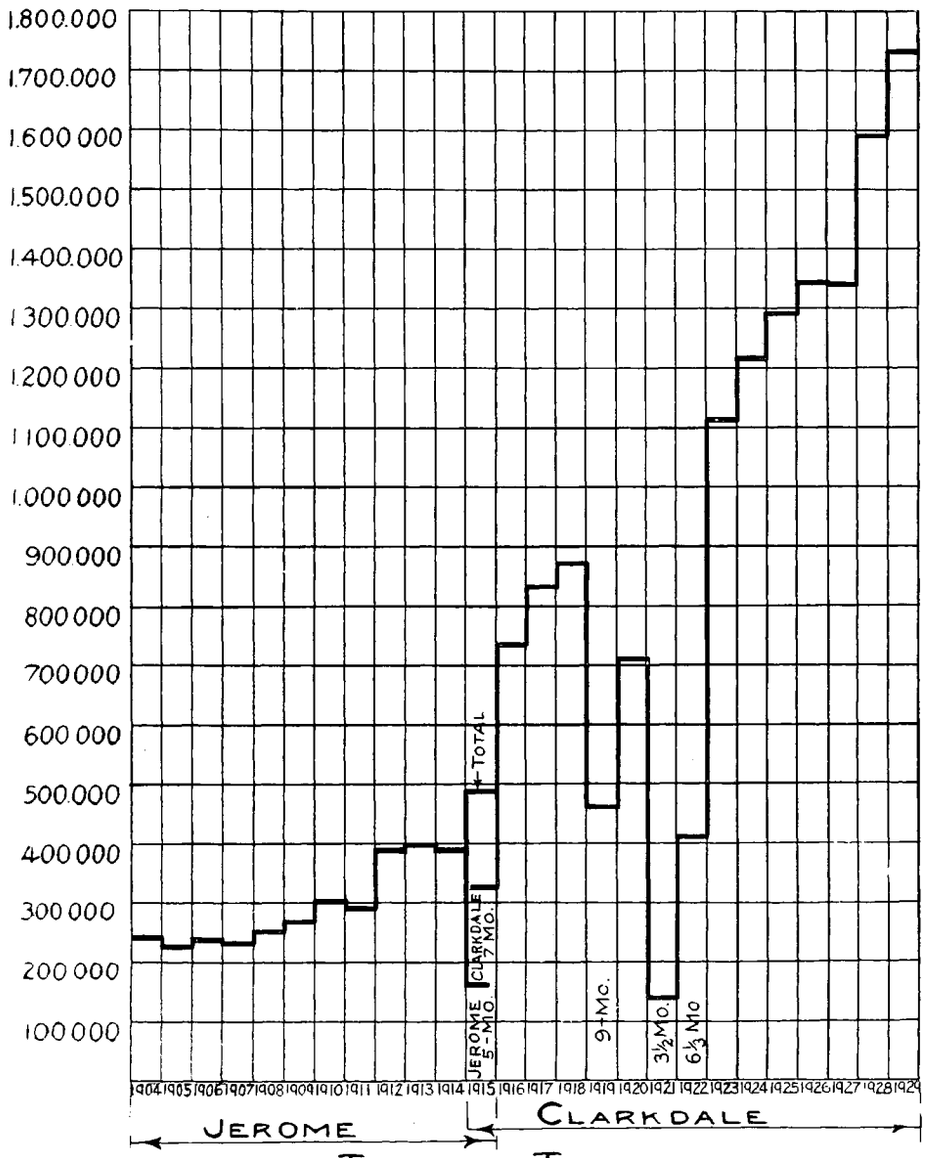
LEGEND—ORIGINAL BLDGS SHOWN ▨ WITH DATE OF COMPLETION OF ADDITIONS □

United Verde Copper Company

Buildings

Item	Frame	Walls	Roof	Runways Floors	Special Equipment
CRUSHING PLANT—					
Coarse Crusher	Steel	Corrugated	Corr. Steel	Concrete	25-Ton Crane
Screen Plant	Steel	Corrugated	Corr. Steel	Concrete	1½-Ton Elec. Hoist on Trolley
Fine Crusher	Steel	Corrugated	Corr. Steel	Concrete	25-Ton and 2-Ton Crane
CONCENTRATING PLANT	Steel	Corrugated	Corr. Steel	Concrete	10 Ton and 2-Ton Crane
ORE BINS	Steel	Steel	Corr. Steel	Concrete and Wood	Railroad Tracks Over
CONVERTER AISLE	Steel	Steel	Asbes. Cover, Corrugated	Dirt	3 40-Ton Cranes
BLAST FURNACES	Steel	Corrugated	Asbes. Cover, Corrugated	Concrete, Steel, Earth	Railroad Tracks Over
REVERBERATORY	Steel	Corrugated	Asbes. Cover, Corrugated	Concrete, Steel, Earth	Railroad Tracks Over
BOILER HOUSE	Steel	Corrugated	Asbes. Cover, Corrugated	Concrete	Freight Elevator
PUMP HOUSE	Steel	Brick	Corrugated	Concrete	Hand Crane
POWER HOUSE	Steel	Brick	Asbes. Comp. on Wood	Concrete	20-Ton Crane
MACHINE SHOP	Steel	Corrugated	Composition on Wood	Concrete	10-Ton Crane
ELECTRIC SHOP	Steel	Corrugated and Brick	Composition on Wood	Concrete	5-Ton Crane
BLACKSMITH SHOP	Steel	Corrugated	Composition on Wood	Earth	Stiff Leg Derrick
BOILER SHOP	Steel	Corrugated	Composition on Wood	Wood	5-Ton Crane
PIPE SHOP	Steel	Corrugated	Composition on Wood	Wood	----------
PATTERN SHOP	Steel	Corrugated	Composition on Wood	Concrete and Wood	----------
CARPENTER SHOP	Steel	Brick and Corrugated	Corrugated	Wood	----------
COAL STORAGE	--------	Concrete	Open Top	--------	5-Ton Gantry Crane
COAL DRYER	Steel	Brick	Asbes. Cover, Corrugated	Steel and Concrete	----------
COAL PULVERIZER	Steel	Brick	Asbes. Cover, Corrugated	Steel and Concrete	----------
BOILER & SWITCH HOUSE	Steel	Brick	Asbes. Cover, Corrugated	Concrete	----------
ROASTER PLANTS	Steel	Corr. Iron	Corrugated	Steel	Freight Elevator
COTTRELL PLANT	Steel	Brick	Corrugated	Steel	----------
COTTRELL POWER HOUSE	Steel	Brick	Composition on Wood	Concrete	Hand Crane
ROASTER DUST CHAMBER	Steel	Brick	Brick Arches	--------	----------
BLAST FURNACE DUST CHAMBER	Steel	Brick	Brick Arches	--------	----------
MOTOR REPAIR SHOP	Steel	Brick	Corrugated Iron	Concrete	10-Ton Hand Crane and Electric Trolley
ROUND HOUSE	Steel	Brick	Corrugated Iron	Concrete	----------
SILICA BRICK SHED	Wood	Open	Corrugated Iron	Dirt	----------
WAREHOUSE	Steel	Corr. Iron	Composition on Wood	Concrete	Fireproof Office and Vault. Freight Elev.
CHANGE ROOM	Brick Walls Steel Truss	Brick	Composition on Wood	Concrete	----------
GENERAL OFFICE	Steel	Brick	Composition on Wood	Concrete and Wood	Fireproof Vaults
ASSAY OFFICE	Wood	Brick	Composition on Wood	Concrete	----------
BUCKING ROOM	Conc. Tile Wood Truss.	Concrete Tile	Composition on Wood	Concrete	----------
RESEARCH LABORATORY	Brick Walls Wood Truss.	Brick	Composition on Wood	Wood	----------
TIME OFFICE	Brick Walls Wood Truss.	Brick	Composition on Wood	Wood	----------
GARAGE	Brick Walls Wood Truss.	Brick	Composition on Wood	Concrete	----------
FOUNDRY	Steel	Corr. Iron	Corrugated Iron	Dirt	10 and 2-Ton Cranes R. R. Track into Building

United Verde Copper Company



TONNAGE TREATED
UNITED VERDE COPPER CO

ments and equipped with hopper bottoms for delivery to blast furnace charge cars, or conveyor system to crushing plant and sampling mill, from which the ore was delivered to the smelting bin, of similar size, capacity and construction. These bins are still in use, serving respectively as coarse and fine ore bins. They are of heavy steel construction with corrugated steel roof, sufficiently high to accommodate railway equipment originally used, and now served by belt conveyors with traveling trippers—delivering ore from the crushing plant built in 1923. An intermediate step in ore handling was taken in 1916, when the crushing plant was built at Hopewell, providing storage of crushed ore ahead of the sampling plant at the smelter.

The first change in major equipment at the smelter came in 1917, when the addition of six roasters doubled the capacity of that department. Two additional 713-hp. boilers were added the following year, replacing the first smaller direct-fired boilers installed, all others being waste heat boilers.

Various shops and accessory plant had been relocated or built, and the power house had been extended to the north, but up to this time the smelting equipment had consisted of four blast furnaces, 48 in. by 320 in. and three reverberators, 19 ft. by 101 ft., which with the four 12-ft. converters and six roasters were served by a single steel stack 30 ft. by 400 ft.

Nineteen hundred and nineteen marked the first radical increase in capacity when three additional reverberators, 25 by 101, were added, each equipped with two waste heat boilers. Coincident with this came the adoption of pulverized coal firing, necessitating the construction of the coal handling, drying and pulverizing plant.

The following year, 1920, the roasting plant was increased approximately 140 percent by the construction of 12 larger roasters. By this addition, the stack and flue capacity became entirely inadequate, and in 1922, the Cottrell dust precipitation plant was installed to treat roaster, converter and blast furnace gases, discharging through a new brick stack, 30 ft. by 430 ft.

The crushing plant at Hopewell, its capacity long since having been exceeded, a new plant was necessary and in view of the requirements for smelter operations as then outlined, and the increasing probability of milling operations, a new 3-stage crushing plant, with intermediate screening plant, was built at the smelter site and put in operation in 1923. This involved moving the blast furnace coke storage pile and trestle, a new coke storage being provided adjacent to the coal storage. This, by the way, was the only move of any important structure from its original location necessitated by the tremendous growth outlined.

Power requirements, both at the smelter and the mine, were gradually mounting and in 1924, a new 7,500 K. V. A. condensing turbo-generator was installed in the power house, requiring a rearrangement of existing equipment and the construction of a cooling pond for condensing water. This resulted in marked economy of operation and as power demand continued to increase, another unit of the same size and type was installed in 1927.

In 1924, the question of milling ores was actively investigated and in 1927, the first two units of the concentrating plant were put in operation. The third unit went on production in 1929, and the fourth, giving a total capacity of 2,400 tons per day, is now in construction.

The accompanying tabulation of structures shows type of construction adopted and in general it may be said that all buildings are of permanent, fireproof construction, with adequate light, both daylight and artificial, good ventilation, and equipped with all necessary safety features both as to building and equipment. All walkways are of sufficient width with guard rails and with stairs between different levels, except in some few cases where ladders are imperative.

The principal shops are painted either white or aluminum inside for better lighting, and the machine and electric shops are equipped with unit hot air heating. Other shops, and offices, are heated by steam except the general office, which has a central hot air plant with air washer.

The plant was designed by Repath and McGregor, consulting engineers, and was constructed under their immediate supervision, Mr. F. J. Brule in charge. Mr. T. C. Roberts, chief engineer of the United Verde at that time, was succeeded in 1917, by Mr. C. M. Hoffman, as superintendent of machinery, in charge of all mechanical work both at the smelter and mine, and for the V. T. & S. Ry. From 1918 to 1925, Mr. J. B. Johnson was chief mechanical engineer for the smelter, followed by the writer. At various stages of development, Mr. George Douglas, Mr. David Cole, Mr. William Rossberg, and Mr. H. Kenyon Burch have acted as consulting engineers, and aided materially in the successful solution of the problems presented.

United Verde Copper Company

By P. C. Keefe

GENERAL FOREMAN OF CRUSHING

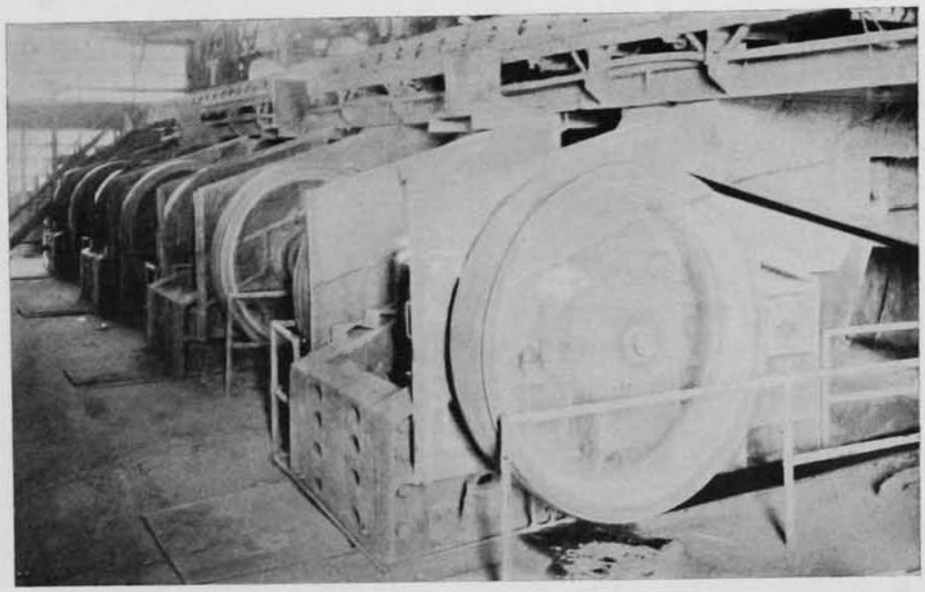


Figure 1. 36 x 48-in. Allis-Chalmers Jaw Crushers

....Smelter Crushing Plant

History

PRIOR to 1923 the crushing for the smelter was done at Hopewell, Ariz., situated about half-way between Jerome and Clarkdale. In addition there was a small crushing plant at Clarkdale known as the Sample Mill. The Hopewell crushing plant, as originally built, was little more than a screening plant. The product obtained was insufficient to satisfy the needs of the reverberatory plant; therefore two 48-in. vertical type Symons crushers were added to the original installation. With later proposed plans for increasing the reverberatory capacity, and consequently the roaster capacity, it was seen that the amount of crushed ore produced at the Hopewell plant would be insufficient for the requirements of the reverberatory furnaces; therefore it was decided to build a new crushing plant. The site chosen was in Clarkdale, and the present crushing plant was put into operation the 21st day of November, 1923.

Theory and Principles

The duty of the United Verde crushing department is quite different than that of the average crushing plant, as this department is not only responsible for all the ore received at Clarkdale, but must deliver to the roaster storage bins a mixture of ore that is almost ready to be smelted. The mixture of ore must be of suitable size and of nearly correct mineralogical content. In order to accomplish this latter problem it is necessary to keep in close touch with the mining and steam shovel departments at all times.

From careful study of furnace conditions it was found that the keeping of a definite ratio between schist and quartz content of the roaster feed was of vital importance to economic furnace operation. Therefore the ore delivered to the roaster storage bins is kept, as nearly as possible, in the proportion of 17 percent schist to 19 percent quartz.

Also this department crushes all the ore milled at the concentrator. Here, again, this department is faced with another problem of extreme importance. As the same bins are used both for the smelting and the concentrating ore it is vitally necessary to use great care in cleaning the receiving bins after a run of smelting ore and prior to a run of concentrating ore. The reason for this is the fact that the largest part of the smelting ore received, at the present time, is coming from the steam shovel workings, called the pit; and as this ore has been burning for some time, and is partially calcined, it has varying

Figure 2. 48-inch Vertical Symons Disc Crushers

United Verde Copper Company

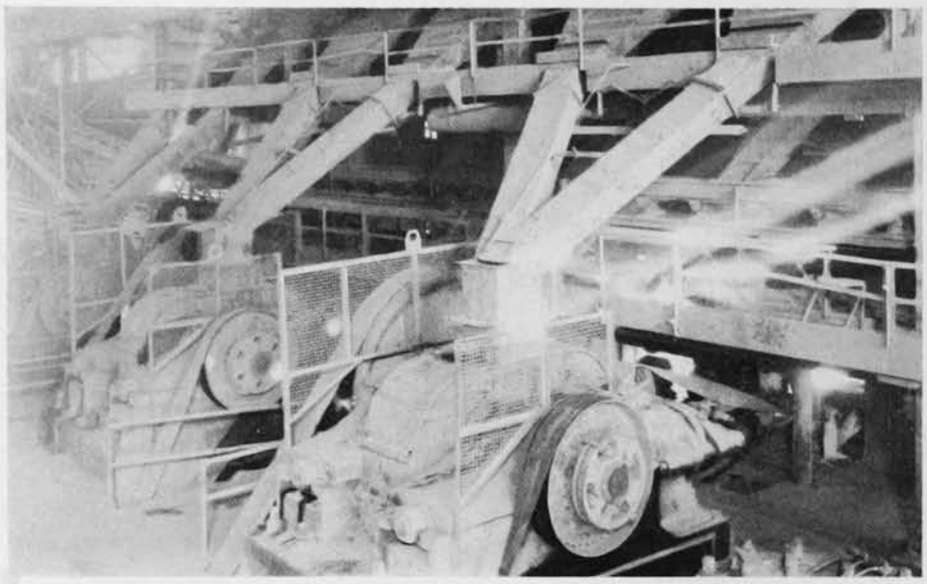


Figure 3. 56 x 24-in. Traylor Rolls

amounts of soluble salts which, of course, have a detrimental effect upon the flotation of sulphide ores.

In addition this department furnishes the converter department with necessary fluxing material.

Location, Design and Construction

The crushing plant is located in the smelter yard, at Clarkdale, west of the roaster storage bins and east of the concentrator. The primary crushers are Allis-Chalmers 48-in. x 36-in. Blake type, all-steel construction (see *Figure 1*). The intermediate crushers are 48-in. vertical type Symons disc crushers (see *Figure 2*) and 56-in. x 24-in. Traylor rolls (see *Figure 3*) are the finishing units. A system of belt conveyors and elevators is used to carry ore to, or from, the various units. (*Continued on page* 69)

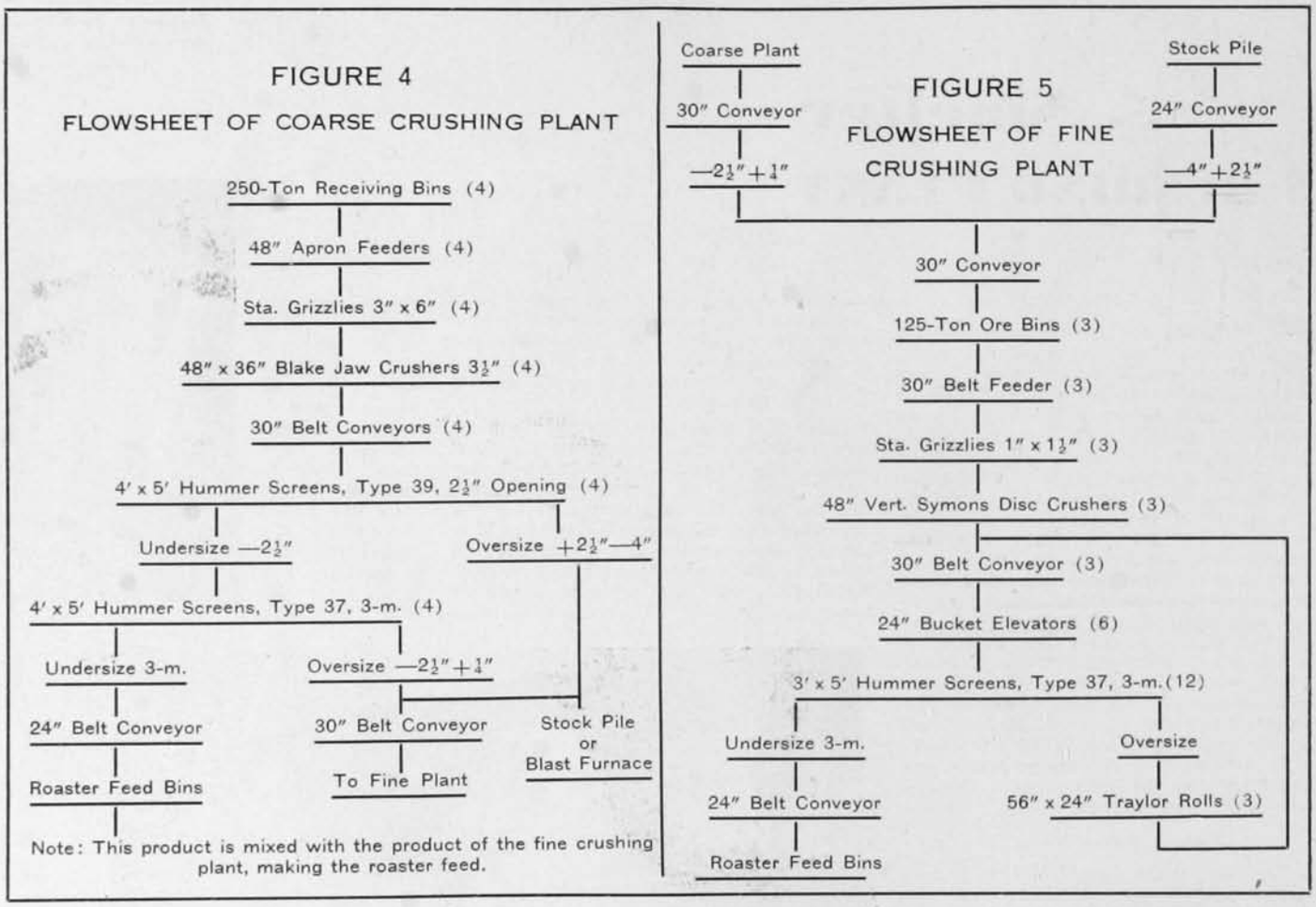


Figure 6. General View of Crushing Plant, taken during construction

A. Two Surge Tanks
B. 60,000-Ton Stock Pile
C. Cottrell Dust Collector
D. Fine Plant
E. Screen House
F. Coarse Plant

United Verde Copper Company

By L. M. Barker

CONCENTRATOR SUPERINTENDENT

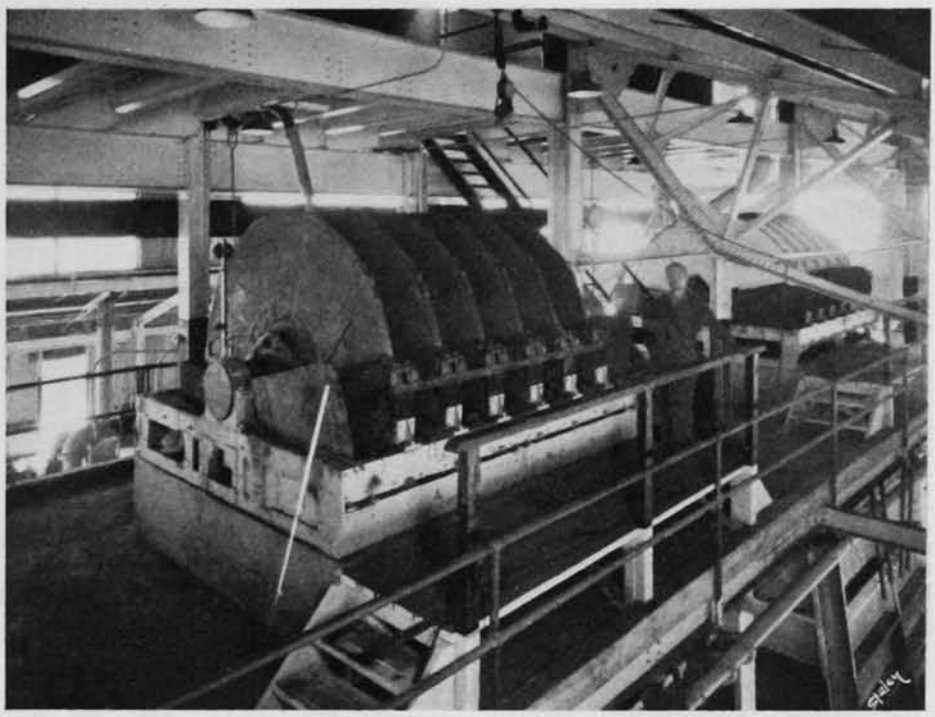


Figure 5. Concentrate Filters and Bins.

CONCENTRATING PLANT

of United Verde Copper Company

HISTORY

THE United Verde concentrator was constructed because of recommendations predicated on certain novel reasons which were quite different from those usually justifying an expenditure of such an amount.

Concentrator operations started in March, 1927, although the metallurgical staff had considered for several years the eventual addition of a concentrator department.

United Verde ore had always been considered a direct smelting ore and the cost of producing copper by such methods satisfactory. However, pyro-metallurgical methods alone resulted in certain smoke nuisances due to the use of all sulphides in the ore as fuel in the smelting operations. Apparently it was logical to avoid smelting these large quantities of iron sulphides, to store them for the future in the hope that sooner or later an economical process would be developed yielding a profit on this iron and sulphur.

The separation of iron sulphides from such ores by differential flotation methods had been conceived, but was not even in a state of embryonic development; nevertheless laboratory flotation tests were undertaken which disclosed that such a separation might be reasonably expected.

Due to the degree of dissemination of the minerals, which are largely microcrystalline in character, it was evident that the problem of extremely fine grinding was a ponderous one which could only be solved by determination of costs upon a large scale. There was also considerable doubt in the economy due to lessened recoveries of silver and gold as compared with direct smelting. The pitfalls were, therefore, so ominous that the metallurgical staff did not feel justified in recommending the construction of a large mill for the differential flotation of the direct smelting heavy sulphide ores. Believing, however, that subsequent improvements in flotation metallurgy and in grinding practice would eventually solve the problem, the staff sought justifiable reasons for recommending the construction of a concentrating plant which could be used economically for selected portions of the ore body and, at the same time, be utilized for working out the major problem of the differential flotation of the heavy sulphide ores.

Those portions of the ore body containing large amounts of a black, slaty-looking rock locally called "schist" yielded about 1,000 tons per day of ore which, from the pyro-metallurgical point of view, was more refractory than the main mass of the ore body; therefore, a laboratory flotation investigation was made of such schist ores. While this investigation was entirely successful from the flotation point of view, the predicted combined costs of flotation concentration plus concentrate smelting did not show enough profit compared with the cost of the direct smelting of this schist ore to justify the expense of a concentrator installation. However, metallurgical calculations did show that the effect of the schist ore on the cost of smelting the average run of mine ore, of which the schist ore constituted only about 25 to 30 percent, was important. Metallurgical data and calculations indicated that if the schist ores were mined separately and then concentrated, the smelting costs on all *the other ore* from the ore body would be materially reduced. About 105,000 tons of ore per month were being produced by the mine, of which 30,000 could be classed as schist. Calculations showed that from 15 to 25 cents per ton on the total ore mined could be saved at the smelting plant if only the 30,000 tons per month were concentrated for the removal of the black schist min-

eral. A recommendation to build a concentrator was then approved and executed, with the result that the expectations have been more than realized in the handling of the schist ore problem and the company has been provided with excellent means of studying the major heavy sulphide problem.

The Problem

The United Verde ore bodies represent a replacement of Pre-Cambrian schists by sulphides. Degree of replacement is variable as is also the character of the replacing sulphides, which condition necessitates classification of ore types both with respect to schist replacement and the quantitative relations of mineral types in any given area. Thus, ores are locally classified as schist, silicious, silicious massive sulphides and massive sulphides. The latter are further classified with respect to valuable metal content as zinc massive, copper massive and copper-zinc massive. The term massive sulphide as herein used implies that pyrite is a major constituent of the ore rather than having any reference to massiveness of the valuable minerals. Such a classification is necessarily broad and refers only to outstanding mineralogical characteristics. All of the ores contain schist, quartz and sulphides of copper, zinc and iron in variable amounts.

Mining operations, though attempting segregation of ore types, can not be so closely conducted that mixing of types can not occur, and influence of such mixing is reflected in the character of the schist ores and massives in general as delivered to the reduction works. Conditions within the ore body dictate the mining methods followed and have a direct bearing upon the problem of segregation. Methods employed may be broadly classified as underground and open pit and both are concerned with all ore types. The latter operation is of particular interest because it involves the ores which for many years were unavailable, being in a zone of early underground workings bulkheaded off on account of an uncontrollable fire. In this zone, though some ore may be found in place, much has been displaced by subsidence of old workings and is mixed with the stope filling of these workings. The whole is chemically altered by fire conditions, the alteration varying, in the case of the sulphides, from superficial tarnishing to complete calcination. The schist ore in this area is not as extensively altered as the massives, but to mine it cleanly is difficult.

Initially concentrator operations were concerned solely with the unburned schist ore. In recent months sulphide content of the concentrator feed has been increased by admixture of massive sulphides and at times pit schist ores have constituted a large percentage of the tonnage of ores treated. The following table reflects those conditions and offers data as to metallic values and mineralogical form of metals and gangue as shown by analyses of monthly composites of concentrator feed:

Figure 4. Two of the small drum feeders

southwest. In the schist ores dissemination of the sulphides in the gangue is such that —65 mesh grinding effects liberation. The association of the chalcopyrite with the marmatite and the pyrite is, however, more complex. This is particularly true in the massives. With a need for separation of these minerals finer grinding becomes necessary. The extent to which such grinding must be carried varies for the different ores from reduction through an .074 mm. 200 mesh opening to .046 mm. 300 mesh to effect unlocking and on some ores locking is still in evidence in the 19 michon, roughly 800 mesh sizes. Metallurgical results are therefore influenced both by degree of dissemination of the copper and contamination of the ores with altered ores from the fire zone of the mine.

Location, Design and Construction

The location for the plant was selected

Period	% Cu Total	% Cu Ox.	Oz Ag	Oz Au	% SiO_2	% Al_2O_3	% Fe	% S	% Zn	% CaO
August, 1927	3.00	....	.632	.011	37.2	13.1	17.5	10.7	1.0	.9
September, 1929	3.06	.17	.738	.016	31.2	11.4	20.4	16.1	1.2	1.6

CALCULATED MINERALOGICAL ANALYSES

Period	% $CuFeS_2$	% FeS_2	% ZnS+	% Schist	% Quartz	% Hyd.Fe*	% $CaCO_3$
August, 1927	8.6	13.5	1.5	52.4	23.0	2.9	1.6
September, 1929	8.9	23.2	1.8	45.6	18.9	1.6	2.9

+ Zn is calculated as sphalerite but is present as marmatite.
* Hyd. Fe=Hydrated iron oxides.
Specific gravity of the ore varies between 3.3 and 3.5.

The variable nature of the ores treated, which is shown by the above table, prevents presentation of a composite picture of ore character. It will be noted that in the above table schist has been classified as a mineral. This is permissible under local conditions because of the chemical uniformity of this gangue material. Following is an approximate analysis of the schist:

	Percent
SiO_2	28.0
Al_2O_3	25.0
FeO	18.0
CaO	0.8
MgO	17.0
Ignition loss	10.0

In the above ores copper is present principally as chalcopyrite, although minor amounts of covellite and bornite have been noted. Oxide copper minerals are negligible in the ore derived from underground mining but reach serious proportions, from the ore dressing standpoint, in the ores from the fire zone of the mine. Inasmuch as the oxides present undoubtedly represent the products from partial calcination of sulphides, concentration of such minerals in the finer sizes of the mill feed would be expected. Oxide content of the ores milled in September, 1929, was above the average and the +100 mesh portion of the mill feed showed 0.118 percent Ox. Cu and the —100 mesh portion O.31 percent. Moisture content of the ore is low, seldom exceeding 1.5 percent and offers no difficulties in milling operations. Aside from the alteration due to fire conditions in the pit, no other alteration incident to mining methods has been noted.

Degree of dissemination of the valuable minerals in the ores of the several types is variable. In general it may be classed as extremely fine as compared with most copper ores treated in the

adjacent to the crushing plant. A plan of the whole Clarkdale works is shown in the paper on the smelter growth by Mr. J. E. Lanning, appearing elsewhere in this issue. Inasmuch as the contours of the ground are not evident it should be stated that the ground profile is uphill from the crushing plant to the concentrator, which necessitates delivery of the crushed ore from a lower to a higher elevation, which is different from usual practice. In this case there was no other suitable location. It is interesting to note, however, that the increment of extra cost for delivering the ore uphill is not over a quarter of a cent per ton. The concentrating plant expenditure did not include the expense of a crushing department because, as will be explained in the paper on "Crushing," by Mr. P. C. Keefe, this department had already been functioning and was of sufficient capacity to crush the amount of feed required by the concentrating department. In this particular setup the concentrator has been in the nature of a minor department with respect to the smelter and the crushing plant was designed in a flexible arrangement to permit preparation of ores and fluxes to meet a variety of conditions in the smelter departments. While the crushing plant has been very efficient from the smelter point of view, it is not an ideal crushing plant for serving a concentrator and some small excess crushing cost, therefore, has to be borne by the concentrator department, but this is not equivalent to the amount of fixed charges which would result from the construction of a crushing plant for the concentrator service alone.

The general location of the concentrating plant is such that tailings can be delivered by gravity through a pipe line to a very desirable location for tailings dis-

posal. The plant is also located favorably with respect to all services such as railroad, water, power, and the like.

The type of construction is that usually found in the concentrating plants of the larger companies in the southwest, being concrete, steel and galvanized iron with abundance of windows for lighting.

The equipment of the plant is not at all complicated. No gravity concentration is included and inasmuch as the feed to the concentrating department is previously prepared by crushing through ¼-in., the only grinding provided by the original flow sheet was single stage. This was followed by flotation in Mineral Separation machines, by thickening and by filtering. The ball mills are 8 ft. by 12 ft. and are of all-steel construction, made by the Llewellyn Iron Works, of Los Angeles. The thickening plant consists of Hardinge thickeners and the filtering plant of American filters.

Flow Sheets

Figure 1 is a flow-sheet of one of the two units of the original mill, as originally built, and shows the extreme simplicity. This flow was adequate for the treatment of schist ores described above. As work began to be performed on the differential flotation of the heavy sulphide ores by mixing them into the feed it became evident that finer grinding facilities would have to be added, and additional flotation capacity. As a result, the arrangement shown in *Figure 2* was adopted for Unit No. 1. This additional equipment, used in experimental as well as in a commercial way, provided considerable new data for subsequent expansion of the mill. *Figure 3* is the flow used at present and illustrates how the additional equipment, shown in *Figure 2*, has been utilized to serve both units of the mill. At the present time another substantial addition in grinding, classifying, and flotation equipment is being made adjacent to original Unit No. 2 and about the summer of 1930, the mill will be equipped for handling still further additions of the massive sulphide ore; thus, in the space of approximately three years the concentrating department has demonstrated its economy on the type of ores for which it was built and has made considerable progress in the attainment of the other original objective; namely, the successful treatment of massive sulphide ores.

While practically all the equipment listed on these flow sheets above described is of standard types, there are some novel features in the design and construction of this plant which can be mentioned.

The supply or feed bin has a capacity of 6,000 live tons; the bin being of two compartments. It might be described as a cross between an ordinary bin and a bedding system inasmuch as the bin can be loaded by a traveling belt conveyor distributor in either horizontal or inclined strata and can be discharged onto the ball mill conveying system in a flow fairly uniform as to size and chemical composition through the use of 96 small drum type feeders under the flat bin bottom. *Figure 4* is a view of two of the small drum feeders.

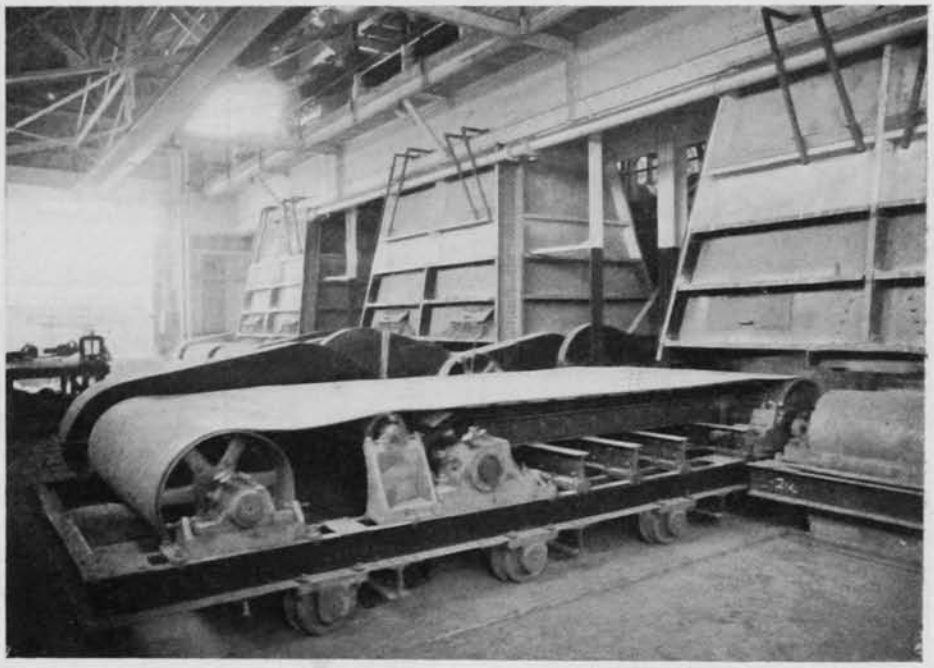


Figure 6. 80-inch Concentrate Feeders

DATA ON TREATMENT AND PRODUCTION

ASSAYS		1929	1928	To Date
Feed	% Cu	3.39	3.13	3.22
	% FeS_2	20.48	16.75	17.57
	Oz. Ag	0.7062	0.6429	0.6816
	Oz. Au	0.0109	0.0087	0.0101
Tails	% Cu	0.316	0.279	0.283
	% FeS_2	17.68	12.41	12.12
	Oz. Ag	0.2602	0.2244	0.2195
	Oz. Au	0.0056	0.0048	0.0046
Concentrates	% Cu	16.03	15.48	14.63
	Oz. Ag	2.6721	2.7181	2.6358
	Oz. Au	0.0315	0.0282	0.0322
	% SiO_2	4.71	4.63	4.55
	% Al_2O_3	2.76	2.77	2.63
DRY TONS CONCENTRATES PRODUCED		92,459	81,551	242,870
MILL RECOVERY				
Cu	%	92.4909	92.7520	93.0352
Ag	%	71.377	73.731	75.596
Au	%	57.854	57.523	64.193
MILL REJECTIONS				
Pyrite	%	69.438	60.210	54.818
Schist	%	95.377	95.760	95.533
RATIO OF CONCENTRATION		5.1119	5.316	4.8887
POWER				
Total K. W. hours		8,277,682	7,122,373	20,116,893
Total K. W. hrs per ton		17.489	16.429	16.987
Distribution				
Conveying and feeding		0.462	0.317	0.382
Grinding		10.329	9.319	9.652
Flotation		3.510	4.148	4.082
Thickening		0.521	0.554	0.584
Filtering		0.387	0.340	0.354
Concentrate handling		0.191	0.195	0.183
Tailing disposal		.....	0.001	.....
Water		0.191	0.173	0.176
Sampling and testing		0.221	0.199	0.206
Lighting and unclassified		0.535	0.415	0.468
32 Oz. air		1.142	0.768	0.900
WATER				
Fresh—gallons		207,848,000	194,151,000	523,308,000
Total—gallons		357,893,000	288,193,000	818,844,000
Fresh gallons per ton		439	447	442
Total gallons per ton		756	665	691
Reclaimed water—percent of total		41.92	32.63	36.09
REAGENTS		Lbs.	Lbs./ton	
Lime—burnt (high calcium)		3,879,000	8.1957	
Cyanide—crude calcium		20,300	0.0429	
Xanthate—Amyl		26,006	0.0549	
Xanthate—Ethyl		32,924	0.0696	
Pine Oil—Yarmour		33,986	0.0718	

GENERAL	1929	1928	To Date
Concts. to roasters—dry tons	93,049	82,665	243,891
Concts. to roasters—percent moisture	12.923	12.93	11.984
Equivalent lbs. Cu in roasters	29,624,579	25,073,070	70,733,639
Equivalent lbs. Cu in process	309,860	300,279	309,860
Equivalent lbs. Cu in tailings	2,405,930	1,965,548	5,318,491
Equivalent lbs. Cu in mill feed	32,040,090	27,118,378	76,361,990
Equivalent lbs. Cu in concts. prod	29,634,160	25,152,830	71,043,499
Ball consumption—per ton milled	1.907	1.565	1.720
Liner consumption—per ton milled	0.312	0.222	0.238
Percent plus 100 mesh in tailings	12.561*	18.308	16.063

* Additional fine grinding equipment was available only 6 months of this year.

United Verde Copper Company

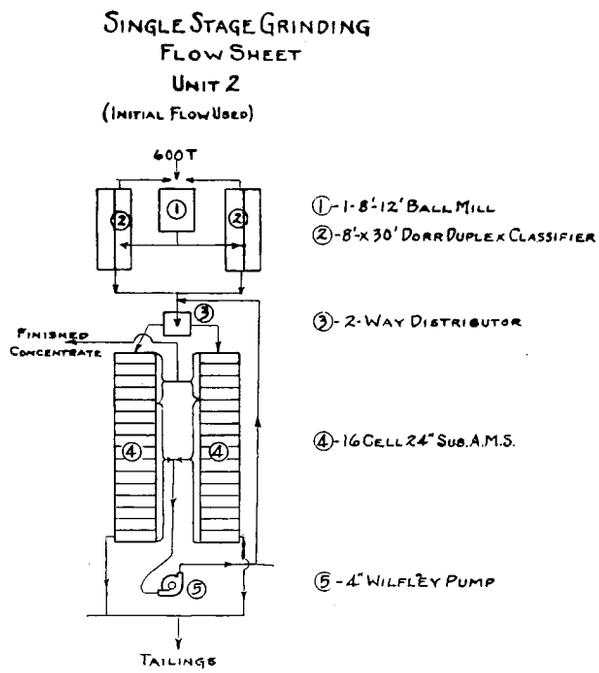


FIGURE 1

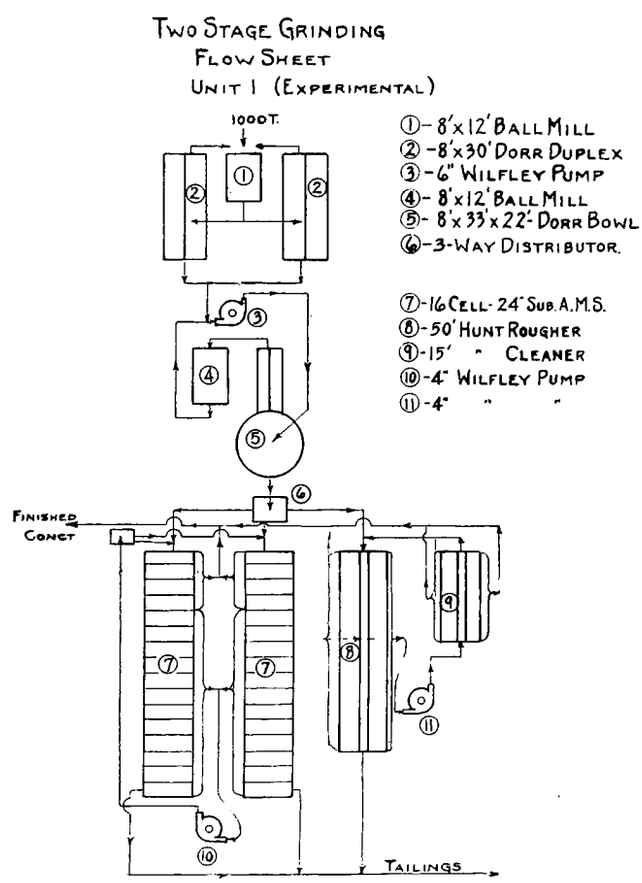


FIGURE 2

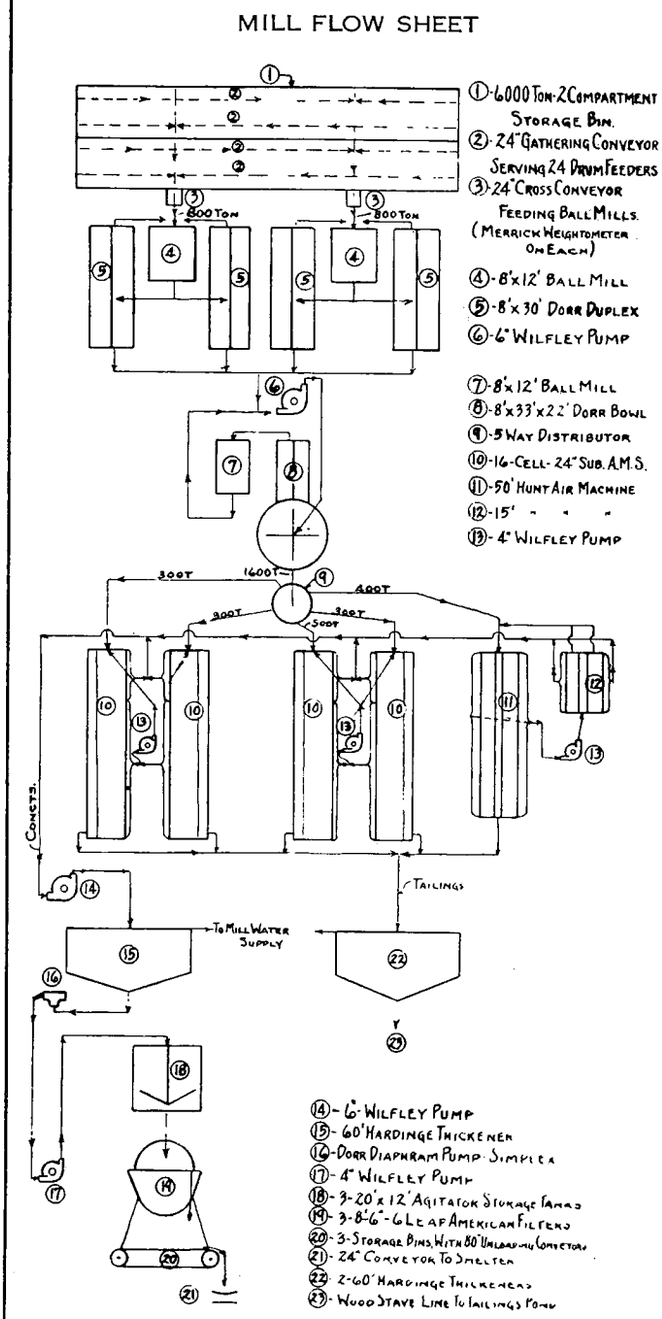


FIGURE 3

United Verde Copper Company

Another novel feature of this plant is the handling of the filtered concentrates and this feature is the result of necessity to coordinate operations with the smelter. *Figures 5* and *6* illustrate the concentrate handling step, which is not found in other plants. It will be noted that the American filters are superimposed over bins of about 100 tons capacity which have downwardly diverging sides and no bottoms; the load of the concentrates resting on 80-in. rubber conveyor belts, one of which is shown in *Figure 6*, pulled out onto the repair floor. The normal position of the belt is under the bin. Two such concentrate feeder belts are provided under each of the three filters. This arrangement provides a storage of filtered concentrates which can be withdrawn when needed, without excess labor, at the convenience of the roaster department. The 80-in. belts travel at a very slow rate of speed and are loaded with a deep bed of concentrate. All six of these feeder belts and discharge onto one high-speed 42-in. conveyor belt which delivers the concentrate to the roaster plant where it is mixed with dry crushed ore as the latter is being fed to the roaster hoppers. In deciding upon this installation it was apparent that the fixed charges on this investment would be less per day than the extra cost of handling concentrates to the roasters on each of the three shifts rather than on one shift, which is the most economical practice from the roaster point of view.

Treatment and Production

The concentrator was originally built for 1,000 tons per day and was actually operated with the original equipment at the rate of 1,200 tons per day on schist ores with a flotation feed containing 18 percent plus 100 mesh. The additional equipment, shown in *Figure 3*, has permitted the concentration of 1,600 tons per day with somewhat finer grinding. The tonnage has been variable according to conditions. The following tabulation gives all the data on treatment and production by years up to the end of 1929:

Tonnage	1929	1928	To Date
Dry tons ore on hand first of year	5,946	6,564	.....
Dry tons ore delivered to ore bins	476,947	432,908	1,193,823
Dry tons ore treated	473,295	433,526	1,184,225
Dry tons ore on hand end of year	9,598	5,946	9,598
Tonnage rate per 24 hours	1,297	1,185	1,156
Average number of units running	1.958	1.972	1.899
Average tons per unit per 24 hours	662	601	609

(*Refer also to tabulation on page* 365)

Costs

The concentration costs are as follows:

	Dollars per ton
Receiving and crushing to minus ½"	.203
Conveying and feeding	.023
Grinding	.164
Flotation	.176
Thickening	.018
Filtering	.016
Concentrate handling	.005
Tailings disposal	.005
Water	.027
Routing, sampling and testing	.033
Unclassified	.067
	.737

Power in the above costs was approximately 75/100 cents per K. W. hour; tons per man shift, 38.8.

SMELTER CRUSHING PLANT

(*From page* 64)

Figure 4 shows the flow-sheet of the coarse plant, and *Figure 5* shows the fine plant. *Figure 6* is a general view of both divisions and of the stock piles, surge tanks, dust collectors, etc.

The plant is divided into two units called the coarse plant and the fine plant with the screening house between.

Practice

The ore received at the crushing plant is by means of ore cars handled by the V. T. & S. Railroad. These cars are of the bottom dump type and are of about 90 tons capacity. The size of the ore received is run-of-mine material. During the present practice of all reverberatory smelting, all the ore received at Clarkdale is crushed to 3-mesh size. This includes direct smelting ore and concentrating ore. The one exception to this is quartz ore, used in the converters as a flux. The latter product is crushed to 1 in. in size and all fines under ¼ in. in size are screened out. This latter procedure is purely for physical reasons, as it helps to regulate the dust conditions in the converter aisle and does not "hang up" in the chutes, over the converters, as readily as if mixed with the natural fines.

During the operation of the blast furnaces the product delivered to them as feed is 4 in. x 1½ in. in size.

During crushing operation all ore is crushed to 4 in. in size at the primary crushers. This product is then passed over Hummer screens, type 39, with from 1½-in. to 2½-in. opening. The screen opening is dependent upon smelter practice, whether blast and reverberatory furnaces are in operation or reverberatories alone. If the blast furnaces are in operation part of the oversize of this screen is the feed to these furnaces. If not, this product goes to a stock pile and is used as feed to the fine crushing plant.

The undersize of the above screen passes over a second screen with approximately a 3-mesh opening. The undersize of this second screen goes directly to fine storage as this is part of the finished product to be used as roaster feed. The oversize of this second screen is a middling product and goes directly as feed to the fine crushing plant; this feed being mixed with ore drawn from the coarse stock pile. The accompanying flow-sheet will show this operation more clearly, perhaps.

The fact that there is one separate stock pile for each class of ore permits the fine plant to draw from the particular pile necessary to correct the ore to the proper mineralogical analysis for a roaster charge.

The fine plant feed is first passed to Symons 48-in. disc crushers, vertical type, and is crushed to about ⅞ in. in size. By means of elevators it is then passed over Hummer screens, type 37, with a 3-mesh opening. The oversize of these screens is the feed for the 56-in. x 24-in. Traylor rolls. The product of the rolls is returned to the same screens. The undersize of the screens is the finished product and joins the natural fines from the coarse plant. After this mixture from the both plants combined is passed over several conveyors it is delivered into the fine storage, or roaster bins. While the ore is being delivered to the bins the distributor runs back and forth over the bins, being filled to give, as nearly as possible, a uniform mixture.

Treatment and Production

There are three classes of ore received from the mine and pit and are as follows:

(1) *Smelting Ore.*—Direct smelting ore and is a heavy pyritic material with a spread in the silica content from about 17 percent to about 25 percent. These limits vary considerably from time to time, but the above figures are a fair average.

(2) *Concentrating Ore.*—This is crushed and delivered to the concentrator to be milled, the chemical and screen analyses being given in another section of this paper.

(3) *Quartz.*—Silicious material used as a flux in the converters and mixed with the smelting ore to raise the silica content preparatory to roasting.

In addition to the above ore, lime-rock is crushed from time to time to be used for fluxing purposes.

An average of about 3,000 tons of smelting ore is crushed each day to 3-mesh size, and during the crushing operation either silicious or basic material is added, as practice demands, to bring the fine ore delivered to the roaster bins to an approximately correct mixture for the roaster charge.

Approximately 1,400 tons of concentrating ore are delivered to the concentrator each day. This product is also 3-mesh in size and is the feed to the ball mills.

An average of about 300 tons per day of quartz, 1 in. in size, is prepared for the converters, and the amount of quartz used each day in the roaster charge depends entirely upon the silica content of the smelting ore as received from the mine and pit.

Costs

The following figures are an average cost per ton for the first 10 months of the year 1929:

	Coarse Plant	Fine Plant
Repairs	$0.0339	$0.1012
Operation	0.0317	0.0768
Total	0.0656	0.1780
Tonnage	1,664,168	975,767

Considering the fact that about 20 percent of the tonnage of ore can be screened out in the coarse plant and delivered as finished product to the storage bins, with only coarse plant charges against it, the average cost of all ore delivered at 3-mesh size is approximately 21 cents per ton.

Conclusions

While it might appear that the use of one crushing plant for both the smelting ore and the concentrating ore is unusual, it is being worked out to good economical advantage in this case. As the present plant was in operation and supplying the smelter before the concentrator was built, and as the operation of the concentrator may decrease some of the ore taken by the smelter, the present method seems to be the most logical and most economical plan to follow.

Because of the very strong spirit of cooperation between the Mining and Smelting Departments, the scheme of mixing the ore for the furnaces by the Crushing Department has worked to very good advantage, and is reflected in the economical furnace operation during the past year.

United Verde Copper Company

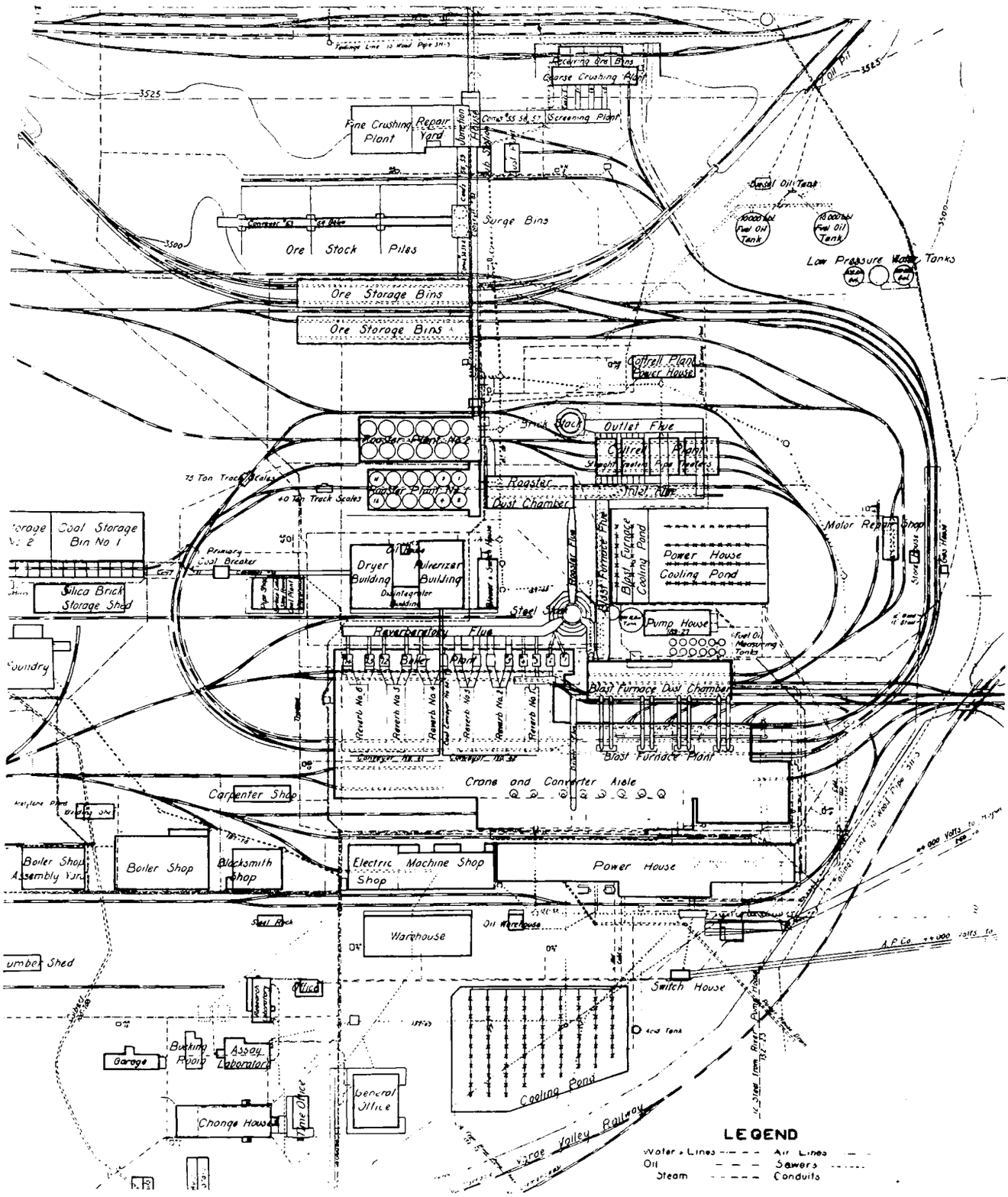


FIGURE 1

GENERAL PLAN OF SMELTER

United Verde Copper Company

The ROASTER PLANT

By F. H. Parsons

ASSISTANT SMELTER SUPERINTENDENT

History

THE original design of the roaster plant called for six roasters of the Wedge type, 21.5 ft. in diameter with six hearths. This installation was completed and put into operation in July, 1915. It was soon discovered that this equipment did not produce sufficient calcine for the reverberatories, so six more roasters of the same type and size were constructed and put into operation in 1917. These 12 roasters, called No. 1 roaster plant, were sufficient until the enlargement program of 1921, called for additional capacity, so 12 more units of the Wedge type, 22.5 ft. in diameter with seven hearths, were constructed and put into operation in 1922. This group is called roaster plant No. 2. The present roaster department therefore consists of 24 units—one of which has been used this past year as a drying furnace for concentrates, and one is usually down for repairs—leaving an average of 22 roasters supplying calcine for the reverberatory charge.

Theory and Principles

The roasting problem of the United Verde smelter is that of preparing for matte smelting, what is ordinarily known as direct smelting ore. This is different from the roaster problem of most smelters today, in that in the United Verde practice only about 10 percent or less of the roaster feed consists of flotation concentrates, while in many other smelters the current practice is the roasting of a feed which is almost entirely flotation concentrates.

In order for the reader to visualize this problem, the following mineralogical analysis is given of the smelting ore, of the flotation concentrates, and of the silicious fluxing ore:

APPROXIMATE PERCENTAGES

Percent	Feed	Water	Chalcopyrite	Sphalerite	Pyrite	Schist	Quartz
Smelting ore	80	2	13.5	4.8	46.2	14.4	17.7
Concentrates	10	9	43.5	10.8	42.4	1.6	1.5
Silicious flux	10	3	5.8	2.6	9.0	32.8	48.7

To further visualize the roaster feed, it should be noted that the smelting ore is almost dry.

The following tabulation shows the size of the grain particles in the roaster feed:

Mesh	Percent	Cumulative
Plus 3	2.1	2.1
" 20	62.5	64.6
" 65	15.4	80.0
" 100	3.6	83.6
" 200	5.4	89.0
Minus 200	11.0	100.0

From the above description of the roaster feed one can readily see that the problem is one, metallurgically, of eliminating a large amount of iron per unit of copper, which is another outstanding difference from the practice of smelters treating principally flotation concentrates.

Situated as the connecting link in the flow of ore, between the crushed ore bins and the smelting furnaces, it is the function of the roasters to supply calcine in a continuous flow as the reverberatories demand feed. These calcines must be delivered as hot as possible, with just sufficient sulphur to give the right amount of matte for the converter department's use. The furnaces must be operated with as little excess air as is consistent with proper sulphur elimination. An under-roasted calcine will produce too much low grade matte for the converters to handle, and an over-roast will cause the formation of an excess of magnetite and a decided slowing of reverberatory performance. Too much excess air will cause excessive dust losses with the resultant overburdening of the Cottrell smoke treating plant. Also, this condition will cause the cooling of the roaster walls and hearths necessitating the use of oil to supply the heat, which should have been conserved from the oxidation of the sulphur of the charge. The function of the roasters, therefore, is to strike a happy medium in operation, supplying the smelter unit with sufficient feed, of the proper sulphur content, but at the same time conserving heat and producing minimum dust losses.

Location, Design, and Construction

Figure 1 shows in plan the relative location of the roaster department with respect to the crushed ore storage bins, preceding, in the flow sheet, the roasting operation, and with respect to the reverberatory department, following it in the line of flow. On page 80 is a general view of the roaster building.

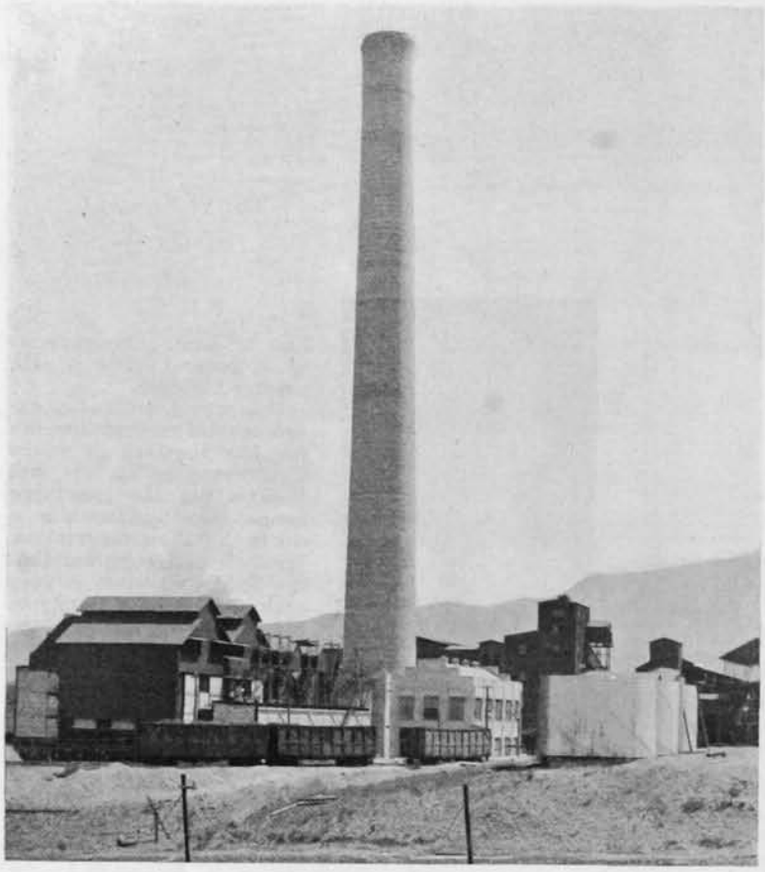
One would ordinarily expect to learn that the ore for the roasters is mixed by a bedding system, but such is not the case, and hence, the local system is worthy of some description. While it is recognized that the bedding system is very desirable in the majority of cases, still considering the great tonnage of ore which is handled in the United Verde plant daily, a very extensive bedding system would be necessary and it is doubtful if the great expenditure necessary to build such an extensive system would be justified, inasmuch as very good results are obtained with the present facilities.

The crushed ore is delivered to the storage bins in lots of about 500 tons, each bin being carefully sampled and analyzed. The predetermined lots which are to make up the roaster feed are bedded on a conveyor belt in proper percentages to make the correct charge and the mixture is then delivered to the roasters. the three thousand tons of feed daily is drawn and delivered to the roasters in one shift of eight hours, showing that the system is very economical.

As indicated in the introductory paragraph of this article, the furnaces are standard Wedge roasters. The tabulation (page 75) will show the construction of the No. 1 and No. 2 plants in more detail.

The usual speed of revolution in practice is 85 seconds for the roasters in the No. 1 plant, and 103 seconds for the No. 2 plant.

The use of the variable speed motors for driving the roasters has made a very decided improvement in continuity of operation. The feed opening is now fixed and the speed of the feed is regulated by the speed of revolution of the furnace arms. Due to the fact that the quantity of material on the hearths is now always constant, the operator can increase or decrease the calcine produced by speeding up or slowing down the roaster revolutions, with the further advantage that he gets immediate results, and does not have to wait until a cutting down of the feed is reflected, about two hours later, by a slowing up of calcine produced. The constant depth of material on the hearth makes it easier for him also to judge accurately the degree of roast at any given point, and leaves the manipulation of the quantity of air introduced as the only variable requiring control to main- (*Continued on page* 75)



Photograph A. General view of Cottrell Treaters, Substation and Stack

Cottrell Plant

By Frank W. Denny

FOREMAN

IN 1918 it was obvious that dust losses were excessive at the United Verde Copper Company's smelter at Clarkdale. A cooperative survey made by the Copper Company and the Western Precipitation Company showed a total daily dust loss of approximately 100 tons by the various departments. The Cottrell plant was immediately designed and constructed as soon as convenient.

Three types of treaters were built. The down draft pipe type were put in operation on July 22, 1922, and, except for a few minor alterations, are still in service as built. The Hayden type, transverse pervious flow treaters, were started on August 26, 1922. After a short time it was found that acid was rapidly eating the steel springs holding the chain receiving electrodes and these treaters were soon down for repair. The springs and chains were replaced by ⅛-in. standard steel pipe. The Tooele type, parallel flow plate treaters were put in operation on January 18, 1923. These treaters gave very satisfactory service for several years, but as the corrugated copper bearing steel plates, the receiving electrodes corroded, it was impossible to replace them without keeping each treater shut down for several days. To overcome this difficulty the plates were replaced by a screen of ⅛-in. standard pipe which is easily renewed. The alteration also changed the treaters from an impervious to a pervious type.

The Problem

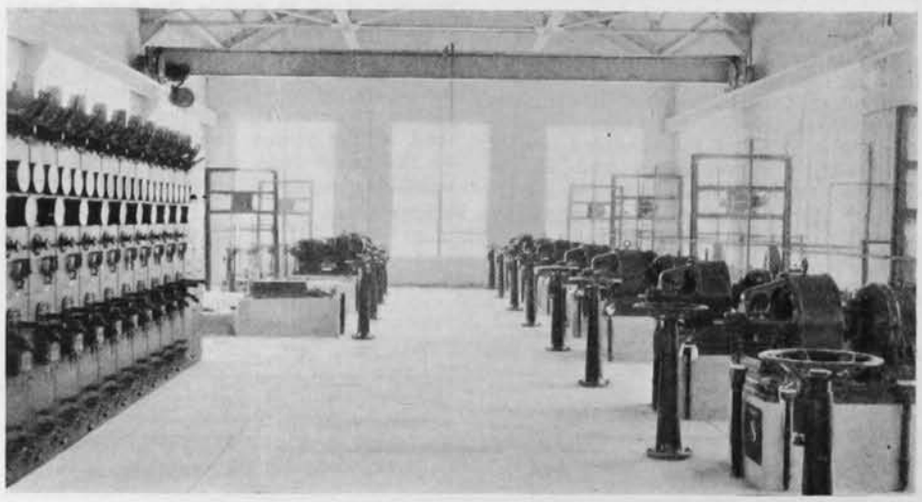
The survey of 1918 disclosed that the roasters were receiving dry crushed ore that was very dusty. The roasting operations gave a gas volume of approximately 232,000 cu. ft. per minute containing 64 tons of dust per day. The combined blast furnace and converter gases were 735,000 cu. ft. per minute and contained 35 tons of dust per day. The reverberatory gases did not contain sufficient dust to need treatment. The total copper lost in these gases was approximately 11,000 pounds per day. The Cottrell Plant was built as a solution to the problem of how to save these losses.

Location, Design and Construction

The location of the Cottrell Plant and its general arrangement is shown by *Figure 1*. Photograph A shows a view of the treater building, Cottrell stack, and substation.

The pipe treaters consist of 12 treater units each containing 120 12-in. diameter pipes 16 ft. long. All units are in parallel and are down draft. A small chain, the discharge electrode, hangs down the center of each pipe. (See Photograph B.) As the gas passes through the pipe most of the dust is precipitated on the inside of the pipe, but a small amount is collected on the chain. Inlet and outlet dampers to each treater are raised and lowered by an air lift. (Photograph C.) These treaters are hand rapped.

In the transverse flow treaters the gases pass horizontally through transverse vertical curtains or screens of small pipes and rods arranged in sets of one screen of discharge rod electrodes in the center between two screens of grounded pipes. The pipes are hung vertically and the rods horizontally. (See *Figure 2*.)



Photograph D. Cottrell Plant Substation, Rectifier Floor

of United Verde Copper Company

There are three treaters of this type, each containing three banks of electrodes in series. Each bank has 10 sets of screens. Each discharge electrode screen has 60 ¼-in. twisted square rods or 3,050 ft. of rods. The spacing between rods is 6 inches. The receiving electrode screen has 42 ⅛-in. standard pipes each 15 ft 6½ in. long. The spacing between pipes is 2¾ in. The clearance between receiving and discharge electrode is 4 in. Each treater chamber is 60 ft. long, 10½ ft. wide, and 18 ft. ft. deep.

The cumulative length of effective electrical field through which the gases must flow is 20 ft. The cross-section of the effective electrical field is 16 ft. by 9½ ft. or 152 sq. ft.

The treaters are rapped mechanically and have air hoists for operating dampers.

In the parallel flow treaters the gas also flows horizontally, but instead of passing through the vertical electrode screens it flows parallel to them. The discharge screens are made of horizontally hung ⅜-in. twisted square rods 18 ft. long and are alternated by screens of vertically hung grounded ⅛-in. standard pipe 12 ft. 1½ in. long. As the pipes wear out they are being replaced by ⅜-in. round steel rods. (See *Figure 3.*)

Like the transverse flow, there are three treaters, each containing three banks in series. Each bank has 16 discharge electrode screens or 272 electrode rods. There are 17 screens of receiving electrodes or 1,326 pipes in each bank. The spacing between pipes is 1⅞ in. and that between discharge electrodes is 8 in. The clearance between discharge and receiving electrode is 4 in. The chambers are 20 ft. long, 10½ ft. wide, and 13 ft. deep.

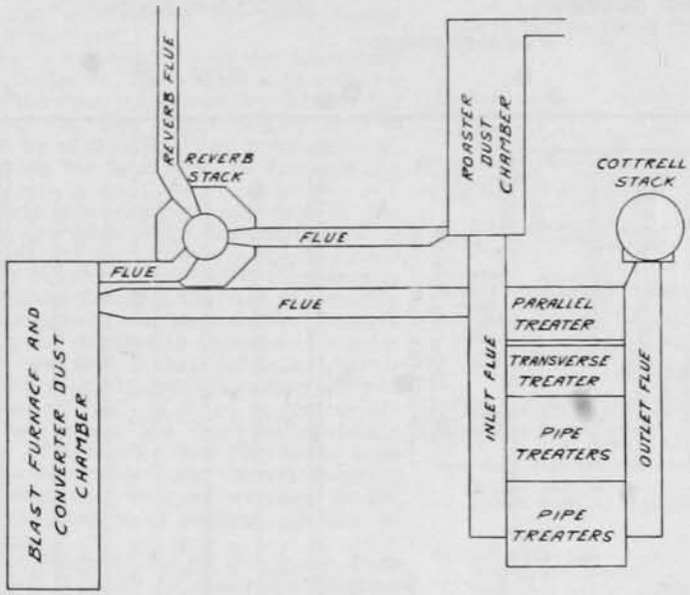
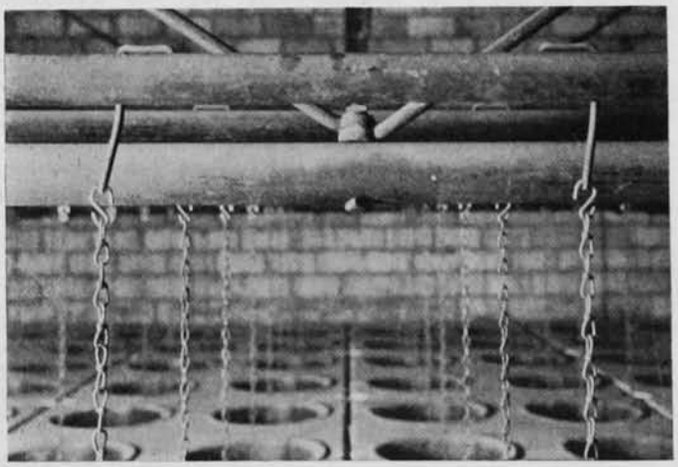
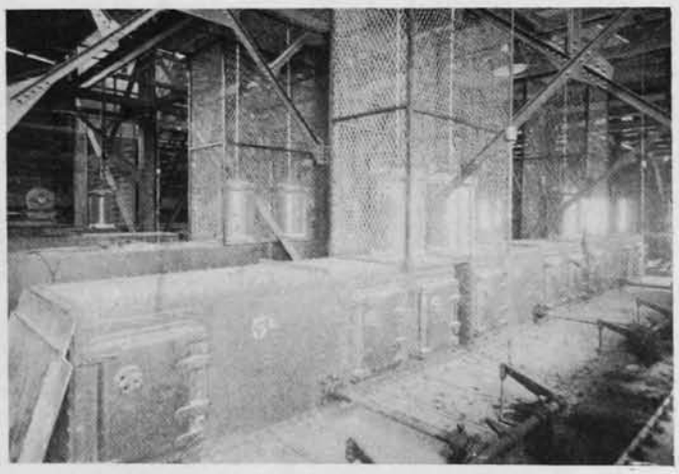


FIG. 1



Photograph B. Discharge Electrodes of Pipe Treaters



Photograph C. Transverse Flow Treator, Top View

United Verde Copper Company

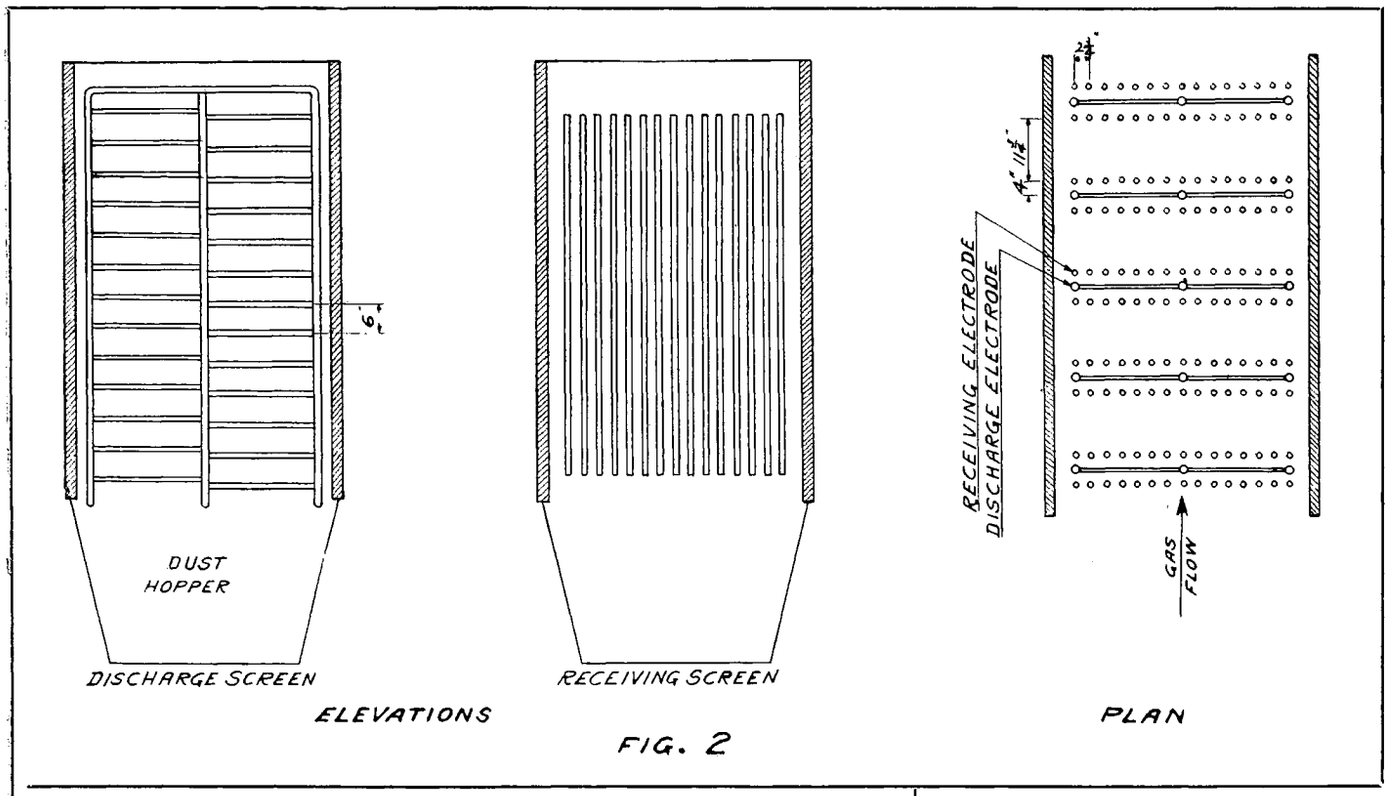


FIG. 2

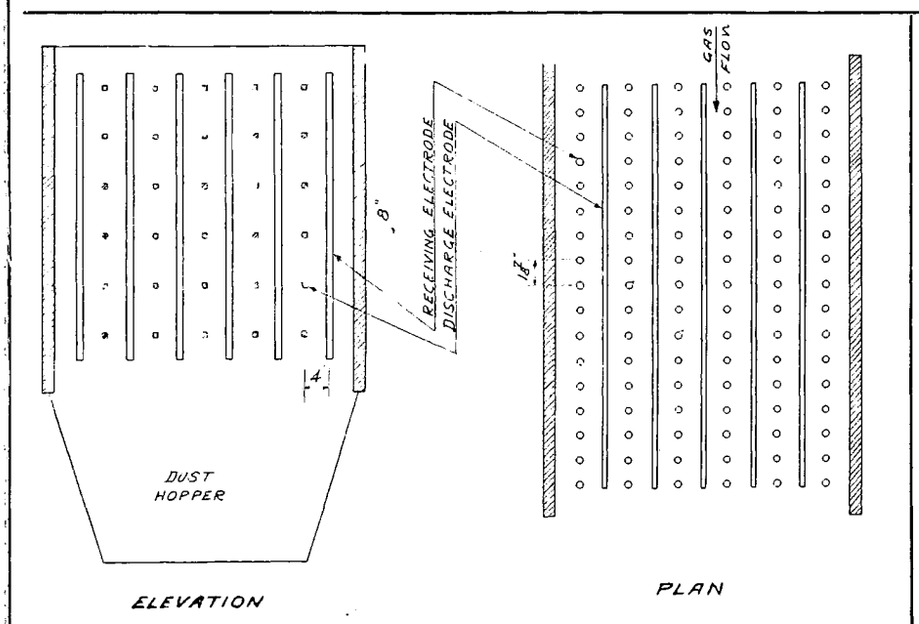


FIG. 3

The cumulative length of effective electrical field through which the gases must flow is 54 ft. The cross-section of the field is 12 ft. by 10 ft. or 120 sq. ft. These treaters are also rapped mechanically and have dampers operated by air hoists.

All flue and treater walls, with a few exceptions, were built of perforated brick laid in cement mortar. Acid corroded the brick and mortar in many of the walls so that when rebuilt they were made of vitrified paving brick laid in a mortar composed of sodium silicate and ground silica. The steel walls of some of the treater flues were also corroded so badly that they were replaced by brick.

The stack is 430 ft. in height and has an inside diameter at the bottom of 36½ ft., and at the top it is 29 ft. It is also built of perforated brick and cement mortar.

Power is furnished to the treaters by 10 mechanical disc rectifiers, two of which are auxiliaries, and each is coupled to a 25 K. V. A. motor generator set. The motor is 40 hp., alternating current. Two exciters furnish the field for the generators. These and the switch board are located on the upper floor of the Cottrell sub-station. (See photograph D.) All transformers are on the lower floor of the same building. Voltage from 50,000 to 100,000 may be furnished from any of the rectifiers.

Practice

Except for a few short periods it has always been the policy to mix the different gases before they are sent to the treaters. The mixing is accomplished by damper control in the inlet flue. Two advantages are gained by mixing. Much of the excess acid in the roaster gas is neutralized by the zinc oxide in the converter gas. The second advantage is the dilution and acidifying of the converter gas which is very difficult to handle alone. The average temperature of the roaster gas as it enters the inlet flue is 512° F. The blast furnace and converter gases averaged 411° F., for 1929, but are subject to much variation due to the number of blast furnaces in operation.

Pipe treaters are normally operated on the 62,500-volt tap, and transverse and parallel type on the 50,000 tap.

The Cottrell crew consists of one shift boss, six operators (two to a shift), one repair man and helper, and one handy man. Testing and clean-up will average one more man a day; making a total of 11 men.

Treaters are rapped from three to four times a shift by operator, who also sees that dust is correctly drawn, insulators cleaned, and floors, and removes minor grounds from treaters. The sub-station operator makes 45-minute readings from switch board, keeps rectifiers and motors oiled and repaired, and keeps building clean. The repair man removes the major grounds and does the smaller repair jobs. The larger repair jobs are done by the mason crew or the boiler shop.

The dust collected is drawn out daily into calcine cars and taken to the reverbs.

United Verde Copper Company

Tests are made by-monthly at all treater outlets to determine their efficiency and to check the work of the section.

Production

During 1929, the average daily recovery of treaters and inlet flue dust was 145 tons of dust carrying from 7 to 8 percent copper, from 2½ to 3 ounces of silver, and about .04 of an ounce of gold. The recovery of copper was 91.5 percent.

The average daily dust recoveries by treaters were: pipe 76.9 tons, transverse flow 24.4, parallel flow 28.6, and inlet flues 15.1 tons. The average volumes were: pipe 431,000 cu. ft. per minute, transverse 217,000, and parallel 238,000, making a total of 886,000 cu. ft. per minute. The average outlet temperature for this period was 346° F.

The parallel flow treaters showed a 2 percent higher dust recovery than the transverse flow which were 1 percent higher than the pipe. In copper recovery the parallel were 2 percent higher than the transverse, and 6 percent higher than the pipe.

Costs

The total construction costs of the Cottrell plant per cubic foot of gas per minute for the various treaters were: pipe, 118.5 cents; transverse pervious, 78 cents; parallel impervious, 72 cents. The repairs up to August, 1929, on the same basis were: pipe, 15.1 cents; transverse, 11.5 cents; and parallel, 16.6 cents. The above figures are based on average volumes handled by each type of treater. Based on 1929 averages approximately 10 man-hours are required daily to handle gas flowing at the rate of 100,000 cu. ft. per minute. Dust was precipitated at the rate of 1 ton per .68 of a man-hour and required 22.4 kilowatt hours of power per ton. Each million cubic feet of gas used 2.28 kilowatt hours of power. Pipe treaters required 3.32 kilowatt hours per million cubic feet, transverse 1.39, and parallel 1.27. Kilowatt hours required to precipitate one ton of dust by treaters were: pipe 26.6, transverse 17.8, and parallel 15.2.

The parallel pervious type treater seems to be the most efficient and economical for the kinds of gas treated at this smelter. The Cottrell plant has amply justified the cost of its installation.

ROASTER PLANT

(*From page* 71)

tain uniform results.

Referring to *Figure 1* it will be noted that both plants discharge their gases into a common dust chamber 140 ft. long, 60 ft. wide and 30 ft. high. This chamber has one zone 20 ft. long hung with wire baffles. From this chamber the gases enter the Cottrell system and are there treated.

Practice

In order to get the best roaster and reverberatory results it is primarily necessary that the proportioning of the ingredients of the roaster feed be accurately figured. The method in use at the United Verde smelter is to calculate the charge based on the mineralogical composition of the ore, rather than on the chemical analysis. A careful study made some years ago of the mineralogical ingredients of the ore compared with the chemical analysis makes it now possible to write the percentage of chalcopyrite, sphalerite, pyrite, schists, quartz, and hydrated iron oxides in any given United Verde ore. By a comparison of furnace results over a long period of time it was found that if the roaster feed was made up to contain 17 percent schist, and 19 percent quartz, the best results were obtained in both roaster and reverberatory operations. There is little difficulty in attaining this ideal charge day after day.

The two roaster plants are supervised by a foreman. Each plant is handled by a roasterman, helper and one laborer for each shift. The charge mixing crew is made up of 5 men; these, with one oiler, make up the total crew of 22 operating men.

There are certain conditions in the plant operation which demand the use of fuel oil. If the sulphur on the feed falls below 24 percent we find it necessary to burn some fuel. If the rate of smelting at the reverberatories increases, necessitating an increase in tonnage of roaster feed, we find it impossible to deliver enough properly roasted calcine without the use of fuel. A delay in roaster operation causes the temporary closing down of a roaster, and also costs some oil to bring back the correct roasting temperature. Over an average month, the oil used is .4 gallons per ton of charge.

The temperature of the gases from the No. 1 plant is about 500 degrees Fahrenheit. Those from the No. 2 plant are about 700 degrees.

The dust discharged from both roaster plants and deposited in the roaster dust chamber amounts to approximately 45 tons per day or 1½ percent of the total roaster charge. The dust precipitated by the Cottrell smoke treater from roaster gases approximates 125 tons per day or 4 percent of the total roaster charge.

The gases from No. 1 plant will show approximately 4 percent SO_2; those from No. 2 plant 4.5 percent SO_2.

	Plant No. 1	Plant No. 2
Number of roasters	12	12
Location of feed	Inside of hearth	Outside of hearth
Drying hearths	1	1
Roasting hearths	6	7
Drying hearth arms	4	2
Roasting hearth arms	2	2
Drying hearth blades	6	5
Roasting hearth blades	8	8
Method of cooling arms	1,000 cu. ft. circulating air per minute per furnace.	1,600 cu. ft. circulating air per minute per furnace.
Speed of revolution	Max. 90 sec. Min. 240 sec.	Max. 75 sec. Min. 180 sec.
Feed per 24 hours per roaster	100 tons	150 tons
Calcine produced per roaster per 24 hours	75 tons	115 tons
Area of drop holes per hearth	Outside—18 sq. ft. Inside—22 sq. ft.	25 sq. ft. 25 sq. ft.
Total number drop holes	Outside—6 Inside—1	14 1
Drive	2 batteries of 6 roasters each driven by variable speed 20 hp. motors.	4 batteries of 3 roasters each driven by variable speed 15 hp. motors.

Treatment and Production

The average analysis of the feed both chemically and mineralogically is as follows:

	Percent
H_2O	3.5
Cu	5.3
SiO_2	23.6
Al_2O_3	4.3
Fe	26.2
S	26.3
Zn	3.0
CaO	0.9
$CuFeS_2$	14.4
ZnS	4.2
FeS_3	36.3
Schist	17.2
Quartz	19.0
Hyd. Fe	4.6
$CaCo_3$	1.6

The calcine produced from the above feed is as follows:

	Percent
Cu	6.0
SiO_2	27.5
Al_2O_3	4.8
Fe	31.2
S	10.5
Zn	3.5
CaO	1.4

The tonnage of ingredients on the roaster charge for a typical month is as follows:

	Tons
Smelting ore	70,341
Quartz	6,456
Concentrates	8,577
Ore and concentrates	85,374
Lime rock	1,674
Cleanings	3,788
Total charge	90,836
Calcine recovered	72,661
Shrinkage	18,175
Tons charged per furnace per day—No. 1 plant	100
Tons charged per furnace per day—No. 2 plant	150

Costs

The average cost of operation per ton charge is as follows:

Operating Belt Conveyors—	
Wages and bonus	$0.0301
Power, repairs, supplies	0.0135
Total	$0.0436
Operating Roasters—	
Wages and bonus	$0.0490
Fuel	0.0181
Power, supplies, repairs	0.0630
Total	$0.1301
Miscellaneous—	
All charges	$0.0133

Total cost of calcining, $0.1870 per ton of roaster feed.

Conclusions

Our experience has disclosed certain facts which are of importance in discussing roaster problems. Among those facts are these: That the most economical tonnage which should be put through a roaster is not the maximum tonnage possible; exceeding the optimum results in higher fuel costs and greater dusting. Emperical formulae deduced from observation on practice indicate that dusting is proportional to the square of the tonnage per furnace day and is also approximately inversely proportional to the percent of moisture in the feed. Heavy sulphide ores prepared for roasting by crushing through 3 mesh yield satisfactory economy in roasting and satisfactory uniformity in roaster product.

We believe, and we think that the foregoing data will convince the reader that the United Verde roaster problem has been economically met.

United Verde Copper Company

Blast Furnace Smelting

By C. R. Kuzell

SMELTER SUPERINTENDENT

IN COPPER smelting plants in general there has been a very marked tendency towards abandonment of blast furnace smelting of sulphide ores for the production of copper matte. Certainly a smelting plant employing only blast furnaces for the production of matte is pursuing obsolete practice. There are certain conditions, however, which may prevail in plants which employ reverberatory furnaces for the production of matte justifying, at least intermittently, the supplemental use of blast furnaces. For this reason and also for the benefit of record, before all copper blast furnaces go into the discard, it was thought to be worth while to present this paper describing the United Verde blast furnace practice.

History

Up until the abandonment of its Jerome smelting plant and until the blowing in of its Clarkdale smelting plant in 1915, the United Verde Copper Company employed, for the production of matte from sulphide ores, only the blast furnace method of smelting. The practice during the last years of the life of the Jerome plant was described by Richard H. Vail in an article published in the *Engineering & Mining Journal*, August 23, 1913.

The new plant at Clarkdale was equipped with four blast furnaces, when constructed in 1915, and also with three reverberatory furnaces. Apparently it was the intention that even in the new plant the blast furnaces would be more important in the production of matte than the reverberatory furnaces and that the latter would be used mainly for the treatment of fine ore and flue dust. The four blast furnaces have not undergone any remodelling of importance during the intervening 15 years; only a few slight improvements having been made on the original design. Three of the four furnaces are still maintained in condition for immediate operation should conditions demand their use. During the past few years, one and sometimes two of these furnaces have been operated during times of heavy production.

Location, Design and Construction

Figure 1 in the article on "Roasting," which appears in this issue (page 70), shows in plan the location of the blast furnace department with reference to the other departments of the smelter. It also shows the arrangement of the furnaces and settlers of the blast furnace department.

Figures 1 and *2*, accompanying this article, are, respectively, vertical cross section and vertical longitudinal section of a furnace and settler.

All of the furnaces are identical. The inside dimensions at the tuyere level is 48 in. wide by 26 ft. 8 in. in length, making a horizontal area at this level of 106.7 sq. ft. The height of the furnace from the center line of the tuyeres to the charge floor is 17 ft. Other dimensions may be read from the drawings from *Figures 1* and *2*.

The furnace walls are constructed of steel plate water jackets with 6-in. water spaces. The inside walls, each consisting of 8 jackets 3 ft. 4 in. wide and approximately 17 ft. in height. Each side jacket is perforated for three tuyeres 1 ft. 1¼ in. center to center. The end walls are also water jacketed and one of the end walls is provided with an opening for the spout through which the matte and slag discharges into the settler. The water jacketed walls and the weight of the smelting charge in the furnace is carried on cast iron sole plates which, in turn, are supported on steel beams resting on concrete piers, there being an air circulation between the sides of adjacent piers and an air space of 1½ ft. between the pier tops and the sole plate. A refractory hearth of magnesite brick is laid between the jackets and on top of the sole plate. The thickness of this hearth being 21 in. at the back end and 18 in. at the front end. The feed plates are inclined and are made of cast iron. These plates as well as the entire furnace super-structure and charge floor are carried on steel building columns. The super-structure or hood of the furnace is made of steel plate. The sides of this hood, however, are hollow steel U-shaped pipes through which the air blast is circulated for preheating of the air. This furnace top or hood is connected by two steel goose-neck flues 6½ ft. in diameter to a dust chamber which serves all four furnaces as well as the gas from the converter plant. This dust chamber is 220 ft. long, 60 ft. wide and 48 ft. in height. One zone 20 ft. long of the dust chamber is hung with wires to assist in the separation of dust from the gases. However, the gases, after leaving the dust chamber, are treated in the general Cottrell plant for the final removal of the solid contents. One blast main of graduated diameter serves all four furnaces. The branch blast pipe for each furnace is 36 in. in diameter, supplying a bustle pipe of smaller capacity. The bustle pipe is designed so that its section is a combination of a horizontal beam surmounted by an inverted semicircle giving sufficient beam strength so that it can resist the outward thrust of the side wall jackets to which the bustle pipe is connected by means of jacks.

The United Verde blast furnace spout construction is worthy of note because it is what may be called a dry spout, i. e., non-water cooled. It is a simple iron casting of rectangular cross-section consisting of a bottom and two side walls open at the top and at both ends and provided with lugs on the sides through which long steel bolts are inserted for the purpose of bolting the spout in place against the furnace. The spout is 5 ft. long, 30 in. wide, and 23 in. high. The thickness of the metal is 2¼ in. It is lined with magnesite brick in such manner that a diverging slot from 4 to 7 in. wide open on the top and both ends remains inside the lining; the slot dimensions being 5 ft. long and 16 in. high. When the spout is in place the edges of one end and of the brick abutt tightly against the magnesite brick lining of the furnace which constitutes the breast wall. The aperture in this breast wall is rectangular in section, 4 in. by 4 in. The settler end of the spout is filled with an inclined dam made with a plastic mass of clay of such height that the matte and slag retained between the dam and breast aperture will be sufficient to prevent blowing of the furnace gas out through the breast opening.

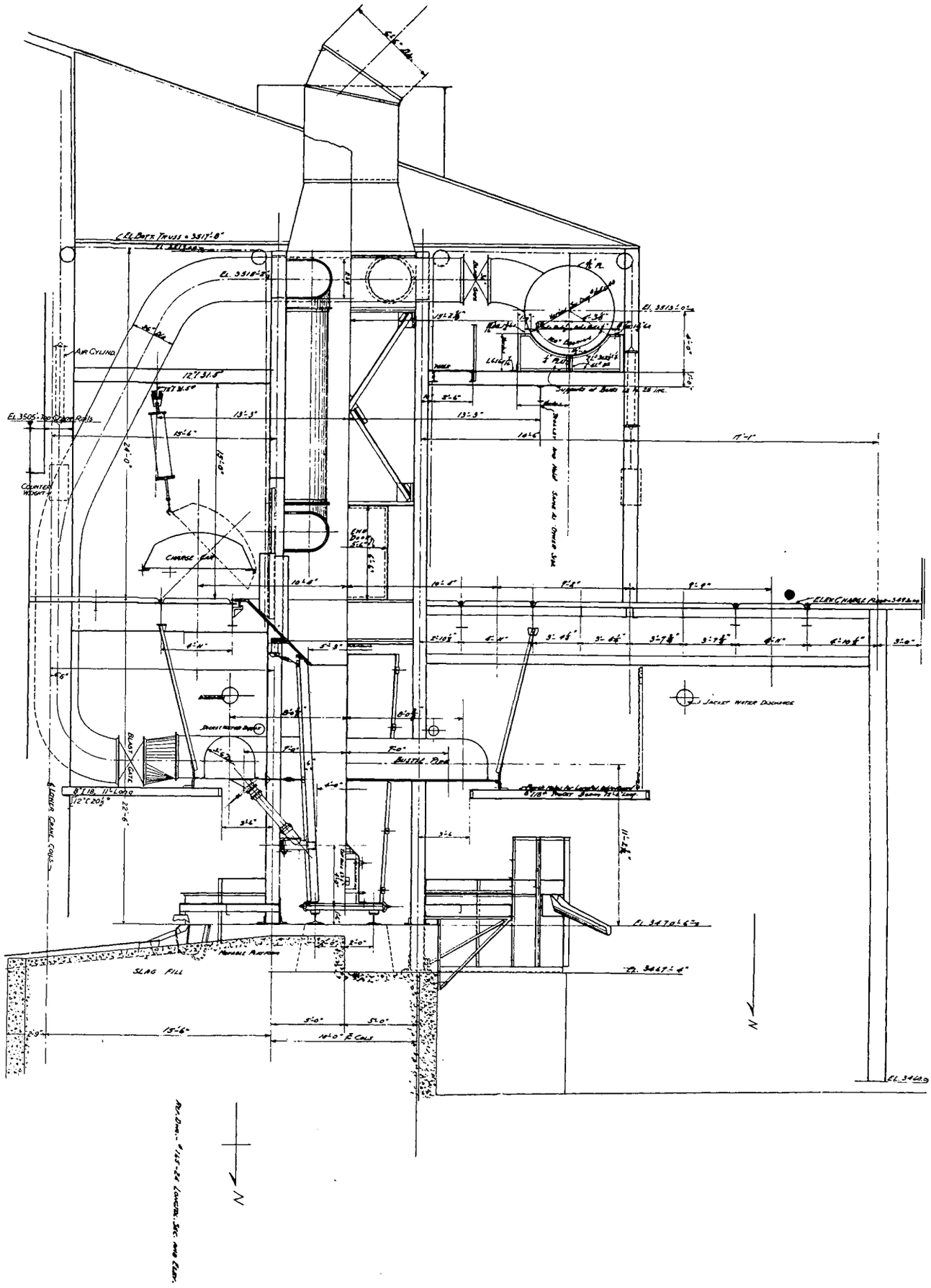


Figure 1. Blast Furnace—Vertical Cross-section and Elevation

United Verde Copper Company

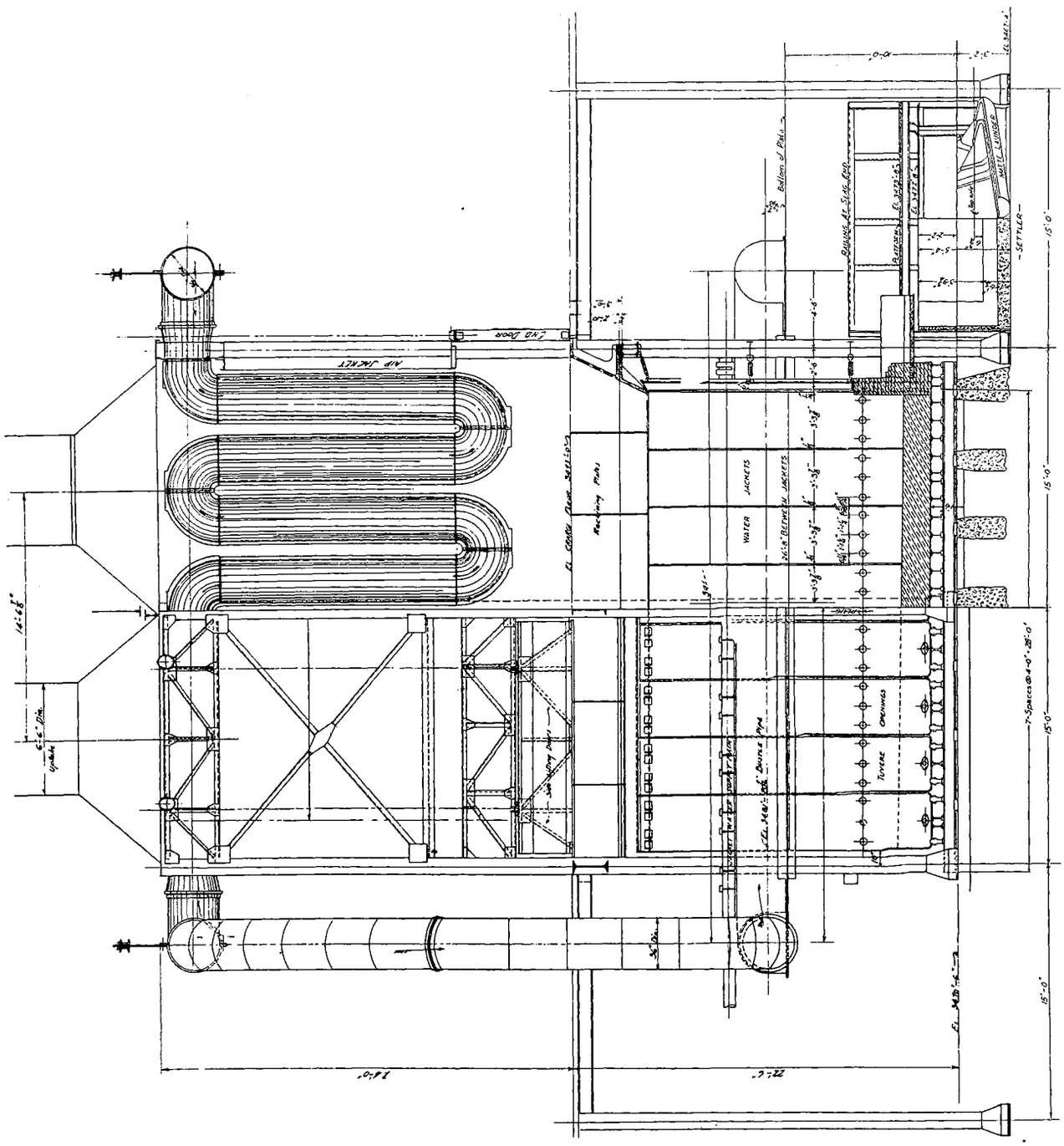


Figure 2. Blast Furnace—Longitudinal Section and Elevation

United Verde Copper Company

There is only one settler per furnace. Each settler is elliptical in shape with its long axis at right angles to the furnace axis and to the long axis of the converter aisle. This arrangement permits matte to be tapped from one end of the settler into a ladle in the converter aisle and permits the slag to overflow from the opposite end into the bowls of the slag cars. Slag cars are standard gauge, carrying bowls of 225 cu. ft. capacity; the branch track to each settler being divided "Y" shape so that two cars may be spotted, one under each of the two slag overflow spouts. The matte is received in 160 cu. ft., cast steel matte ladles from either of two tap holes located at the opposite end of the settler. The settler is built on a concrete foundation and consists of a steel shell lined with magnesite brick. The settler makes its own roof from frozen slag during operation. Each settler is equipped with a super-imposed inclined spout projecting out into the converter aisle through which molten converter slag may be introduced or through which solid cupriferous clean-up material may be fed for melting in the settler at times when the matte and slag content is hotter than necessary.

The accessory charging equipment consists of smaller standard gauge flat bottom charge cars of the Anaconda type with a capacity of 7,800 pounds each. The cars are arranged for side dumping and are tilted by air operated hoists. A charge track serves each side of the furnace. A third standard gauge track on the feed floor is used for receiving coke in standard gauge 40-ton coke cars. The center line of this track is about 29 ft. from the center line of the furnaces and requires the use of large push buggies for the delivery of the coke from the box cars to the furnace.

The charge cars are operated in trains of eight, hauled by 10-ton electric trolley type locomotives.

The air supply comes from the general smelter power house where it is produced by Roots blowers; the maximum allowable pressure of the equipment being 40 ounces per square inch.

Theory and Principles

A glance at the following tabulation discloses at once that considerable improvement has been made through a period of years. This improvement resulted from the establishment of a number of policies each having bearing on the furnace performance.

PERCENT COKE AND FURNACE TONNAGES

Year	Percent coke	Tons per fce. day
1917	8.10	561
1918	9.05	560
1919	8.45	621
1920	6.21	786
1921	5.30	751
1922	4.77	871
1923	4.97	776
1924	4.81	811
1925	4.68	809
1926	4.77	845
1927	3.77	1,045
1928	4.38	927
1929	4.80	887

The problem is that of smelting a sulphide ore containing approximately 5 percent copper, 22 percent silica, 30 percent iron, and 30 percent sulphur, and of smelting cupriferous clean-up material of a lumpy nature originating at various points in the smelting plant and containing approximately 10 percent Cu 20 percent SiO_2, 44 percent Fe (mostly oxidized), and 6 percent S. The smelting ore and the clean-up material, locally called "seconds," are fluxed with quartz ore, available of variable chemical composition from the United Verde mine, and with limerock, also available from the quarry on the company property located between the mine and the smelter.

Principles which must be adhered to in order to attain good performance may be stated as follows:

(a) The smelting, or sulphide ore, must be properly sized by efficient crushing and screening so that the particles will pass through a 4-in. screen but not through a 1-in. screen. It is believed that such sizing will increase the smelting capacity of the furnace at least 1.5 tons per square foot of hearth area per day compared with the smelting of mine run ore.

(b) The quartz fluxing ore must be sized by crushing and screening so that the particles will pass through a 2-in. screen, but not through a ¼-in. screen.

(c) The limerock flux should be crushed so that the particles will pass through a 6-in. screen and, if the fines can be utilized by the roaster-reverberatory system, the particles minus ½-in. in size should be separated and be so diverted.

(d) The coke should be of good metallurgical quality free from fines and handled carefully from the railroad cars to the furnaces with a minimum number of operations to avoid breakage.

(e) If a bedding plant is not available as is the case at the United Verde the prepared ores and fluxes should be stored in suitable sized lots and the chemical composition of these lots determined from samples taken during the preparation of such lots.

(f) The furnace charges must be calculated so that the mixture of ores and fluxes will have a chemical composition to give the optimum result in smelting. The proper chemical composition may be forecast within limits by the use of general metallurgical knowledge, but is not finally determined except by close observation and tabulation of results based upon a study of a number of factors, some of which may be peculiar to the local problem.

(g) The coke should be fed into the furnace by itself, and not mixed with the charge, so as to give a distinct layer of coke in the descending column of stock in the furnace. Twenty to 25 pounds of coke per square foot of horizontal column area is satisfactory. This means that approximately 2,400 lbs. of coke is fed into the furnace at each time. The charging of this coke should precede the charging of the ore and fluxes and should be uniformly distributed over the charge column.

(h) The mixed ore and flux should be introduced into the furnace and uniformly distributed immediately after the charging of the coke. On top of the quantity of coke above described 30 tons of mixed ore and fluxes is introduced.

(i) The height of the charge column should be as high as possible consistent with the height of the furnace, available blast pressure, and the porosity of the charge column. The United Verde blast pressure is limited to 40 ounces. There is no difficulty in maintaining a charge column of a height, immediately after charging, of 15 ft. above the tuyeres which is about 2 ft. below the level of the feed floor.

(j) The blast pressure should be maintained as high as necessary to deliver into the furnace the maximum volume of air which can be efficiently utilized without the formation of blow holes. The United Verde practice is limited by the maximum available pressure of 40 ounces which amounts to 375 cu. ft. per square foot of area at the tuyere section per minute.

(k) The tuyeres should be kept open by punching and cleaning frequently.

(l) It is believed, but not proved, that the rapid draining of the accumulation of matte and slag on the hearth of the furnace is beneficial at least once or twice per shift. This operation is easily accomplished by and, in fact, is necessary with the United Verde dry type spout due to the burning out of the clay dam, which forms the lip of the spout.

(m) The development and observations of these principles has resulted in the economy disclosed by the tabulation given above.

Practice

In outlining the theory and principles of operation as stated above, the practice has been partially described, however, additional notes on practice may be of value.

Metallurgical considerations and observation of results have taught that a certain amount of limerock flux is essential to smooth operation. Previous experience of the writer indicated that the ratio of CaO to Al_2O_3 should be about 2.50. Probably due to the presence of MgO in the United Verde ore this ratio of CaO to Al_2O_3 has been found to have been satisfactory at 2.0. In round numbers the Al_2O_3 in the slag is carried at 6 percent and the CaO at 12 percent, while SiO_2 is carried at 33 to 35 percent. The ratio of 5½ parts of SiO_2 to one of Al_2O_3 is another criterion to be observed. This information, together with knowledge of the oxidizing capacity of the furnace and together with the knowledge disclosed by the mineralogical analyses of the ore, is sufficient to guide one in calculating the proper furnace charge. The mineralogical analysis is of importance as an aid to this calculation because it portrays the relationship between free quartz in the charge and the complex mineral locally called "schist" which is a hydrous-magnesian-ferrous-alumino-silicate.

The water supply for the cooling of the furnace jackets is a matter of importance if furnaces are expected to operate in long campaigns. The local practice is to use water in a closed circuit with a reservoir; the makeup to this reservoir being soft water with a hardness to two to three grains per gallon. The makeup water is softened by the lime soda process. The discharge of the jackets has a temperature of about 140° F. With this quality of circulating water and also because the jackets are well constructed, the details of which can not be given here, it is not difficult to obtain continuous campaigns of over a year in duration.

The air blast, as indicated above under "Design and Construction," is preheated. The rise in temperature above atmospheric temperature averages about 70° F., which is not very great because the preheater design is not adequate for good efficiency in the transfer of heat from the furnace gases to the blast. However, it is believed that this 70° of

	Percent	Cu	SiO_2	Al_2O_3	Fe	FeO	S	CaO
Smelting ore	68.0	5.1	22.1	3.5	29.2	37.6	30.4	0.9
Quartz ore	6.0	1.6	57.9	5.5	13.7	20.4	7.1	0.8
Limerock	11.4	..	4.8	1.1	2.4	....	....	50.4
Seconds	14.6	9.8	19.5	2.2	44.4	....	6.0	....
Charge	100.0	5.0	21.9	3.2	27.4	....	21.9	6.6
MATTE	.....	21.7	....	...	44.5	....	23.8	0.6
SLAG	.....	.26	33.6	5.9	32.7	....	1.3	11.6

preheating makes a worth while reduction in the amount of fuel which would otherwise be used.

The labor force or crew for operating the department when only one blast furnace is blowing consists of the following for 24-hour period:

1 foreman and metallurgist at $6.60 base rate per day.
3 shift bosses who also act as feeders at $6.60 each.
6 coke wheelers at $3.47 each.
3 furnace men at $4.79 each.
3 helpers at $4.07 each.
3 helpers at $3.47 each.
3 charge weighers at $3.85 each.
3 charge motormen at $4.07 each.
3 charge switchmen at $3.47 each.
3 slag motormen at $4.07 each.

Extra labor per 24 hours averages about $5.

The compensation of all men on the force is augmented by earnings under the bonus system for efficiency. The bonus earnings are variable, but average about 20 percent of the base wages. During periods in which the selling price of copper is higher than 15 cents per pound, the base wages are increased according to the general schedule used in the industry.

Treatment and Production

In presenting the following data the year 1927 has been chosen as being most representative of the best one furnace operation. Only one furnace was operated during this year and only for a period of 133 days. Makeup of charge and analyses of matte and slag produced was as shown in the table above.

During the above period the tons of charge including limerock, but excluding coke, were 1,045 per furnace-day which amounted to 9.7 tons per square foot of hearth area (as measured by the horizontal area at tuyere elevation). Percent of coke used per ton of charge was 3.77.

The amount of flue dust recovered from the dust chamber common to the blast furnace and converter departments was approximately 35 tons per day and approximately 22 tons per day of the dust precipitated in the general Cottrell plant is estimated to have originated at the blast furnace.

Costs

Blast furnace smelting costs follow:

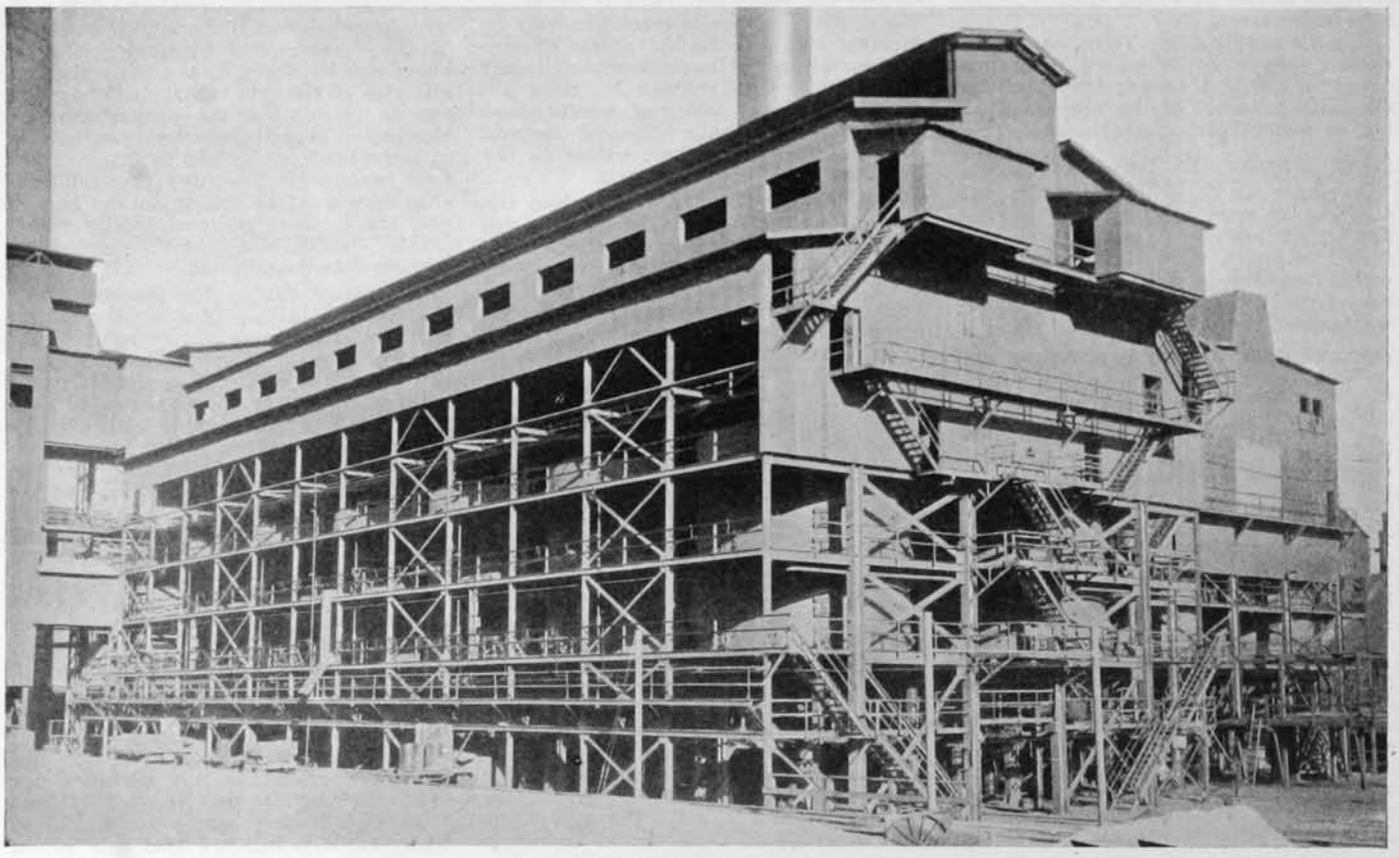
	Average per ton charge
Tramming and weighing charge	$0.075
Tramming slag	.045
Operating wages and bonus	.105
Shift bosses wages	.030
Sundry supplies	.005
Coke	.600
Limerock, flux	.090
Blast	.130
Blast	.130
Circulating water	.015
Light	.001
Mud mill	.006
Repairs, bldg., fce., flues	.066
Repairs, tools	.015
	$1.183

The "ton of charge" includes limerock which is the only barren charge ingredient and which makes up 13 percent of the charge.

Conclusions

The United Verde blast furnace practice is considered to be good and the cost of smelting to be low. However, such results can only be obtained in a plant which employs reverberatory furnaces for the treatment of fines and flue dust.

In the varying schedule of operations at the United Verde it is more economical to operate by the roaster-reverberatory system up to the point at which the scale of production would require more than three reverberatory furnaces. At this point the waste heat steam is just about sufficient to supply all power for mine and smelter. The additional waste heat from a fourth furnace could not all be utilized under present conditions and therefore its value could not be credited to the fourth reverberatory. To supply it with feed the roaster plant and Cottrell plant would be overloaded and as dusting at the roasters increases as the square of the tonnage per roaster day, the loss of values in dust would be appreciable. The periods of peak production which require a fourth or fifth furnace for the production of matte are not sufficiently high to justify further capital expenditures which would be required to enlarge the roaster and reverberatory departments. These factors together with consideration of the fact that a blast furnace uses coarse screened ore which would entail a cost of 18 cents per ton if crushed to minus ¼-in. size for roaster feed, lead us to the use of the blast furnace for the larger smelting program during periods of peak production.



General view of Roasters described on page 71

United Verde Copper Company

Reverberatory Smelting

By F. X. Mooney

GENERAL FOREMAN, REVERBERATORY DEPARTMENT

History

AT THE United Verde Copper Company's smelter at Jerome, all ores were treated in blast furnaces, which process left the treatment of fines, flue dust and converter slag unsolved. This and the growing popularity of matte smelting in reverberatories, with waste heat boilers, decided the management to add reverberatories when the new smelter was built at Clarkdale.

Three reverbs were built with the new Clarkdale plant, blown in May 26, 1915, and are 101 ft. x 19 ft., inside measurements.

Later when it was decided to add to this equipment, larger furnaces were decided upon. Three were erected between the year of 1918 and 1920, and are 101 ft. x 25 ft., inside measurements. The smaller furnaces are numbered 1, 2, and 3; the larger 4, 5, and 6.

The furnaces were blown in in the following order:

No. 1—July, 1915.	No. 4—June, 1920.
No. 2—Oct., 1916.	No. 5—Sept., 1920.
No. 3—Dec., 1916.	No. 6—Mar., 1928.

Reverbs 1, 2, and 3 were designed for center and side charging, but before 4, 5, and 6 were built it was decided to adopt side wall charging and at the present time all furnaces are so equipped.

Fuel oil was the only fuel used until the year of 1919, when the pulverized coal plant was completed. Since that date the furnaces have been equipped for either fuel, and since that time pulverized coal has been used almost exclusively, although the change to oil can be made without loss of smelting time. All accessory equipment in the reverberatory group has expanded in proportion.

Theory—Principles

Experience has demonstrated that certain conditions are necessary to obtain maximum smelting results; some of which are enumerated:

(a) Ores should be crushed to at least 3-mesh before roasting.

(b) Different grades of ore that go to make up the roaster charge should be mixed as intimately as possible and the mineralogical as well as the chemical composition of the charge must be balanced. How this is accomplished is described under the article on roasting.

(c) Calcine should be delivered from roasters to reverberatories as hot as possible. The average temperature is from 850° to 900° F.

(d) Combustion of fuel must be complete or very slightly on the CO side. Not more than .1 to .3 percent CO. This is controlled both by the primary air and draft. The practice is to use approximately 40 cu. ft. of primary air per pound of coal and maintain a draft of from .01 to .025 in. of draft at the uptake.

(e) A grade of matte must be maintained that will not require over-roasting at the roaster on the one side, and not throw too great a burden on the converters. The rule adopted at this plant is to make the percent Cu in matte produced five times the percent of copper in the roaster charge.

(f) The pulverized coal must be only as dry and fine as economy indicates and must be injected into the furnace at a constant rate and with sufficient turbulence. The above varies with different kinds of coal. If the sub-bituminous coal used here at present is dried much lower than 5.5 percent H_2O we have much trouble with fires in the mill bins. This type of coal dried to 5.5 percent H_2O flows satisfactorily in screw conveyors, feeders and burners. Satisfactory combustion is had when pulverized to the following:

On 100 mesh	15%
On 200 mesh	25%
Through 200 mesh	60%

Location, Design and Construction

The relative location of the departments allied to the reverbs is shown in *Figure 1* of the article on the Roaster Plant, appearing on page 70 of this issue.

Calcine, Cottrell dust and flue dust is hauled from hoppers under dust chambers and roasters in bottom discharge cars of 25 tons capacity over a curved trestle, double tracked, to the reverb charge floor. Haulage is by electric trolley type locomotives.

Matte is tapped into ladles of 160 cu. ft. capacity mounted on a truck, traveling in matte tunnels and delivered to crane in the converter aisle.

Slag disposal equipment consists of 225 cu. ft. side dumping slag bowls electrically operated, generally run in trains of three cars, by 25-ton electric locomotives.

Molten converter slag is delivered to reverbs, by converter cranes, through launders introduced in firing end wall of the furnaces.

Pulverized coal is delivered from coal plant by a system of screw conveyors and stored in steel bins of 60 tons capacity, each above the burner openings.

Fuel oil is piped to all furnaces ready for use whenever wanted.

A conspicuous feature of United Verde reverberatories is that there is no droop to the arch. In its long dimension it is level from the burner end to uptake.

The dimensions of the furnaces under operating conditions are given in *Figure 1*. *Figures 2, 3, 4* and *5* are exterior views of 25-ft. x 101-ft. furnace.

The furnaces are built on a slag fill, a foundation wall approximately 4 ft. high and 4 ft. thick of silica brick extends from the slag to the top line of the silica bottom, at which point the side walls are stepped back to 30-in. thickness and the end walls to 18 in., at which thickness they are carried up to the arch.

The arch is made of 20-in. silica brick.

The uptake is the full width of the furnace and opens to two separate flues connecting to individual waste heat boilers. A system of cross flues makes it possible to connect to two or three waste heat boilers; the path of gases being controlled by a system of water-cooled dampers.

There are two matte tap holes in each furnace located 25 and 31 ft. from skim end.

The tap plate consists of a square magnesite block 6 in. x 6 in. x 4 in. perforated with a 2-in. hole set in a copper block, which block is fitted into a large water-cooled plate 11 ft. x 3½ ft. x 4 in. thick.

The slag outlet is in the middle of the flue end of the furnace and is equipped with water-cooled iron castings.

The charging pipe hoppers and buckstaying is of the standard type, with some local adaptations.

The waste heat boiler plant consists of 12 Sterling type boilers rated at 713 hp. Two other boilers of the same type are equipped for direct oil firing.

Practice

Distribution of charge in furnace is very important. (See *Figure 6*.) Approximately 90 percent of the charge is smelted in the 35 ft. nearest the burners. Just enough charge is dropped at the burner end to keep it fettled.

Slag skimming is as nearly constant as the layout of slag tracks will permit, or from 60 to 70 percent of the time.

Matte tapping is regulated by a system of electric lamp signals from the converter floor. The converter foreman calls for matte as he needs it.

Combustion is regulated by gas tests and temperatures. A chemist is always at the furnaces on day shift to test the flow of coal and make gas tests.

The draft on furnaces is regulated by dampers between boilers and reverb flue.

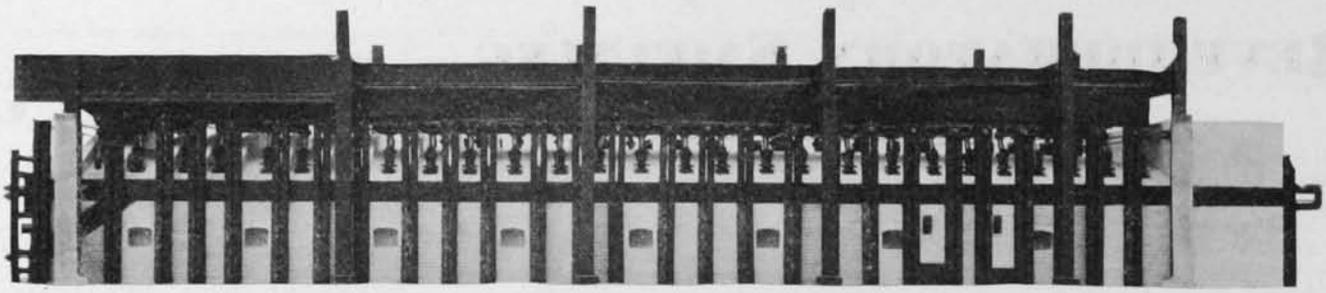


Figure 2. Side view of 25 ft. x 101 ft. Reverberatory Furnace

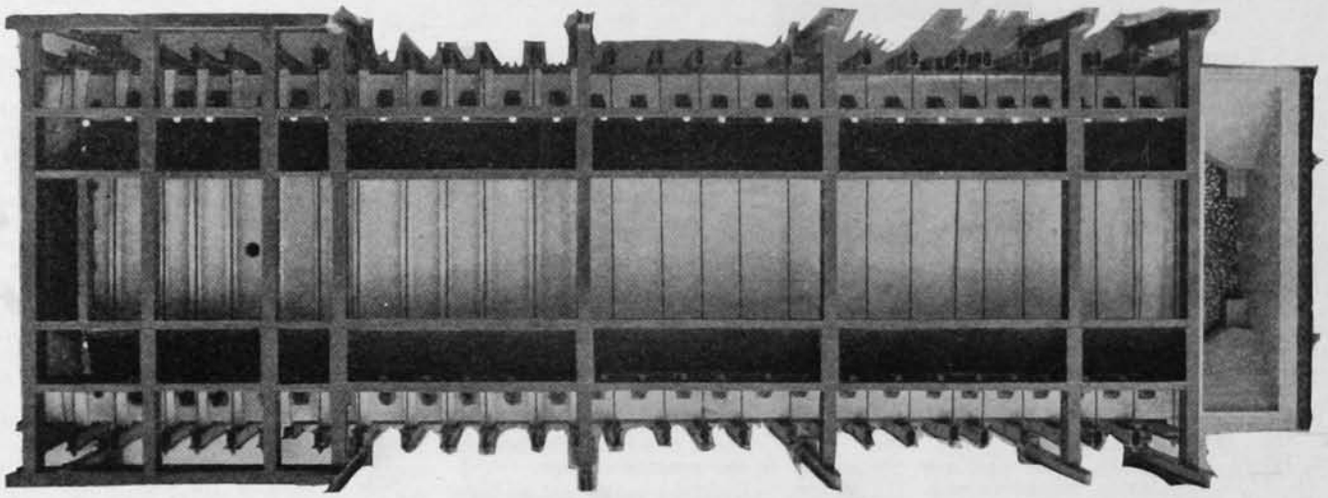


Figure 3. Top View

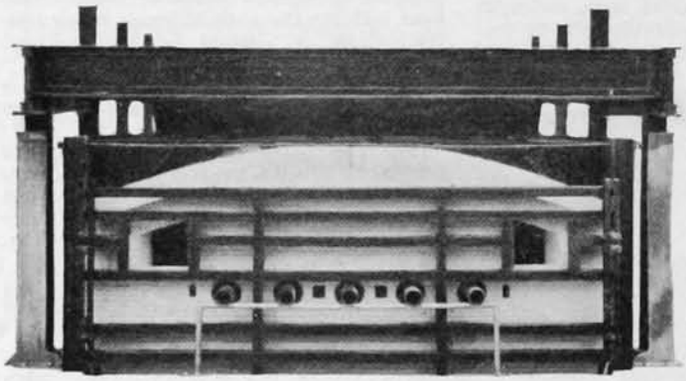


Figure 4. View of firing end

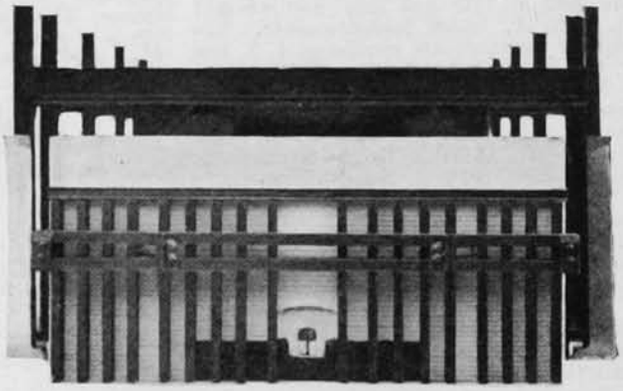


Figure 5. View of skimming end

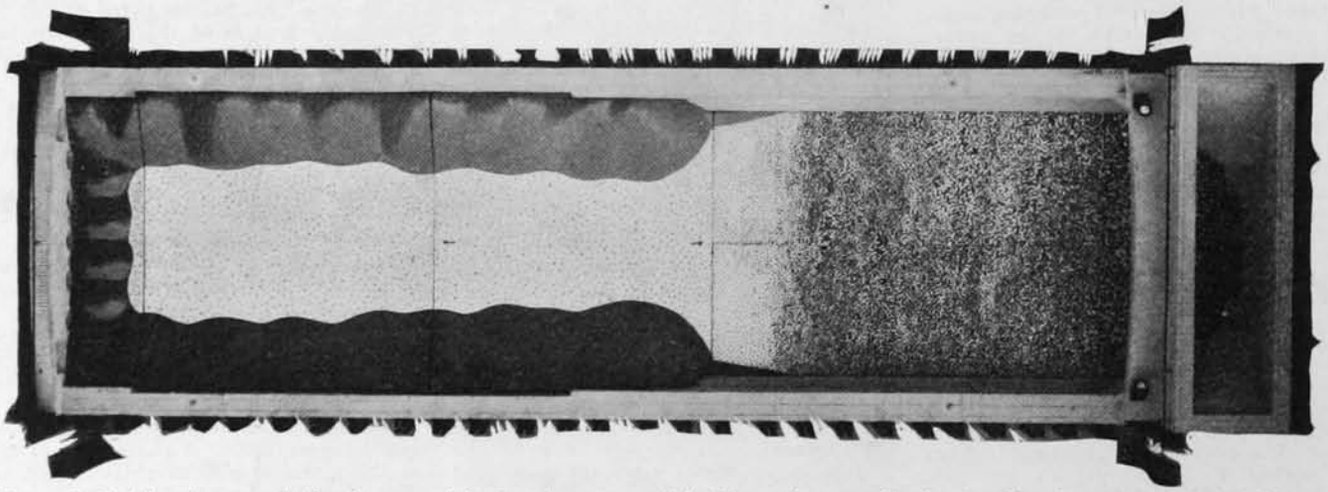


Figure 6. Inside view, vertically downward (taken from a model with roof removed) showing distribution of side-fed charge

United Verde Copper Company

Temperature of gases to boilers, 1,950° to 2,050° F.

Temperature of gases from boilers, 600° to 700° F.

Liquid converter slag is poured into the reverbs from 160 cu. ft. ladles by the converter cranes and distributed as evenly as possible to the furnaces operating.

Primary air to coal burners is from Turbo blowers housed in a separate building at the coal plant. The pressure at blowers is kept between 14 and 15 ounces.

Ordinary type coal burners are used, supplied by coal dust feeders with 4-in. variable speed screw conveyors.

During the furnace campaign, side wall and fettling holes are repaired. A high-grade clay fire brick is found to be preferable to silica brick for that purpose, since it does not spall when placed in the hot walls.

The length of furnace campaigns vary with the charge. At one time the large furnaces made campaigns between arch repairs of two years and more; the smaller furnaces campaigns of one year or more. At present the larger furnaces average about one year and the smaller type about nine months. There are two factors to account for the change, the increased quantity of fines on the charge, and the larger tonnages smelted per furnace day. The recent years have brought flotation concentrates into the furnace feed and smelting ore from the mine fire zones containing a greater amount of fines.

The operating crew, per shift, with three furnaces operating, is as follows:

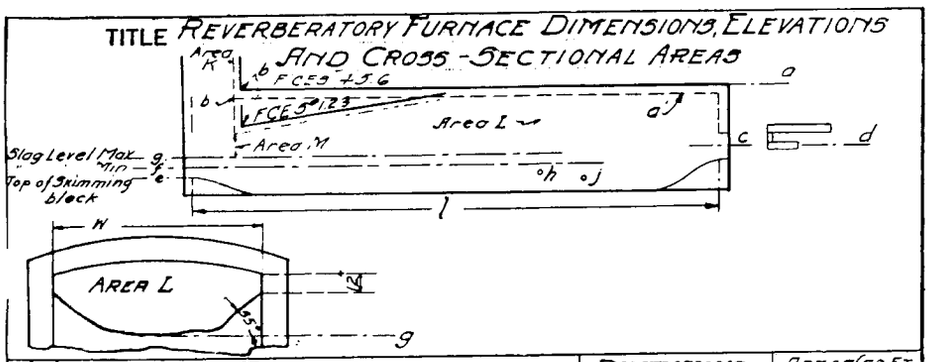
One shift boss, 3 calcine motormen, 3 calcine brakemen, 9 furnace tenders, 1 slag motorman, 1 slag brakeman, 2 cleaning and tending converter slag launders, 1 operating hoist in matte tunnels, 6 extra laborers on day shift, cleaning at furnaces, tracks and building, 4 men working afternoon and night shift cleaning connecting flues to waste heat boilers.

Treatment and Production

Materials smelted in reverberatories October, 1929, two large and one small furnace in operation, were:

	Tons	Percent of charge
Raw ore	105	0.1
Calcine	69,877	86.6
Blast furnace dust	844	1.1
Roaster flue dust	1,507	1.9
Cottrell treater dust	5,429	6.7
Converter flue dust	211	0.2
Dried concentrates	2,577	3.4
Total materials	80,550	100.0

	Total	Per ton solid charge
Tons wet coal as weighed at Gallup, N. Mex.	10,618	.132
Tons wet coal after deducting losses in transit, storage and processing.	10,193	.126
B. t. u. per pound wet coal		10,500
B. t. u. per ton solid charge smelted		2,650,000
Tons molten converter slag treated		17,000
Tons reverberatory matte to convertors		17,700
Tons reverberatory slag produced		84,000
Per furnace day:		
Charge to large furnaces		916
Charge to small furnaces		767



FCE No	Elevations a	b	c	d	e	f	g	h	j	a'	b'	Dimensions a'-g	b'-g	a'-c	f-h	Areas (sq. ft) L	M	K
1	85.29	83.11	78.38	79.06	76.09	76.76	77.26	74.86	74.69	83.62	81.44	6.36	4.18	4.64	1.90	96	35	68
2	85.39	83.57	78.34	78.94	75.99	76.68	77.16	75.04	74.68	83.78	81.90			4.78				68
3	85.69	83.83	79.06	79.06	76.14	76.81	77.31	74.91	74.74	84.02	81.16	6.71	3.75	4.96	1.90	99	33	68
4	85.28	83.96	78.61	78.61	75.41	76.08	76.58	74.73	74.39	83.61	83.69	7.03	7.11	5.00	1.35	138	100	175
5	85.55	83.54	78.50	78.45	75.30	75.97	76.47	74.51	74.11	83.88	83.87	7.41	7.43	5.32	1.46	142	102	175
6	85.17	83.00	78.51	78.61	75.23	75.90	76.40	74.53	74.14	83.50	83.33	7.10	6.73	4.99	1.37	136	94	175

Fuel Burning Capacity 24 hrs

FCE	L	W	Kind of Fuel	Wet Tons Coal	Bbls Oil	B.t.u./lb	B.t.u./Second
1	101'-0	19'-0	Oil or Pow. Coal	105	343	10500	25100
2	101'-0	19'-0	Oil or Pow. Coal	"	"	(COAL)	"
3	101'-0	19'-0	Oil or Pow. Coal	"	"	19000	"
4	101'-0	25'-0	Oil or Pow. Coal	135	437	(OIL)	32500
5	101'-0	25'-0	Oil or Pow. Coal	"	"		"
6	101'-0	25'-0	Oil or Pow. Coal	"	"		"

Figure 1. Furnace Dimensions

Costs

Tramming and weighing charge	$0.06
Tramming slag	.05
Operating wages and bonus	.11
Shift bosses' wages	.015
Sundry supplies	.04
Fuel—Gross $0.78	
Credit from waste heat .38	
Net	.40
16-oz. air	.015
Power	.003
Switching, crushing cleanings	.002
High-pressure air	.002
Light	.002
Repairs—Building, furnace, flues	.135
Repairs—Tools	.005
Limerock—Flux	.025
Mud mill	.012
Total direct charges	.876

Repair cost may be subdivided as follows:

Refractories	.072
Masons, helpers, laborers	.036
Shops, supplies and labor	.027
	.135

Analyses	Cu	SiO_2	Al_2O_3	Fe	FeO	CaO	S
Calcine	6.81	31.2	5.8	24.8	31.7	0.9	10.5
Reverberatory slag	0.41	37.2	6.7	37.1	47.9	1.7	....
Molten converter slag	1.82	22.4	...	....	....	...	....
Matte	27.2	....	...	....	....	...	....

Evaporation in waste-heat boilers:

Pounds of water evaporated per month	87,055,900
Pounds of coal consumed	21,236,000
Pounds of water per pound coal consumed	4.0994

Above is from feed water temperature or equivalent to 4.99 lbs. from and at 212° F.

BENEFICIATION OF MANGANESE ORE

Semi-commercial volatilization tests, having for their object the beneficiation of a manganese ore which could not be used in the iron and steel industry on account of an undesirable zinc and lead content, have recently been conducted at the Intermountain Experiment Station of the Bureau of Mines, in cooperation with the University of Utah, Salt Lake City. After 10 days of experimental work, during which time the plant was operated continuously, day and night, a manganese product was produced which met the requirements of the iron blast furnace operator in every way.

Beneficiation of this ore was brought about by the addition of diluents and by heating to a temperature just below the point of fusion of the mixture. The zinc and lead were driven off as fumes and were collected in bags as the oxides of the respective metals. The lead content of the ore was reduced from 3 percent to practically zero; the zinc was reduced from 2.7 percent to 0.2 percent. In this way, states the Bureau, the ore, which is refractory to all other metallurgical methods of treatment, can be converted into a useful saleable material and the undesirable constituents, which have been removed, are recovered as valuable by-products of the process.

General View of Converter Department

By J. J. Williams

GENERAL FOREMAN OF CONVERTING

CONVERTER DEPARTMENT

of the United Verde Copper Company

History

IN MAY, 1915, when the smelter at Clarkdale was ready to be blown in, the equipment in the converter aisle consisted of four converter stands and eight shells. This equipment, within two years, was found to be insufficient to handle the matte produced and one additional stand was built and three more shells purchased. After the enlarged production program of 1922 was under way, it was again found that additional equipment was necessary, and three more stands were built and three shells added to the equipment, making a total of 8 stands and 14 shells, which represents today's installation.

The first crane equipment in 1915, consisted of two Morgan cranes to which a third was added in 1916. These three cranes make up the present equipment.

Two casting machines, essentially the same two as now in use, were in the original design, but when No. 1 and No. 8 converter stands were built in 1924, the chain drags from the sump were reversed, so as to make room for the construction of the stands.

The length of the converter aisle in 1915, was 475 ft. An extension of 62½ ft. was built on the north end in 1917, at which time the crane repair bay and crane transfer hoist was built. In 1919, at the same time the three additional reverberatories were built, an extension of 165½ ft. was built on the south end. This makes the present converter aisle 700 ft. long. The original design called for the skull breaker in the same location as it is now found.

In 1915, an old Nordberg, a Southwark blowing engine, and a Ratteau-Smoot steam turbo-blower, which had been used at the old smelter in Jerome, were installed. As expansion took place in 1917, a new Nordberg blowing engine was installed. In 1924, a new Allis-Chalmers blowing engine was installed. In 1925, the old Southwark and the turbo-blower were dismantled and were replaced with a new Allis-Chalmers engine. From 1915 to 1918, the ladle equipment consisted of small 10-ton ladles taken from the smelter at Jerome. In 1918, new 160-cu. ft. ladles were bought and put into service.

Theory and Principles

The problem of the converter department is to blow into copper the matte produced by the blast furnace and reverberatory departments, prepare this copper for shipment and load it into the cars. This involves the problem of treating matte varying from 20 percent copper from the blast furnaces to 35 percent copper from the reverberatories.

The yearly average grade of the matte (handled by the converters) since 1922, has been very constant holding between 26 and 27 percent.

The problem is further complicated by the great variations of the SiO_2 and Al_2O_3 contents of the quartz ore used for flux. Much of this ore is not ideal converter fluxing ore but must be used on account of its values in gold and silver. Depending on the source of the quartz the silica content has varied from 45 to 75 percent. The air supply for converter operations has always been satisfactory. Due to the great amount of low grade matte handled, the tonnage of cleanings which the converters have been called upon to handle has been excessive.

Due to the great quantity of comparatively low grade matte, a very large volume of converter slag is produced which is poured back and retreated by the reverberatories. It has been found necessary, therefore, to keep the silica content of the converter slags as high as possible so as to avoid throwing too much magnetite and too great a burden on the reverberatory department.

Location, Design and Construction

The location of the converter aisle is shown in *Figure 1* accompanying the "Roaster" article on page 70 of this issue.

The converters, eight in number, are of standard Great Falls type, 12 ft. in diameter with 16 2-in. tuyeres of the standard Allis-Chalmers type. These stands are motor driven, each being equipped with a 50-hp. motor, with reversible control, driving through worm and bull gear. Accompanying this article is a photograph of the converters.

The converter aisle is 700 ft. long and 52 ft. wide. The crane rails are 45 ft. from floor and the crane bridge span is 52 ft. The bales used for handling the matte and slag ladles are of laminated construction locally made. There are 12

ladles of cast steel, 160 cu. ft. capacity, used for transporting matte and converter slag. Under each converter stand is located a boat made of boiler plate 9 ft. square, 3 ft. high, to catch any spillage or slag discharged from the converter. The department is also supplied with 12 2½-ton "cleanings" boats for handling cold slag and copper chips back into the converters. There are also two 10-ft. dust boats for handling hood cleanings from behind the converters. Three cast steel collar pullers of the Taylor type are used to keep the converter throats clean.

The McGregor skull breaker, located in the south quarter of the aisle, constructed by the Allis-Chalmers Company when the smelter was first built, is of pile driver type, and has given very satisfactory service during the 14 years of operation. The slag broken by the skull breaker through grating with 12-in. apertures, falls into standard gauge railroad cars, and is hauled back to the crushing plant.

The gases from the converters escape through a portable hood into a common smoke box which discharges into a balloon flue leading to the blast furnace dust chamber and from there to the Cottrell smoke treater. This balloon flue is cleaned with a drag chain, motor driven, delivering the accumulated dust into a convenient place for rehandling.

Air is supplied to the stands through a common blast pipe, 36 in. in diameter, from which 10-in. laterals lead to each stand. The air is controlled by a slide valve. In order to prevent as far as possible the entrance of excess air into the smoke box system, automatic dampers are installed above each converter. These are so designed that they automatically close when the air is shut off from the converter and open as soon as the control valve is opened. Above the smoke box are located the bins for the quartz flux. The flux is transported from the storage bins by motor and 20-ton cars to a 100-ton storage bin, and from there to the flux bins above the converters by an elevator and conveyor system and from these charged into the converter when desired, through a 14-in. pipe leading from the bin, over the portable hood, into the mouth of the converter.

As mentioned in the history of the converter department the converters are served by two Morgan cranes with a third kept in reserve in case of a breakdown on either operating unit. The cranes are 40-ton, with two 40-ton hoists. The main hoist is threaded with 460 ft. of ⅞-in. 6-37 blue center Roebling steel extra flexible cable. The auxiliary hoist is equipped with 520 ft. of ¾-in. 6-37 blue center Roebling steel cable. Although the term "mainhoist" and "auxiliary hoist" is used, it must be understood that each hoist has the same power and is used for any work interchangeably.

Plastic clay is mixed in a mill of the Chilean type at the north end of the aisle. The mud used in the different departments is transported to a convenient platform by the cranes.

Practice

The practice at the United Verde Copper Company has little to distinguish it from usual converter operations. The method used is to blow two matte taps successively nearly high with the required amount of silica to make about a 24 percent silica slag. As required, slag is skimmed from the converter, more flux added and the blow continued. Then the third tap of matte is added and blown with much less silica flux than used with the first two; the slag skimmed off when necessary, and the total metal blown high to the white metal stage. This operation is carried on in two or three converters at the same time and when all the metal is high and skimmed clean, it is transferred into one converter and blown to blister copper.

J. J. Williams

During the whole of this operation the converter is constantly punched and the tuyeres kept clean. Up to the point in the operation before the transfers are made into one shell for the finishing blow, cold cleanings from the floor containing copper values are added, thus utilizing the excess heat from the chemical reactions for the smelting of high grade cleanings, and also preserving the magnesite brick of the linings by depositing thereon a thin layer of magnetite. The finished copper is then poured into the copper ladle, transferred to the casting machine and poured into bars. These are cooled, cleaned and loaded into standard railroad cars, and are ready for shipment to the refinery.

The crew necessary to operate a shift in the converter aisle at times of normal production, is made up of a shift boss, three to four skimmers, six to eight punchers, three cranemen, four crane attendants and about 10 laborers.

Treatment and Production

An analysis of the matte from smelting units is as follows:

REVERBERATORY MATTE

	Percent
Cu	28.0
Fe	38.2
S	24.8

BLAST FURNACE MATTE

	Percent
Cu	20.0
Fe	47.0
S	26.0

An average analysis of converter slag is:

	Percent
Cu	2.0
SiO_2	23.0
Al_2O_3	2.3
Fe	52.0
FeO	66.7
S	2.0

Average converter flux:

	Percent
Cu	3.0
SiO_2	58.0
Al_2O_3	5.9
Fe	13.3
S	12.0

Average analysis of cleaning:

	Percent
Cu	8.0
SiO_2	17.0
Al_2O_3	2.0
Fe	41.3
S	6.6

The bullion produced during 1929 has averaged 12 million pounds per month. The average analysis of which shows:

	Percent
Cu	99.30
Ag	31.1
Au	0.90

To produce this copper the converter handled about 23,000 tons of reverberatory matte and used approximately 9,000 tons of converter flux monthly. This produced about 18,000 tons of converter slag monthly, which was returned to and retreated by the reverberatories. The air required monthly for the converter operation averages 1,740 million cu. ft., showing a consumption of about 290,000 cu. ft. per ton of bullion produced. In comparison, this air consumption with other plants using large converters, these figures compare very favorably. In fact we fail to find any other plant which exceeds ours in efficiency of utilization of air.

Costs

For converting a ton of copper:

Crane operation	$0.8388
Converter operation	5.5655
Lining converters	1.7499
Handling product	.8949
Unclassified	.5952
Total	$9.6446

Conclusions

Although the cost of producing a ton of bullion in the converter department of the United Verde Copper Company may seem to be high in comparison with other smelters, it must be remembered that this operation handles a great quantity of comparatively low grade matte in small converter vessels. Due to the tonnage handled during high production periods, little time is allowed the skimmer to handle his converters carefully enough to get the longest life and cleanest results from his operation. The amount of slag and matte transported by the cranes in connection with the other work demanded of them, cause a considerable delay in the continuity of the converter operation. In conjunction with these facts, the very erratic silica content of the flux available for the converters, and the fact that the Converter Department in times of forced production is the "neck of the bottle," causes the operation to be more expensive and less efficient than could be desired. During periods of lower production there is no doubt that the cost per ton of bullion will compare very favorably with any other smelter handling the same grade of matte.

The Bureau of Mines has published recently an information circular (No. 6246) entitled "Data on Metal Mine Ventilation in 1929," by D. Harrington. It discusses conditions brought to light during the year concerning factors in metal mine ventilation; deals with occurrence of gases, effect of blasting on air, fires—their causes and the methods of preventing and handling them, air conditioning, health as effected by ventilation and up-to-date methods of forwarding and controlling air flow.

United Verde Copper Company

RESEARCH

at the United Verde

By Oliver C. Ralston

DIRECTOR OF RESEARCH

THE problems demanding research at the United Verde can be best understood if the ore is first described. The ore deposit is a huge crescent shaped pipe of varying cross-section inclined at a high angle. Most of the main crescent consists of microcrystalline massive pyritic ore carrying copper and zinc minerals, mainly chalcopyrite and marmatite. A portion of the ore is disseminated in the schist which is the main rock replaced by ore. Even in the massive pyritic ore the schistosity of the original rock is usually evident.

Flotation concentration has in the past been practiced mainly on the disseminated schist and the main bulk of copper production has been from the more massive ore. In portions of the deposit are to be found sufficient silicious ore to act as smelter flux for the heavy iron ore. In earlier days the mine was regarded as one "long" on iron and "short" on silica for properly balanced smelting but this is not true at present. Continually increasing rejection of pyrite by differential flotation may swing the balance the other way.

The reasons why the massive pyritic ore has not been previously concentrated by flotation are as follows: The grinding problem is terrific. On an average the ore must be ground to pass 800 mesh to obtain over 90 percent liberation of the copper and zinc minerals from pyrite. As 300 mesh is about the commercial limit, that portion of the ore which fails to pass 800 mesh will contain much middling and concentration will be a compromise, at best. Another reason for failure to adapt the flotation process has been the peculiar frothing characteristics of the ore. To the unpracticed eye nothing is noticeable in a laboratory size machine but in the large scale the ore "over-froths," with most ordinary flotation reagents.

The research problems at the United Verde may now be stated:

(1) To develop flotation reagents better adapted to the frothing characteristics of the ore.

(2) To study the grinding problem and develop flotation treatment of the massive copper ore.

(3) To adapt flotation to the recovery of the zinc minerals, either in a separate product or in a mixed product from which subsequent metallurgical processes could recover the zinc.

(4) To either mitigate or abate the nuisance created by the large amount of sulphur smoke released during smelting.

(5) To consider means of recovering a part or all of the sulphur and iron in the ore, now wasted in large quantities.

The frothing problem solved itself during investigation of a wide series of frothing and collecting reagents. The usual condition being one of over-frothing, the operating and research staff were always watching for the reverse condition of underfrothing or over-flocculation. Several reagents were discovered which caused over-flocculation and then by proper proportioning of an over-flocculating reagent with an over-frothing reagent excellent control over the working characteristics of the flotation froth was obtained. While the two types of froth are known to operators their causes were obscure. Also mixtures of reagents were known but the reasons why mixtures worked better were not known. A patent has therefore been applied for covering control by this means. In the present mill circuit the sodium ethyl xanthate causes over-frothing while potassium amyl xanthate is used as the over-flocculator.

A word of explanation about over-flocculation may be needed. An over-flocculated froth forms from flotable minerals which rise to the surface in large feathery floccules. Minerals which repel water can do so more effectively if they gather together in floccules or "raspberries," the condition of flocculation being one where each particle is less effectively surrounded by water. If floccules are very large they interlock in a froth and draw together so tightly that each bubble acquires too heavy a load. Excess air merely goes through the over-burdened froth, water gradually drains from it and large patches of froth lose air until they sink and must be broken up again and rafted to the surface with new air. In the reverse condition, over-frothing or under-flocculation, the froth is voluminous and poorly loaded and works best in cells where it is allowed to accumulate for some time before discharge (providing space is available) in order to drain and drop more of the undesired mineral.

The grinding problem was attacked by the aid of the microscope, samples of different grinds being sized or elutriated, the sizes analyzed and also briquetted in suitable binders for preparation of polished sections. Whereas a polished section of solid ore often caused undue alarm about the probable grinding needed, a polished section of individual grains produced by grinding often showed that a commercial quantity of the desired minerals was freed, the remaining locked mineral if included in the concentrate did not contaminate it beyond an acceptable grade. It was very definitely established that flotation middling rejected during retreatment of concentrate contained a high percentage of locked grains and was therefore a true middling. This is useful knowledge. During the sizing work, elutriation products were prepared in which the sulphide minerals were present in 1,600 mesh size and were briquetted and studied separately. This is close to the limit of resolution by optical methods when using the visible spectrum.

Following this study more grinding equipment was placed in the flotation concentrator and increasing amounts of massive sulphide ore added to the disseminated schist which does not require the grinding demanded by the massive ore. At present, as high as 30 percent pyrite is found in the feed and the proportion can be increased if tonnage is cut sufficiently to produce the necessary grind. Expansion of the concentrator to take an increasing tonnage of pyritic ore is under way.

Separate flotation of the zinc minerals was a more difficult problem. Not until control of the froth was obtained could any differential separation of zinc and copper minerals be attained. However, the reagents which were satisfactory with the schistose ore proved to be well adapted to the massive zincy ores. None of the zinc ore is free of copper and none of the copper ore is free of zinc. A certain portion of the pyritic mass contains both high zinc and copper. Clean fresh sulphide ore from deep underground was found to separate easily. The copper was present as chalcopyrite which floated easily while the zinc was present as marmatite of about 10 percent FeS content and required activation with copper sulphate to encourage it to float.

Activation of the marmatite was delayed until after chalcopyrite could be floated off. A study of activation showed that the zinc sulphide particles became coated with copper sulphide by reaction with the soluble copper salts and the flotation reagents adapted to flotation of copper minerals were then efficient in flotation.

A special problem was encountered in the upper level ores where oxidation of the surface ore had caused solutions of copper sulphate to descend through the ore beneath, reacting and leaving chalcocite, covelite and bornite in small to large portions in the zinc ore. Differential separation of such ore is almost impossible. Addition of chalcocite to an

otherwise well behaved clean zinc-copper ore will activate the marmatite particles to almost the same flotability as the chalcopyrite, even exceeding it slightly. Ore which has been near a mine fire zone is likewise "self-activated" and permits of preparation only of a bulk copper-zinc concentrate. Fresh ore after standing a year subjected to atmospheric influences acquires increasing activation of the marmatite but the use of cyanides will restore it to proper condition. A study of the action of cyanides showed that the principal action was to clean the mineral surfaces by dissolving films. Not only could activated marmatite be cleaned by a cyanide solution and thereby de-activated, but the flotation of chalcopyrite was enhanced by this cleaning and also the silver and gold minerals could be bettered in their flotation characteristics. Excess cyanide is a detriment but if used only in amounts sufficient for dissolving slight films, as is the usual practice when using cyanide, it could be left in the pulp without difficulty. For badly filmed mineral particles larger amounts of cyanide are needed and the excess cyanide, or its reaction products, must be removed and fresh water substituted. In other words, it was found that a regular hydrometallurgical operation could be performed and thereby leave a clean ore ready for differential separation. This new understanding of the action of cyanide in differential flotation increases one's control of the operation.

For the past year a test mill rated at 500-lb. daily capacity has been in operation on ore from various portions of the mine, collecting data for construction. Usually it is operated at 200 lbs. per hour as means have been found for feeding reagents steadily in sufficiently small amounts to permit this.

An extension of the concentrator, the first zinc concentrating unit, is now under construction.

The sulphur problem of the smelter is well on the way toward solution. When smelting as high as 5,000 tons charge daily, a huge amount of sulphur dioxide is vented but with increasing diversion of pyritic ore to the flotation concentrator more and more pyrite is going into tailing and is not being smelted. With the copper ores about 70 percent rejection of pyrite into the mill tailing can be attained and with the zinc ores, 90 percent.

In addition, a certain amount of work has been done on conversion into useful products of part of the sulphur which must enter the smelter. Sulphuric acid can not be shipped to any available market. Likewise the present markets for liquid sulphur dioxide are too far distant to make such a venture attractive. Elemental sulphur seems to be the only product which could be shipped any distance to a market sufficiently large to assure sales. Most sulphur recovery processes call for large amounts of fuel. Unfortunately, at Clarkdale all forms of fuel are relatively expensive. Therefore, several processes have been under investigation which do not call for fuel and which use only intermediate products already available in the smelter. Technical success has been attained but the economic aspect of the problem is still far from being reassuring. The large mitigation of the sulphur nuisance through increased flotation concentration of the ore will probably prove to be sufficient solution of the problem.

While some study of iron recovery has been made, nearly all available resources have been expended on the problems discussed above. Yet, it is recognized that a large tonnage of iron is ultimately slagged which during the process of smelting can be brought to intermediate products of high iron concentration, if desired.

The conversion of zinc-bearing products to metallic zinc is being studied. Apparently electrolytic conversion of concentrated zinc products is the best metallurgy. Zinc concentrate of 45 percent Zn and 2 percent Cu can be prepared and it is possible that the grade will ultimately be higher than this. The copper concentrate prepared from the zinc ore contains about 15 percent Cu and 8 to 10 percent zinc. This must be smelted together with residue from the electrolytic zinc plant, and the molten products de-zinced. Several adaptations of the regular copper converter to the de-zincing of molten products have been studied. One period of operation in a 12-ft. Great Falls type converter has shown that by proper manipulation as much as 55 percent of the zinc in such a charge can be fumed off and recovered as zinc oxide without the use of any fuel beyond the burning of the iron and sulphur in the charge. Another smaller converter of special design with up to 10 tons capacity, has been used in study of de-zincing with various reducing agents blown through the charge. A combination of copper matte and slag can be more easily de-zinced than either alone. The ability to de-zinc a molten charge permits less care in concentration and less care in roasting zinc concentrate and leaching the soluble zinc. The de-zincing converter makes up for the shortcomings of previous operations and permits cost cutting in all of them.

The electrolytic conversion of zinc concentrate and zinc oxide fume into metal is soon to be studied in a 200-lb. per day pilot plant. It is already known that the principal difficult impurities to handle are antimony, tin, nickel, and germanium. There is also an unusual amount of copper in the unpurified solutions. New methods of purification are being studied.

The above outlines the main activities of the United Verde research department. Numerous other smaller matters occupy attention. Treatment of ores offered from other mines must be studied. Any unusual happening in the smelter is likely to call for work from the research department. Failure of parts of machinery demand metallographic examination. The department has participated in the development of rammed linings for basic converters, an innovation which is still in rapid state of change and improvement. Metallurgy is never so well done that there is not plenty of opportunity to devise means of improving it.

Housing of the research work is somewhat different than in the usual research department. A small brick building at Clarkdale accommodates the headquarters and most of the work is done in temporary shacks or in spare space of other buildings. Each job is so placed that adequate services and supplies are available. The flotation test mill is housed in space left for a spare filter in the flotation concentrate filter building. The de-zincing converter is placed under the foundry crane-way and is served by railway tracks coming direct from the reverberatory furnaces and also from the ore bins. The electrolytic zinc pilot plant is adjacent to the power house. By this organization an expensive permanent research laboratory building is not needed. The headquarters building contains main office, director's office, library and conference room, microscopic laboratory, a medium sized general laboratory with glass blowing bench and torches for small welding, cutting, brazing, lead burning or quartz blowing, and a hood with four hot plates for special analytical work or small scale experimental work. The south side of the headquarters building is a covered porch with cement floor which serves as an out-door laboratory for most of the year in the equable climate and also permits of using small oil or coal fired furnaces without creating the usual sooty or smokey condition that these furnaces cause in-doors. All analytical work except special determinations is referred to the assay office of the smelter. Most of the shop work is done by machine, boiler, blacksmith, carpenter, electric, paint or pattern shops or by the foundry, facilities being available for building almost anything needed. Occasional day labor is supplied by the smelter labor gang. With the exception of one clerk-stenographer and a part-time janitor, every one of the 15 employes of the research department is a technical man doing productive work.

APPLYING VARIOUS GEOPHYSICAL METHODS OF PROSPECTING TO SAME TERRITORY

The comparative advantages of applying several geophysical methods of prospecting to the same territory are discussed in Information Circular 6235, by F. W. Lee, senior physicist, recently published by the United States Bureau of Mines. The paper is written primarily for the mine operator who contemplates the employment of geophysical methods for determining the lateral extensions of known ore bodies and adjacent areas of mineralization. It represents the combined results of investigations conducted by the Canadian Geological Survey and the United States Bureau of Mines in the Sudbury District of Canada, the work being done on portions of the Falconbridge and Errington properties in this region.

The geophyslcal methods employed involved magnetometer, self-potential, and resisivity measurements. The advantages, as well as disadvantages, of the several methods applied to this territory with its diverse geological conditions are explained and discussed, and the results of these methods are illustrated by two concrete examples which show how certain geological factors may influence these results. The results of the investigation show the advisability of first making a preliminary geophysical survey, in order to test the particular value of each method in a given area.

Attention is called to certain criteria that may be used for the evaluation of geophysical data and which may indicate when the results are valuable, when doubtful, and when worthless. Were the geophysical prospecting companies to furnish the mine operator with more data in their geophysical reports than mere maps and charts of supposed mineralization, this would do much to invoke mutual confidence, it is pointed out.

United Verde Copper Company

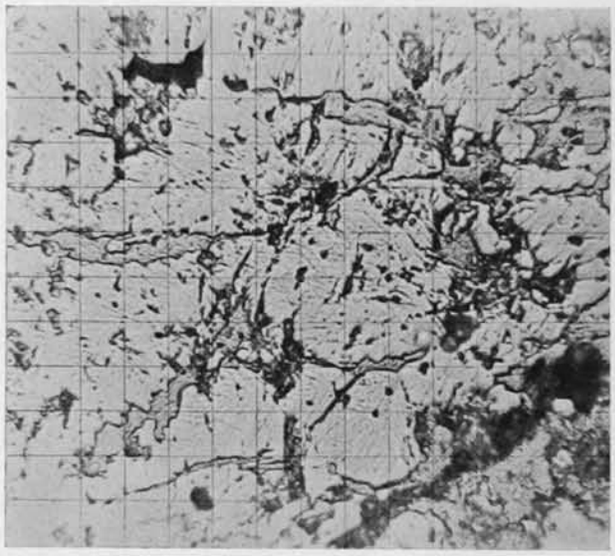


Figure 1. Fine veins of chalcopyrite (gray), in a groundmass of pyrite (white). Squares are 0.019 mm.=800 mesh

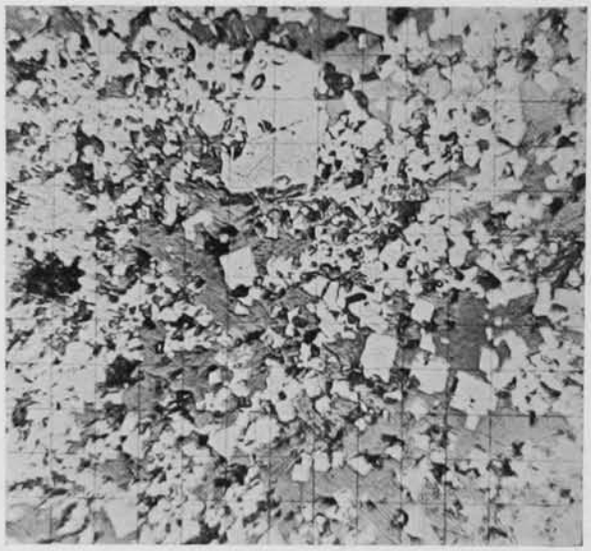


Figure 2. Crystals of pyrite (white), in a groundmass of marmatite (gray). Squares are 0.019 mm.=800 mesh

Ore-Dressing Microscopy

at United Verde

By Morris Slavin
MICROSCOPIST

WHEN the flotation research work of United Verde copper-zinc ores was being planned, prior to the organization of the research department, it was anticipated that difficulties would be encountered because of the extremely fine texture of these ores. With the naked eye it was often impossible even to differentiate between minerals. It was thought that the reflecting microscope might be of help, and in order not to overlook any tool that would aid in the work a well-equipped microscopic laboratory was included as part of the flotation equipment.

Although the reflecting microscope had been suggested innumerable times in the technical press as an instrument that could be applied with profit in the solution of ore-dressing problems, its actual use in this field appears to have been limited, judging from the very few papers that have appeared in the literature. R. E. Head stands almost alone in this field, and his experience was adopted as our starting point. We started, therefore, with a practically clean slate, working out or adapting methods and correlating structures under the microscope in terms of flotation causes and effects.

The microscopic laboratory consists of three small rooms in the research headquarters building. The main room contains the optical bench and photomicrographic apparatus, files for specimens, closets for accessories, and a large workbench. The polishing room contains the polishing machine, a sink, and the apparatus for imbedding crushed mineral grains in bakelite. The third room is a photographic darkroom.

The equipment was chosen with the help of Mr. R. E. Head, of the Salt Lake City station of the Bureau of Mines. The microscope is a Leitz metallurgical type MO with a set of apochromatic objectives ranging in focal length from 32 mm. to 2 mm. oil immersion. The apochromatic objectives have been found to be very satisfactory for, although more expensive than the usual achromatic type, with their high corrections for color it has been possible to determine without difficulty the colors of very small mineral areas, a prime necessity in the identification of minerals. A low-power Leitz binocular microscope for examining rough mineral surfaces and performing other work ordinarily done with less convenience with a hand lens, is included. An optical bench and camera, together with various accessories, complete the optical equipment.

The polishing machine, made by the Warner & Swazey Company for general metallographic work, has four horizontal laps driven by a countershaft through friction drives and with provision for changing the speeds. For our purposes it was necessary to increase the speed range by replacing the driven pulley with a smaller one, and to substitute a plain cast-steel disk for the carborundun wheel accompanying the machine.

The United Verde ores, with the exception of the schist ore now being concentrated, are all of the heavy sulfide type. They are classified for testing purposes as high zinc or high copper ores, although none are entirely free of either of these two metals. The copper occurs most frequently as chalcopyrite in the form of narrow, ragged-edged veins disseminated through a massive pyrite groundmass. *Figure 1* shows a typical structure. The zinc occurs as the ferruginous variety of sphalerite known as marmatite, whose composition, determined by means of a quantitative estimation with the microscope, corresponds to the formula 8 Zn S. FeS. Our marmatite has, in general, the same structure as the chalcopyrite, although it occurs also in larger masses, which, however, invariably contain scattered crystals of pyrite in the form of tiny cubes (*Figure 2*). The texture, therefore, is just as fine as in the case of the chalcopyrite.

The first work consisted in polishing and examining a large number of hand specimens taken from various parts of the mine. From this we were able to learn the mode of occurance of the copper and zinc. the minerals present in the

ore,* and a rough estimate of the degree of intergrowth, or texture.

Polishing and examining individual hand specimens is a long and tedious process. Besides, this method of examination suffers from two serious defects; first, a very large number of specimens must be viewed in order to give a fair sample of the ore; and, second, it gives only partial indication of the degree of grinding necessary to make a satisfactory flotation feed. Consideration of the second objection immediately leads to the conclusion that the only way to determine how readily an ore liberates in grinding is to grind it. If, then, we could grind carefully taken samples to various degrees of fineness and find some means of examining the crushed grains to determine the degree of liberation of the various minerals, the second objection would be overcome directly, and the first, relating to sampling, indirectly, for viewing several thousand grains taken from a carefully mixed sample is both easier and the results are more representative than viewing a score or a hundred hand specimens taken at random.

The problem of viewing the tiny crushed grains was solved by mounting them in a bakelite matrix or binder † and polishing the whole mass. The mineral grains in the polished surface appear in section, the areas being on the average a quantitative volumetric representation of the various minerals present. An obvious objection is that while the grains are examined in section the flotation process is based on surface phenomena, and it is the surface, therefore, that is of interest. This is true, but the condition of the surfaces can be inferred from the appearance of the section; besides, it is impossible, at least with our ores, to differentiate between minerals by observation on the surfaces of any but the very largest grains of a grind suitable for flotation, whereas in the case of briquetted grains, particles of the order of one micron diameter can be polished and the color of the section surface, and therefore the minerals, distinguished.

A more serious objection to this method of examining crushed grains in a briquette is the impossibility of knowing the composition of the portions of the grains which were ground off to form the section and the portions which lie below the polished surface. A grain, therefore, which in section appears to consist of but one mineral may in reality be made up of several minerals, although a grain which appears locked—that is, composed of more than one mineral—can be nothing but what it appears to be. Due allowance in estimating liberation is made on account of this imperfection in the method.

The briquetting method has been extended to coarse grains (minus 6-mesh, plus 35-mesh) also. This size is imbedded in a hard brown wax confined in brass rings of about 1⅛ in. inside diameter and polished in the usual way. The edge of the ring is bevelled to prevent catching in the polishing cloths. In our case these grains are large enough to exhibit all the original structures of the ore that the hand specimens show, and the work of mounting and polishing is trifling compared to polishing a like number of hand specimens.

The procedure for the preliminary examination of an ore can be outlined as follows: The sample for flotation testing is delivered from the mine in large carbide cans. It is reduced in a jaw crusher and rolls to —6-mesh (the size for flotation testing) and riffled down to about 1,500 grams. The main bulk of the sample goes to the flotation laboratory, the 1,500-gram portion is riffled once more, one-half going to the assay office and the other half is used for the microscopic examination. This last portion is sized by screening from +35 to —300 mesh.

The +35-mesh fraction is mounted in brown wax in a brass ring, as described above, polished and examined. In this way the general characteristics of the ore are learned and the minerals present determined. With this information the quantitative mineral composition can be calculated from the assays.

The remaining sizes are individually briquetted in bakelite and examined for liberation of minerals. Viewing a series of sizes in this manner, in the order of decreasing size, the first few may show no appreciable unlocking, then one is encountered which shows signs that at this size some liberation has taken place, and the remaining sizes show an increasing proportion of liberation, until, presumably, a size is reached which contains no locked mineral, although with United Verde ores this size has never been reached, as it is far beyond the finest available sieve and our supplementary sizing methods do not reach to it. The size at which the ore begins to liberate is taken as the size to which to grind. This estimate is checked and changed, if necessary, when the flotation test products are examined.

Microscopic examination proceeds parallel with flotation testing. Products of the more promising tests are mounted in bakelite and examined, with the chief attention given to the state of liberation of the minerals in the various products.

Our principal problem is the rejection of pyrite without entailing too great a loss of copper and zinc. As the intergrowth of the chalcopyrite and marmatite is mainly with the pyritic groundmass, a large proportion of chalcopyrite-pyrite and marmatite-pyrite middlings form on grinding, and this requires a delicate balance in grading up of the concentrates in order that the concentrate shall be sufficiently high grade without too great a loss in the tailing. Pyrite can be rejected without difficulty to any degree by the use of lime, but too great a rejection results in dropping of the middling grains of which chalcopyrite or marmatite is a part. This balance of concentrate grade against tailing loss can be checked very neatly by means of the microscope.

This study of locked grains and their effect on flotation has led us to set up certain criteria in terms of middlings. A good concentrate is one which contains no free gangue grains (principally pyrite, in our case). As middling grains are wanted in the concentrate, their presence there is not considered detrimental. A good tailing is defined as one containing no free grains of the mineral being concentrated, and only a small proportion of the middling grains. Intermediate products are rated on the same basis, with due allowance made as to whether their position is toward the concentrate and of the flotation series or the tailing end. The intermediate products toward the tailing end (second concentrate, cleaner tail) should contain less free mineral than those close to the final concentrate (recleaner tail).

It has been possible, by examining tailings in this way, to distinguish flotation tests that are poor. This is evidenced by the appearance of free valuable mineral in the tailing. If the tailing contains no such free mineral and its assay value in copper and zinc is still too high, this is an indication that further grinding is necessary. A similar examination of the concentrate immediately reveals whether further cleaning steps could raise the grade without sacrifice in recovery. In this way series of tests can be checked and the action of various changes of reagents and of flotation conditions determined.

To summarize, then, the work of the microscopic laboratory—

(1) The general structure of the ore is determined.

(2) The quantitative mineral composition is calculated from the assays.

(3) A preliminary estimate of the fineness of grind is made.

(4) The products of flotation tests are examined for:

A. General efficiency of the test.
B. Suitability of grind.
C. Possibility of grading up to the concentrate by further cleaning.
D. Direction which succeeding tests should take.

The microscope has been of help in solving other, more special problems, which lack of space precludes from describing here, but the main bulk of the work has been as outlined above. Its great use has been in abolishing a large part of the uncertainties as to what actually takes place in a flotation circuit, enabling the operator to devote his thought to the remaining uncertainties of this puzzling process.

The microscopic laboratory is also provided with a petrographic microscope for study of transparent minerals and slags. All manner of service is given, including metallographic examination of alloys, but are not discussed under the subject of the present paper.

* The method of determining the opaque minerals is described, with tables, in Davy and Farnham "Microscopic Examination of the Ore Minerals," McGraw-Hill (1920).

† The method and apparatus are described in a paper by R. E. Head and Morris Slavin, "A New Development in the Preparation of Briquetted Mineral Grains." (Soon to appear in the Eng. and M. J.)

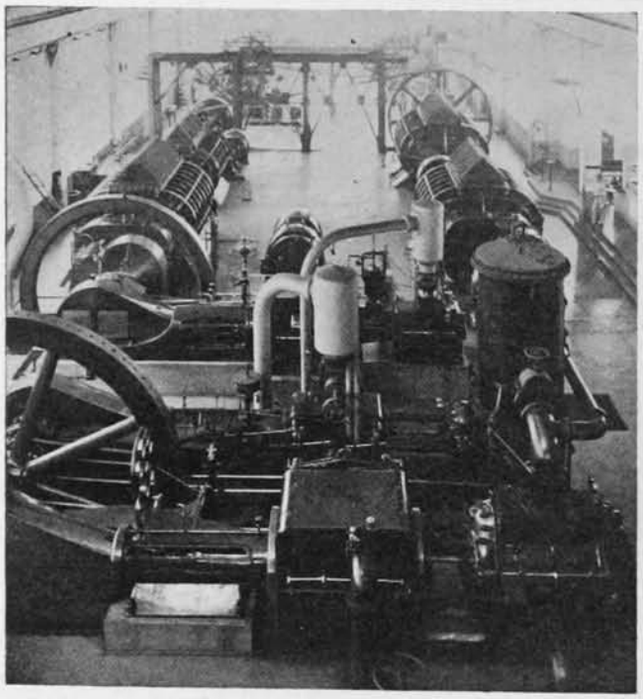
REPORT ON CAESIUM, RUBIDIUM AND LITHIUM

A report on the characteristics, occurrence and uses of "Caesium, Rubidium and Lithium" has been issued by the Bureau of Mines as Information Circular 6215, by R. M. Santmyers, mineral specialist. The minerals from which caesium, rubidium and lithium may be extracted occur in only a few regions in the United States, principally in California, South Dakota, New Mexico and Maine, and although distributed rather widely in nature, the known minerals and deposits rich enough to furnish large supplies are few. Most of the minerals contain at least a trace of all three elements.

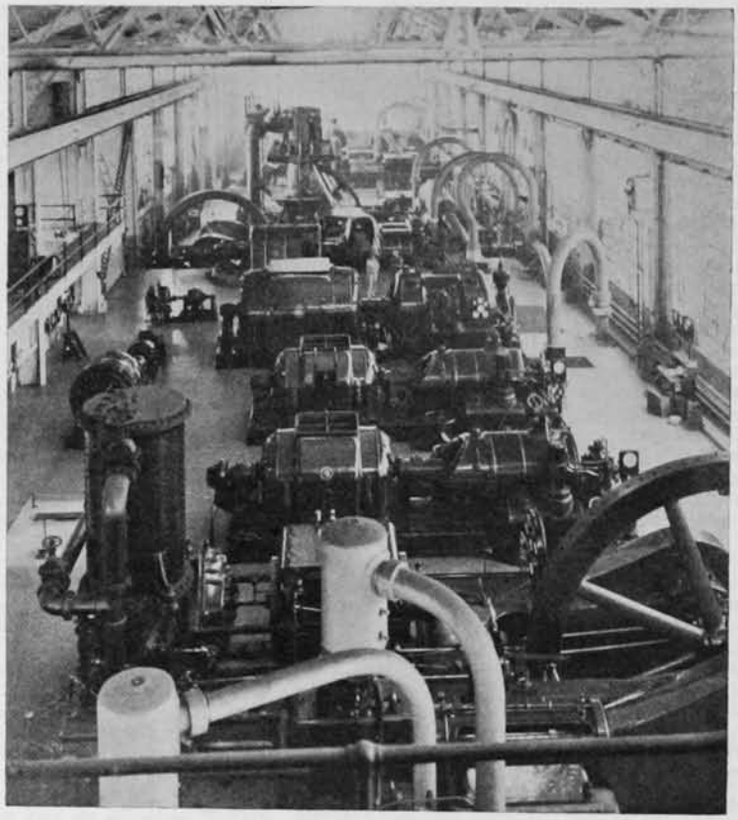
CADMIUM RESEARCH

Because of the increasing importance of cadmium, research on the recovery of cadmium from zinc mineral concentrates is being conducted at the Intermountain Experiment Station of the Bureau of Mines, in cooperation with the University of Utah.

United Verde Copper Company



Interior Views of the Power House, looking North, on the left; and on the right, South



United Verde POWER HOUSE and EQUIPMENT

By Frank Avis

CHIEF ENGINEER OF POWER HOUSE

POWER generating equipment of the United Verde Copper Company has gone through various stages of development as the plant has grown, the first installation in 1915 consisting of six waste heat boilers taking waste heat from three reverberatories then installed, and two Parker direct fired boilers supplying steam for various blowing engines and turbo generators, most of which were moved to Clarkdale from the old plant at Jerome.

The power plant produces air for blast furnaces and converters, and electric power for use at the smelter and mine, as well as a small amount of steam for heating the various offices and shops in the plant, and the major portion of the steam is developed from waste gases from the reverberatory furnaces. As these furnaces operate at a uniform rate of 24 hours per day, while the power demands vary from time to time, most of the demand ordinarily coming in the day shift, the result is that without proper co-ordination it would be necessary to waste considerable amounts of steam at night and either burn fuel under the direct fired boilers or purchase power from outside sources during the period of high demand. Accordingly, one of the problems of the Operating Department is to equalize as much as possible the demand on the power plant to flatten out the demand curve. It is, of course, impossible to provide a uniform load coinciding with the steam production and a connection is provided with the power lines of the Arizona Power Company and close contact is established with their operating department in order that our excess demands may be provided for and to enable us to supply excess power to them when it is available and required.

The boiler room equipment at present consists of fourteen 713-hp. Stirling boilers, twelve of which are waste heat boilers and two direct oil fired. These boilers are operated at 185 pounds pressure at a total temperature of 525 degrees, and the waste heat boilers on the larger furnaces operate at approximately 100 percent rating, while those on the smaller furnaces operate at around 75 percent. The boilers are equipped with Diamond Soot Blowers for removing accumulated ash and dust deposited by the furnace gases. Feed water is treated at a lime-soda ash softening plant, and is heated by exhaust steam from feed water pumps and steam oil pumps. When the blast furnaces are operating the blast furnace jacket water is used as feed water, increasing the feed water temperature from an average of about 110 degrees to about 130.

The boiler house is a steel frame-corrugated building, built as a part of the reverberatory building and immediately adjacent to the furnaces. Gas temperatures from the furnace range from 1,800 to 2,000 degrees, and the gases leave the boilers at from six to seven hundred degrees, and discharge through the 30 x 400-ft. steel stack. The boiler pump house is a steel frame building with brick walls and houses the various feed and circulating water pumps, feed water heaters and meters, and steam driven oil pumps.

There are four turbine driven feed pumps installed at various stages in the

development of the plant, only one of which is ordinarily operated, and two plunger type feed pumps used for stand-by service. The feed water lines to the boilers are in duplicate and with the two feed water meters cross connected to either line, enable us to keep separate records of direct fired or waste heat boiler feed as occasion demands.

There are three motor driven circulating pumps delivering water to blast furnace and reverberatory jackets and one steam driven pump for this service that is operated when there is danger of interruption to the electrical service. There is a complete loop of steam lines from the boilers to the pump house to insure continuity of service. The 10-in. line from the boilers to the power house is also arranged in a complete loop, cross connected to each boiler, fully insulated and trapped to insure the delivery of dry steam to the power house units.

The power house equipment is shown in the accompanying illustrations.

The electric generating units in the power house consists of two 7,500 kva. turbo generators, each with its surface condenser and accessories and one 2,000 and one 2,500 turbo generator operating on barometric condenser system.

For converter air there are four steam driven compressors having various capacities from 18,000 to 23,500 cu. ft. of air per minute at 16 lbs. Blast furnace air is provided for by Roots rotary blowers of various capacities from 225 to 400 cu. ft. per revolution and with the exception of two, these machines are all motor driven. One is direct connected to a tandem compound Corliss steam engine, and another is driven by rope drive from a duplex 3-cylinder Diesel. This unit was originally installed at Jerome and is now held as a stand-by unit.

High pressure air is provided by one 2,500 cu. ft. synchronous motor driven compressor and one steam driven cross compound Corliss two-stage compressor.

There is a total of 800 kws. of direct current generating capacity consisting of two, 200 and one, 400 a.c. motor driven unit supplying direct current power to the converter cranes and the electric trolley industrial railway system. Various small steam or electric driven exciters together with condenser pumps and accessories complete the power house equipment. All reciprocating engines operate condensing on the barometric system, and are served by a common gravity oil system with centrifugal purifiers.

The power house building is a steel frame, brick buidling with composition roof, 450 ft. by 60 ft. in plan, with an extension to the east wall 20 ft. wide by 150 ft. long, for switchboards, house transformers, office and locker room.

Approximately 75 percent of the outside walls are steel sash, a large proportion of which is ventilated. The basement is of sufficient height to accommodate condenser circulating pumps, piping and accessories and the main engine bay is served by a 20-ton crane.

The operator is stationed on the switchboard balcony about 10 ft. above the main engine room floor and is in close contact with all departments of the smelter by a signal system, indicating automatically the number of converters blowing and other important information as to load demand. He is also in contact with the distributing station at the mine and with the Arizona Power Company by telephone, so that production of power can be closely coordinated with demand.

Of the total amount of steam used in the power house in 1929, amounting to something over 1,000,000,000 pounds of steam, 71 percent was converted into electric energy at the switchboard, the steam driven units using 3 percent for high pressure air, 19 percent for converter air, and 7 percent for blast furnace air. Electric power amounted to over 53,000,000 kw.h. of which 62 percent was used at the smelter, the balance going to the mine and outside consumers. Direct steam cost, not including taxes, depreciation or administration expense, was 1.37 cents per boiler horsepower, charging the boilers their share of fuel and crediting to reverberatory furnaces. Electric power, on the same basis, cost approximately 8 mils per kw. h. at the switchboard.

The safety record of the power department is excellent, and every effort is made to keep it so and improve the general efficiency by close cooperation with all departments of the smelter.

MICHIGAN COLLEGE OF MINING AND TECHNOLOGY TO CONDUCT UNDERGROUND RADIO EXPERIMENTS

The Electrical Engineering Department of the Michigan College of Mining and Technology has received a permit from the Federal Radio Commission to install short-wave radio equipment, which will be used in experimental work and for purposes of instruction. It will be housed in the Metallurgy Building until the completion of the new Mechanical-Engineering Building, which will be erected this year and which will be largely devoted to electrical engineering instruction. Application will now be made for an amateur license, which will permit the college to employ short waves in its radio work. An important series of experiments will be conducted underground to determine the degree of absorption of radio waves by various rocks and rock structures. An unusual opportunity is provided for research of this character in the great depth of the copper mines of the district. Communication underground by means of radio short waves will be given a practical test in this way.

Another view of 500 Level Surface Plant

United Verde Copper Company

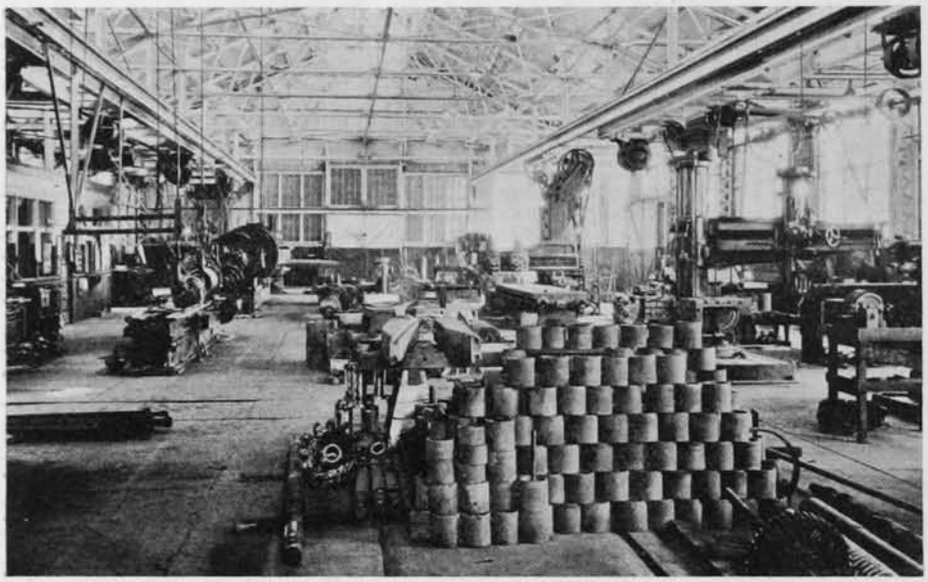


Figure 1. Interior of Machine Shop

By George Mieyr

SMELTER MASTER MECHANIC

THE SMELTER SHOPS

of United Verde

THE Clarkdale works is equipped with a full complement of shops for performing a great variety of services. Due to the remote location of the property with respect to available supplies the shops and their equipment are probably more extensive than is found in the usual industrial works of similar size, but of more favorable location. The remoteness of location has also put the company in an unfavorable position with respect to the sale of scrap metal of all kinds which accumulates in large quantities over a period of years, and this accumulation of scrap has been a factor in the expansion of the shop equipment, because it has resulted in the establishment of a well-equipped foundry for melting and recasting of the large supply of scrap metal.

The shops not only perform repair service but also fabricate and manufacture a great many kinds of mechanical supplies. It is the policy of the company, however, to fabricate and manufacture only for its own consumption and only such supplies which can be economically produced. Due to the large amount of scrap available, the availability of waste heat electric power and the saving in the freight expense, a great many articles can be made at a profit.

The shops consist of a foundry and pattern shop, machine shop, plate and structural shop, and blacksmith, carpenter, pipe, and electric shops. The electric service is described in an article by Mr. A. I. Greenwood, which will be found elsewhere in this issue. Each of the shops is housed in a separate building and the location and general design of

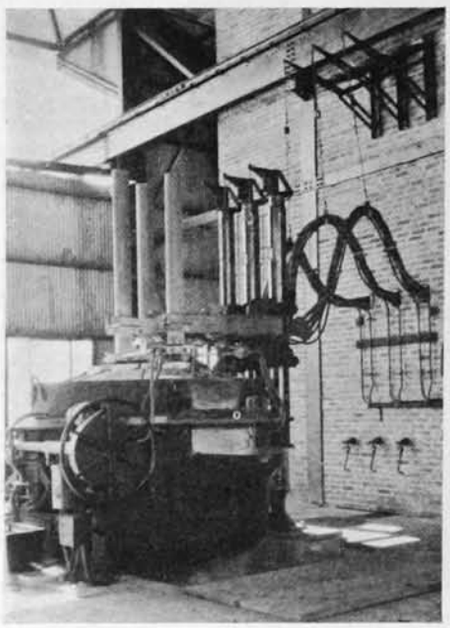
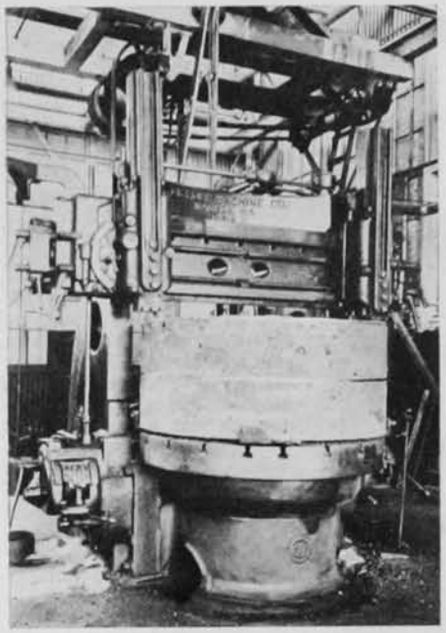


Figure 2. Electric Steel Furnace

Figure 3. Pouring Roll Shell

Figure 4. Cutting Risers off Roll Shell Casting



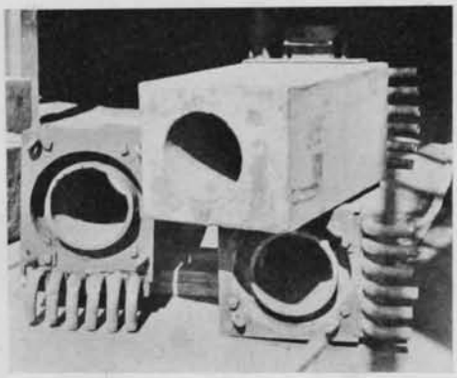


Figure 6. Water-cooled Copper Castings Used for Furnace Drop Holes

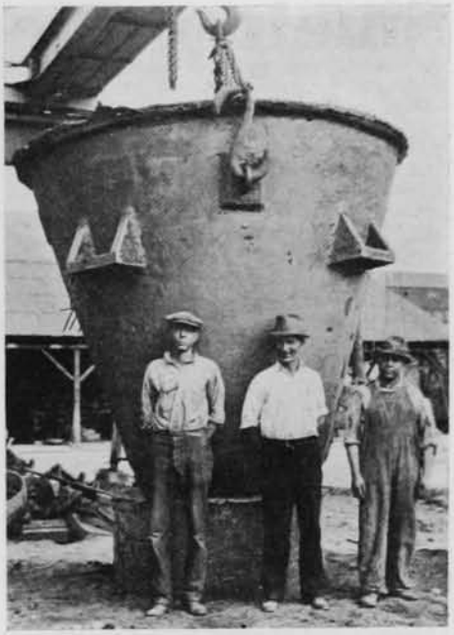


Figure 5. Machining Roll Shell

Figure 7. Cast Iron Slag Pot, 17,000 lbs.

these buildings are described in an article by Mr. J. E. Lanning, chief engineer, also found elsewhere in this issue.

The following is a concise statement of the equipment of the various shops; space does not permit detailed description, and such description would be superfluous because most of the equipment is standard and well known:

MACHINE SHOP

Shop equipment (see *Figure 1*) consists of one 60-in. Gisholt boring mill, one 72-in. Bennet lathe, five smaller lathes ranging from 14 in. to 25 in., two crank shapers, one 4-ft. x 4-ft. x 18-ft. Detrick and Harvey planer, three drill presses, one 13-in. x 16-in. high-speed saw, two milling machines, one universal grinder, one 400-ton hydraulic press, one bolt machine, one Brinell tester, and one 10-ton crane to handle material to the various machines.

The work ranges from the machining of converter frames (13,000 lbs.), roll shells (7,600 lbs.), down to the smallest of repair parts.

PLATE AND STRUCTURAL SHOP

Shop equipment consists of three pairs of shears, two punches, one set of 12-in. rolls, one hydraulic flanging press, one drill press, four portable electric welding machines, one stationary welding machine, and a general assortment of miscellaneous small tools.

The introduction of electric and Oxy-Acetylene welding and cutting torches has revolutionized operations in the boiler shop. It practically eliminates the use of pneumatic drills and hammers. The shop is at present engaged in the fabrication of an extension for the mill building, which will be an entirely welded job.

FOUNDRY

The foundry equipment consists of one 1½-ton per hour Moore 'Lectromelt steel furnace, one 300-lb. Detroit brass furnace, one 42-in. cupola with a capacity of 12,000 lbs. per hour, one electric annealing furnace large enough to handle a 24-in. x 58-in. roll shell, two core ovens, one scrap steel cutting shears and the usual miscellaneous equipment of sand mixers, casting cleaners, emery wheels, sand blast, etc. Casting production at present is 225,000 lbs. per month.

BLACKSMITH SHOP

This shop is well equipped with tools, steam hammers, etc., but the work has been cut to a minimum by the use of cutting and welding torches. Heavy forgings are either substituted by steel castings or built up of laminated plate and welded. The present work of this shop consists mainly of dressing of shop and smelter tools.

CARPENTER AND PATTERN SHOPS

These shops are well equipped with band saws, rip saws, mortise machines, saw filing machines, scrapers, planers, and miscellaneous small tools necessary to handle the variety and volume of work which they are called on to do.

Figure 1 shows an interior view of the machine shop. *Figure 2* a view of the electric steel furnace. *Figures 3, 4,* and *5* are views of steps in the manufacture of crome nickel roll shells. This operation is of interest in that the casting which weighs 7,600 lbs. finished requires a charge of 11,700 lbs.

Figure 6 shows a large water-cooled copper casting used for furnace drop holes. *Figure 7* is a cast-iron slag pot weighing 17,000 lbs., also made locally.

Figure 8 shows a large hydraulic flanging press. This press has been installed for many years and, among other uses, it has served in the fabrication of all blast-furnace water jackets. It was built by the local shops under the direction of Mr. J. P. Harrington, boiler-shop foreman.

We believe that the Clarkdale shops have justified their existence in the large amount of service rendered to both the mine and smelter.

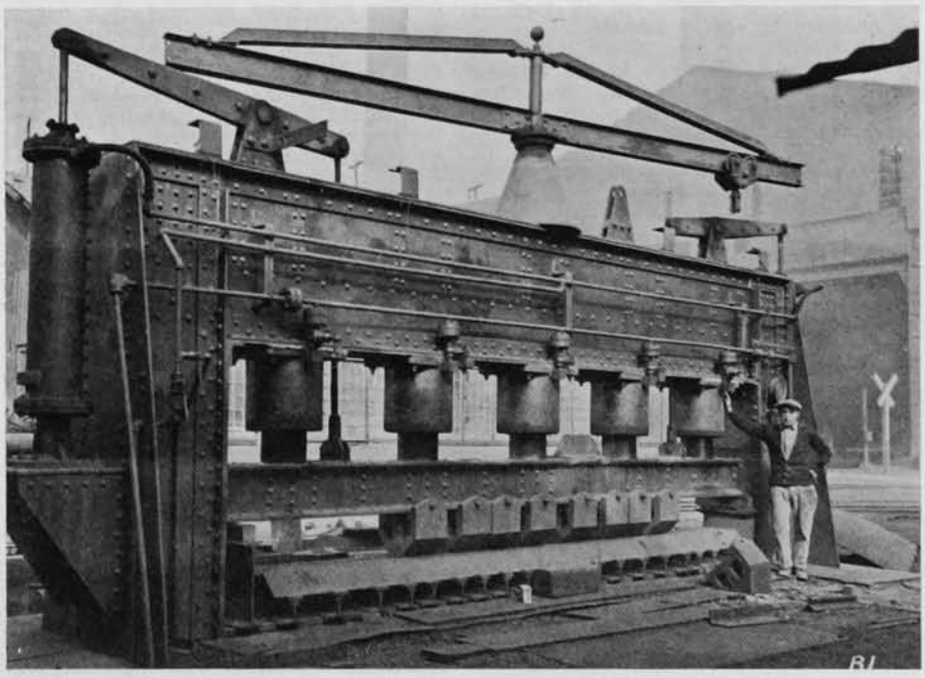


Figure 8. Hydraulic Flanging Press

United Verde Copper Company

Smelter Electrical Service

By **A. I. Greenwood**

CHIEF ELECTRICIAN

THE amount of electric equipment used in a large mine, mill, or smelter under present-day conditions is altogether different from what it was 25 years ago. Then the lighting features and a few motors constituted the electric equipment. Today electric equipment is conspicuous on every hand. The power for the Clarkdale plant of the United Verde Copper Company is furnished by a steam-driven generating plant, and also by the Arizona Power Company, the two systems operating in parallel the greater part of the time. Most of the steam is generated by waste heat from reverberatory furnaces, but two oil-fired boilers can also be cut in to furnish additional steam when needed.

The generating equipment consists of two 6,000-kw. and two 2,000-kw. turbo-generators. Power is supplied to the mine and also to the tie with the Arizona Power Company through a 6,000-kva. transformer bank stepping up from a generated voltage of 2,300 to a line voltage of 44,000. Distribution within the Clarkdale plant is at 2,300 volts, three phase, to four substations placed at various load centers in the plant. From the substations, distribution, at both 2,300 volts and 460 volts, is through underground conduits, which extend from manhole to manhole. In the plant are about 25 manholes.

All alternating-current motors of 50 hp. and over are operated at 2,300 volts; those under 50 hp. at 460 volts. In addition, many motors taking less than 1 hp. operate from the lighting circuits. The overhead cranes in the converter aisle, all haulage locomotives, and several shop cranes are operated on 250 volts, direct current. The total number of motors in the plant is about 500, and the connected horsepower about 17,500, exclusive of fractional-horsepower motors.

Slag haulage is effected by two 25-ton electric locomotives and one of 18 tons. Ore for the blast furnaces is hauled from the storage bins by two 10-ton electric locomotives. Hot calcines are hauled from the roasters to the reverberatory furnaces by two 10-ton locomotives. In addition one 18-ton and one 10-ton locomotive are held in reserve and for extra work. The traveling cranes for handling the ladles in the converter aisle are 40-ton units, each equipped with two hoists. Dust is removed from the smelter waste gases by a Cottrell installation, which uses rectified alternating current at a potential of from 50,000 to 70,000 volts. Most of the conveying of material is done by belt or screw conveyors; and, where elevation is necessary, by bucket elevators.

The electrical maintenance work as well as installation of electric equipment at this plant are in accordance with several objectives or ideals. The four most important are (1) safety, (2) continuity of service, (3) convenience, and (4) economy. Safety is a fundamental requisite of all industrial operations. Any plant with a bad accident record can not be considered successful, no matter what the apparent results in production. Safety should therefore be the first consideration, not production. The training of men for safety and the installation of safe equipment are continuous, for no matter how much has been accomplished in the past, additional refinements and improved equipment become available from time to time.

About seven years ago, when safety work was first started at this smelter, many electrical hazards existed, such as open knife switches, even 2,300-volt switches, and exposed wiring. At that time no safety knife switch could be found in the plant; that is, one which could be operated by a handle on the outside of an inclosed case. With the exception of power house and substation switchboards, there is now no open-knife switch in the plant except in a few lighting circuits, and these are all soon to be changed to safety switches.

All underground circuits are tagged in every manhole, the tags giving voltage and service of each circuit. Switches, circuit breakers, panels, starters, and machines are labeled in good, legible letters, usually in yellow paint so as to prevent accidents arising from a connection with the wrong circuit. A complete set of maps shows all circuits, with numbering to correspond to that in the manholes.

Accidents have occurred through improper use of manually controlled starting equipment. Many starters have been changed during the last few years from hand to automatic control, and practically all new equipment put in is automatic. All 2,300-volt manually operated, reduced-voltage staters have been equipped with a boiler-plate guard in front, with a rod attached to the operating handle, extending through the guard. Since these guards were installed, two starters have blown up, but in neither occurrence did injury result to a workman.

All distribution lines in the plant are in conduit, except the overhead trolley system, yard lights, and two short stretches of feeder lines. The policy now is to have all wires protected by conduit right up to the motor frame, using a steel or iron box in which to make connections to the motor. All 2,300-volt motors in the plant are now so protected, but many 460-volt motors still have exposed wires from the conduit outlet to the motor. All of the 460-volt motors will eventually be equipped with conduit boxes.

In the installation of new equipment, care is exercised to have all parts needing inspection or repair provided with plenty of room and conveniently arranged. One important requirement for safety in any plant is good lighting. During the first two years of our safety campaign the lighting of nearly all parts of the plant was reorganized. A foot-candle meter proved to be of considerable service in determining proper lighting in various sections of the plant. At present the lighting is good in all departments and complies with the major safety conditions.

For the safety of men in the electrical department, rubber gloves are provided for all "hot work." These are tested at regular intervals. An iron-clad rule insists that whenever hot work is to be done on any circuit over 110 volts, at least two journeymen electricians shall be on the job. "Square D" voltage testers are used for all low-voltage testing, test lamps being considered very dangerous. For testing 2,300-volt circuits, a tester made up of a potential transformer and lamp mounted in a box is used.

The men are given good material and good tools with which to work, and are urged to keep their tools in good repair. The safety idea is strengthened by having the men of the electrical department get together in a "safety meeting" at least once in two weeks. A safety bulletin board upon which new posters are placed every few days is maintained. The number of men employed in the elctrical department is about 25, and they are all proud of the fact that for two years and six months to date no lost-time accident has occurred.

Continuity of electrical service is of prime importance in an industrial plant, and especially in one where a flow of materials through the plant takes place at a steady rate. A few minutes' delay on one machine may cause a production loss for a whole department. To guard against costly delays owing to failure of electrical equipment, several methods are in use. One of the most important of these is duplicate service, which is maintained in regard to practically all important drives in the smelter plant. Beginning with the generating equipment, two large and two small generators are available. One large and one small machine can carry the total load. In the

power house are two generator buses and two distribution buses.

The transmission line to the mine consists of two circuits. An extra step-up transformer in the transmission line supplying the mine is so arranged that it can be connected in the place of any one of the three in service by the simple expedient of throwing switches. Two circuits feed the converter cranes. The converter tilting motors are similarly served, and each motor is equipped with a double-throw switch, so that it can be connected to either line in a moment. Duplicate lines also supply the crushing plant and concentrator. In many places, where considerable time would be lost in changing motors, two motors are permanently mounted in place, the spare motor being placed in service by simply putting on a belt and changing connections to the circuit. This arrangement is used for the motors driving the pulverized-coal burners supplying the reverberatories.

Four sets of power transformers in various substations, furnishing 460 volts, are arranged for interconnection on the low-tension side. In this way all 460-volt circuits can be kept in service, even though one of the transformer banks is out of commission.

Provision for necessary spare parts and a knowledge of where they are is also important when it comes to giving uninterrupted electrical service. For equipment that is used in various parts of the plant, spare parts are kept in the warehouse. For electric equipment that requires special parts, such parts are kept in a locker conveniently placed near the equipment on which they are to be used.

One factor that is important in preventing delays is a good system of records. We have a complete data file covering all electrical equipment, and we use one large manila envelope for each drive. In the envelope is complete information on the drive. One card has complete name-plate data of the motor and such necessary information as bearing sizes, and grade and size of brushes. On the card is indicated what spare parts should be kept in stock and where they are to be found. A similar card is made out for the control equipment. The envelope also contains complete drawings of the control equipment and records of any tests which may have been made. Such a system of records is of importance in keeping an adequate stock of spare parts and also gives the maintenance electrician complete information about the equipment. This is a necessity on automatic equipment when "shooting" trouble. Besides these records on the various drives, maps of the distribution systems are also provided.

Probably the most important factor in giving continuous service is through maintenance. We have found that the best method for accomplishing this objective is to give an electrician a certain part of the plant to inspect and maintain, and then to hold him responsible for keeping his equipment in good operating condition. All of these men are on bonus. Some are paid a flat bonus of 25 percent, with deductions for delays. Others participate in a departmental bonus, based on the cost of production for the month.

Another factor which helps in maintaining continuous service is that of proper equipment. If a particular type of equipment gives considerable trouble, and a newer and better type is on the market, to discard the obsolete equipment and install the new is often a profitable procedure. The most notable example of this fact in this plant is in the use of contactors equipped with thermal relays in place of fuses, or in place of oil switches, released by dash-pot type of overload relays.

Convenience of operation of electrical equipment is thought to be worth while. We are slowly appreciating the fact that happy, contented employes are a valuable asset to any establishment. A machine that is easily operated helps to keep the operator in such a state of mind, and also means an increased output. In line with this idea, all new equipment installed during the last three years has full automatic control. Instead of the operators being required to remember a certain sequence of operation of various switches, to press a button is all that is necessary. Much of the old equipment which has been changed over to give more reliable service also gives more convenient operation. Properly placed equipment is also important. In installing any new equipment the operator is always kept in mind and the equipment is arranged for his convenience.

Accessibility for inspection and repair should not be overlooked. An electrical installation that is crowded is not only inconvenient for the repair man but is a hazard that may cause a serious accident. Signals and interlocks add greatly to convenience of operation. These will be found almost everywhere, from one end of the plant to the other.

Economy of operation is, of course, a most important factor in the operation of any industry. It is the measuring stick which indicates whether the industry will continue to do business or whether it will eventually succumb. The foregoing phases of electrical work tend to lower costs. Safety always pays. Accidents always increase cost. To secure continuity of service increases the cost, but delays would involve a greater expense.

Probably the largest item of cost of the electrical department of any plant is repairs, and along with this is the cost of the delays which occur when a machine fails. To keep repair costs down to a minimum, each maintenance man is required to make out a daily report. In this way any equipment that is giving trouble can be quickly identified and the cause of the trouble investigated. Some causes of trouble which have been remedied are tight belts; bearings not properly protected against dust; overloaded equipment; and excessive dust. To overcome the last-named trouble, a system of forced ventilation is being used for some of the larger motors in dusty positions. Line transmission losses are an important item, and these have been reduced by the installation of synchronous motors, and in the transmission line to the mine such losses have been diminished by means of a synchronous condenser. Transformer-core losses are important, and they are kept to a minimum by proper loading.

About 230 watt-hour meters measure the power used throughout the plant. The flow sheet is arranged to show all line and transformer losses. If any of these losses are high at the end of the month, the cause can be investigated. The cost of power at the switchboard has been reduced by the installation of turbine generators with a water rate of 14 lbs. per kilowatt-hour as compared to 19 on the old turbines.

The cost of original installations is being reduced by installing motors which will start directly across the line wherever possible. This greatly cuts down the cost of starting equipment. The largest motor now started on full voltage is a 350-hp. self-starting synchronous motor driving a ball mill.

The use of new types of equipment in place of old is also important. A few illustrations are Rectox rectifiers in place of wet-cell batteries; new-type magnetic brakes instead of old; thermal-overload relays in place of the magnetic type; line starters in place of oil switches and fuses; and moulded rubber cords for extension lights and portable tools, instead of packing-house cord. Such automatic features as float switches, interlocking of conveyors, and automatic starters all reduce operating costs by making possible a reduction in the number of operators needed.

In investigating means of reducing costs, standardization should not be overlooked. Standardization in this plant has been carried out in motor brushes, resistance grids, lamp sizes and voltages, lighting fixtures, condulets and motor speeds.

The men who comprise the electrical department of the Clarkdale plant of the United Verde Copper Company feel that they have a large responsibility and are a vital factor in the organization. Their desire is to cooperate in every way with all other departments and to bring the electrical department to as high a standard as possible. This attitude in itself is likewise an important factor in cost control.

Electric Shovel Loading Dump Trucks

United Verde Copper Company

RELATIONS with EMPLOYES

By Robt. E. Tally

PRESIDENT

MUCH has been said and written during recent years upon the subject of industrial relations. These writings and discourses touch upon every conceivable phase of the question, from the simplest principles to the most involved theories regarding the psychology of the workman and the scientific application of the economics of management.

My own view of the matter has always been that no intricate conceptions need be evolved to make the relations between employer and employes satisfactory. It is simply a question of a real desire to understand the essential facts and possibilities, together with an earnest effort to apply them to the conditions at hand.

Cooperation between the employe and the employer to the fullest extent possible is, of course, the first requisite to satisfactory industrial relations. Confidence must be established. The employe must be satisfied of the good intentions of his employer, and the employer must be confident that he will receive the sincere support of his men if he gives them a square deal.

It was in this manner that the officials of the United Verde Copper Company years ago approached the proposition of improving its relations with its employes. Today I believe the company has as contented, as efficient, and as loyal a group of men as can be found anywhere.

The principles used are in themselves, of course, fundamental and ethically sound, and from this standpoint need no further comment. However, the method of applying them may be of interest, and the outlining of these methods forms the burden of this article.

To keep employes in a contented frame of mind they must have good wages, kind and just treatment from their employers, good working and living conditions, plentiful opportunity for physical and mental recreation, and good churches and schools.

Compensation of men employed by the United Verde is based upon a standard scale, varying with the class of work performed. However, we recognize the principle that efficient and skilled workmen are entitled to compensation in accordance with their results, and that mediocre workers, rendering but average service, are not entitled to as much compensation as those doing more and better work. Bonus systems have therefore been established, both at the mine and smelter; standards are set for the various jobs, and the men are thus given an opportunity to increase their compensation to the extent that their results exceed those which may be expected of the average worker.

This system attracts a satisfactory type of worker; it encourages efficiency among new men as well as among the more experienced employes; it develops initiative, and is responsible for many constructive ideas and improved methods. The success of such a system depends on accurate cost estimating and honest administration, and trained men are in charge of this work. Benefits both to the company and to its men have resulted.

Recognizing that good working conditions are of prime importance, the officials of the company are always on the alert to make improvements along these lines. To maintain clean, safe plants and the best equipment is our ambition, and this work is assisted by well-organized safety departments, both at the mine and the smelter.

Living conditions and means of recreation are kept always in mind. Several hundred modern houses have been built for employes, and these are rented at the lowest possible rate. Dormitories have been built for the single men. Clubhouses, built and equipped in an exceptional manner, are maintained both in Jerome and Clarkdale; a fine golf course has been provided; tennis courts, ball parks, playgrounds for children, swimming pools, and other aids to recreation, rest, and amusement are supplying in a most satisfactory manner the means of occupying the spare time of the men and their families.

We have lent our encouragement to the establishment of school systems as good as can be found in the state, and have also encouraged and assisted in the building of churches in Jerome and Clarkdale.

Yet the mere fact that these advantages are supplied would not in itself create the spirit of mutual confidence and satisfaction which is so desirable in an organization. A real problem is the contacting of the workmen with the management and taking them into the confidence of the officials to the extent that they have the proper appreciation and understanding of the company's plans and policies. In an organization as large as ours the higher executives and operating officials rarely come in direct contact with the rank and file of the employes, the result being that there is little personal touch except that which is established between the workmen and their immediate superiors. It is, therefore, an established policy of the United Verde thoroughly to train all its officials, from the highest to the lowest, not only in the details of their work but in the policies of the company, to the end that the company through these officials may secure the confidence and cooperation of the men under them. This training is conducted mainly through conferences and mass meetings of the key men, at which pertinent questions are taken up and discussed.

As to the men themselves, means are provided by which any employe may present a grievance and have the same considered in a sympathetic manner.

Another policy of the company designed to further the welfare of its employes and increase good will is to fill vacancies, in all possible cases, by promotion from within the organization. When positions are not in sight for good men capable of holding better jobs, assistance is given them in finding advanced positions with other companies.

Despite our interest in the welfare of our men, we oppose a paternalistic attitude, and give our employes the free exercise of their own beliefs and the right to live their lives accordingly, providing their conduct is not such as to interfere with their work nor the lowering of the standards of their community.

We believe in taking an active interest in worthy civic movements, and are always willing to cooperate in any constructive program for the benefit of the community, county, state, or nation. We do not believe that our responsibilities cease with the comfort and contentment of our employes.

A group insurance plan has been provided whereby the company assists in carrying insurance for its employes at a low premium rate.

Further detail regarding the working out of our various policies and plans will not be given here, as these are being fully described elsewhere.

Boiled down, our entire policy with regard to our relations with our men is simply an endeavor to have every official or supervisor, major or minor, establish the same relations with his employes as he would wish his superiors to establish with him. In other words, a simple application to industry of the philosophy of the Golden Rule.

EMPLOYMENT and WELFARE
at the United Verde

By J. C. Harding
EMPLOYMENT AGENT, SMELTER DEPT.
C. L. Guynn
EMPLOYMENT AGENT, MINE DEPT.

THE United Verde policies and methods developed for the purpose of recruiting and hiring a high type of labor, together with our policies aiming to create a more harmonious relation between the management and the employe, were adopted in their present form early in 1925. Our policy in this regard is predicated on the belief that centralization of all responsibility relative to carrying out the company's labor policy will insure not only equal justice to the workman but lower labor costs to the company as well. To assure the uniform administration of all labor matters throughout the entire organization, two separate offices are maintained; one connected with the smelter at Clarkdale, the other serving the mine at Jerome. Each office is in charge of a man familiar with the particular requirements of the branch he serves.

EMPLOYMENT

As a general rule, the available supply of skilled and unskilled mine and smelter labor is equal to the demand, and with the exception of an occasional shortage of experienced miners and smeltermen we experience no difficulty in obtaining sufficient labor necessary for the operation of our mine and smelter plants.

The foregoing statement should not lead the reader to believe that there is always a large army of unemployed men in the Verde mining district or that a first-class man capable of filling any vacancy in the organization may always be selected from among the men seeking employment; quite the reverse is true, and a majority of the men applying for work are of foreign birth and inexperienced in mine and smelter work.

Because of the fact that a majority of all applicants are what may be classed as "green men," it will readily be seen that the big problem constantly confronting the employment agent is one of selecting only those men who may be expected to remain with the company for a long period of service and ultimately develop into valuable employes.

By carefully selecting each man entering the organization and giving all to understand that there is an opportunity for advancement for those who remain with the company and apply themselves to their work, we are able to hold within the organization a group of men who are semiskilled and who act as a reserve, from which most jobs requiring certain training and knowledge of our operations may be satisfactorily filled.

As an additional safeguard against unforeseen demands for labor, the Employment Department at Clarkdale maintains an "Extra Board." This group of men are carried on the pay roll and are inactive unless they are called to replace some absentee, or to meet some unusual demand for labor. After members of the Extra Board have been hired, and pending the time when they are assigned to regular work, they are given particular instruction in safety and other company policies.

Each applicant for employment is given the privilege of a private interview, and it is endeavored to make the man seeking work feel that he is welcome the minute he enters the employment office. Each applicant during his interview with the employment agent is given sufficient time to state his qualifications in his particular line of work and also make known his reasons for desiring to enter the services of the company. Those men who, in the opinion of the employment agent, would make desirable employes are instructed to report at the employment office each morning, and they are hired in the order in which their appilcations are received. Men who for some reason or another are seemingly not qualified for work in the organization are told that they are not acceptable, and no undesirable applicant is permitted to linger in the hope that he will eventually be employed. Each day the men needed are selected from among the group previously interviewed, and after they have executed their personal record cards and insurance applications they are referred to the Medical Department for physical examination. After the newcomer has satisfactorily passed the physical requirements he is given a copy of the general rules and safety regulations, and also brief verbal instructions relative to the company's policy toward its employes, and also the compulsory safety equipment required, after which he is told to report to his foreman for duty.


J. C. Harding


C. L. Guynn

TRANSFER AND DISCHARGE

If for any reason a foreman or shift boss is not satisfied with a man's work the employe is referred back to the employment department with a recommendation for discharge or transfer. The employment agent then gives the employe an opportunity to state his case and, taking into consideration the foreman's recommendation, decides whether to discharge the employe or transfer him to another department for a second trial. Thus the foreman or boss is permitted to remove from his department those men with whom he is not satisfied; however, no employe is ever dismissed from the service of the company without the approval of the employment agent.

The wisdom of this policy governing transfer and discharge has been proven many times over when men who failed to perform satisfactorily in one department have developed into valuable employes when transferred and given a chance in some other line of work.

EMPLOYES' TRAINING

The practice of rewarding those employes who have performed loyal and efficient service is followed very strictly. Whenever a vacancy which may be considered out of the ordinary occurs, some deserving man from within the organization is selected to fill that vacancy. Our policy relative to promotion has been very profitable to the company as well as the employe, and in the last few years we have developed not only many promising men into skilled workmen of a high order but many shift bosses and foremen as well.

Considerable progress has also been made in training young mining engineers in the practical side of mining. Each spring a number of young engineers are brought into the organization and placed in the operating departments. They are all compelled to start at the bottom, and as their knowledge of mining in general and our methods in particular broaden, they are given jobs carrying more responsibility, until ultimately they are subject to promotion to a position in the bossing organization or an important position on the staff.

We also (*Continued on page* 103)

United Verde Apartment House

By R. K. Duffey *

HOUSING and SERVICE for Employes

THE Upper Verde Public Utilities Company, a subsidiary of United Verde Copper Company, serves the residents of Jerome and Clarkdale with power, light and water and also owns and controls the town of Clarkdale and handles the housing facilities for the United Verde Copper Company in the town of Jerome.

Realizing that a well-housed, contented employe is an asset to the company and the community, a building program was inaugurated several years ago with the plan in view of furnishing comfortable living quarters at a rent commensurate with the incomes of the average mine and smelter employe. The dwellings recently constructed are three, four, five and six room cottages, concrete tile or stucco. All dwellings have tub or shower bath, hard maple floors, composition or tile roofs, sleeping porches, laundry trays, and where the topography permits, fenced-in yards. Company houses are wired for electric ranges and have convenient outlets for lamps, vacuum cleaners, etc. These dwellings rent as low as $15 per month for three rooms and bath, and as high as $45 per month for the larger houses with Arcola heating installations. The employes seem to be entirely satisfied with the type of houses furnished and there is always a waiting list of applicants. The pay roll turnover of employes living in company houses is around 5 percent.

Power, purchased from the United Verde Copper Company, is transmitted by a 2,300-volt line to the distribution lines where it is transformed to 440, 220, and 110 volts, and metered to the consumers. Electricity for lighting is sold on a sliding scale, 11 cents to 4 cents per kw. hour, depending upon consumption. For cooking purposes there is a flat rate of 4 cents per kw. hour. Both towns are lighted by series of incandescent street lamps, this type being used to eliminate radio interference. Electric ranges are sold at cost and installed free of charge in all company dwellings. The saturation point for electric ranges is higher in these towns than in any other community in the United States.

Spring water, flowing from the base of Mingus Mountain, is piped to large storage tanks above both towns. From these tanks it is distributed through 6-in. mains and metered to the consumers. Due to the elevation of these tanks the pressure is ample, not only for distribution to any part of the community, but also for fire fighting purposes. Water is sold on a sliding scale from 90 cents to 18 cents per thousand gallons, depending upon consumption. During the summer months, all residents desiring to maintain lawns are furnished 15,000 gallons of water per month free of charge. Due to the character of the soil and sliding ground in the town of Jerome it has been found that copper service pipes are much more satisfactory than iron. All replacements during late years have been made with copper pipe.

The town of Clarkdale is located in the Verde Valley at an elevation of 3,500-ft. above sea level, on the Verde River about one mile from the United Verde Smelter and five miles from Jerome. It was laid out in 1914, for the purpose of housing the employes of the United Verde smelter and subsidiary companies. It is generally spoken of as a model town. The streets are wide and are either gravelled, or paved with concrete or oiled macadam. In the center of the town is a large plaza, dotted with trees and shrubs and covered with grass, for use of the people. (There are no "Keep off the grass" signs.) All places of business are housed in modern brick buildings. At present there are three grocery stores, two dry-goods stores, one furniture store, two cafes, two cigar stores and pool rooms, music store, drug store, garage, transfer company, four oil companies, bank, moving picture theater, laundry, tailoring and cleaning establishment and five churches.

There are 520 dwellings and two hotels, which house most of the employes. However, due to the contented spirit of the community and the virility of the employes in general, not excluding Thomas Taylor, general superintendent, the birth rate is exceedingly high and more and larger houses are in demand. An additional 40 houses are now under construction.

Clarkdale is not incorporated. The entire management is handled by the Upper Verde Public Utilities Company, including police force, street maintenance, sanitation, etc. Garbage is collected four times weekly from the residence section and daily from the business section. The company requires that all premises and yards be kept clean

* Manager, Upper Verde Public Utilities Co.

Two dwellings at Clarkdale, Arizona

and the sanitary conditions in general are excellent. There is a sewer connection for every building and sewage disposal is handled through an up-to-date septic tank.

An ice plant of 20 tons daily capacity serves both towns. Ice is shipped to Jerome in refrigerator cars and in Clarkdale is delivered direct from the ice plant.

A volunteer fire department is maintained. Due to the efficiency of this department and the type of house construction, there have been no serious losses from fire since the town was first started.

The Clarkdale School District has six modern brick buildings, including domestice science, manual training, domestic art and auto mechanics. The high school has a four-year course and is accredited by the North Central Association of Secondary Schools and Colleges.

Jerome is an incorporated town of about 10,000 population, established in about 1888, when the late Senator Clark took over the property of the United Verde. Its affairs are ably administered by a town council and manager. The construction of a modern town upon a steep hillside would, perhaps, be a revelation to those unaccustomed to building methods in mountainous regions. Jerome is located near the base of Mingus Mountain, which rises to an elevation of 8,200 ft. above sea level and 3,000 ft. above the town of Jerome. The present town little resembles the early day mining camp with its open saloons, dance hall girls and red light district. Today it is a modern community. The business streets are paved with concrete and all residence streets are oiled macadam, or gravelled.

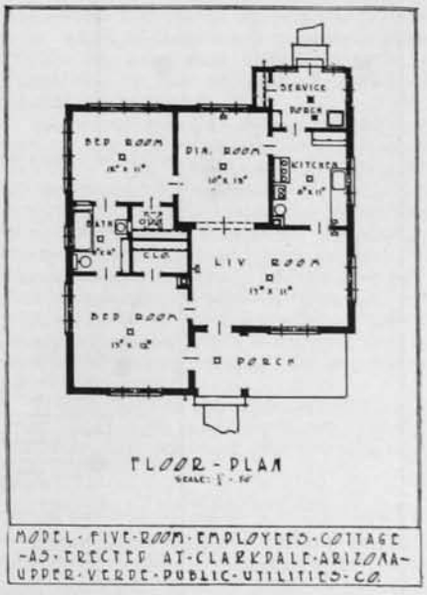
Until recent years the company has maintained only a few residences in the town of Jerome for some of the officials and foremen. In the last four years 117 new houses have been added to extend housing facilities not only to officials and foremen, but to the workmen. Owing to the irregularity of the country surrounding Jerome, it has been necessary to locate this construction in the most advantageous places, which resulted in four small townsites outside of the town limits, but adjacent to the mining property. These sites are on waste dumps in some instances and cut out of sharp hillsides in other cases.

This company maintains a four-story mercantile building, housing several independent businesses, large ball room, and lodge room. It also owns and operates a dormitory which will house about 80 single men. This building is steam heated, has hot and cold water in each room, tub and shower baths.

A class "A" concrete apartment building is located on one of the main streets of Jerome, containing 13 three-room and 20 four-room apartments, together with offices for the Utilities Company and the children's welfare clinic. The building is steam heated, has a laundry room equipped with stationary tubs, wringers, dryer, etc., for use of the occupants; each apartment being furnished with refrigerators, electric ranges, folding beds and many built-in features.

Altogether about 330 mine employes are housed under company roofs.

A branch line of the Atchison, Topeka & Santa Fe Ry. Co., furnishes freight and passenger service for the district. Auto freight and passenger stages also connect both towns with Flagstaff, Prescott, and Phoenix, via the state highway. This highway extends from Nogales on the Mexican border, through Prescott, the Verde District, on to Flagstaff and the Grand Canyon Bridge to Fredonia on the Utah line. There is also a first-class county road connecting with the Globe-Miami mining district.



United Verde Copper Company.

UNITED VERDE COPPER CO. SHEET NO
Jerome Bonus Department
MINE BONUS
MONTH December
LEVEL 500 PLACE 21-A Drift.
CHARGE ACCT. No. 202-A W. O. 6090
APPROVED

CLASS	SHIFTS BONUS	SHIFTS CO. T.	CLASSIFICATION / CONDITIONS	WORK DONE	STANDARD	TOTAL TIME ALLOWED	TOTAL TIME REC'D.	RATING %	BONUS %
TIMBERMEN									
TMB. HLPRS.									
MINERS									
PLUGGERS									
CARMEN	48		18 Cuft. Cars Mucked	442	8 Cars	55.2			
JIGGERS			Tramming Average distance 610 ft.	442	42 "	10.5			
MOTORMEN						65.7	48	137	18½
SWITCHMEN									
TOTAL									

No.	NAME	OCCUPATION	RATE PAYROLL	TOTAL SHIFTS	BONUS RATE	TOTAL BONUS	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
243	Garcia, Juan	Carman	4.40	24	0.814	19.54	888888 888888 888888 88 888 88
691	Blas, Jesus	"	4.40	24	0.814	19.54	888888 888888 888888 88 888 88
					~~0.814~~	~~19.08~~	

UNITED VERDE COPPER CO. SHEET NO.
JEROME BONUS DEPARTMENT
MINE CONTRACT
MONTH December
LEVEL 500 PLACE 21-A Drift.
CHARGE ACCT. No. 202 A W. O. 6090
APPROVED

CLASS	SHIFTS CONTRACTS	SHIFTS CO. T.
TIMBERMEN		
TIMBER HLP.		
MINERS	23	
CARMEN		
TOTAL		

FORM 633 SW 8-13-29 VCH.

WORK DONE	PRICE	TOTAL
22 ft. 6x8 Poly. Dft.	3.75	82.37
66' 6x8 " "	2.00	264.00
GRAND TOTAL		348.37
TOTAL DEDUCTIONS		291.53
AMOUNT DUE CONTRACT		56.84
LOSS ON CONTRACT		
DUE PER SHIFT		
Miner		2.471

DEDUCTIONS	
23 MINERS @ 4.91	113.83
" " CARMEN	
TMBR. HLP	
PLUGGERS	
CARMEN	
HCK. SUP	
TOTAL LABOR	113.83
22.21 ex. POWD. @ 8.8¢	177.68
TOTAL DEDUCTIONS	291.53

NO.	NAME	OCCUPATION	RATE PAYROLL	TOTAL SHIFTS	BONUS RATE	TOTAL BONUS	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
168	Smith, Joe	Miner	4.95	23	2.471	56.83	888888 888888 8 8888 8 888 88

FORM 632 SW 8-12-29 HS-12820

U. V. COPPER CO.
Sheet No. 42
Month Dec 1929
JEROME BONUS DEPT.
Shop Machine

Class	SHIFTS Bonus	SHIFTS Co. T.	Bonus No.	DESCRIPTION	Estimate	Actual	Rating %	Bonus %
			9	Testing Nordberg Shovel	4½	3	142	21
			10	Babbit 1 Motor Bearing	1½	1	150	25
			11	Dismantle 2 Main hoist Cylinders	4½	3	150	25
			12	Press out shaft of rotor 50-B Shovel	1½	1	150	25
5.23	3		13	Assemble #5 Sinking Skip	21	15	140	20
			14	Make Key For Yard Crane Clutch	1½	1	140	20

No.	NAME	Occupation	Bonus No.	Acct. No	Wage Rate	Total Shifts	Bonus Rate	Total Bonus	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
32	Smith John	Machinist	9	4051	5.23	3/8	1.098	41	3
			10	X-1	"	1/8	1.307	16	1
			11	X-80	"	3/8	1.307	49	3
			12	X-75	"	1/8	1.307	16	1
			13	6106	"	1 7/8	1.046	1.96	8 7
			14	X-43	"	1/8	1.046	13	1
								3.31	

BONUS ESTIMATE

____ Shifts @ $____ $____
____ Shifts @ $____ $____
____ Shifts @ $____ $____
____ Shifts @ $____ $____
Total Wages $____

Note: This sheet is first made out in duplicate as shown. The workman receives the duplicate. After the job is completed the Bonus Eng. fills it out and files it.

Date Dec. 27
Dep't. Machine Shop
W.O. No. 6106
Bonus No. 13
Work Started Dec. 30
Work Finished

DESCRIPTION
Assemble Sinking Skip For #5 Shaft.

	Estimated Time	Actual Time	Eff. %	Bonus %	Wages	Bonus	Total Labor	Unit Cost
	21 hrs.							

I have inspected this job and found the work satisfactory
FOREMAN.

Employes' Record and Bonus Estimate Sheets used by the United Verde Copper Company

United Verde Copper Company.

EXTRA-INCENTIVE SYSTEM
of the United Verde Copper Company

By *W. P. Goss,* *CHIEF BONUS ENGINEER, MINE DEPARTMENT*

F. H. Jones, *CHIEF BONUS ENGINEER, SMELTER DEPARTMENT*

F. H. Jones

W. P. Goss

THE United Verde Copper Company used several contract and bonus methods of payment prior to 1921, but no systematic plan had been tried or adopted. However, enough data was obtained to indicate that some such method of payment would produce several desirable results. Therefore, in 1921, work was started towards evolving a plan which would be definite, systematic, comprehensive, and fair.

The plan adopted was modeled on a system used in the shops of the Santa Fe Railroad, the assumption being that the average worker is ordinarily 66 2/3 percent efficient, but that it is possible for him to do one and one-half times as much work as is his custom, thus reaching an efficiency of 100 percent. The man above average is able to produce at a rate above 100 percent, while the subnormal man can not reach 100 percent efficiency. It was believed that an efficiency above 110 percent would probably be reached only because of an incorrect standard. Therefore, a 1 percent increase in wages was paid for each 2 percent increase in efficiency between 66 2/3 percent and 110 percent, but above 110 percent efficiency the percentage increase in wages was gradually diminished.

This plan probably provides a fair and equitable method of payment for work in certain shops where there is not a great deal of difference in the productive ability of the men, or where the nature of the tools and product control the output to such a degree that one man can not produce a great deal more than another. These conditions do not exist at this property, and we discovered that exceptionally capable men could and would produce four times as much or more than the least capable, and twice as much or more than the average. This proved true when the average was taken as the actual accomplishment over a long period of time previous to the adoption of a bonus payment plan. Therefore, the system first adopted proved unfair to the most efficient workmen, and it discouraged them from doing their best. Also, because the payments began at a point below 100 percent, or standard, it was very difficult for the workman to understand how his bonus was figured or to figure for himself how much bonus he had earned.

After trying the above plan during 1921 and 1922, it was decided to make several major alterations in the system, and early in 1923 the plan under which we are now working was adopted. It is based upon the assumption that an average man, working at normal speed, equal to what would be expected of him under a day's pay plan, is 100 percent efficient. Such an amount of work is taken as standard for a shift. For every 1 percent increase in efficiency above 100 percent, the man is given one-half percent times his wages as a bonus payment. This 50-50 division goes on indefinitely, regardless of the efficiency attained. The amount of bonus paid for a job is computed according to the formula $B=\frac{1}{2}(E-100)W$. The values for "E" and "W" are obtained as follows: $E=\frac{T_2}{T}$; $T_2=\frac{P}{S}$ when S=standard; P=work accomplished; T=time required for doing P; T_2=time allowed for doing P; E=efficiency expressed in percent; W=wage rate per unit of time; B=bonus paid per unit of time.

Under certain conditions we have found it more convenient to use money rather than time as a unit for the standard. When this is done, the efficiency of the job is affected by the rate of day's pay as well as the amount of work accomplished in a given time, and the practice of having a highly paid class of labor do work which could be as well done by a lower paid class of labor is discouraged.

In the Metallurgical Department it is often possible to accurately measure the quality as well as the quantity of the work produced, so that standards may be set which take both of these factors into consideration. Most of the work outside of the machine, boiler, and pattern shops is of such a nature that the efficiency of individuals can not be computed. Because of this, group bonus plans have been adopted. The maintenance of a certain important mechanical unit is paid for by allowing the men responsible 25 percent bonus, providing the equipment operates without a breakdown throughout the month. They are penalized a given amount for delays in the operation of the machines.

The blast - furnace operation standard is 26 2/3 tons per furnace per hour, with a 1 percent bonus for each 0.01 percent copper below 0.30 percent copper in the slag.

The calcining crew bonus standard is a given cost per ton for labor and fuel.

The reverberatory furnace bonus is figured from three standards based upon tonnage smelted, percent copper in the slag, and waste heat produced. This latter item is included because the furnace men have control of the gases going to the waste heat boilers.

The power-house bonus depends upon the cost of producing electric energy, blast furnace blowing air, and converter blowing air.

The Concentrator Department of the mill is paid a bonus depending upon the ratio of concentration, percentage of extraction, and delay due to plant shut down.

The foundry men are grouped and the standard is in terms of tons of finished casting per man-shift.

The machine and boiler shop mechanics are given individual standards for each job. Most of their work is comparatively simple to rate and the standards are functions of the volume of metal displaced or replaced.

Whenever possible, the standard is set before the job is commenced. A bonus estimate form (shown on opposite page) is made out in duplicate. The duplicate is given to the workman and the original is given to the foreman. It shows a description of the work to be done and the standard allowed. After the job is completed, the foreman fills out his copy and sends it to the time office, where the efficiency of the job is computed and any bonus earned is credited to the account of the workman.

Standards for the open pit are set for individual jobs and workmen whenever possible, but some of the work is rated for a group.

Examples of Open Pit Standards

Type 50-B electric shovel operation standard is 250 cu. yds. loaded into trucks per shift per crew of three men.

Churn drilling standards are 0.75 to

4 ft. of hole drilled per hour per crew of two men, depending on the formation.

Truck hauling standards from the shovels are 58 to 143 cu. yds. hauled per truck per shift, depending on the distance hauled.

Truck repairs standard is $3.60 allowed against the pay roll of the repair crew for each eight-hour truck haulage shift.

Jackhammer drilling and blasting crew standard is based upon cubic yards of material broken.

Pipe work installation and maintenance standard is $23 pay roll allowed per 24 hours. This last item is an example of the efficiency of a group bonus plan on a job which is difficult to rate if the individual jobs are considered. The pipe crew serves various crews of drillers and shovel operators throughout the pit. The work was formerly being done by a crew which cost about $26 per day for wages. After a study of the work was made by the Bonus Engineer, it was decided that the job had been operating below 100 percent efficiency, and $23 was agreed upon as being enough to allow as a standard for this work. The first month after the standard was set, the pay roll plus the bonus paid was only $18, and the service was better than it had been before. The head of the crew had weeded out several subnormal workers and had encouraged the few picked workers left to do a better quality, as well as a larger quantity, of work than before.

The shop work bonus at the mine is practically the same as at the smelter. Some of the standards used are as follows:

Electric Shop—The standard for hanging trolley and installing crossarms underground is 5.5 ft. per hour. The standard for installing a charging panel for storage battery locomotives is 32 hours.

Boiler Shop—The standard for making one set of dipper stocks for a 50-b electric shovel is 320 hours. The standard for building one 18 cu. ft. side or end dump car is 56 hours. The standard for building a body and dump rack for the 10-ton open pit trucks is 400 hours.

Blacksmith Shop—The standard for dressing picks is six minutes. The standard for making and tempering a 12-lb. hammer head from locomotive tire steel is 88 minutes.

Drill Steel Sharpening

The bonus for sharpening drill steel is not based on the number of pieces of steel sharpened, because when this method was tried the operators burned the steel in an effort to increase their production. This burned steel did not "stand up" underground and it was soon back in the shop for resharpening. This tended to increase the work in the sharpening shop and the men were being paid a premium for poor work. Under the present method, the steel sharpening shop pay roll is allowed $1.15 for each operating machine drill shift. When good steel is sent underground, it will have a maximum life before requiring resharpening and the sharpening can be done with a minimum crew; thus, the premium will be paid for good work rather than for poor work.

Underground, the bonus is figured separately for each man wherever his individual work can be measured. However, such work as is performed by a group of shovellers in a stope cannot be divided according to the accomplishment of individuals, so that the bonus for the entire group must be computed as a unit. Mining standards are in terms of volume broken for stoping, and in terms of feet advance for drifts, crosscuts, and raises and shafts. The unit of volume used is the cubic contents of one square set, which is 5 ft. 6 in. by 5 ft. 6 in. by 7 ft. 2 in. Timber work is paid for according to the unit built or repaired. Chutes, manways, chute pockets, square sets, and drift sets are standard in size, shape, framing, etc., throughout the mine, so that uniform standards prevail. Repair work standards must be set to fit the individual job because of the lack of similarity of such work.

Practically all of the new timber work, drifting, crosscutting, raising, is now being paid for under a contract system. This system is really a modified bonus rather than a true contract, because the agreement is verbal, the worker assumes no obligation, the company retains the right of complete supervision, and the man is guaranteed a day's pay regardless of his output. Each man on the contract is responsible to the company only, and is paid separately by the company. Examples of the method used in computing the contracts are:

A timber crew built 10 chutes in five days, for which the price allowed was $10 each.

Credit 10 chutes @ $10		$100.00
Deduct 5 shifts @ $5.23	$26.15	
Deduct 5 shifts @ $4.68	23.40	
Total deductions		$49.55
Balance due contractors		$50.45

Due timberman in addition to day's pay—53%×$50.45=$26.75.

Due timber hpr. in addition to day's pay—47%×$50.45=$23.70.

The division of profit is made according to the ratio of the day's pay rate of the two men.

A drift is driven 100 ft. in 20 shifts, with 30 boxes of powder. The price allowed is $5 per ft. for mining labor and explosives.

Credit 100 ft. @ $5.00		$500.00
Deduct 20 shifts @ $4.95	$99.00	
Deduct 30 boxes powder @ $8.00	240.00	
Total deduction		$339.00
Due contractor		$161.00

The miner receives $161 in addition to his day's pay for 20 shifts.

Two advantages of this type of contract over the bonus system in use here are that, first, the contract is more easily understood by the miner and he visualizes his units of work accomplished in terms of dollars. This acts as a stimulant to greater industry because the man can see the dollars piling up. Second, in the case of development work, the miner is rewarded for saving explosives, while under the bonus system the tendency is to be wasteful of explosives.

A great deal of care must be taken to set a correct standard price under this method of payment, because the company does not participate in the bonus earned and, if the price is set too high, the cost to the company will be too high. Most of the contract prices have been arrived at from bonus standards taken in conjunction with the average bonus earned over a long period of time on a given class of work. For instance, the price of a drift is arrived at as follows: The bonus standard for the drift might be 2 ft. per shift; the average bonus paid for drift work had been about 50 percent. A miner would have to average 4 ft. advance per shift in order to make 50 percent bonus, and thus the cost for labor would be $4.95 plus (50% x 4.95) = $7.42, or $1.855 per ft. The allowance for powder might be $1.90, so the contract price would be set at $1.85 plus $1.90, or $3.75 per foot. If the miner broke four feet per shift and used the allowed amount of powder, he would make $2.475 per shift, which is the same as if he were on bonus. If he does better than this, he will make more money on contract than if he were on bonus; if he falls below the efficiency illustrated, he will make less money on contract than if he were working on bonus.

If the contractor is not doing reasonably well, an investigation is made to determine the cause. If unforeseen difficulties have arisen, the contract price is changed to meet the conditions; if the fault is found to be with the workman, he is transferred to some work for which he is better fitted, or else he is discharged.

The contract price is subject to change at any time if conditions over which the contractor has no control are altered. For example, the price varies according to the formation encountered, and it may be changed to meet variations in temperature or humidity due to the ventilation. If an error is made in setting the price or standard for a job, due to misjudgment on the part of the bonus engineer, the change in price is retroactive to the beginning of the job, providing an increase in price is to be made; if a cut is made, the change takes effect on the day the workman is notified. This practice works to the disadvantage of the company in one way, but it pays by keeping the confidence of the men.

Most of the underground work and all of the mechanical and open pit work is on bonus rather than on contract. The standards are expressed in various units according to the type of work considered, some of which follow:

DRIFTING STANDARDS IN FEET PER MINER SHIFT

	Holes per round				
Drilling Speed	10	13	16	20	24
15″ and up per min.	3.4	3.1	2.9	2.7	2.5
12″	3.1	2.9	2.7	2.5	2.3
9″	2.9	2.7	2.5	2.3	2.1
6″	2.5	2.3	2.1	1.9	1.7
4″	2.0	1.8	1.6	1.4	1.2
3″ and less	1.4	1.2	1.1	1.0	0.8

Note: These standards are for normal conditions. Where the temperature is excessively high or the working place is remote, the standard is made to fit the conditions.

Mucking

Stopes—10 to 20 18-cu. ft. cars per shift, according to the chute spacing, floor conditions, etc.

Drifts and crosscuts—8 cars per shift in porphyry and schist; 7.2 cars per shift in iron sulphide.

Hand Tramming

	18-cu. ft. cars per shift
150 ft.	65
170 "	64
190 "	63
210 "	62
230 "	61
250 "	59
270 "	58
290 "	57
310 "	56

And so on, one car less being required per shift for each 20-ft. increase in distance.

Stope Mining

Square set—Remove ground and stand timber—0.75 sets per miner shift.

Cut and fill—Break ground—1.5 to 2.5 sets (5′6″x5′6″x7′2″) per miner shift, depending on the formation and the hardness of the ground.

United Verde Copper Company

Whenever it is possible to set a standard for each detail operation, the bonus system functions at its maximum efficiency, but sometimes it would cost more to obtain sufficient data from which a standard could be set than the total cost of the job. In such cases no attempt is made to set a standard and the method of handling this work depends largely upon circumstances. The job may be considered as only worth day's pay and be treated accordingly. If the men doing the work have been regularly making bonus on jobs, the efficiency of which could be figured, then they are paid a bonus on the unrated job commensurate with what they have been making and in accordance with the opinion of their boss of their efforts on that job. Of course, this practice does not inspire better or faster work, but it prevents much dissatisfaction among the workers and is cheaper than trying to obtain a standard. Also, we are constantly on the alert to prevent abuse of this practice and to keep the number of jobs so figured at a minimum. Because some of the work about the plant is of such a nature that it is impractical to put it on a bonus basis, this work is done on day's pay basis.

The bonus is figured on the first of each month for the work done during the past month. The men and their bosses are immediately notified of the amount of bonus earned, but they are not paid this bonus until the second regular pay day, which is the twenty-first of the month.

The Mine and Open Pit Departments, with their attendant shops, are located at Jerome; the Mill and Smelter Departments are located at Clarkdale, which is about four miles from Jerome. This physical separation, as well as inherent differences between the work done at the mine and metallurgical plants, has made it necessary to maintain two organizations for handling the bonus work.

The organization of the Bonus Department at the Mine consists of a chief bonus engineer, an assistant, an office clerk, who posts the time of the workmen on the various job sheets, four bonus engineers, who handle the underground work, and two bonus engineers who look after the surface shops. Each engineer computes the efficiency of all the jobs under his jurisdiction and the computations are then checked by the head of the department or his assistant. At Clarkdale, a chief and two bonus engineers keep track of the work and set the standards. The time office keeps account of the time and computes the bonus.

We are constantly revising our old methods and standards, as well as setting new ones, because of further knowledge or because of changing conditions surrounding the work. The installation of better equipment calls for a review of the operations concerned and a revision of standards. Naturally, it is impossible to convince every workman that he is being treated fairly when his standards are changed, but this distrust can be cut to a minimum if the bonus engineers explain the reasons for making the change in a clear, tactful, and patient manner. The success of a bonus system depends largely upon obtaining and keeping the confidence of the workmen. If the bonus engineers are experienced workers themselves, who are known by the workmen to be capable and honest, they can do much towards assuring the success of the bonus system.

When a dispute arises between a workman and the bonus engineer which cannot be settled at once, the workman is invited to take his grievance to the head of the bonus department. If he is not satisfied with the decision given by the chief bonus engineer, he is at liberty to go to the mine superintendent.

We have found that the bonus system of reward for extra accomplishment, whereby a workman is compensated in proportion to his efficiency, has lowered our costs. It has attracted and retained a high class of labor which is prosperous and contented because of the fair pay it is receiving for a fair day's work.

Acknowledgment is made of the free use of two previous papers, viz, *The United Verde Copper Company's Bonus Methods*, by W. V. DeCamp, assistant general manager, U. V. C. Co., printed in the December, 1924, issue of THE MINING CONGRESS JOURNAL, and *Group Bonus System for Smelter Departments*, by C. R. Kuzell and J. R. Marston, Smelter Department, U. V. C. Co., presented at the meeting of the Arizona Chapter, American Mining Congress, Jerome, Ariz., March 14 and 15, 1927.

EMPLOYMENT AND WELFARE

(From page 97)

maintain an apprentice school in connection with the mechanical department at both the mine and smelter. Boys, usually high-school graduates who are anxious to learn a trade, are placed in the shops and serve a four-year term of apprenticeship. Vocational classes in shop mathematics, mechanical drawing, and general shop practice are conducted as a regular part of the apprentice training plan, and all apprentices are compelled to attend these classes. At this point it should be mentioned that when hiring mechanics we at all times endeavor to hire only those men who have served their time as apprentices.

FOLLOW-UP PLAN

The superintendent of the Smelting Department has adopted a follow-up plan whereby all new men are interviewed by him shortly after they enter the service. At this interview an effort is made to ascertain the extent to which the man has been instructed in safety regulations and company policy, as well as what steps have been taken to train the employe relative to our particular operating problems. This heart-to-heart talk between our new men and the superintendent is made in an effort to impress the employe that in his place as laborer, helper, or mechanic he is just as important to the success of the organization as a whole as any member of the staff.

WELFARE

The welfare of our employes and their families, and particularly those who are disabled due to sickness or injury, is of vital concern to the management, and our policy in this regard is one designed first to relieve those who because of accident or sickness are in need of assistance, and then for the general comfort of the entire organization. Employes injured in the course of their employment are satisfactorily provided for and receive adequate compensation during their period of disability under the terms of the workman's compensation law, and although the company has no jurisdiction regarding the amount of compensation paid or any final settlement which the Industrial Commission may make with the disabled employe, we are, however, able to assist the injured man by providing a high type of medical treatment and by cooperating with the Industrial Commission and the employe in order that the case may be promptly and justly settled. Furthermore, we are often able to assist in the rehabilitation of the injured employe after he has sufficiently recovered to permit his return to light duty, and the employe who due to some permanent disability is unable to resume his former occupation is trained to perform some gainful work, which often has made the man more valuable to himself than he was before the injury.

To protect the employe who, due to sickness or injury off the job is compelled to leave his work, we have adopted a group insurance plan, whereby every man who has been in the service of the company for six months or longer receives weekly benefits of approximately 50 percent of his wages during the period of disability. This group plan also covers the employe in case of death from any cause. In order that this insurance may be furnished to our employes at a very low rate, the amount of the monthly premium is divided equally between the men and the company.

Employe clubhouses have been built at both Clarkdale and Jerome. These clubs provide libraries, ladies' lounge rooms, men's billiard and card rooms, bowling alleys, and auditoriums for large parties, dances, and employes' gatherings. The clubs are governed by employes' committees, while the expenses in connection with their operation are borne by the company. Four swimming pools, two at Clarkdale and two at Jerome, are available for all residents of the two communities during the summer months. Community playgrounds are maintained and a recreational area at Peck's Lake is greatly enjoyed during the spring and fall months, a golf course, baseball fields, and tennis courts have also been provided. The problem of housing our employes is rapidly being solved by the construction of modern three and four room dwellings, which are rented to the married employes at a very low rental.

That our efforts directed toward the induction and retention of a higher type of workman within the United Verde organization have been liberally rewarded is unquestioned. Our labor turnover has been greatly reduced, the number of unnecessary absences has been cut to the minimum, and our program for the prevention of accidents has met with a marked degree of success. At the same time the cordiality of the relations between the management and the employe has been developed to a degree which 15 years ago would have been considered impossible.

United Verde Copper Company.

By J. C. Harding
SAFETY AND EMPLOYMENT ENGINEER
Oscar A. Glaeser
SAFETY AND VENTILATION ENGINEER

SAFETY POLICY *and* ACCIDENT PREVENTION

ACCIDENT prevention is not a one-man job. On the contrary, it is squarely put up to every man as a part of his work. Foremen and shift-bosses are held strictly responsible for the safety of the men under their supervision. A good safety record is considered one of the qualifications of a good boss. The Safety Department renders all the assistance possible in bringing about the desired results.

When a man is employed he is informed of the equipment which he must have before he can go to work. This consists of protective headdress and hard-toe shoes in the case of mine employes and proper footdress and clothing for smelter work. Leggings, spats and respirators are also required in certain smelter departments. Each new employe is furnished with a pair of goggles, which he is required to have on his person during hours of employment, and a book of safety rules. He is also instructed in the company's general safety policy.

Safety training is carried on in the smelter department by teaching those who are on the "extra board" or the "absentee reserve" ways and means of doing work safely and efficiently. Such training is not carried on in the mine.

Guarded by protective clothing and armed with the knowledge that safety should at all times receive his first consideration, he is sent to his boss. The bosses' first instructions deal with safety. The new man is instructed in standard practices, such as wearing goggles when breaking rock or slag with a hammer, when collaring a drill hole, or when blowing out a drill hole. Barring down roof and muckpile and the correct manner of handling heavy material is stressed. The necessity of using only tools in good condition is emphasized. The miner is informed of the location of "Safety Chambers" and when to use them, and then he is specifically told about his particular job. It is felt that by this time the new employe will have gained some idea of the value this company places on the lives of its employes, and that the first seeds of a safety consciousness have taken root.

There are two important elements to any accident prevention policy. The first is a safe plant or mine, as the case may be, with all the protective devices and safeguards necessary to eleminate and reduce industrial and occupational hazards. The next step is to induce men to work safely. The use of safeguards and the latest and best protective devices is an old established policy with the United Verde Copper Company. Methods of mining are changed to afford the greatest protection to the miner. The main shaft and shaft stations are of steel and concrete; this has been done to eliminate that dreaded hazard, shaft fires. Manways and all other vertical or inclined openings are covered or properly guarded. Safety belts are kept at grizzlies, and grizzly men are required to wear them. Blasting is done on a combined time schedule and signal system which extends from level to level. The main haulage systems operate on a modified block signal system, and haulage motors are equipped with heavy plate guards to protect the motormen. All loading chutes are supplied with hinged loading platforms which have practically eliminated the foot injuries common to loaders. All hoisting cables, cages, dogs and crossheads are inspected weekly. The large man cage is inspected every morning before the shift goes down. A fire drill involving all the men in the mine is held once every three months.

There are many other features too numerous to mention which make up the safeguards and practices in the mine and plant. It is the foreman's responsibility, assisted by other supervisors, to see that these safeguards are at all times in first class working order. The work of safeguarding mine and plant has been in process for years, and because of the nature of the industry, will continue as long as ore is mined and smelted.

Safeguards are of no less importance in the smelter. Here again protective clothing and goggles become an all-important part of a man's equipment. Safety devices, such as water seals, have been installed in an extensive system of acetylene gas lines designed to prevent backfire into the mains and generator. Steel ladders, stairs, and railing have replaced all wooden structures of this kind. Every possible effort has been made to guard all moving machinery. Steam lines are all covered with asbestos. This is not only an economic measure but has eliminated burns which at one time were quite common. Safety regulations have been posted in each department, designed to inform the workman of, and guard him against, the particular hazards to be found there. Here again the foremen are considered the key men, and as such are held strictly accountable for accidents.

While accidents are the bosses' responsibility and their prevention his duty, the safety engineer has a responsibility and a duty which is equal to the bosses' in this regard. The boss instructs his men in the work to be done, and it is assumed that these instructions include the necessary safety admonitions, followed by enforcement where this becomes necessary. It is difficult for a boss, harassed by production problems, to go much deeper into accident prevention. Some one with more specialized training, and, therefore, better equipped for this work, can be of material aid to a boss in carrying out an accident-prevention program.

In the United Verde organization the safety departments render this assistance. Its members make periodic inspections and recommendations resulting therefrom. They supervise the transportation of men in the mine and arrange the blasting schedule. The safety departments issue safety rules and instructions; test and supply approved safety equipment; pass on all safety suggestions; and are charged with the responsibility of proper mine ventilation and fire prevention. All of this is necessary if the mine and smelter are to be safe.

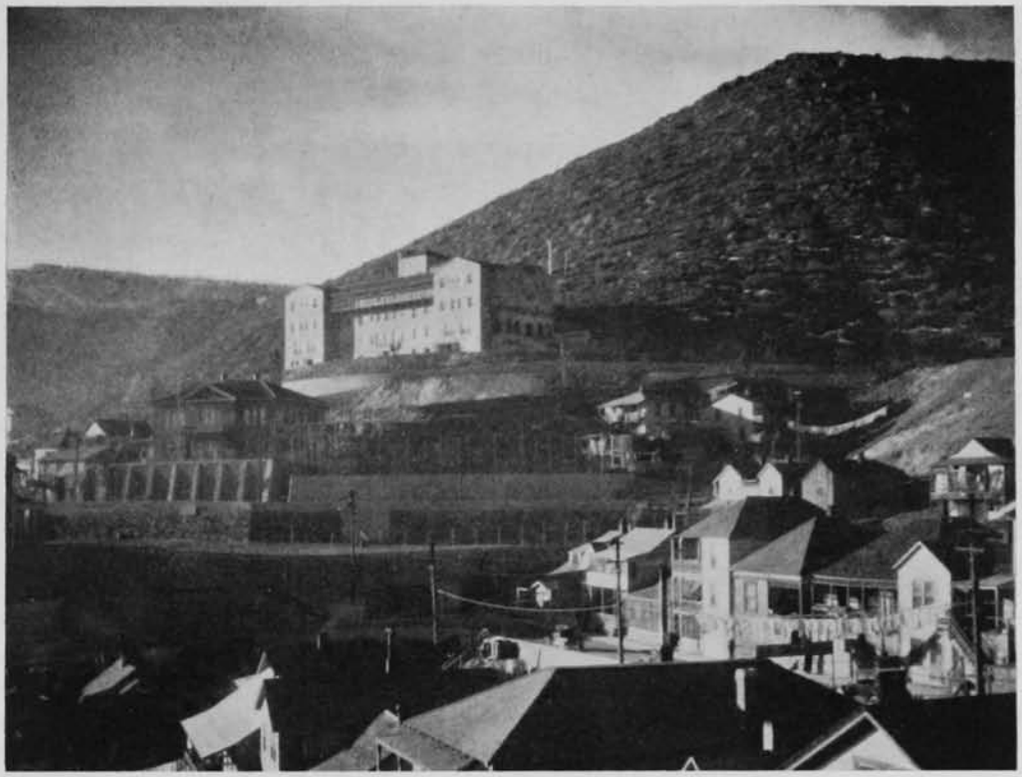
More difficult is the task of controlling the human element to the extent of inducing men to become safe workers. Education and safety propaganda are two means available to establish a safety consciousness among the men. In the operations of the United Verde the two are more or less interwoven, yet each serves its purpose separately. Education is carried on principally through general and departmental meetings. At the mine lunch hour safety talks with the men are held. A group of men is brought together; they are given time to eat their lunch; cigarets are passed, and the meeting begins.

At most of these meetings the safety engineer or his assistant gives a talk, occasionally a meeting is thrown open for discussion, and at other times some employe gives a prepared talk. The talks range in subject matter from practical applications on one or more jobs to inspirational talks. They have all proven popular and profitable. These meetings are easily arranged for in the mining department.

The smelter department relies for its success in this matter on evening mass meetings held in the clubhouse, at an hour when the whole family can attend. In many respects, the latter is propaganda of the finest kind, in that the entire family becomes concerned about the husband's and father's welfare while at work.

The smelter is further divided into four divisions. Each department holds its own meet- (*Continued on page* 112)

United Verde Copper Company.



The New United Verde Hospital

By Dr. A. C. Carlson
CHIEF SURGEON

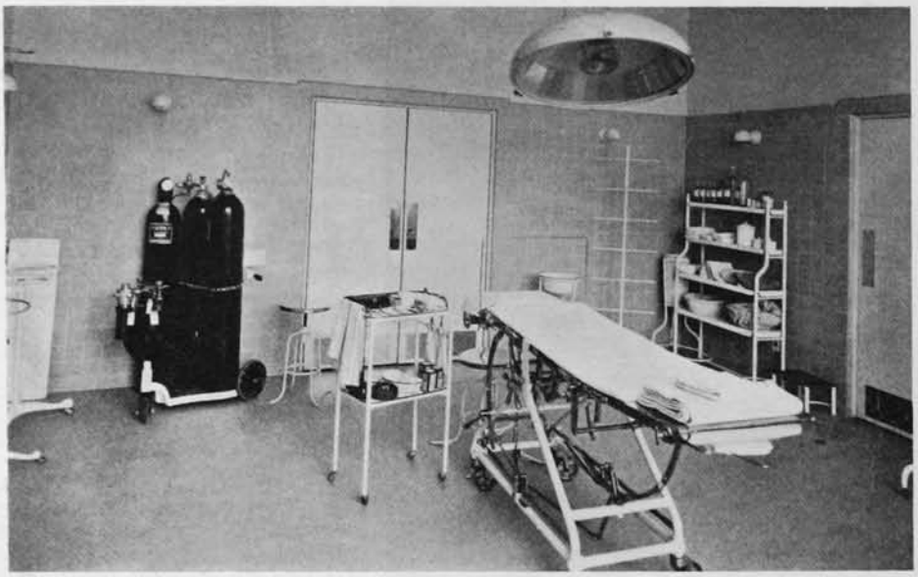
MEDICAL DEPARTMENT
of the United Verde Copper Company

I AM well aware that opinions differ as to how far the activities of industrial medicine should extend. Personally, I feel that in the cities where a general hospital and specialists in every line are available, the activities of the industrial surgeon should extend only to the occupational injuries and diseases, and to the plant as far as safety and sanitation. On the other hand, in small communities such as we have in Arizona, where as a rule, the community actually exists because of the presence of the industry, the activities should extend to the entire health supervision of the employe and his family. Some critics would call this a combination of industrial and community medicine, but, regardless of what it is called, it is the ideal plan for the mining communities of this state. Industry renders this extensive service at a heavy cost in dollars and cents, but this is actually a sound investment to both employe and employer. The employe, for a small monthly hospital deduction, receives free medical and surgical care, as well as free hospitalization for himself for all conditions, except venereal. For his family he receives free medical care with a 50 percent reduction on hospital and surgery. This health supervision is a problem of human conservation—the conservation of the lives and limbs of the employes as well as the reclamation of the disabled workman in his daily strife. It is the soundest foundation for maximum production of any industry, for aside from its economic value it fosters a sense of loyalty as well as making the men's work more attractive.

The activities of the Medical Department of the United Verde Copper Company in carrying out this program of health supervision are as follows: Physical examination of employe, care of accident cases, medical and surgical care of the employe and his family, hospitalization and infant welfare clinic. Indirectly, the duties extend to the Safety Department through close cooperation with the safety engineer in his work with safety and sanitation. Also cooperation with the Employment Department in reclamation of the disabled, as well as placing new defective employes at suitable work.

The first activity—physical examination—was primarily instituted to protect industry against the dishonest and unscrupulous applicant who would attempt to collect damages for minor or major defects he had prior to entering the employ. Today, with our present compensation law, this protection is not necessary. However, the true value of this procedure to both employe and employer is appreciated. The examination continues more thorough than at first and is repeated at periodic intervals. The Medical Department is familiar with the work and working conditions of the various departments. When an employe is found with defects great enough to prevent him from performing the duties for which he was hired, he is either rejected or recommended for work that he is physically able to perform, with efficiency to the company, as well as work which will not further aggravate his condition.

The physically perfect individual is a rare specimen. Some physical fault can generally be found in any one. It therefore becomes necessary to employ men with certain impairments. I will not attempt to classify these impairments, other than to state the problem of placing or rejecting the defective workman depends upon whether the defects are major or minor; whether they are correctable or non-correctable. The major non-correctable case should not be placed in an industry such as mining and smelting. In the majority of cases the minor defects are passed, as well as a certain number of major correctable cases. Often, conditions are found that the employe was not aware of, for which either treatment or timely advice is given.



Operating Room at Hospital

THE CARE OF ACCIDENT CASES

The majority of injuries are of a minor nature, principally injuries to the fingers, hands, and toes. In all injury cases, either minor or major, our aim is restoration of the most perfect function possible, striving for the best economic end result as well as medical end result. This treatment is rendered with genuine friendliness and kindness. We want the workman to know that we are sincerely anxious to get the best possible results. The close contacts and friendships develop a feeling of satisfaction and confidence in the employe regarding the care he will receive should he meet with an accident.

MEDICAL AND SURGICAL CARE OF EMPLOYE AND FAMILY

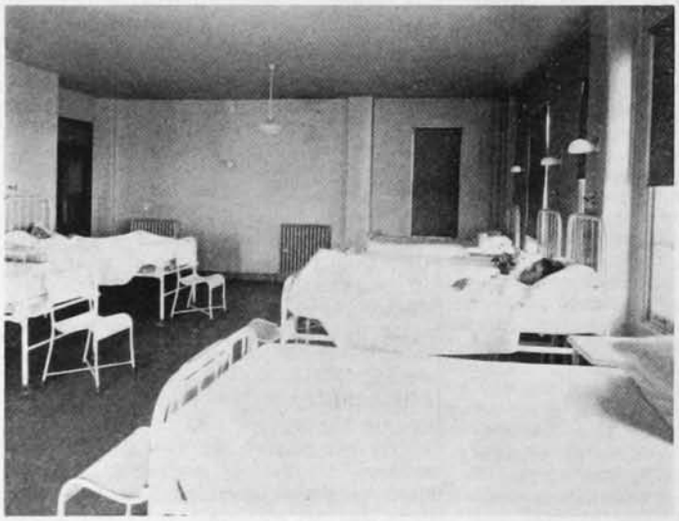
The employe is given free medical and surgical care, as well as hospitalization for any condition that may develop, except venereal. The family of the employe receives free medical care and free surgical care for minor conditions. Operations are charged for at one-half the regular rate, with a maximum charge of $75.

Hospitalization is furnished the family at $2.50 per day for private room, and $12.50 per week for the ward. All medicines and dressings are furnished free of charge. Many medical men will state the last named duties belong to the family physician. However, in the small community in which we live we are the family physicians to the great majority of our employes. As to the economic side to the employe, it is a comfort to him not to have worries over doctor bills when sickness comes in his family.

HOSPITALIZATION

We have a modern institution, three and one-half stories above the ground or basement floor. The building is reinforced concrete construction with hollow tile partitions, metal lath and plaster. On the ground, or basement floor, we have an engineer's room, storage battery room (which automatically cuts in should the regular power line cut out). The emergency lighting system supplies the main surgery and corridors. There is a refrigerator room where we make 200 pounds of ice per day and which also cools the main kitchen refrigerator. Also a boiler room in which we have heat, sterilizing and hot water boilers, water softener and incinerating plant. Adjoining this room is the ambulance entrance, laundry room, locker room and two spacious steam-heated garages. On the first floor we have the out-patient department, waiting room, doctors' offices, large dressing room, fully equipped with a sterilizing unit and pharmacy. Adjoining the dressing room we have a physical therapy department, fully equipped and under the supervision of a capable physical therapy nurse. On this floor we also have the hospital storeroom, main kitchen, nurses' dining room, convalescent patients' dining room, kitchen store room, rooms for kitchen help and porters' utility room. The kitchen is under the supervision of a capable dietician. We also have here a first and second cook, dish washer and combination waiter and cook's aid. The kitchen is equipped with electric range, steam table, (*Continued on page* 111)

Sun Porch



Ward and Private Room at United Verde Hospital

United Verde Copper Company

East Terrace of Clark Memorial Club at Clarkdale, looking toward Smelter

By Noel Pegues

LIBRARIAN, CLARK MEMORIAL CLUB

RECREATION in the Verde District

IF ONE of the miners who worked at the United Verde back in the 90's, when it was still in its infancy, could return to the scene, he would find in use different methods of mining and smelting, but there would still be shafts, tunnels, drifts, cross-cuts and blast furnaces, as in his day. There is little similarity, however, in the way in which the employes, then and now, spend their leisure hours.

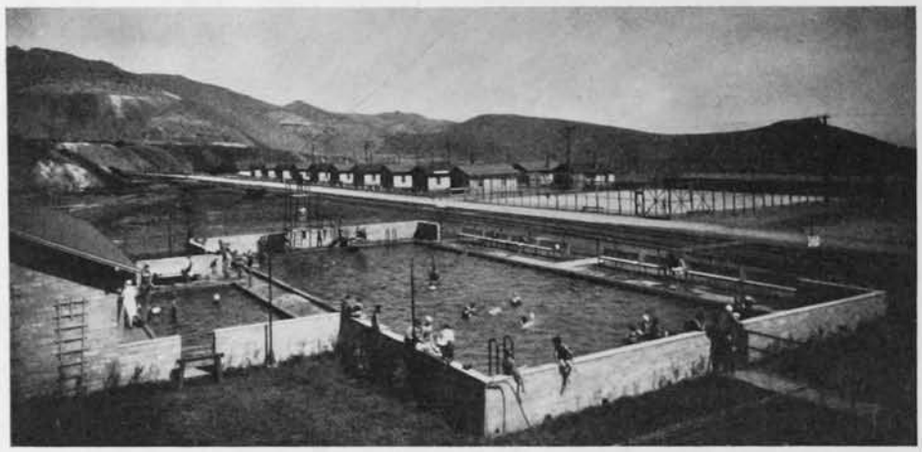
In his day, the saloon was the working man's club; the opportunity for outdoor sports was limited. Today, numerous facilities for recreation are provided in Jerome and Clarkdale. These facilities are equal, if not superior, to those found in the larger cities, and there is no reason to regret residence in a so-called smelting or mining camp.

Through a $100,000 bequest made by the late Senator Wm. A. Clark, the town of Clarkdale has a beautiful club house which was dedicated to the employes and their families in appreciation for long years of faithful service. This club house, as well as the one in Jerome, is available for the use of all employes and their families. There are no dues or membership fees, and the facilities, with the exception of bowling, the soda fountain and the pool and billiard tables, are furnished without charge. The concessions are operated on a non-profit basis.

In addition to the bowling alleys and billiard room already mentioned, the Clarkdale club house has a large, attractive men's lounge, a public library and reading room, where 40 current magazines and newspapers are always available, a ladies' lounge, an auditorium, and change rooms which are used in connection with the swimming pools and gymnasium. This club is as beautifully furnished as many a metropolitan club, and its comfortable lounging chairs and divans, beautiful draperies and floor coverings, as well as the complete and luxurious culinary equipment, are a constant joy to the residents.

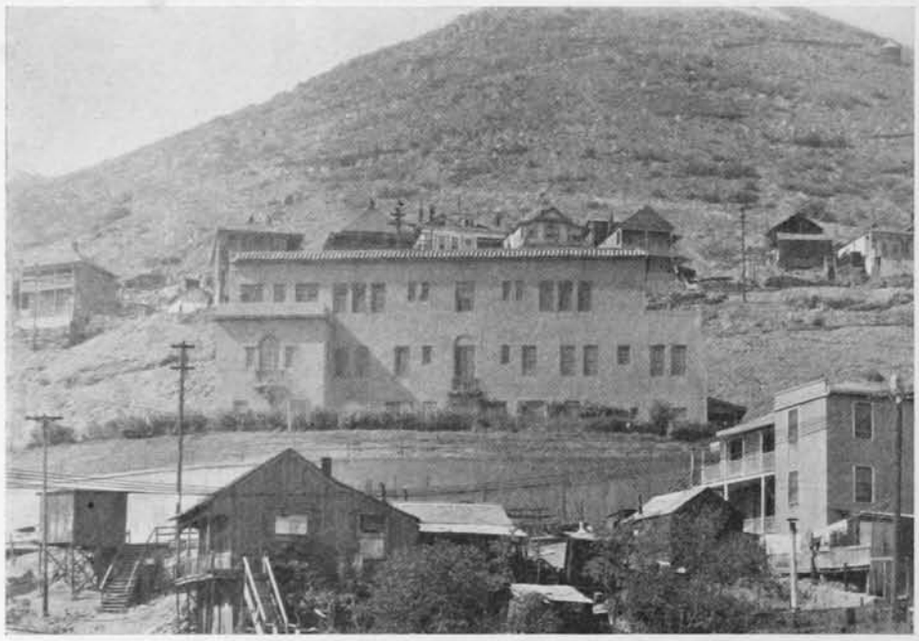
The auditorium, with a seating capacity of 600, has an excellent stage and dressing rooms, and is also used as a dance hall. The Clarkdale schools utilize the auditorium for a gymnasium and basketball court.

At Jerome, the club house has a large men's lounge containing pool and billiard tables, a soda fountain and card room, a ladies' lounge and card room,



Swimming Pool at Jerome (Wading Pool for Children at left)

Club House at Verde Valley Golf Club, Clarkdale



United Verde Club House at Jerome

United Verde Copper Company.

and a small ballroom which is used for dinners and dances. The Jerome Public Library is also housed in this building, and the United Verde Copper Company assists the library in the purchase of current books and magazines.

The excellent facilities provided by these club houses have been the inspiration for an unusual number of activities among the citizens of the two towns.

Swimming Pools

The younger generation has not been overlooked with respect to recreational facilities. Four swimming pools have been built and are operated and maintained by the Copper Company. At both Jerome and Clarkdale pools are provided for the Mexican population as well as the American. There are club houses at both towns for the Mexican employes, and these too are operated without profit by the company.

At Peck's Lake, around which the Verde Valley Golf Club course is laid, a public playground has been established. Swings, teeters, merry-go-rounds, and other paraphernalia for childish enjoyment have been installed. Similar equipment is provided in the public park on the 300-ft. level in Jerome.

During the course of steam shovel operations at Jerome, a large gulch, located about a mile from Jerome and several hundred feet above the town, was filled in. This filled ground was levelled off and is now used as a baseball park and football field. Another excellent park for football and baseball is located in Clarkdale, and both parks have adequate grandstands.

There are four tennis courts in Jerome, located near the park and swimming pool, and in Clarkdale there is a like number, as well as a group of horseshoe courts.

Golf Course

The outstanding feature for the recreation of residents of Clarkdale and Jerome is the nine-hole golf course, within a mile of Clarkdale, and only six miles from Jerome. This course has all grass fairways, and being built around Peck's Lake, has the only natural water hazards of any course in Arizona. It is considered by many to be the sportiest nine holes in the Southwest, and many local and state tournaments are played at the Verde Valley Golf Club.

A beautiful club house, with men's and women's locker rooms, a large lounge, dining room and kitchen, is part of the golf club plant. Near the golf club house, and on the shore of the lake, is a dance pavilion, erected as a part of the Wm. A. Clark Memorial. This pavilion is used in the summer months for public and private dances.

Both motor and row boats are housed at Peck's Lake, and the horseshoe shaped waterway, a mile in length, affords an excellent course for the speedy crafts.

Nearby Resorts

Numerous mountain and lake resorts within a few hours' drive from both Clarkdale and Jerome offer many possibilities for vacation and week-end trips.

Oak Creek Canyon, famous for its magnificent scenery and fine trout fishing, can be reached in an hour and a half over a splendid highway. Cottage resorts, inns, and camp sites are available for motorists throughout the 12-mile canyon.

Stoneman and Mormon Lakes are also popular vacation resorts nearby, where

good bass and perch fishing may be enjoyed. These lakes are situated in one of the largest virgin pine forests in the world, and are ideally located for summer resorts.

The United Verde Copper Company has cooperated with the United States Forest Service in providing a recreation area on the summit of Mingus Mountain, about half an hour's drive from Jerome and at an elevation of 8,000 ft. The road to this area is maintained by the Forest Service, and picnic tables and benches, fireplaces, etc., are provided by the Copper Company. In the summer this cool park among the pine trees is a popular spot, as the elevation of Mingus assures a delightful temperature at all times.

Flagstaff, located 65 miles north of the Verde Valley, has modern hotels, a fine golf course, and numerous facilities for spending a pleasant and inexpensive vacation.

Prescott, about 40 miles from Jerome and Clarkdale, has a delightful summer climate, and excellent accommodations for tourists and vacationists. The Smoki Snake Dance, held on the second Friday in June, and the Frontier Days, celebrated early in July, at which a galaxy of cowboy contests are held, are annual events which attract many visitors from the Verde District.

During the past few years a number of dude ranches have been established in the Verde Valley, which is also the center of the district in which many features of prehistoric interest exist, such as Montezuma Well and Castle. All these points are popular objectives of week-end and holiday excursions.

The old-timer referred to in the opening paragraph might not approve of his co-workers tramping the fairways in golf knickers, or doing the jack-knife at the pools in gaudy bathing suits; but their physical fitness, and the contentment both of the men and their families, should soon convince him that the change has been for the better.

Clark Memorial Club House at Clarkdale

Ladies' Lounge in Clark Memorial Club at Clarkdale

Men's Lounge in United Verde Club at Jerome

Jerome Ball Park

United Verde Copper Company.

Jerome Supply Department, Exterior, and Rock Drill Parts Storage Section

Supply Department Procedure at United Verde

By D. L. Bouse *

A SLOGAN has been adopted by the supply department of the United Verde at Clarkdale and Jerome. It is "This is a Service Department."

It is our business to serve the operating departments of this company with material and supplies needed to carry on the mining, milling and smelting of copper and still not allow our stocks to reach unreasonable proportions.

One of the first steps toward accomplishing this service is to pay careful and prompt attention to our requisitions for we know that "A requisition in time saves a rush order."

* General storekeeper, United Verde Copper Co., Jerome and Clarkdale, Ariz.

Requisitions to Purchasing Agent

We make every effort to regulate the issue of requisitions on a schedule of dates set aside for the various departments. This plan is more or less flexible but is important in maintaining a regular flow of these requisitions to the purchasing agent's desk.

Requisitions are originated and signed by a department head for special equipment and by one of our storekeepers for all stock items. All requisitions indicate how shipments are to be made as to freight, express, parcel post or truck; approximate date of delivery; where to have goods shipped and the approximate price to be paid for the entire list of material on the requisition (this information for the superintendent, and the general manager).

The following description of the articles, quantity, unit, page and catalogue number from which the selection of material is made.

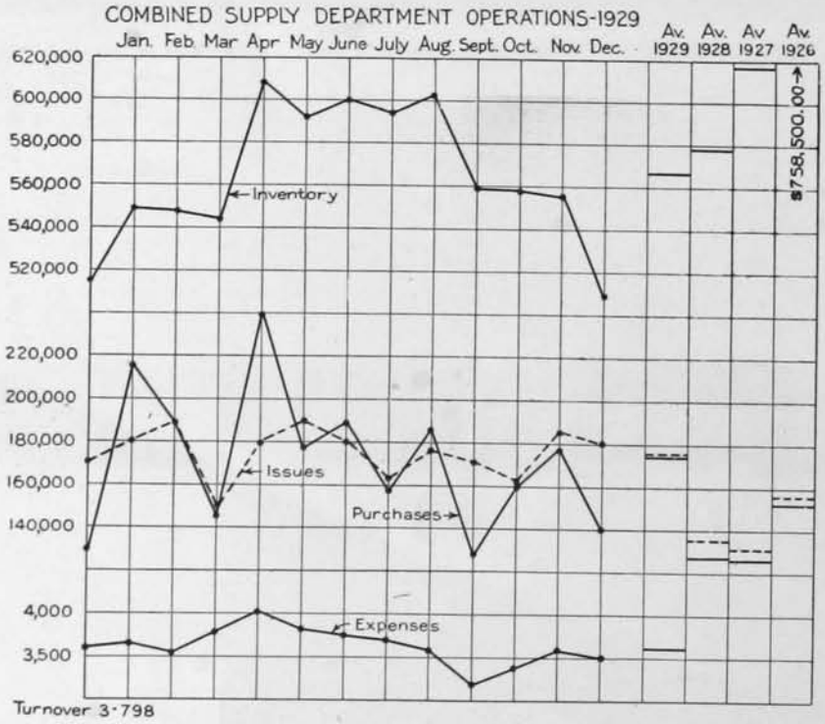
All purchase requisitions are referred to the general storekeeper before the purchase order is written as a protection against buying material which we may be able to furnish from stock at Jerome or Clarkdale or from our salvage stock department.

Methods of Storing and Issuing Warehouse Stock

Upon receipt of material at either of our supply departments (Clarkdale or Jerome), the receiving clerk checks the packages and weights, then tabulates this information on a receiving report form which gives the following information: shipper; our purchase order reference and date received; freight or express bill number or note if parcel post, and the total weight.

Each article is listed on this receiving report with the individual weight, and remarks showing just what items have been received and what ones are still enroute.

When properly filled, this report is attached to a copy of the purchase order and passed on to the invoice clerk to be checked against invoice for same. Invoices are passed for payment by the general storekeeper's office to the accounting department, and the receiving report is the basis of our entire system



United Verde Copper Company.

of stocking material and payment of same.

The Rand Visible Record system, consisting of an issue card and a receipt card is used for record of each article in stock. This record consists of approximately 15,000 articles in the combined system.

Withdrawals of stock are issued only on a supply department requisition signed by a department head or foreman who has been authorized by the management.

These requisitions show the article or material wanted, where it is to be used, the charge account or work order number, and name of party signing for same. All these slips are priced and recorded on the stock cards and from these tickets we make up a daily report showing the entire record of the previous day's business together with cumulative totals for each department or each foreman. With this daily report we aim to keep everyone interested and informed as to his material costs.

Our perpetual inventory record supplied by our stock record control, to a large degree, regulates the amount of investment in supplies and guarantees sufficient stock to meet all operating conditions.

The combined inventories of supply stocks at the close of the year 1929 was $510,000 as compared with approximately two million dollars tied up in the same class of material on December 31, 1920. The money thus tied up in inventory stock is only so many "loafing dollars," and we have had close cooperation between the operating heads and supply department in order to make the large reduction noted above. This was gradually accomplished each year even under increased production and expansion from the beginning of the year 1922.

Our stocks consist of 34 classifications including everything used by the mine, mill and smelter from bolts and nuts to steam shovel and locomotive parts. However, fuel oil, coal and lumber are not listed in the above figures as they are carried under separate accounts.

For the year 1929, just passed, our total purchases were $2,102,822.67, with a combined yearly turnover of nearly four times.

Our simple system and ingenious methods help our department perform its duties promptly and efficiently. System and methods, however, are not sufficient to account for our performance record. It is made possible only by the hearty spirit of team work within the department itself and through the utmost cooperation from the management and the various departments served.

Summary of Combined Supply Department Operation for Year 1929

To summarize the operation of our department for both Jerome and Clarkdale, we show a graphic chart illustrating how we keep posted on monthly purchases, issues and inventory. The peak periods caused by increased development programs or construction work of all kinds show for themselves in this picture. If we were on a steady operating basis without continued new construction the curves in this chart would develop into straight lines.

With the cooperation afforded our department by all concerned we are able to keep our expenses at a minimum and one of the greatest money and time savers of all is a bill of material or a list which is furnished us by our engineering departments anticipating their requirements for new work. And another advantage in planning the work together—we are in position to supply material when it is needed.

MEDICAL DEPARTMENT

(*From page* 106)

electric kitchen aid and toaster. Every endeavor is made to serve food hot, which is done by the electric dumbwaiter, capable of handling 10 trays a trip to all floors. On the second floor the majority of bed patients are cared for in four wards of eight beds each, and two private rooms. Two wards are located on the south end of the building and two on the north. On both ends are glassed-in solariums, furnished with cushioned wicker furniture. The wards are divided into clean surgical, dirty surgical, medical and convalescent. The two private rooms on this floor are adjoining, with bath between. On this floor we also have a treatment room, laboratory, chief surgeon's office, X-ray department, diet kitchen, history filing room, hopper room, porter's utility and nurses' chartroom. In the laboratory we do the routine work only. All tissue examinations and Wassermann's are sent to outside laboratory. The X-ray department is equipped with a double disc, remote control X-ray machine, a motor driven tilt table with fluoroscopic attachments, all-metal stereoscope and bedside unit. Exposures are filed in metal filing cabinets. This is a very busy department as any injury with any possible fracture, as well as chests in periodical physical examinations, are checked here.

Regarding histories, we comply with the requirements of the College of Surgeons; a complete history being filed for every case.

On the third floor we have a solarium on the south end, 11 private rooms, diet kitchen, hopper room, treatment room and a surgical pavilion on the north end. In the surgical pavilion we have a work and supply room, instrument room, sterilizing room, main surgery, scrub room and delivery room, all completely equipped. The main surgery is tiled with a light green flat tile. The lighting is from skylight and reflected light.

The top half-floor has two sun rooms on the south end, five private rooms, diet kitchen, nursery and hopper room. We place all obstetrical cases on this floor, requiring the patients to supply special nurses.

The total capacity is 52 beds and 5 bassinets. Every bed, solarium, nursery, surgery and doctor's office is equipped with silent call system and convenience outlets. An elevator of sufficient size to easily handle a bed, runs from the ambulance entrance to the top floor. It is impossible to do justice to our hospital with this short description. Our capacity is only 52 beds, which is ample to care for the community, but considering the completeness of the building and equipment, it compares with the best in any of our cities.

The staff necessary to care for this institution and work consists of six doctors, supervisor of nurses, nine general duty nurses, surgical nurse, dressing nurse and physical therapy nurse.

Infant welfare clinics are carried on both in Jerome and Clarkdale. Each clinic is conducted by a nurse with special training in this work, under the direct supervision of a pediatrician. The clinics are open for certain hours each day, the balance of the nurses' time being occupied with home calls. At the clinic, mothers are given formula instruction and taught the care of infant and child of pre-school age. Weights and heights are taken and recorded, diets given, as well as to teach them the value of sun baths and proper clothing. In the home calls, home and personal hygiene are taught, nursing service rendered, as well as formula and baby instructions.

In formula instruction they are taught the care of bottles, nipples and utensils; how to combine and boil the ingredients and the care of the formula after it is prepared.

In baby instruction, the value of breast feeding, the care of the mother to increase breast milk, the care of the mother when weaning baby, clothing, sun baths and the value of checking weights are fully gone into.

The pre-school instruction takes up the necessity of checking weights and heights, diet regulation, proper clothing and proper posture. A complete record is kept of every case that comes to the clinic. This record shows the weight and height, formula, diet, and in all cases seen by the doctor, his findings and instructions.

In home calls, wherever it is necessary, the nurse teaches the value of fresh air, sanitation and cleanliness. In the nursing service she carries out treatment as ordered by the doctor. She is always prepared to take cultures and on the lookout for any contagion, reporting any possibility to the doctor. Instruction on formula and infant care is also given in cases where the mother is unable to come to the clinic.

The object of the medical department is the care of the industrially injured, as well as to supervise and care for the health of the employe and his family. The humanizing influence of this work is a stimulus to seek additional means to further maintain the health, comfort and contentment of the employe. As to the physician in this work, a personality based on friendliness, love and sympathy for his fellow man is just as essential as ability. He must put himself in the place of the injured employe and treat him as he, himself, would want to be treated.

The properly conducted medical department of any industry has an opportunity to protect and maintain health and productiveness, and therefore happiness, of large groups of individuals in a manner almost impossible by other means or connection. This, we feel, is being accomplished by the medical department of the United Verde Copper Company.

United Verde Copper Company.

SAFETY POLICY

(*From page* 104)

ings, and in turn the key men of each department attend periodic divisional meetings. When the occasion requires, special meetings are called of such departments as may be involved in some common problem.

While excellent results are obtained in this manner, meetings in themselves are not enough. Experience has shown that mass meetings must not be held too often, lest they become tiring; and even when they are held only three or four times a year they should be varied. They are supplemented by bulletin boards and by chalkwork on blackboards which have been placed in prominent positions. Advice, good and bad practices, safe and unsafe methods, causes and results of accidents, and slogans are featured in this manner. The blackboards are changed periodically and a special effort is put forth to make them most attractive and instructive. These boards are largely relied upon to keep alive the safety consciousness between meetings.

The company is a member of the National Safety Council and avails itself of every facility this worthy organization has to offer. In addition it is a subscriber to a nationally known safety service organization. Bulletins from both these sources are used to good advantage. These bulletins and news items are again supplemented by photographs taken in the various departments and in the mine. Such pictures show safe and unsafe practices and conditions, and are very effective for the local color they give to the bulletin board. A considerable library of such pictures has been accumulated, and it is now planned to procure a stereopticon machine and show them on a screen in connection with meetings. Moving pictures are also being tried out. All literature sent out by the Arizona Industrial Commission is given wide publicity.

Both the desire to excel and the pride resulting in the success thereof are used in the furtherance of safety. Individuals and groups of men with exceptional safety records are photographed; enlargements are made and framed and displayed in prominent places at the entrance to the change room. Occasionally they are published in some periodical which is later exhibited on the bulletin board. Safety statistics are used in a similar way. Large signs, worded as indicated below, have been hung on shaft stations of the various levels and brought up to date weekly:

> ——— SHIFT BOSS
>
> This crew has worked —— days without a lost-time accident. A total of —— man-shifts.

Similar bulletins have been placed in the various departments of the smelter.

The interest manifested in these figures is surprising. It is with considerable pride that members of certain departments point to the ever-mounting number of consecutive days that they have gone without a lost-time accident.

The safety committee plan is also used. In the smelter two committees function. The inspection committee systematically inspects the entire plant for housekeeping conditions, fire hazards, unsafe conditions and practices, lighting and ventilation. A report is rendered after each inspection. The investigation committee receives reports of all accidents, even though no injury may have been sustained, fixes the responsibility, and makes recommendations to prevent the recurrence of such accidents. It also meets for the purpose of solving problems of plant safety in general, with special attention to safety equipment and practices. Each shop and operating department has one safety committee man whose duties are to correct unsafe practices among the men and report to his foreman all unsafe conditions which come to his attention. It is also his duty to accompany the foreman and members of the inspection committee when going through the department.

The Steam Shovel Department has a workmen's safety committee consisting of four men, with the department superintendent as chairman. They inspect, investigate, make recommendations, and receive suggestions.

The Mine Mechanical Department has shop committees whose duties are similar to those of the steam shovel committee. The mine is divided into districts, each district having one committeeman. The members of the mine committee make periodic inspections, accompanied usually by some supervisor, in order that official action can be taken at once on any question that may arise. They also receive and pass on to their boss or the safety engineer any suggestions or complaints from the workmen.

In order to further stimulate safety effort among the employes a safety contest, with the various mine crews as teams, was held during the period July 1 to December 31, 1929. Nineteen crews, with a total of 980 men, participated. Seven crews, with a combined total of 375 men, finished the race without a single lost-time accident. These seven groups of men worked a total of 66,198 man-shifts during the year without a lost-time accident. The contest was in the form of a horse race, each horse representing a crew. A racetrack was painted on a board 6 x 12 ft. The horses, made of cardboard and tinted, were fastened to the board with thumbtacks and were moved ahead several times each week. The reward at the end of the race was a leather belt and silver buckle with an original emblem embossed on the buckle. Much interest was shown in the race.

A safety bonus is paid to all operating bosses in the mining departments of the company. The bonus is paid monthly and is based on lost-time accidents. The jigger bosses receive a safety bonus of $5 per month whenever no man under their supervision has sustained a lost-time injury during the month. If one such accident occurs the bonus is not paid. The shift bosses are paid at the rate of 2 cents per man-shift and are penalized at the rate of 500 shifts, or $10 per lost-time accident. It is seldom that they lose their entire bonus during any month. There is, however, a further condition imposed; namely, that their accident-frequency rate for the year to date must not exceed 10 per 10,000 man-shifts. If this figure is exceeded, no bonus is paid until the rate is again below this figure.

The safety work is in charge of two men. At the smelter the safety engineer has also charge of employment; and at the mine, ventilation and mine fire are grouped under the safety department. All accident reports and records are kept by the respective safety departments of both smelter and mine. Statistics are compiled and weekly and monthly reports rendered to the department heads and the management. One technical office assistant with a fluent knowledge of the Spanish language is employed. He is required to take care of all mine safety equipment for which this department is responsible.

The realization of the increasing importance of safety in the organization is reflected in the decreasing tendency of both the severity and frequency rate in accidents. The smelter established the lowest severity rate in its history during 1929 and showed an improvement over 1928 of 31 percent in the frequency rate of all lost-time accidents. The mining department showed an improvement of 40 percent in the frequency rate of 1929 over 1928, with 52 percent of the year's total accidents occurring in the first four months.

Typical dwellings for workmen, under construction at Jerome

United Verde Copper Company.

PURCHASING

for the United Verde

By Dave Hopkins
PURCHASING AGENT

THE method and practice of purchasing for the United Verde Copper Company does not differentiate, in general, from those adopted by other large and well-organized companies. The character of the work is similar, though may differ slightly in detail and minor procedure.

At present the responsibility of purchasing is entrusted to the purchasing agent, who as the plant increases in size and production diversifies, may find that the attendant and various activities may so increase his work that the need of recommendation for specialization will be felt.

The plan of this article includes only reference to a few of the essential features in connection with the functions, responsibilities, classification of stock, and major policies of the department.

The prime requisite, prefacing a purchase of minor supplies, is the presentation of a requisition to the purchasing department, properly approved; for large and costly installations of machinery and equipment, the specific ratification on the requisition by the head of the department affected, and approval of the general superintendent and general manager.

In all matters pertaining to a contemplated purchase involving a considerable outlay of money—special or out-or-the ordinary purchases — specifications covering are prepared and sent out with a request for bids which particularly solicits information on the major points, such as price, delivery, construction, shipping weight, and point of shipment. Manufacturer's specifications on the proposed offer are also requested. The information thus received is tabulated in convenient form and, in conferences with department heads affected and officials interested, a full, free, and frank discussion covering the construction, quality, refinement, merit, and efficiency of the offer ensues. Upon agreement, recommendation is made to the purchasing agent, who is one of the conferees, and steps are immediately taken to close the business at hand. If a contract is involved, the instrument must be approved by the general manager.

All contracts covering commodities of which we are larger purchasers and consumers, such as fuel, lumber, oxygen, etc., are negotiated on as long a time basis as is possible to obtain and are approved by the general manager.

The variety of purchases needed to the safe and economical operation of a plant of this character is broad in scope and embraces all supplies, repair parts, accessories, machinery, equipment, etc., essential to the various units of operation involved, such as mining, transporting, crushing, concentrating, and smelting. A yearly compilation of orders covering these purchases will aggregate close to 5,000.

The more or less general classification following excludes machinery, equipment, and certain repair parts not carried in stock and discloses, in a measure, the variety and approximate yearly amounts of supplies, etc., purchased for stock to insure a degree of operation in accordance with managerial desires:

1. Bolts, rivets, and washers.	$51,711
2. Brick, lime, and cement..	180,621
3. Pipe and boiler tubes....	93,516
4. Rails and fittings.....	33,345
5. Rope and twine.....	44,397
6. Steel and metals.....	216,441
7. Hardware	32,982
8. Belting	26,571
9. Electric wire.....	73,272
10. Electrical supplies.....	162,735
11. Hose and fittings.....	10,968
12. Oils, waste, and greases..	14,115
13. Paints	13,956
14. Plumbing supplies.....	3,678
15. Pipe fittings.....	29,190
16. Window glass.....	708
17. Valves, cocks, brass goods.	33,828
18. Packing	15,804
19. Stationery	7,593
20. Machines and tools.....	236,145
21. Miscellaneous	38,064
22.	
23. Power drill parts.....	42,402
24. Track shifter parts.....	2,256
25. Powder, fuse, and caps..	204,000
26. Lubricators	1,260
27. Mine safety supplies.....	8,595
28. Drill sharpener and heat treater parts.....	5,727
29. Locomotive parts.....	1,149
30. Air-brake parts.....	4,254
31. Hopewell ore-car parts...	8,709
32. 50 B electric shovel parts.	29,373
33. Nordberg underground shovel parts.....	1,308
34. Sundries	13,809
35. Gears and pinions.....	18,108
36. Fuel (coal, Diesel, 14 deg., coke)	933,020
37. Lumber (Jerome, Clarkdale)	284,300
38. Lime rock.....	2,100
Total.....	$2,880,010

Coordination with all departments is necessary, as the responsibilities of maintaining the safe, economical, and efficient operation of the various units are delegated to officials accountable to the general superintendent, who, in turn, is answerable to the management. Therefore, a close contact is maintained with all departments, in that the purchasing end may at all times be properly informed of their respective needs.

A fairly complete catalogue library is maintained and is at the disposal of any employe of the company. Treatises of a technical nature are, upon receipt, placed immediately into the hands of those interested.

The actions of the purchasing department must mesh with the various activities of the operating and construction department, else results not conducive to economy in operation and speed in construction are obtained. The department, therefore, occupies the position of a service department and stands in readiness at all times to devote its activities in behalf of any specific unit or general plant welfare.

The purchasing agent is "held responsible for effective buying, and his accountability ceases where his authority ends." Under this condition, "there is danger that he will take a restricted viewpoint of the organization's interest," in that his perspective is encompassed within his own department. This tendency is not conducive to the best interest of the company, as practically every activity of the purchasing department concerns the organization as a whole.

The reputation of the company among vendors depends largely upon the conduct of the purchasing department. The regard and estimation of any firm depends to a considerable degree upon the conduct of those with whom contact is made, therefore an investment of peculiar trust is reposed in the department. The successful protection of this trust depends upon gaining and retaining the good will of the vendor, which is some times difficult under certain circum stances, yet the department must be fully prepared at all times to protect the interests of the company in all dealings with outsiders. The aim is toward the cultivation of effective and pleasant relations with salesmen, as this happy connection is recognized as being of tangible worth at all times, and unless consistently nourished is very likely to be found wanting when most needed.

The general policies of the department are similar to those of its associate departments, and special attention is given throughout the organization to the creation and maintenance of good will in external relations and protection of the company's interests.