SAND FILLING AT THE COPPER QUEEN

by

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"SAND FILLING AT THE COPPER QUEEN"

In 1877, a scout attached to the U. S. Cavalry located a mining claim on a showing of oxide copper ore just a few feet south of the present route of U. S. Highway 80 as it circles the business district of Bisbee. Since that time, 40 shafts have been sunk, nearly 1,000 miles of crosscuts driven, and well over a billion dollars worth of copper, lead, zinc, and precious metals extracted from a mineralized zone about 9,000 feet wide by 13,000 feet long.

Today, the Copper Queen Branch of the Phelps Dodge Corporation employs some 750 men in its underground mining operations at Bisbee. Three production shafts, the Campbell, Cole, and Dallas, service over 100 stopes currently being mined from the 500 Cole level in the southwest to the 3100 Campbell level in the northeast. Several other shafts are used for pumping and ventilation.

Orebodies are intermittent, and vary in size from a few hundred tons to - in exceptional cases - over a million, with the majority in the 20,000 to 30,000 ton category. The host rock is the Paleozoic limestone, which dips 20° to 30° to the northeast, and the preponderance of ore occurs as a replacement of the lower Escabrosa (Carboniferous), the Martin (Devonion), and the upper Abrigo (Cambrian) limestone beds.

The shape of the orebodies is nearly always influenced by the bedding of the limestone which they replace. Deposits in the thin, shaly beds of the Abrigo are practically always tabular in conformance with the limestone beds. In the Martin, with its dirty but more massive beds, the deposits are usually bedded but have greater relative thickness than in the Abrigo. Deposits in the clean Escabrosa limestone are usually thick and massive, with a tendency for the vertical dimension to be greater than the horizontal, but even they plunge with the bedding, although normally at a greater angle.

The type of limestone in which the deposits occur also influences the ground conditions for mining purposes. Orebodies found in the Abrigo and Martin limestones usually require some timbered method of mining due to the poor bond between the beds along the shaly partings, whereas the thick, massive orebodies of the Escabrosa usually stand better and thus permit some open type of mining.

At the present time, some 87% of stoping is by square-set methods, and the remainder by open cut and fill. Ore is broken by holes drilled with push feed machines, and blasted with prilled AN-FO. Muck is transferred to chutes by air or electric slushers, and transported to the shaft pockets by rail haulage. Stope miners work under an incentive system, with standards, in terms of tons per man shift, based on the type of mining and conditions in the stope. Exploration and stope development programs require the drilling of 7,000 to 8,000 feet of diamond drill holes and the driving of about 2,250 feet of drifts and raises per month. Although the mine is not wet, several fault areas make about 4000 gpm.

Sand filling at the Campbell shaft began May 1, 1964, as a result of production efficiency studies made during 1963. The usual factors of faster filling, better ground support, and a reduction in the amount of raising and drifting required by the use of development waste for stope backfill, recommended the change to hydraulic fill. The fact that Campbell development was rapidly moving to levels below current stoping, so that some supplemental source of fill material must soon be made available, was also an influential consideration.

Compared with many other mines using hydraulic sand for stope backfill, the Copper Queen enjoyed a number of advantages: a plentiful supply of sand due to tailings from an 18,000 ton per day concentrator, the existence of relatively little-used service compartments in each shaft for installation and maintenance of sand lines, and stoping areas within a reasonable distance of the shafts yet sufficiently far from the pump stations to permit sand and slimes carried in water draining from the stopes to settle in the ditches before reaching the sumps. Possibly the major drawback to the use of sand fill in Bisbee was the erratic nature of the orebodies. Although some similarities exist, no two stopes are alike, and to obtain the maximum benefit from hydraulic fill, supervision must plan the work much more carefully than would be the case where regular orebodies allow the development of standardized mining and filling schemes.

By the end of 1964, it had become evident that the change to hydraulic fill at the Campbell had been worthwhile. Although some time was required to teach bosses and miners the new techniques, efficiency in those stopes which had passed through at least three mining cycles using sand for back fill had increased 13%. Raises and drifts for waste fill had been eliminated, as experience had shown sand could be placed as much as 120 feet above level lines without difficulty.

A good example of the support provided by hydraulic fill was given by an area on the 2700 Campbell which had been extensively stoped by a combination of square-set and cut and fill methods, and required the constant services of two repair crews merely to maintain access for ore haulage. Two unused raises were bulkheaded, a filter wall built along one end of the area, and 5,500 tons of sand pumped to tighten the old gob. No further repair work has been needed in this section.

Experience in this area and others has shown that from 30 to 60 tons of sand are required per 100 tons of original fill to tighten old gobs in stopes which have not caved.

At the Copper Queen, particularly strong emphasis has been placed on careful and thorough preparation of stopes for fill, not only because sand spilled in the drifts is an expensive nuisance, but so that once filling is begun in a stope, it may be concluded as rapidly as possible and the sand crew moved to the next working place.

Although an effort is made to see that several stopes are preparing for fill at any given time, precise scheduling is not practical due to differences in crew experience and work rates, the complexity of most orebodies, and the intricacy of some bulkheads. Often, to avoid elaborate bulkheading, an attempt must be made to completely mine out a section of indeterminate size before filling. However, the system has been designed to meet occasional abrupt increases in sand requirements. Considerable storage capacity is available, and sand lines are of sufficient size to pass sand at high rates without greatly accelerating wear.

The rate at which sand is run underground is controlled by manually-operated rubber sleeve valves located at the surface tank discharge. In stopes with limited drainage, or with only a small area to fill on each mining floor (so that turbulence created by the sand entering the stope tends to prevent settling), rates may be held to as little as one dry ton per minute. In larger stopes, the sand may be allowed to run at rates up to 2.5 tons per minute.

With the exception of a few serious breakouts involving considerable tonnage, all of which could be traced to errors or oversights in bulkheading, the average amount of spill per stope has not exceeded one-half of one percent of sand run.

It is now felt that any stope can be sand filled provided the cost of preparatory work is warranted under the circumstances. For instance, a stope on the 2833 level of the Campbell in heavily crushed and broken ground could only be filled by completely lining the four sides and bottom with burlap. Although preparation of this section required a week's work and considerable burlap, the alternative of filling with gob would have involved at least five weeks work by four men to repair an old gob raise and three weeks work by two men to reopen a drift on the level above.

The encouraging results obtained from sand filling at the Campbell led to the decision to extend the new method to the other production shafts. The Dallas system, servicing the lower Cole levels as well as the Dallas mine, began operation in May of this year; the Cole system, supplying sand to upper levels of that shaft, was placed in service during September.

Sand Preparation

In the sand preparation plant located at the collar of the Campbell shaft, 800 gpm of concentrator tailing at 45% solids are deslimed and partially de-pyritized to produce about 65 dry tons of sand per hour.

Although the character of the mill tailing is primarily dependent on the area currently being mined in the pit, and may vary from a pulp high in slimes and difficult to process for sand to a relatively coarse material readily amenable to treatment, the average screen analysis is as follows:

Mesh	%
+ 65	12
+ 100	10
+ 150	8
+ 200	9
+ 325	13
- 325	48

After two-stage wet cycloning, the screen analysis is approximately as follows:

Mesh	%
+ 65	16
+ 100	13
+ 150	11
+ 200	12
+ 325	13
- 325	35

The percolation rate of this deslimed sand varies from 6 to 9 inches/hour.

Pyrite in the tailing averages about 18%, but on a day-to-day basis may range from as low as 4% to over 30%. To avoid a possible generation of heat and SO_2 by oxidation of pyrite in the fill, it was felt the amount of pyrite in sand placed underground should be limited to a maximum of 14%, with an average of no more than 10%. On the other hand, it is not desirable to eliminate all pyrite from the sand since in time it will tend to cement the fill. For these reasons, pyrite is removed from only a portion of the pulp handled by the treatment plant.

About 1/5 of the normal flow of mill tailing, passing through an 18" diameter concrete pipe within 50 feet of the shaft on its route to the settling ponds, is pumped directly to a distribution manifold supplying three 10" hydro-cyclones. Overflow from this first stage of desliming, at 25% solids, is returned by gravity to the mill tailings line a short distance downstream from the plant intake. First stage underflow, at 70% solids, falls through 3/8" mesh screens into a sump, where it is diluted to 50% solids. A frother and a collector are added to the diluted underflow from two of the cyclones, and this pulp flows by gravity to a 4-cell flotation machine with the intermediate cell partitions removed, for pyrite recovery. Pyrite concentrates, at 40% solids, and with an average assay of about 0.80% Cu and 87% FeS_2, are pumped through a 3" polyethylene pipe to one of two settling ponds about 400 feet south of the plant.

Underflow from the third first-stage cyclone bypasses the flotation machine to combine with the flotation tailing. This pulp is diluted to 25% solids, is passed through a 3/8" mesh screen, and is pumped to three 10" hydrocyclones atop a 15' diameter by 20' high steel storage tank. Underflow from the second stage of desliming, at 70% solids, falls into the tank where it is held in suspension by a 60" diameter, 6-blade, 150° pitch, rubber-covered, turbine-type propellor, rotated at 64 rpm by a 30 hp motor.

A flocculant is added to the second stage overflow, which, at 10% solids, is transferred by gravity to a 30' diameter by 15' high steel tank. Here slimes settle toward a central cone in the tank bottom from whence they are discharged at 35% solids to join the first stage overflow for return to the tailings line. Clear water overflowing the tank rim is caught in a full peripheral launder and conducted to an 8,000 gallon surge tank, from which it is pumped to an 800 gallon head tank 40' above the plant. The head tank supplies an 8" pipe header running the length of the plant just under the peak of the roof. Use of a head tank and large header prevents a minor adjustment in dilution water to one of the sumps from being reflected in flow to the other sumps.

With the exception of pump gland water, sufficient water is reclaimed to completely supply plant needs.

Piping arrangements permit recirculation of pulp from the sand storage tank to pumps feeding the second stage cyclones so that the extremely slimy tailing occasionally encountered may be passed through a third stage of classification.

A 6" thick reinforced concrete pad was placed in the bottom of the sand storage tank to serve as a wearing surface, and to date has shown no deterioration. To eliminate a rhythmic surging of the pulp in the tank in tempo with the propellor rotation, four 1/2" thick by 15" wide vertical steel fins, or vanes, were bolted to the inside of the tank 90^{o} apart. Twelve 1/2" inlets for compressed air are spaced around the bottom of the tank to provide agitation in case of power failure or if the agitator must be stopped with a tank full of sand.

Reagents are hauled from large mixing and storage tanks at the concentrator to 1200 gallon ground-level storage tanks at the sand plant, and are forced from these tanks up to rotary bucket-wheel feeders in the plant by air pressure. If pH of the mill tailing reaches levels detrimental to pyrite flotation, barren tail water from the nearby precipitation plant with a pH of about 2.7 is added to the flotation machine feed as required.

Shaft Lines

Shaft sand lines are victaulic-coupled 4" Schedule 40 steel pipe lined with 3/8" thick rubber of 45 Shore A Durometer hardness, and were installed exactly on plumb using shaft plumbing equipment and a template. The majority of joints are 21' - 0" long, but a sufficient number of 10' - 6" and 5' - 3" increments were used to permit exit on the different levels without use of special lengths. Each joint is separately supported by at least one heavy clamp-type hanger welded into place so that replacement pipe will also be plumb.

At about 152' intervals in the Campbell and Dallas shaft lines, restricting orifices lined with an abrasion-resistant ceramic were installed (Figure 1). These orifices eliminate excessive velocities in the shaft columns by converting one long drop into a series of short free-fall full-flow steps. Periodic inspections have shown no appreciable wear to date in either the rubber-lined pipe or ceramic orifices.

Sand is routed from the shaft lines to main distribution lines on the various levels by removing a short length of shaft pipe and swinging a rubber-lined ell feeding the level line into its place.

Level Lines

Over six miles of 3" Schedule 40 plain end pipe were required to complete the primary network of level sand distribution lines (Figure 2). Level lines are installed in 5', 10' or 20' lengths; curves are made with standard 36" radius sweeps, bent from 3" Schedule 40 pipe to 5°, 15°, 30° and 90° angles in the pipe shop. The plain end couplings used are rated at 12-1/2 tons end pull. Each joint of pipe is individually supported by a clamp-type hanger to facilitate replacement.

Usually a "wye", controlling the direction of flow by simple plug valves, is installed in the sand line in or near the area being filled to allow diversion of the flush water run before and after every pour of sand. Although a "wye" may not always be needed in some of the larger stopes, its use will minimize sand loss in the event of a serious leak.

Whenever possible, sand is conveyed from the main distribution lines to the stope being filled through 100' lengths of flexible polyethylene pipe connected with victaulic couplings. This plastic pipe is rated at 50 psi working pressure, but pressures as high as 70 psi have been handled without incident providing additional couplings are placed over the pipe about 6" on each side of a connection and the entire joint wired together.

An effort was made to reduce pipe wear by particular attention to four factors:

1. Elimination of foreign material and tramp oversize in the sand.
2. Retention of sufficient slimes in the sand for pipe lubrication.

3. Running sand at the highest practical density. (Of course, this also increases the tonnage of sand delivered at a given velocity and decreases the amount of drain water to be handled).
4. Positive control of line velocities.

Although some portions of the level lines have passed more than 40,000 tons of sand, no pipe has worn out to date. In fact, the only four instances of pipe failure in the level lines could all be attributed to improperly made connections with the pipe failing to butt tightly.

Stope Preparation

To prepare a stope for sand fill, the mine foreman first marks the location of the filter bulkheads required. Then, side lagging are removed and loose rock cleaned from the stope walls at their juncture with the bulkhead.

Typically, filter bulkheads to contain sand in the stopes are built by nailing 2" x 10" lagging, gapped about 3", between the 8" x 8" or 10" x 10" stope posts. A 7-1/2 ounce burlap, purchased in 54" wide by 100 yard long rolls, is stapled over the lagging in loosely-draped horizontal strips working from the bottom of the bulkhead upwards and with each ascending layer lapped about 4" over the one beneath.

Gaps between the lagging are staggered on adjacent sets to permit as nearly continuous drainage as possible. If the filter bulkhead is less than two sets wide, 3" x 10" lagging, with 10-1/2" gaps left between the individual lagging, will be placed vertically between caps or girts and covered with light poultry fencing before burlapping. When filling has been completed, additional 3" x 10" lagging will be inserted in the gaps.

The sides of a bulkhead are sealed to the stope walls by tightly fitted 2" boards trimmed to approximately fit the ground contours. Since no drainage is desired at the edges of a bulkhead, a roll of burlap will be wedged into the crevice between the ends of the boards and the ground, and the entire joint covered by an additional double thickness of burlap.

Wherever caps or girts enter the bulkhead from the area to be filled, they are wrapped loosely with short pieces of burlap which are ruffled back against the bulkhead before the horizontal strips of burlap are applied.

If the section to be filled lies over a previous sand fill, a trench about 15" wide and 24" deep is dug down in the old sand and the bottom layer of burlap brought down into and across the trench and covered with loose sand.

If the section to be filled lies over solid ground, a trench about 24" wide will be dug along the inner toe of the bulkhead until a solid footing is exposed. The bottom layer of burlap is brought down across the trench and its edge glued to the ground with an inexpensive waterproof plastic cement. If the bulkhead must be erected at the edge of a raise, the trench will usually be dug a set or two out in the stope and burlap will be carried from the toe of the bulkhead across the floors to reach the trench.

Occasionally, to avoid extensive bulkheading or to obtain a tight fill in a stope which is caved, one or more extraction raises will be sealed with solid timber bulkheads 8" thick and covered with a double thickness of burlap. Water is drained from the fill through 10' sections of 12" diameter 14 gauge steel pipe, with 2" holes punched in it on 3" centers, which are wrapped in a single thickness of burlap and erected over an opening in the raise bulkhead. Brief runs of sand are made until the bulkhead is covered to a depth of 15 to 20 feet, then full-scale filling is resumed.

Leaks

Minor leaks encountered while running sand are usually stopped by thrusting patches or wads of burlap into the leaking area; major leaks may require that sand be stopped until the necessary repairs can be made.

Occasionally, stope sections in broken ground have been particularly difficult to fill due to inaccessible leaks in the hanging wall. In such a case, shredded rock wool insulation scattered on the surface of the pulp as it drains toward the leaks is drawn down to seal crevices and cracks. At one time cotton gin refuse was used in a similar manner, but since that material is highly inflammable, it has been replaced by the rock wool.

Communications

Dependable communications between the surface plants and the stopes being filled are vital to satisfactory sand filling. At the Campbell, the regular mine phone system was extended into the stoping areas, and an additional 3-wire network servicing powered signal horns was installed in parallel to it. Portable telephones and horns in the stopes are connected to the basic network with extension cords.

At the Cole and Dallas, telephone and horn signals are carried by a network of salvaged 21-conductor cable. Several of the idle circuits in this cable will be used for transmission of remote control signals when the surface facilities are automated.

Running Sand

Before starting sand underground, the surface plant operator admits compressed air at about 15 psi to the sand lines for a few minutes. Reception of this air at the stope end of the line is evidence that the line was adequately flushed following the previous pour. While waiting for a report from the stope, the operator injects an air-water spray to repulp settled sand in the tank bottom cone discharge just above the main sand valve, and checks and records density of the pulp in the storage tank. As sand is run at near critical densities an accurate determination is important.

To begin running sand, the operator flushes the line with water for from 1 to 4 minutes, depending on the distance from the shaft to the place being filled, in order to wet the line and purge any sand left from the last pour. He then opens the sand valve and closes the flush water valve. Sand is always run in a closed system with no outlet to atmosphere, and pressures at the top of the shaft line normally range from 15" to 25" Hg vacuum.

To stop running sand underground, the operator opens the flush water valve, then closes the sand valve. Flushing is continued for from 1-1/2 to 5 minutes, and is followed by compressed air for from 5 to 8 minutes to break the vacuum in the shaft line and to force the plug of flush water through the level lines.

Sand Trucks

Sand fill is transported from the preparation plant at the Campbell shaft to the surface storage facilities at the Cole and Dallas shafts in tank trucks built from 25-ton muck trucks no longer used by the pit department. To convert a pit truck to a tanker, the bed was removed, the frame lengthened and drive line extended, and a steel tank, W-shaped in cross-section, and approximately 15 feet long, 8 feet high, and 10 feet wide was installed. The W cross-section was resorted to after a V-shaped section used in early experiments resulted in an excessively high center of gravity once the tank sides had been steepened sufficiently to shed sand.

Pulp, at 68 to 70% solids, is held in suspension in the tanks by compressed air passing through slitted hose jets spaced at 15" intervals lengthwise along the double-bottoms of the tank. Compressed air is furnished by a six-cylinder industrial gasoline engine with three cylinders compressing air and three cylinders providing power, which is mounted on one side of the truck. The compressor is rated at 87.5 CFM at 100 psig at sea level, but in service actually supplies 150 CFM at 20 psig.

Sand trucks are loaded with about 38 tons of pulp, or an equivalent 26 dry tons of sand, at the Campbell plant in about 2.5 minutes through a spout extending from the side of the sand storage tank over a manhole in the top of the truck tank. The 1.3 mile haul to the Dallas tank, against a 260' rise in elevation, requires about 8.5 minutes; the 2.1 mile haul to the Cole tank, against a 320' rise in elevation, requires about 11.0 minutes (Figure 3).

Trucks discharge through a pair of 6" manually-operated rubber-lined valves at the rear of the tank in about 3.5 minutes. The last few tons of sand are flushed from the trucks by water jets permanently installed in the forward end of the tank. Flush water is obtained from a quick-coupling hose connected to the plant supply. The amount of water added in flushing is just about sufficient to balance evaporation losses and reduce the pulp density to the 68% solids normally run underground at the Cole and Dallas.

Cole-Dallas Surface Plants

Surface facilities at the Cole and Dallas shafts are generally similar: one or more large storage tanks with a service tunnel underneath which contains the "Xmas tree" valving for admission of sand, flushing water, or compressed air to the sand line leading underground (Figure 4).

The upper Cole tank has an effective capacity of 460 dry tons of sand; the lower Cole tank, 240 tons; and the Dallas tank, 300 tons. The three tanks are connected by piping which permits transfer of sand by gravity as required; thus up to 700 tons of sand are available at the Cole and 1,000 tons at the Dallas without operating the sand plant or hauling sand. Adequate storage capacity is important, since mill shutdown days do not coincide with those of the mine, occasionally both sand trucks may be temporarily out of service, and from time to time the total sand production of the treatment plant is needed at the Campbell.

Sand is held in suspension in the storage tanks at the Cole and Dallas shafts by air agitation. Air is distributed through a circular 4" pipe header to 12 individually regulated manifolds, each supplying from 7 to 9 air jets positioned on 24" centers across the bottom of the tank. These jets or vents are made from 6" lengths of scrap 1" air hose, with one end plugged, and a short 3/4" pipe nipple inserted and clamped in the other end (Figure 5). A 3/4" long slit in the center of each hose opens when air pressure is applied and closes when pressure is released, so that sand cannot filter back into the manifolds. With this arrangement, agitation may be stopped when fill is no longer needed underground; the tanks are refilled, and when fill is again required in the mine, the settled sand may be repulped in 20 to 30 minutes.

To avoid an undesirable drain on the regular mine supply, compressed air for agitation in the Cole and Dallas tanks is provided by auxiliary 600-CFM electrically-driven compressors mounted on rotary drills which have been retired from active service in the pit.

Determination of pulp density with this type of agitation is not quite as accurate as with the mechanical means used at the Campbell plant, so the density of sand sent underground at the Cole and Dallas is normally held to a maximum of 68% solids.

Personnel

Mine timbermen have been trained to operate the various phases of the sand system. At the Campbell, one timberman is stationed with the two regular stope miners in the place being filled, while a second timberman operates the preparation plant and regulates the flow of sand underground. Timbermen are employed in a similar manner at the Cole and Dallas shafts, but the surface plant operators at these divisions will soon be replaced with automatic controls, whereby a signal from the sand man underground will initiate the proper sequences of compressed air, water, and sand flow.

Metal and plastic sand pipes are run from the main distribution lines into stopes preparing for fill by the two timbermen during lulls in the filling program. Stope preparation and cleanup of any sand spilled are almost always done entirely by the regular stope crews. Exceptions to this general rule include filling around old fire areas where the workings are prepared for fill by repair crews, and filling stopes which have been completed. In the latter case, stope crews are often moved immediately to new working places, and the abandoned stope is prepared and filled by the sand men as time permits.

One shift boss is assigned full time to supervise pipe installation, oversee treatment plant operation, aid in instructing other shift bosses and miners in the proper methods of stope preparation, and coordinate production and delivery of sand to meet requirements of the various shafts.

A second shift boss is also used, when available, to aid in the instruction of new miners in stope preparation.

Through November 1, 1965, 120,000 tons of sand had been placed in 168 stope sections. Recent consumption of sand has averaged 13,500 tons per month.

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(NOT FOR PUBLICATION)

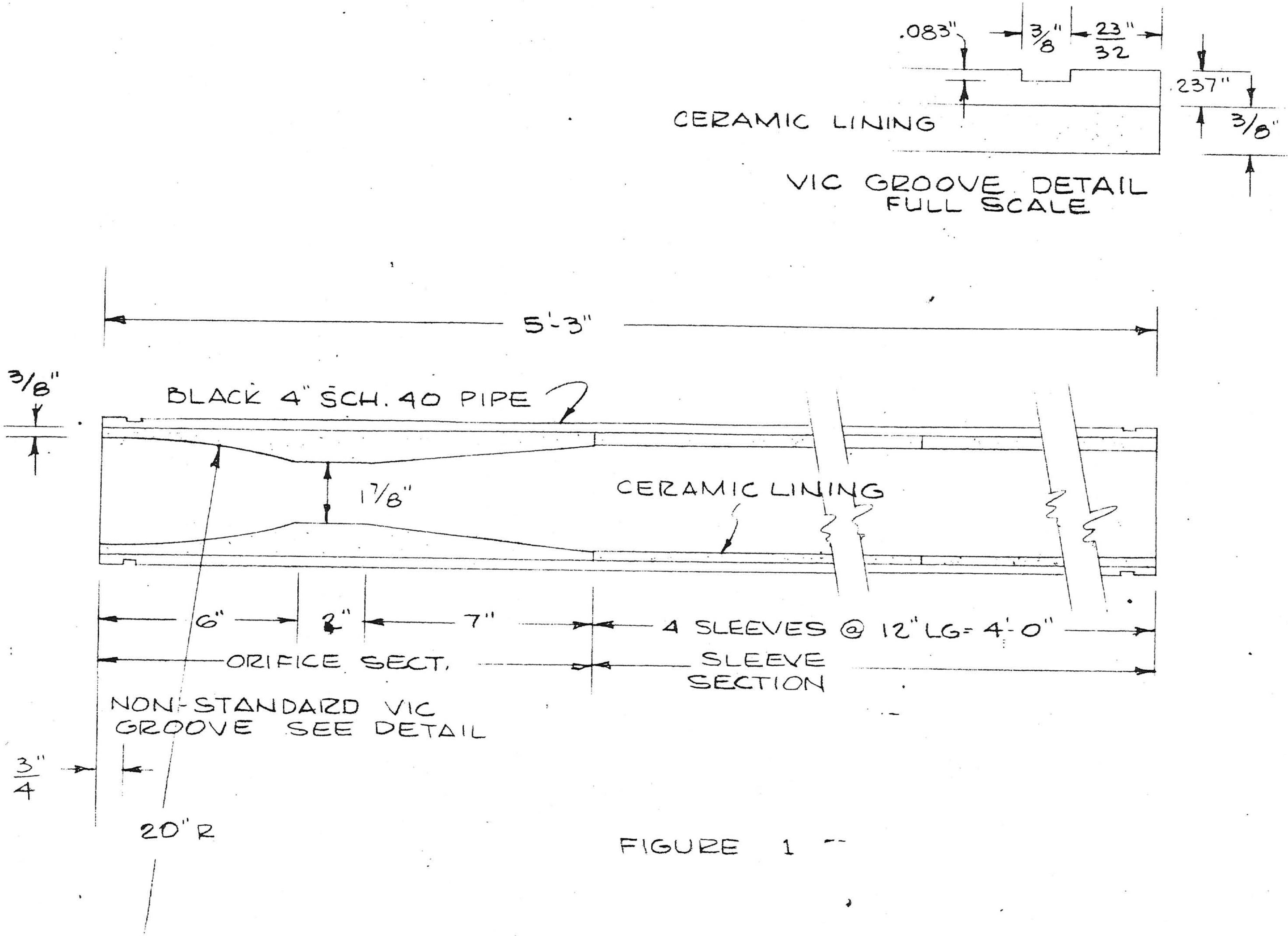

.083"
3/8"
23/32"
.237"
3/8"
CERAMIC LINING
VIC GROOVE DETAIL
FULL SCALE
5'-3"
3/8"
BLACK 4" SCH. 40 PIPE
1 7/8"
CERAMIC LINING
6"
2"
7"
4 SLEEVES @ 12" LG = 4'-0"
ORIFICE SECT.
SLEEVE SECTION
NON-STANDARD VIC GROOVE SEE DETAIL
3/4"
20" R

FIGURE 1 --

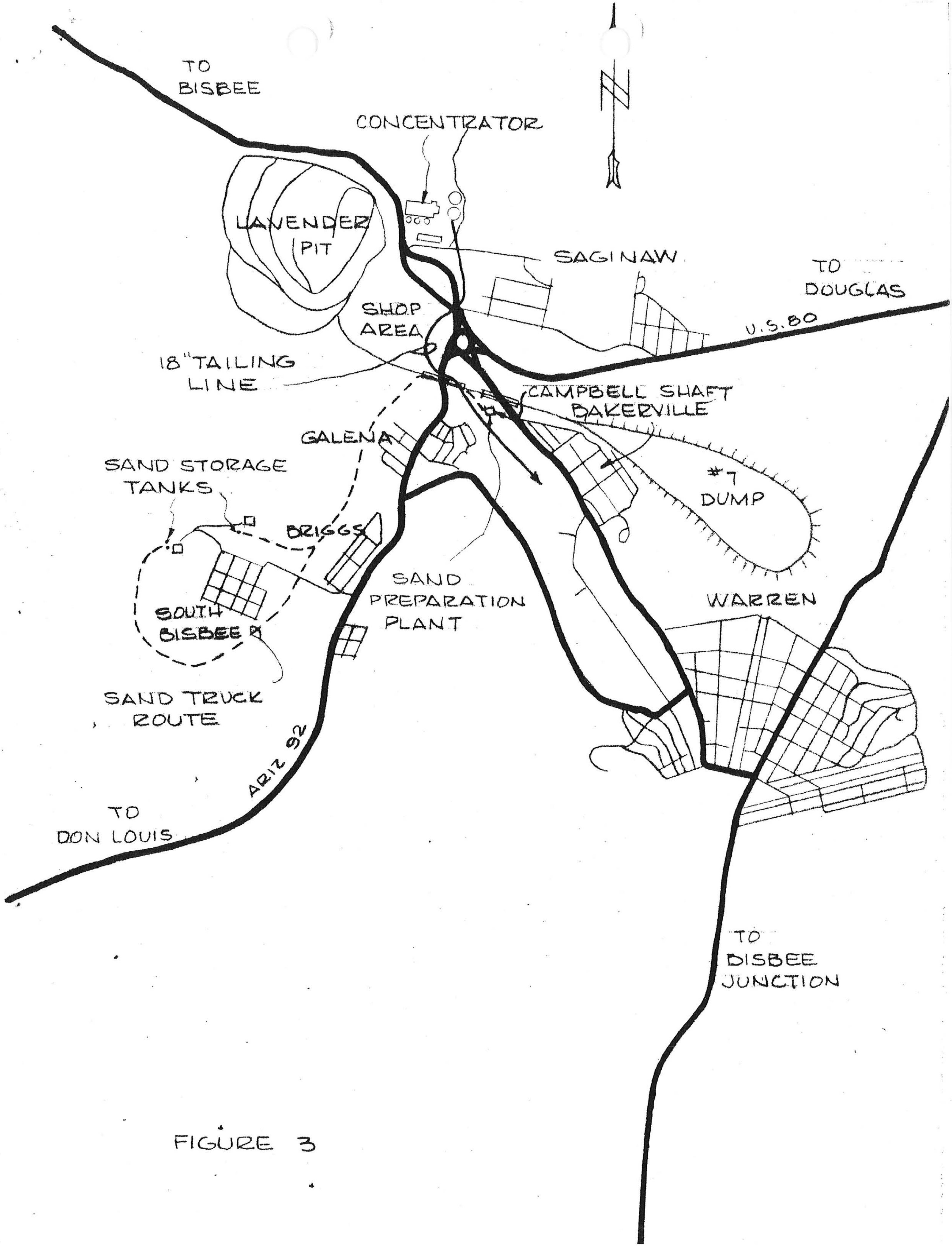

TO
BISBEE
CONCENTRATOR
LAVENDER
PIT
SAGINAW
TO
DOUGLAS
SHOP
AREA
U.S.80
18" TAILING
LINE
CAMPBELL SHAFT
BAKERVILLE
GALENA
SAND STORAGE
TANKS
#7
DUMP
BRIGGS
SAND
PREPARATION
PLANT
WARREN
SOUTH
BISBEE
SAND TRUCK
ROUTE
ARIZ 92
TO
DON LOUIS
TO
BISBEE
JUNCTION

FIGURE 3

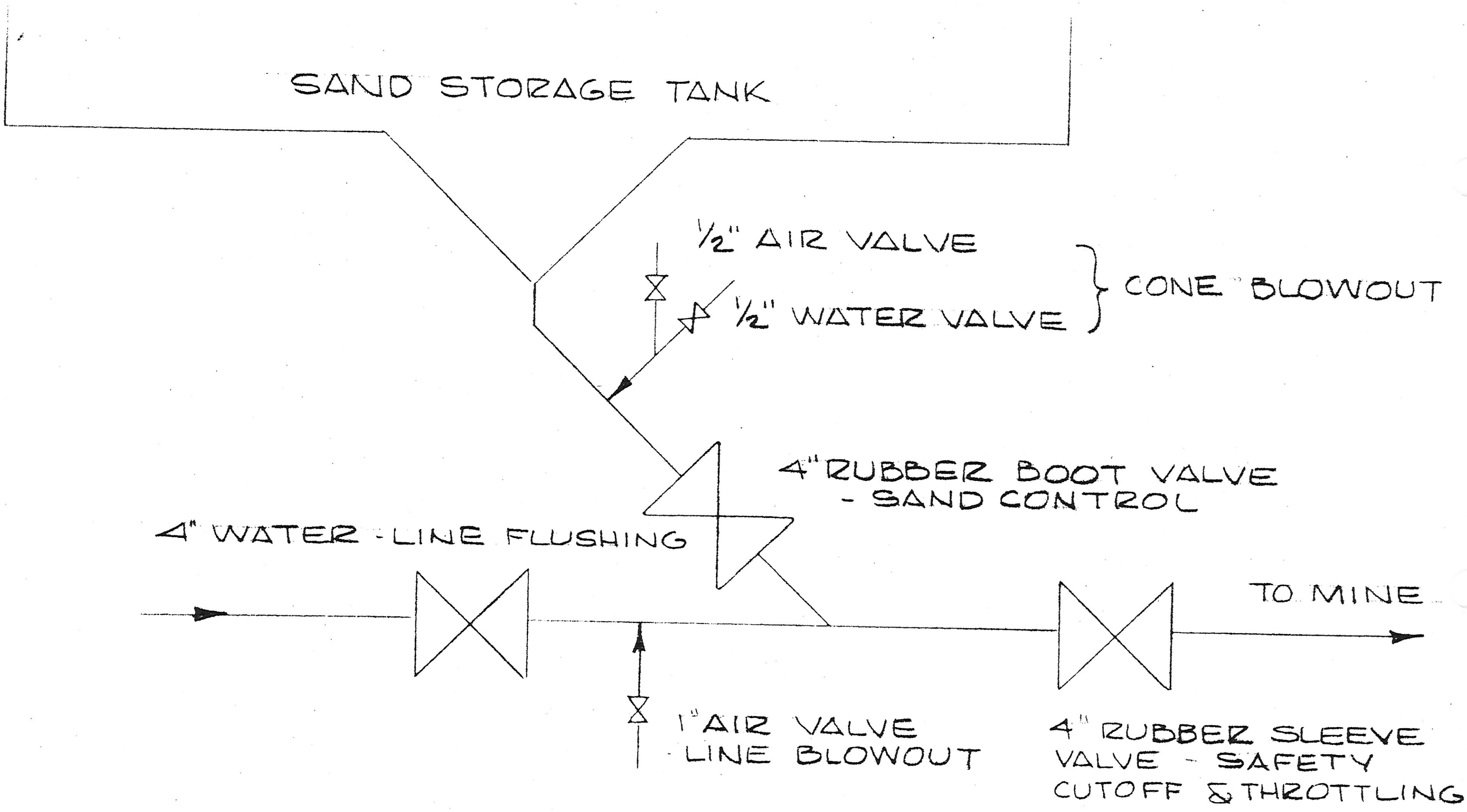
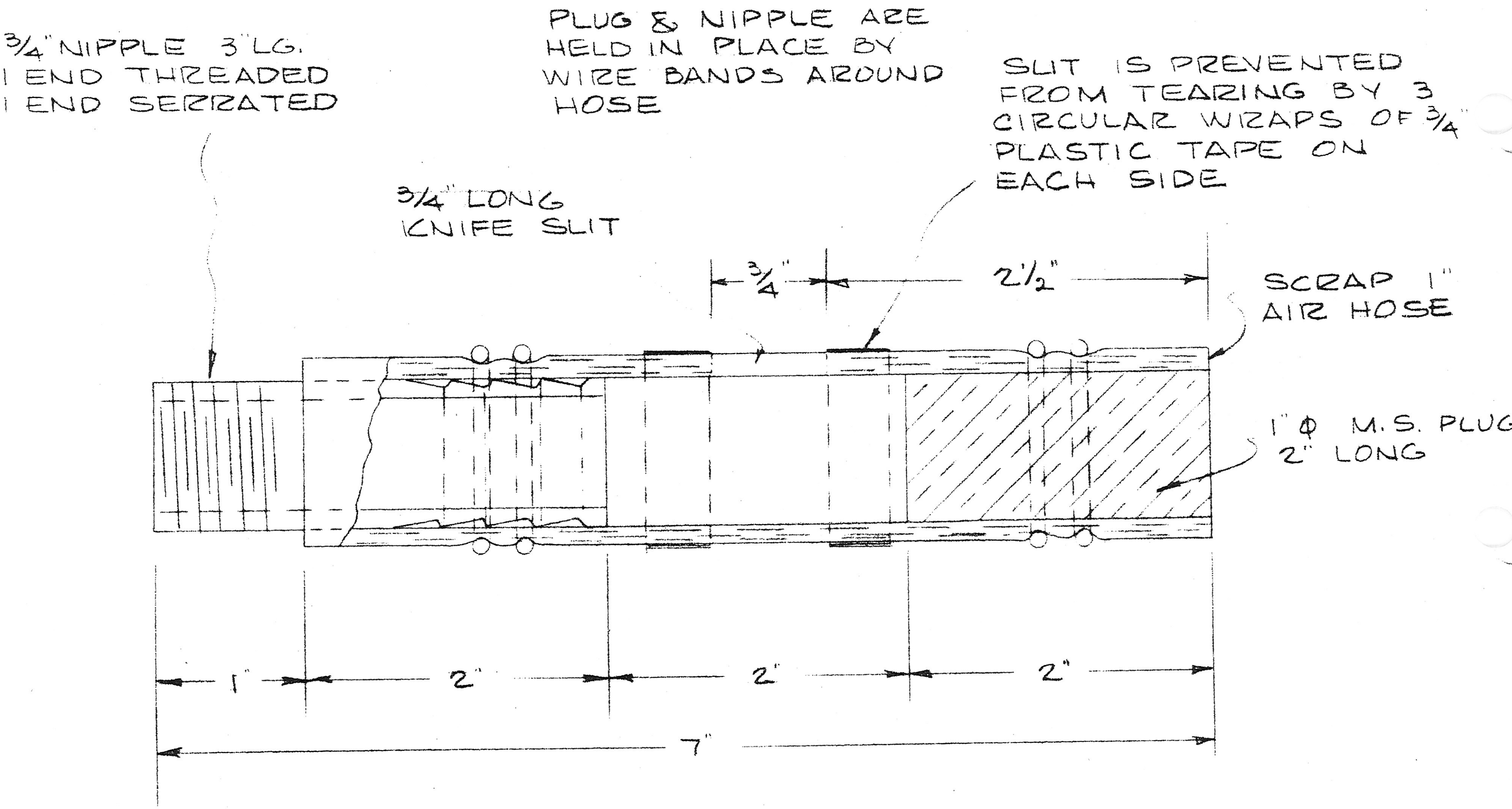


FIGURE 4

4" RUBBER BOOT VALVE OPERATED
BY COMPRESSED AIR AT 90 PSI

COMPRESSED AIR TO BLOWOUTS
REDUCED TO 60 PSI



STANDARD HOSE JET

FIGURE 5

The American Mining Congress — RING BUILDING WASHINGTON 6, D. C.

AUTOMATION AND ELECTRONIC CONTROL AT THE COPPER QUEEN CONCENTRATOR*

By

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SUMMARY

This paper will briefly discuss the instrumentation and control equipment in use at the Copper Queen Concentrator. With mining men everywhere searching for new and better ways of increasing production efficiency and reducing costs, it is to be hoped that some of the material in this paper may prove useful to that end.

The following equipment will be described:

1. Dumping control devices at Primary crusher.
2. Closed circuit television.
3. Automatic Tripper car.
4. Pulse feeding of reagents.
5. Miscellaneous installations.

In gradually improving operating efficiency through Automation, Instrumentation and Control, progress has been slow but deliberate. Each installation has had to prove itself before another is considered and although much advance thinking is done prior to finalizing ideas, actual fabrication is delayed until the previous installation has been completely tested and proven sound.

1 - DUMPING CONTROL DEVICES AT THE PRIMARY CRUSHER

The primary crusher is a 48" double discharge gyratory and delivers the crushed ore directly onto two pan feeders by means of short chutes. New feed is brought to the crusher in trucks of either 25-ton or 35-ton capacities. Traffic signals at either end of the crusher building control dumping. When the signal light at the end of the crusher building is green, the truck backs in and elevates the bed to discharge the ore. When a red light is showing at this point, the truck driver understands that he is to remain away from the dumping position until signalled by either an amber or green light. The amber light allows the driver to back in to dumping position but elevation of the truck bed must wait until the signal is green.

* Presented at the 1961 Metal Mining and Industrial Minerals Convention, American Mining Congress, Seattle, Wash., September 10-13, 1961.

Cover page for reference

Cochise Co.

THE STORY OF BISBEE AND THE COPPER QUEEN

From the time when the Bisbee (or Warren) District began to produce ore (about 1880) until the end of 1951, the district had mined almost 55 million tons of ore containing over 5½ billion pounds of copper, 50 million dollars worth of gold, 56½ million dollars worth of silver, over 300 million pounds of lead and over 360 million pounds of zinc, or a grand total of mineral wealth amounting to over one billion dollars. This would indicate an average value of 20 dollars per ton of ore, which puts the district in the bonanza class, as distinguished from the low-grade porphyry copper districts with their three to four dollar ore.

Since 1951, the Copper Queen underground mine has produced 2,852,574 tons of copper ore from which 297,319,674 pounds of copper have been recovered. This copper at the prevailing price of the metal, had a gross value of about $96,600,000. The value of gold and silver by-products, and of some lead and zinc produced in 1952 and 1953, would bring the gross value of Bisbee's production (not including the new production from the Lavender Pit, which amounted to 11,174,020 tons of ore and 157,225,506 pounds of copper) to about $100,000,000 for the five year period since 1951. The higher price of copper during this period brought the gross value of the new ore to about $35 per ton.

To be sure, the net profit to the operators in producing this mineral wealth is only a small fraction of the gross metal value of over 1.1 billion dollars. Such gross value was only attained after many processes outside the State of Arizona were performed and the final product transported to markets.

History *

The discovery of ore in the Bisbee district was made by an American Army Scout

* James H. McClintock, "Arizona", and Rickard's "History of American Mining."

named Jack Dunn, in August 1877, when he located a claim called the Rucker, near the Mexican border. Dunn's location was named after J. A. Rucker, an army officer, to whom was given a share in the claim. The Copper Queen deposit was discovered by Hugh Jones in 1877, and a claim named the Mercey was located by George Warren, after whom the district is named, on Dec. 27, 1877. This claim was re-located as the Copper Queen by George Eddlemann and M. A. Herring on Dec. 15, 1878, The original locator, Jones, abandoned his discovery because he saw nothing more than "copper-stained rock". A little copper furnace was erected by Warner Buck on the Robb claim owned by B. D. Rea of Tucson, and some matte was produced unprofitably in 1878. The Copper Queen prospect was purchased by John Ballard and William Martin, of San Francisco. They were successful contrators, but entirely ignorant of mining; they had, however, the advice of two competent men, Ben Williams and Lewis Williams, the sons of John Williams of Globe. John Williams had been a Welch miner and was now a partner with Judge Dewitt Bisbee in the noted brokerage firm of Bisbee, Williams & Co. of San Francisco. Bisbee sponsored the new company of Ballard & Martin, and the town was named in his honor. Under the direction of the Williams Brothers, George Center built a smelting-furnace, a 36-inch water-jacketed cupola, in 1880. This little smelter treated an ore yielding 23 percent of copper, and for a time did well. The fuel was English coke, brought by way of San Francisco.

In 1881, James Douglas came to Bisbee and obtained an option on the Atlanta claim, which was next to the Copper Queen. In developing the Atlanta, Dr. Douglas was unsuccessful at first in finding ore, and after he had spent $70,000 in exploratory work it was proposed by his associates to discontinue operations, but on his advice they agreed to advance $15,000 more for development, with the understanding that if this renewed attempt failed to discover sufficient ore, they would abandon the venture. Sinking was resumed, and within a few feet the Atlanta workings penetrated a great orebody, which proved later to be the basis for a magnificent copper enterprise.

Meanwhile, Ballard and Martin had exhausted the ore in the Copper Queen, and in 1884 litigation was threatened between them and the owners of the Atlanta, whereupon the two mines were joined in the name of the Copper Queen Consolidated Mining Company. This was in 1885, and Douglas, who was acting for the firm of Phelps, Dodge & Company, became the moving spirit of this company. In 1890 he engaged Louis D. Ricketts as his assistant, and their association continued for 17 years.

The Copper Queen Company extended its territory by acquiring the Goddard properties and by purchasing outlying claims including the Neptune and Lowell groups. The Irish Mag and one or two other desirable claims, however, were involved in litigation because the owner, an Irishman named James Daley, was a fugitive from justice, and a Mexican wife became claimant to his belongings. Eventually the Supreme Court of the United States recognized her title, which soon afterward passed to Martin Costello of Tombstone. He was willing to sell for $500,000 and Douglas was willing to take a bond at that price provided he could explore the property by extending the underground workings of the Copper Queen mine, whereas Costello insisted that the work be done from the surface of the Irish Mag, so that he could have a shaft in case the deal fell through. When these negotiations failed, in 1901, the Irish Mag was purchased by a group from Michigan and Pennsylvania in the name of the Lake Superior & Western Development Company, which later became the Calumet & Arizona Mining Co., the leaders of which were the Hoatson Brothers, Thomas F. Cole, George E. Tener, Chester A. Corigdon, and Charles Briggs. This company subsequently acquired additional territory and eventually became one of the leading producers of copper in the Southwest. Litigation over apex rights would have ensued between this company and the Copper Queen if Douglas had not possessed the sagacity to arrange with his neighbors to waive any extra-lateral rights in favor of the common law, whereby each company waived any claim to ore in depth that was vertically outside its side and end lines. At the same time an agreement was made giving each company free access, for information, to its neighbor's underground workings. This not only ensured peace but also the opportunity to become informed concerning discoveries of ore, all of which redounded greatly to the prosperity of the district, and to the esteem in which Dr. Douglas was held by his fellow-engineers.

Only oxidized ores were worked by the Copper Queen until 1893, when converters were added to the smelting plant. As early as 1886 a film of matte floated on the bars of copper and the quality of the metal suffered so much that the direct method of smelting had to be abandoned, whereupon matte was made; and reduced in the converter. In 1908 the mine began to produce some lead, and in 1916 some zinc. Since those dates, the district has produced over 312 million pounds of lead and over 366 million pounds of zinc. Gold, amounting to over 1.9 million ounces and silver amounting to almost 81 million ounces, have also been produced by the district since operations began.

* The first smelter of any importance was erected at Fairbanks on the Southern Pacific Railway, 37 miles northwest of Bisbee; and for several years ore was packed thither on mules and burros. In 1888 a railway was built by the mining company from Bisbee to Fairbanks, but in 1900 plans were made for the new reduction works at Douglas, and the El Paso and Southwestern Company, controlled by the Phelps Dodge interests, constructed the necessary railroad. The Copper Queen smelter at Douglas went into operation in 1904, being supplied principally by the Copper Queen mines with rich carbonate ores of smelting grade until 1932.

The blowing-in of the smelter in 1904 was a milestone in the company's history. From a single water-jacketed cupola no higher than a man, its smelting plant had now grown into the most modern structure of its kind in the world.

* Parsons' "The Porphyry Coppers"

Containing five blast furnaces and four barrel-type acid-lined converters, the new works covered three hundred acres with a fifteen mile network of standard gauge railroad tracks connecting smelter, power houses, machine shops and foundry. It was built to handle a production of more than a hundred million pounds per year, but through the years it was constantly enlarged to meet even greater demands upon it. This camp became a thriving town - and in honor of the man who had done so much to develop copper mining in the southwest, it was named Douglas.

In 1917, the name Copper Queen Consolidated Mining Company was changed to the Phelps Dodge Corporation and the assets of Phelps,Dodge & Co., were transferred to the new corporation.

When the other porphyry mines began to cut an important figure in the copper-mining world, the attention of Dr. Douglas and his associates was naturally drawn to the large mass of granite or monzonite porphyry which had intruded into the limestone, in past geologic eras, and with which had been associated the copper-bearing solutions that were responsible for the formation of the rich limestone "replacement" orebodies. As early as 1909 exploration of the porphyry intrusion was undertaken, first by extending the underground workings in the limestone sections of the mines, and later by churn-drilling from the surface. Two orebodies separated by a mass of rock too lean for profitable exploitation, were proven. The so-called West, or Sacramento Hill orebody, was richer, on the average, and the overburden was thinner, ranging from 50 to 350 feet with an average of 250 feet. Steam shovel operations began in 1918.

Sacramento Hill was - before it succumbed largely to the steam shovels - a bold precipitous hump standing in the center of Mule Gulch. Its crest was dark brown from stains of iron; it contrasted sharply with the reddish schist on one side and the gray limestones on the other. The demolition of it was a most spectacular project.

In September, 1929, shovel operations were finally suspended after moving 15,000,000 cu. yd. (equivalent to about 30,000,000 tons) including ore and waste. As the ratio of waste to ore was 2.75 to 1, about 8,000,000 tons of ore was shoveled during the period 1923 to 1929. At least three million tons of ore remained to be won. A glory-hole method was developed that recovered this ore economically. In the meantime, mining of the East porphyry orebody was started. In spite of some misgiving on account of the wet and sticky character of the ore, a block-caving method adapted from Morenci practice was put into successful operation, and a substantial part of the ore going to the concentrator in 1930 and 1931 came from this section of the mine.

The old underground sections of the Copper Queen mine have been mined by the use of both square-setting and top-slicing methods, or modifications thereof. The high grade of the ores mined permitted these more expensive mining methods. The Denn Mine, which was taken over by the Phelps Dodge Corporation in March 8, 1947, also used the square-set and pillar system, the pillars being mined by the Mitchell slicing method developed at Bisbee. The Denn Mine had originally been owned by Lem Shattuck and Maurice Denn. In the early days, Lem Shattuck, like almost everyone in the district, picked up a number of claims, including one southwest of the "Copper Queen". When the boom hit Bisbee, he interested some Minnesota investors in helping him sink a shaft on his property. At the three-hundred-and-fifty foot level, the shaft dug into a body of high grade ore which continued all the way down to the eleven-hundred foot level and made the "Shattuck" renowned as the "biggest little mine" in the area. Then with Maurice Denn and others, he owned another group of claims to the northeast where no one expected to find ore; but Shattuck sank a shaft to the seventeen-hundred foot level and again ran into an enormous body of sulphide ore.

Ore Bodies *

The first discovery of copper ore in the district was in the old open cut

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* From Cark Trischa's article in the Arizona Bureau of Mines Bulletin #145, pages 38-41.

on the hillside above the Bisbee Post Office. Except at the White Tailed Deer Mine, it was the only copper outcrop in the district. The ore here was malachite and azurite (copper carbonates), which for many years was the only kind of ore found or mined. As work progressed downward and southeastward, secondarily enriched sulphides and finally primary sulphides of mineable grade were found.

The ore bodies of the district are arranged in the semicircle around Sacramento Hill and also radiate outward from this center. A commonly accepted idea about the replacement ore bodies in the limestone is that they are tabular, wider than they are high. The idea originated at a time when mining was done mostly in the western portion of the camp. Here oxidation and erosion shrank and cut down the height of some of the ore bodies of this area. In the extreme eastern ore area, height is generally greater than length or width. Oxidation progresses in intensity from southeast to northwest. In small portions of the Campbell area, however, oxidation has penetrated as deep as the 2,300 level.

Practically all of the ore bodies of the district had a central core of of somewhat siliceous pyrite containing small amounts of copper around which sulphides of copper and iron occurred. In the fine grained pyrite core, the pyrite is commonly shattered and becomes ore because of the deposition of small veinlets of copper sulphides in the breaks and cracks. Hematite is frequently associated with the ore along its contact with the limestone. Magnetite is intimately mixed with the pyrite and chalcopyrite in certain areas. In the process of replacement the grain structure, bedding, and the included unreplaced chert lenses of the limestone are frequently beautifully preserved in the resulting sulphide.

<u>Porphyry Ore Bodies</u>

There was a fairly large mineralized area within the stock of Sacramento Hill. These ore bodies were secondarily enriched by chalcocite and were partly in the porphyry mass of Sacramento Hill and partly in the contact breccia around it. The protore contains less than 0.50 percent copper. The stock of Sacramento Hill was highly silicified, sericitized, and pyritized, and the small amounts of chalcopyrite and bornite in the protore are responsible for the copper of the secondary enrichment.

Ore Guides

Granite-Porphyry dikes and sills are guides to ore: By following them on both sides ore may be encountered in the embayments.

Fracture zones, where they are rather steep and dip more or less normally to the bedding, are well worth following if they are at all mineralized.

Manganese oxides as outcrops or along fracture zones can be used as guides. Silica breccia and hematite, or both, are usually closer to ore than manganese.

Limonitic gossans and calcite-filled cracks in the limestone over oxidized slumped ore bodies are direct guides and point down to the possible ore.

Copper Queen Concentrator *

Because of the sulphide character and the low grade of the ore in Sacramento Hill, a concentrator was deemed necessary. Experiments preliminary to the design of such a plant were commenced as early as 1916 in a small test plant constructed for the purpose near the mine. Early in 1918 H. K. Burch was employed to supervise the design of a 3,000-ton mill to be constructed at a site about two miles south of the mine on the slope of the Mule Mountains. Besides affording gravity flow of the ore through the concentrator, the site was exceptionally well situated with respect to disposal of tailing on the valley floor below.

Many obstacles delayed the completion of the mill until 1923. First, the Government declined to release steel necessary for construction; then in 1919 important changes in the design of the plant were made; in 1920 the strike of railway employees in the United States held up shipment of materials; and

* Parsons "Porphyry Coppers"

finally in 1921 the general curtailment in copper production caused a complete suspension of construction, which remained in force until January 1, 1923. However, on April 1, 1923 the first unit of the plant was put in operation, the nominal capacity having in the meantime been increased to 4,000 tons per day.

At the start the concentrator provided a combination of gravity and flotation treatment, flotation constituting an intermediate process between "roughing" and finishing" concentrating tables. Although a number of Porphyry Copper mills had by this time discarded tables, the reason they were retained at the Queen mill was because it was not planned to obtain rich concentrate and a high ratio of concentration. Coarse concentrate was more readily handled at the smelter, and moreover there was the supposed need of iron-bearing minerals at the smelter to flux the large quantities of highgrade ores that were mined from the carbonate orebodies at Bisbee and smelted without concentration. As the minerals of copper and iron in the sulphide concentrating ore were very intimately associated it would be necessary, if they were to be separated, to grind exceedingly fine, and fine grinding is a costly operation. Early operation, however, indicated that radical changes were desirable. Within a year, the gravity tables were discarded, leaving the flotation cells as the only concentrating devices in the plant. The next change was the introduction of finer grinding to permit the removal of more of the gangue constituents of the ore. From these changes came a saving of freight on the concentrate, a reduction in smelting costs, growing out of the treatment of a smaller tonnage, and a reduction of slag losses because of the removal of worthless slag-forming elements in the furnace charge.

The ore mined from the Sacramento Pit increased in grade as the lower horizons of the deposit were reached. For example, the first five years of operation produced an average grade of 1.65% copper as compared with 2.04% for the last four years. The concentrate grade was increased from 7 - 8% copper to 13 - 15%. An excellent recovery was made from the start, averaging better than 88%.

Heap Leaching *

In one respect the Sacramento Hill project is unique among the Porphyries, and that is the process of heap-leaching. Heap-leaching itself consists essentially of: (1) piling run-of-mine ore on a gently sloping hillside to form a bed; (2) "irrigating" this heap with slightly acidulated water; (3) collecting the water (which, in percolating through the heap, has acquired a burden of copper in the form of a solution of copper sulphate) in a pond at the foot of the bed ; (4) passing the pregnant solution over scrap iron to precipitate the copper; and (5) collecting and drying the mudlike precipitate of "cement copper" to be fluxed and melted to produce comparatively pure metal. A distinctive feature of the process is that the foregoing sequence of operations is repeated for a particular section of a heap with long intervals intervening during which the sulphide minerals oxidize by contact with atmostpheric oxygen. The consequence is that a period of years is required to effect a satisfactory extraction, a feature that militates against wider utilization of the method.

Ever since early in 1900, experimentation and research have been devoted to the heap-leaching of Bisbee waste dump and low-grade ore piles in the district, and a considerable amount of low-cost copper has been produced by the process.

Zinc-Lead Deposits

Records of the earlier mines of the Bisbee district show that zinc and lead occurred throughout the district in areas mined for copper. The zinc and lead production of Bisbee since 1939has come largely from the Eastern part of the district, particularly the Campbell and Junction areas. Near the borders of the Campbell copper orebody, sphalerite[1/] and galena[2/] become increasingly abundant.

The deposits vary greatly as to size, shape and mode of occurence, but in general they may be classified as follows:

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* Parsons' "Porphyry Coppers".
1/ Sphalerite - Zinc Sulphide - 67.1% Zn and 32.9% S.
2/ Galena - Lead Sulphide - 86.6% Pb and 13.4% S.

Deposits along structural breaks.
Deposits peripheral to barren siliceous-pyritic bodies.
Deposits intimately associated with, or peripheral to, rather massive pyrite-copper ore bodies.
Deposits associated with porphyry.

The Phelps Dodge Corporation annual report for 1951 has this to say regarding the lead-zinc ores at Bisbee:

"During the year, the change-over from lead-zinc ores to the mining of copper ores as the major source of production was brought to completion. This change-over was started in the previous year as a result of the exhaustion of lead-zinc ores of importance. The tonnages mined in 1951 totaled 490, 184 tons of copper ore and 40,426 tons of lead-zinc ore".

Conclusion

For 42 years, until 1932, the Copper Queen Company relied on smelting ores from its carbonate mines to produce its copper, but since 1927, the Porphyry ores - milling and leaching - have contributed the preponderant proportion of the output. The absorption of the Calumet & Arizona Company by Phelps Dodge in 1931 gave the Corporation enormous proved bodies of ore of direct smelting grade, and there is the ever present possibility of finding still further bonanzas in the famous old district.

SEGREGATION, CONTROL AND SURFACE DISTRIBUTION OF MINE WATER

Phelps Dodge Corporation
Copper Queen Branch

by
A. E. Himebaugh

Mining Division of the Arizona Section - A. I. M. E.
Tucson - December 7, 1964

During the last 15 years underground pumping has averaged approximately 4,000 g.p.m. consisting of 250 g.p.m. of mine acid water from natural ground water leaching in the old pit and underground workings, 1,000 g.p.m. of contaminated water and the balance of clean potable water from high pressure water zones intercepted by crosscutting and diamond drilling in the deeper workings.

Prior to modifications started in 1959 one 1500, one 1000 and two 600 g.p.m. centrifugal pumps elevated the water from the 3100 level Interior station to the 2966 level and then by pipeline to the Campbell shaft where it was collected with water pumped from the Denn and Campbell shafts. Two 2000 and four 1200 g.p.m. centrifugal pumps delivered water from the 2966 station to the 2700 level and across by pipeline to the 2700 Junction Pump Station having five 1500 g.p.m. centrifugal pumps for the 500 ft. lift to the 2200 level main pumping station.

Tail water from the 1800 level precipitation plant handling mine acid water to recover some of the contained copper, was collected in the 2200 level overhead sump. Here dilution water and hydrated lime were added to raise the pH to 6.7. Sometimes as high as 3500 lbs/hr. of lime were added during the rainy season although 200 to 350 lbs/hr. was normal. The resulting mixture containing iron hydroxides was pumped by either one 1250 g.p.m. or one 1000 g.p.m. power pump through a separate column to the surface. At one time this water, because of its high solids content, was wasted. However, in 1954 with the starting of the Lavender Pit leaching operation 600 g.p.m. was used as makeup and the balance was wasted.

Another 2250, one 1500, one 1000 g.p.m. power pump and one 2000 g.p.m. centrifugal pump handled the balance of the water through another column to the surface. Due to a relatively low solids content the water was usable in concentrating and nothing more than acceptable for cooling towers, underground drill water and most general uses other than for drinking, bathing or steam boilers.

In 1955 work was started building bulkheads above the 1800 level Junction to divert, contain and control mine acid water to reduce peak flows resulting from the rainy season which impaired the efficiency of the 1800 level precipitation plant and increased the consumption of lime used for neutralization. This storage system proved invaluable after abandonment of the Junction as a production shaft made a compartment available for a pipe column to pump the mine acid water directly to the surface precipitation plant.

Removal of the underground precipitation cells which utilized an old steam pump station and the pouring of several concrete bulkheads, provided a 300,000 gallon elevated sump. Two 300 g.p.m. 1925 ft. head, 6-stage centrifugal pumps are each fed by a 2-stage booster pump which, in turn, has a positive suction head from the sump.

All pumps, valves and piping are of type 316 or 316 low carbon stainless steel with the exception of the 1800 ft. 6 inch base supported column of fiberglass reinforced plastic construction and surface lines of cement asbestos.

These pumps are automatically primed with thermal protection on both pump and motor and operate from sump electrodes for unattended operation. Signals from the pumps are remotely indicated in the surface power plant along with the recorded inflow on a 30 day strip chart and sump water level as relayed from capacitance probes in a Parshall flume and pump sump respectively.

Hydrated lime consumption at the 2200 level pump station was reduced to 50 lbs/hr. which was needed to neutralize the 25 g.p.m. of acid water seeping through the ground from the 1800 level sump. An automatic pump was installed on the 1900 level to collect this seepage and return it to the 1800 level through a 3 inch stainless steel column with the resulting elimination of lime addition on the 2200 level.

Segregation of mine acid water below the 1800 level was impractical both from a volume standpoint as well as requiring a complex collection system. The balance of the drainage is of two types: clean water for almost any use purchased utility water is used for; and dirty water - a combination of ditch water, mine acid water and some clear water for dilution along with required amounts of hydrated lime.

Clean water is stored behind the 3100 level - 9 crosscut bulkhead and metered into the 3100 level Interior sump where it is elevated to the 2966 level and in turn to the 2700 level by the existing routing. On the 2700 level the water is controlled into the sump by a float operated diaphragm valve with relief being behind the 12 crosscut bulkhead storage which accumulates with the natural flow of 700 g.p.m. behind this bulkhead. Another float valve regulates the flow from this storage to supplement the 2966 level water maintaining the 2700 level sump at the proper level for operating one of two 2000 g.p.m., 2900 ft. head, 6-stage, 1750 H.P. centrifugal pumps formerly used as standby protection, which elevate the water to the surface and by pipeline to the vicinity of the power plant. The pipeline "Y's" at this point and all of the water that is not used in the surface pumping plant continues on to the clean water pumps at the concentrator.

Segregation of dirty water begins on the 2966 level and two 700 g.p.m., 310 ft. head vertical pumps were installed to handle this water to the 2700 level with connection into the pipeline to the Junction shaft and the previously existing system to the surface. 600 g.p.m. of the flow is diverted at the shaft collar for makeup water to the precipitation plant while the balance is delivered to the tailing thickeners where the solids settle out and the water is added to the reclaim water system at the concentrator. The intentional pumping of solids in suspension has not appreciably increased pump maintenance and has more than compensated for the handling of settling sump sludge as previously practiced.

Clean water is stored underground during shutdown periods and only sufficient water is pumped to the surface to supply the precipitation plant and other continuous requirements. Normal pumping is reduced by two-thirds resulting in a considerable power savings and increases available water from storage during operating periods by 300 to 400 g.p.m. 700,000 gallons can be stored from a 600 g.p.m. natural flow behind the 2200 level bulkhead before overflow will take place through raises and crosscuts to the 2700 level storage. The 2700 level has an additional flow of 700 g.p.m. and as its capacity of 2,000,000 gallons is reached the overflow passes through previously unwatered natural courses to the 2966 level and 3100 level storages. Flow behind the 2966 level bulkhead amounts to 200 g.p.m. plus serving as a surge reservoir of 1,700,000 gallons for the 500 g.p.m. from the Campbell shaft sump. The capacity of the 3100 level storage is not known since natural water courses provide the main storage for the 1700 g.p.m. delivered from this source prior to closing the bulkhead door. Over 150 p.s.i. pressure has built up behind the bulkhead and water has been stored for over 14 days. After 70 p.s.i. has been attained the 3100 level pumps are by-passed and water is delivered to the 2966 level pump station under this pressure. Concentrator demands deplete this reserve over an 8 to 12 month period.

All underground pump stations, with the exception of the 2200 level Junction are locally automated using sump electrodes for stopping and starting each pump and remotely indicated at a control panel located in the surface power plant. Sequential time delay on starting eases the electrical load after power outages. Pumps requiring a suction lift are primed through individual priming chambers on a continuous basis from vacuum tanks operated by compressed air piloted eductors. All bearings on 3600 r.p.m. and vertical pumps are equipped with thermal and vibration protection. Vibration protectors have short time delays to avoid false signals due to underground and pit blasting. Individual control panels on each pump have signals indicating malfunctions of a hot pump or motor bearing, vibration and failure to prime which is used to facilitate trouble shooting by the pump maintenance crew consisting of an electrician and machinist.

The 2200 level pump station is not automated primarily because of the size and complexity of power pumps. However, improvements to the air charging system reduced the operation from two men to one. Presently this is the only manned station underground requiring a pump operator on each shift with a jigger boss and two repairmen on day shift.

The 1750 h.p. pumps on the 2700 level Campbell are the only pumps started remotely from the power plant because of the severe electrical starting load. Due to their relative size and importance of supplying the only clean water from underground, supplemental protection of separate priming tanks and automatically controlled cylinder operated check valves to reduce water hammer and prevent reversal of flow on normal shutdowns or power failure have been added.

Field scanners located at each underground and concentrator pump station are synchronized with scanners in the power plant telemetering desired signals to the indicating and control panel. Signals shown for each pump indicate if it is ready, running, malfunctioned or has been placed on "Hold". The "Hold" option provides the operator with control to prevent a pump from starting and is most frequently used to adjust the electrical load after power outages. Release of this "Hold" feature will not start a pump unless all local control conditions are satisfied. Loss of the station priming system, control voltage, pump malfunction and high or low water conditions are audible alarm signals which the operator may silence leaving a lighted signal until he has remedied the particular condition or has contacted maintenance personnel and corrective action is taken.

Paging telephones are located at every pump station, power plant, electric shop, mechanical office, underground maintenance shop and Campbell shaft collar, providing adequate communication.

Pump sump and head tank water level indicators provide the necessary information to the operator who remotely controls strategic valves dispatching water to locations of need in the order of assigned priority. Normally the pumps and distribution valves will function without attention as the system is designed on a "Fail Safe" basis with first consideration given to the safety of the underground mine. However, good operators at the power plant are invaluable in keeping the surface water distribution running smoothly.

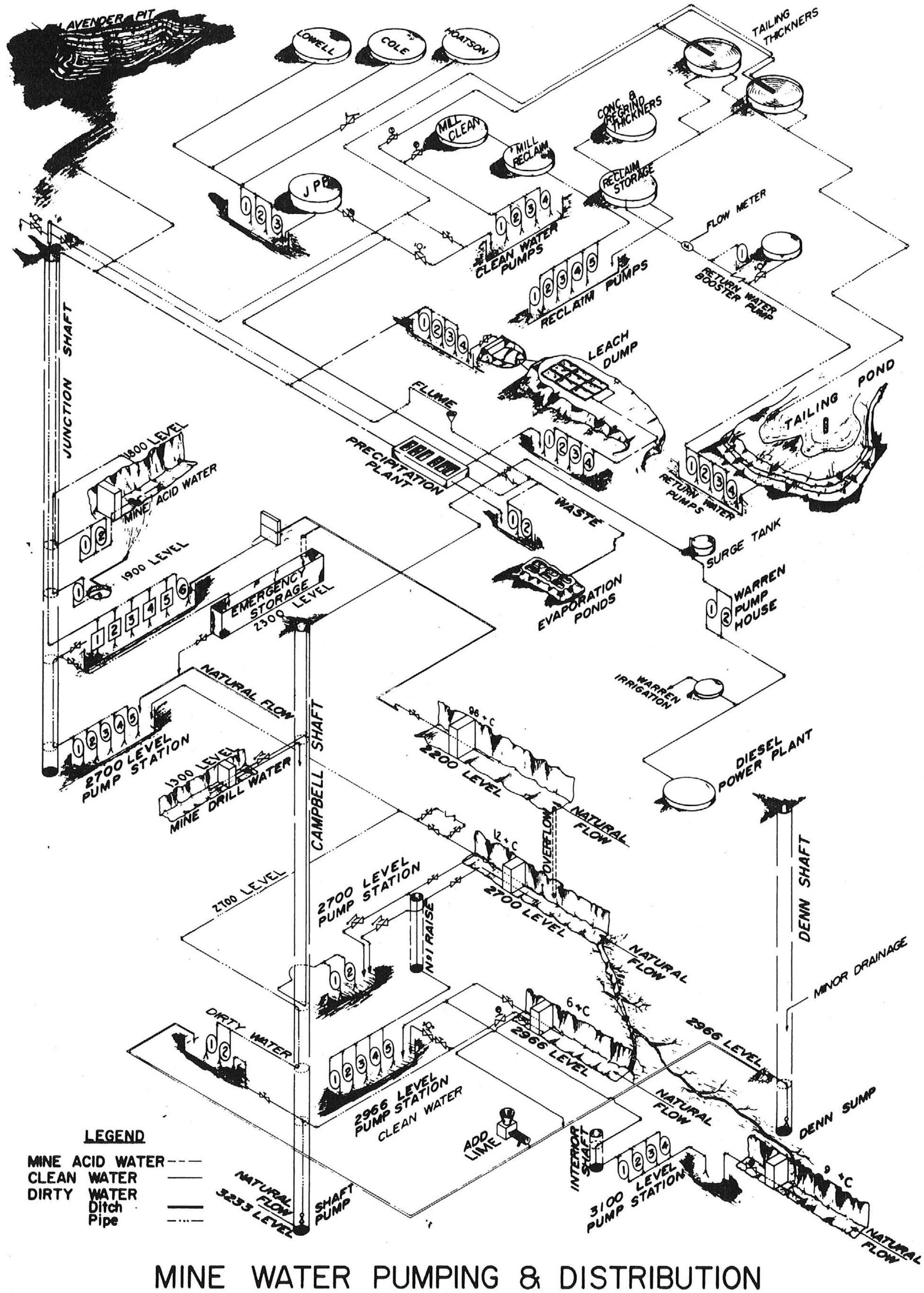
The booster pump on the return water from the tailing dam is controlled over power line carrier from the power plant. The return water pump station is automatically controlled although no indications are telemetered. Proper interpretation of the recorder in the power plant on a magnetic flow meter in the return water line will indicate improper function of the system and necessary action is taken to restore normal operation. Pump malfunctions, low or high water sump, low pH and underfeeding of hydrated lime are reported by a siren and flasher at the top of the tailing dam for the operator to see and take remedial action.

The precipitation plant pumps are automatically controlled locally rather than remotely indicated as plant personnel are chiefly concerned with their operation.

The Lavender Pit drainage pumps are not automated because sump locations are temporary and pumping seasonal.

Since this system has been completed water shortages at the concentrator have practically been eliminated, purchased utility water has been reduced by 6,000,000 to 8,000,000 gallons per month and 23 less pumpmen and operators are presently required.

Bisbee, Arizona
October 8, 1964



MINE WATER PUMPING & DISTRIBUTION

P H E L P S   D O D G E   C O R P O R A T I O N
COPPER QUEEN BRANCH, MINES DIVISION

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Milling Practice at the New Lead-Zinc Concentrator of Phelps Dodge Corporation at Bisbee

R. C. Thompson,
Mill Superintendent

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The lead-zinc mill of Phelps Dodge Corporation, Copper Queen Branch, Mines Division, is located about three miles from the main hoisting shafts of the Junction and Campbell mines at Lowell, Arizona. All of the ore treated in the mill is obtained from these two mines.

The mill was designed for an all flotation treatment of 450 tons of lead-zinc ore per day, and has been operated at that capacity since the start of milling operations on November 17, 1945. At the present time, an expansion program is under way whereby the mill capacity will be doubled. This article describes the original program only and presents results of the 450-ton operation.

GENERAL DESCRIPTION

The milling plant consists of an ore receiving bin, and three steel-constructed buildings housing, respectively, the primary crushing equipment, the secondary crushing equipment, and the ore storage bins, grinding, flotation, and filtering equipment.

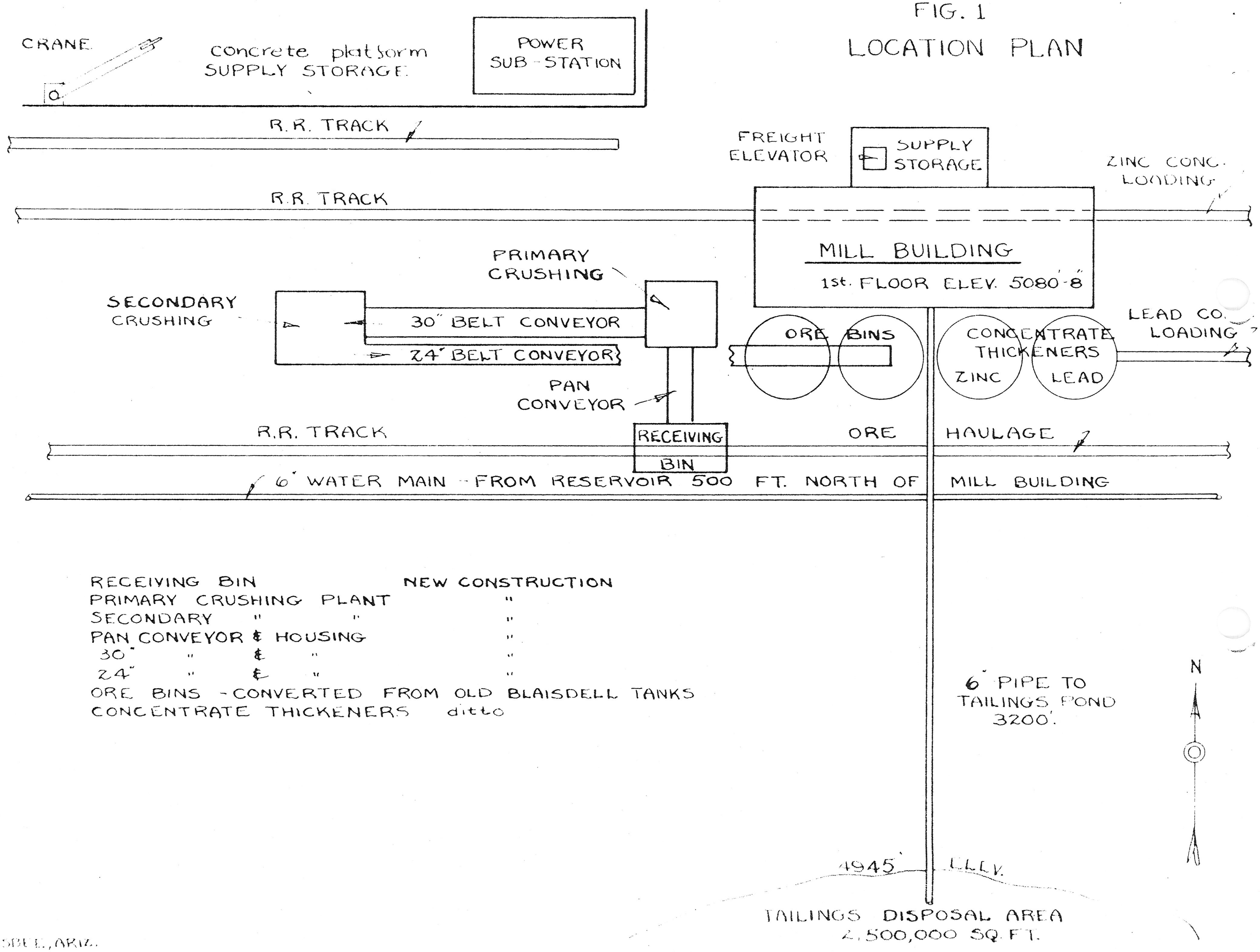
The mill site is the same as was used for a previous copper concentrating plant which was operated over a period of years and then dismantled. There remained from this operation some milling equipment, a reservoir for mill water supply, railroad tracks, tailings disposal ponds, and a steel-constructed building, all of which were incorporated in the design of the present lead-zinc mill. The location plan of the plant layout is shown in Figure 1.

The flow sheet of the mill is based on the conventional lead-zinc treatment scheme; however, several innovations were made necessary in order to utilize the existing building. Placing of equipment was also governed by the design of the building; thus making the mill unique in many respects. A brief description of the mill building as adapted to the flow sheet points out the compactness of the installation and some of its unusual features.

PHELPS-DODGE CORPORATION - COPPER QUEEN BRANCH - MINES DIVISION

FIG. 1

LOCATION PLAN



BISBEE, ARIZ.
1945

The first floor (Figure 2) houses the grinding equipment, with connecting conveyors for ore feeding, also an air compressor, two vacuum pumps, lead and zinc concentrate storage bins with individual conveying systems for loading the two kinds of concentrates. Filtering of concentrates is done on the second floor with the concentrates discharging directly into the concentrate storage bins. On this floor are located the lead and zinc filters, two diaphragm pumps, sample filtering equipment, sample preparation room and a change room for the mill employees. The flotation machines are located on the third floor, at elevations allowing a gravity flow of the lead and zinc concentrates to the thickeners. The annex to the building contains a freight elevator and provides space for all reagent mixing and feeding, also storage of reagents and grinding balls.

All of the water used for milling purposes is pumped from the Junction Mine and none of the water used is reclaimed. Power is brought to the mill from the Company Steam and Diesel Power Plants at 2200 volts and is transformed to 220 and 110 volts for mill use.

ORE TREATED

The lead and zinc generally occur in the ore as galena and sphalerite, while the iron occurs mainly as pyrite. Anglesite and cerussite have been noticed on rare occasions in amounts sufficient to be observed on the pilot tables. The sphalerite varies from a straw-colored variety to a coal-black variety as seen in the ground condition under the microscope, with varying iron content. Of the sphalerite which enters the lead concentrate, the light-colorec varieties predominate. The major portion of copper exists as chalcopyrite with bornite next in frequency of occurrence. Chalcocite is sometimes present in small amounts. Quartz and clacite are the predominating gangue. Ores from various sections of the mines show marked differences in association of minerals and texture. Some ores are so fine grained as to defy liberation of the mineral with any practical grind, and others are fairly coarse grained. In milling, the practice is to grind to approximately 68% minus 200 mesh as a compromise to the demands of the various ores. Sphalerite-pyrite locking is generally more severe than that between other minerals. Activation of sphalerite in place by copper salts, tarnished galena surfaces, locking of minerals, and excessive fineness of sphalerite all play a part in reducing the selectivity of the mineral separations.

MILLING PRACTICE

A flow sheet of the mill is shown in Figure 3. It gives a comparatively complete picture of the progress of the ore through the various stages of treatment. An outline description of the flow sheet follows.

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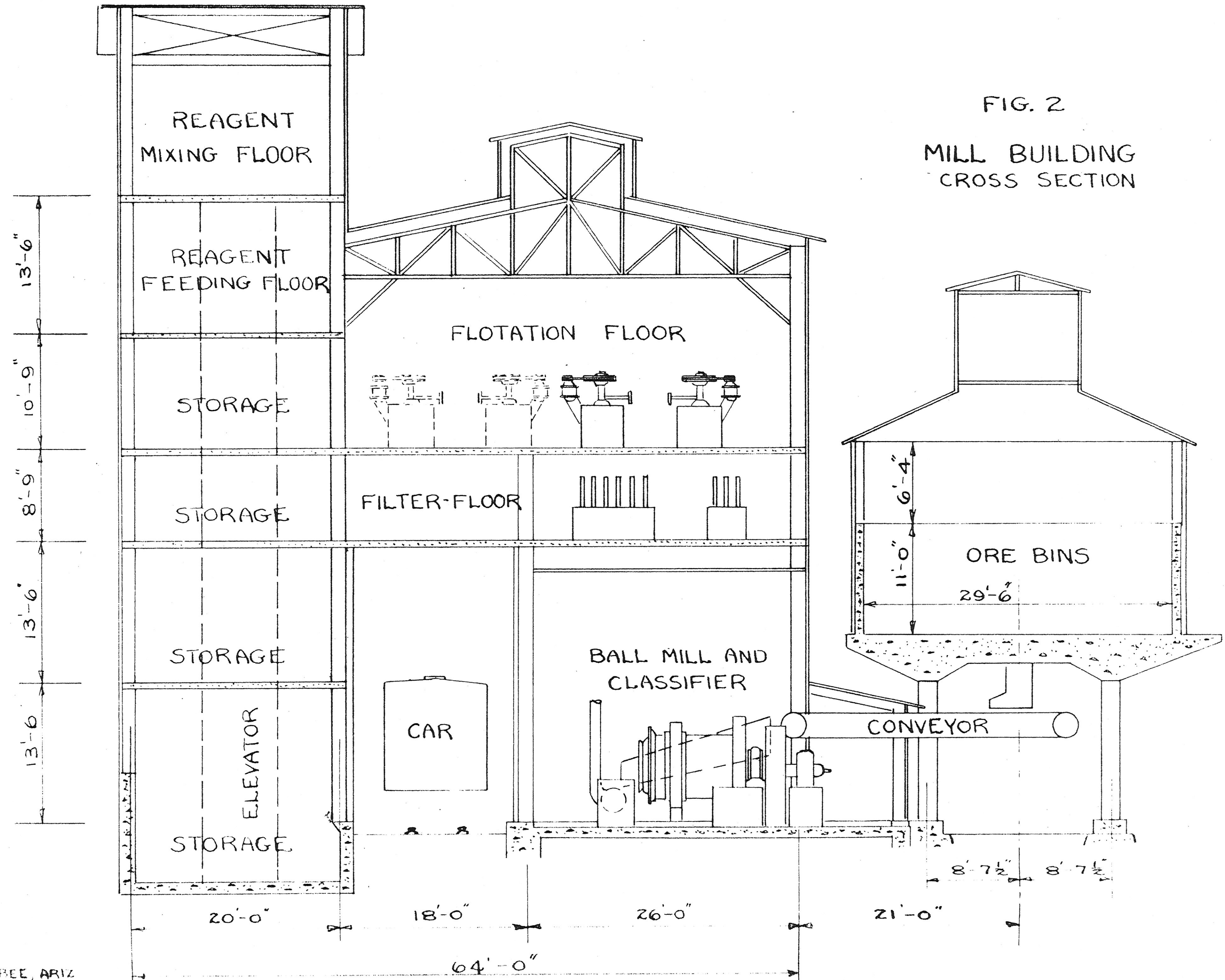
FIG. 2

MILL BUILDING
CROSS SECTION



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Unloading and Crushing

Ore is received from the mines in 50-ton capacity gabled-bottom side-dump cars. Loaded cars are weighed on a track scale in the mill yard. The cars are dumped into the receiving bin, a concrete and steel hopper-bottomed bin in an excavation beneath the ore haulage track. The receiving bin is simply a dumping pocket and was not designed to provide any appreciable ore storage. The ore track accommodates 28 loaded cars above the bin and an equal number of empties below, and serves in lieu of a coarse ore storage bin.

Ore is transported from the receiving bin by a 34-inch pan conveyor to a steel grizzly having 2-inch spacings. The conveyor is 45 feet between head and tail sprockets and is on an incline of 22 degrees from horizontal. The conveyor speed may be varied between 8 and 14 feet of travel per minute. Grizzly oversize is crushed to 2-1/2 inches in an 18 x 36 inch jaw crusher. The crusher operates at 240 R.P.M. and is driven by a 75-H.P. motor through Texropes. Crushed ore and grizzly undersize are delivered to a 30-inch conveyor belt which passes under a magnet and discharges onto a 3-x 8 foot vibrating screen, ahead of a 3-foot cone crusher. The cone crusher is driven by a 150-H.P. motor through Texropes and is set to deliver, with a single pass, a product containing 85% minus 1/2 inch. The screen undersize an crusher product are transported by a 24-inch inclined conveyor to an elevated 850-ton ore bin, which was originally one of our old Blaisdell tanks, 29 ft., 6 in. x 11 ft., of reinforced concrete construction. Depth of the tank was increased to 17 feet by adding a ring of 5/16-inch plate. A steel-framed housing was erected over the tank and four feeders installed in the bottom to complete the conversion into an ore bin with a live capacity of 850 tons.

Grinding

Ore is drawn from the bin by four conveyor feeders and is delivered to the ball mill by a 20-inch conveyor equipped with a weightometer. The grinding unit consists of a 7 x 7 ft. grated end ball mill operating in closed circuit with an 8 x 23 ft. duplex rake classifier. The mill, driven by a 200-H.P. motor, operates at 23 R.P.M. Forged steel balls are used, with the dail make-up proportioned at 60% two-inch and 40% three-inch balls. The ball load is maintained at a level about two inches below the center line of the mill. Ball consumption has averaged 1.23 pounds per ton of ore ground. Classifier overflow is maintained between 32 and 35% solids with a sand return circulating load around 400%. Table 2 shows the grinding product.

Flotation

Classifier overflow is pumped with a 3-inch sand pump to a sampling box ahead of the lead flotation circuit on the thi floor of the mill building. The pump operated at 1200 R.P.M.

against a head of 45 feet. Figure 3 shows the general arrangement of the flotation flow sheet in which thirty-two 43-inch x 43-inch "Sub-A" cells and one 10 x 10 ft. conditioner are employed.

A bank of twelve cells constitutes the lead circuit with feed entering the No. 4 cell. The circuit consists of a roughing section of six cells, a scavenging section of three cells, concentrate cleaning and recleaning sections of two and one cells respectively. Cells #4 to #9 inclusive produce a rougher concentrate which flows by gravity to the No. 2 cell. Cells #2 and #3 perform the first cleaning operation and produce a concentrate which is cleaned in the No. 1 cell, where the final concentrate is produced. Cleaner and recleaner tails return to the rougher section. Concentrate from the scavenger cells, #10, #11 and #12 are returned to the rougher section and enter No. 6 cell. The lead tailing from No. 12 cell flows to the conditioner tank where it is conditioned approximately 13 minutes, and then is split between two parallel banks of ten cells each, which comprise the zinc flotation circuit.

Feed enters the No. 3 cells of the two zinc banks. Rougher conentrates from cells #3 and #4, go to No. 1 for cleaning and from #5, #6, and #7 to No. 2 for cleaning. Scavenger concentrate from #8, #9 and #10 returns to the No. 5 cell. The first 1 or 2 cells, depending upon the zinc content of the feed, produce finished concentrate, and tailings from these cells join the original feed to the zinc section in the No. 3 cell.

In both circuits all middling products are returned to the different cells by gravity flow in launders. Elevations of the lead banks, conditioner tank and zinc banks of flotation machines permit a gravity flow of pulp through the entire flotation installation and of the final concentrates to the thickeners.

Reagents

A mixture of sodium carbonate, thiocarbanalid and zinc sulphate is added to the ball mill by means of a dry reagent feeder. Additional zinc sulphate, as a 15% solution, and sodium cyanide, as a 10% solution, are also added to the ball mill. Cresylic acid and sodium ethyl xanthate are added to the classifier overflow as the pulp enters the pump sump. Additions of sodium ethyl xanthate are also made to the Nos. 7 and 10 cells in the lead bank. Sodium carbonate is added mainly to produce the desired alkalinity. The pH value of the mill water is 8.2 and lead circuit feed is maintained from 7.6 to 7.8. Thiocarbanalid is used as the collector with a little sodium ethyl xanthate used intermittently. Thiocarbanalid has proved to be more selective and to produce a higher grade of concentrate than xanthate.

Hydrated lime and copper sulphate are added to the lead circuit tailing as it flows to the conditioner. Lime is added by means of a dry feeder but preparations are now under way to change the method to a milk of lime addition. Copper sulphate

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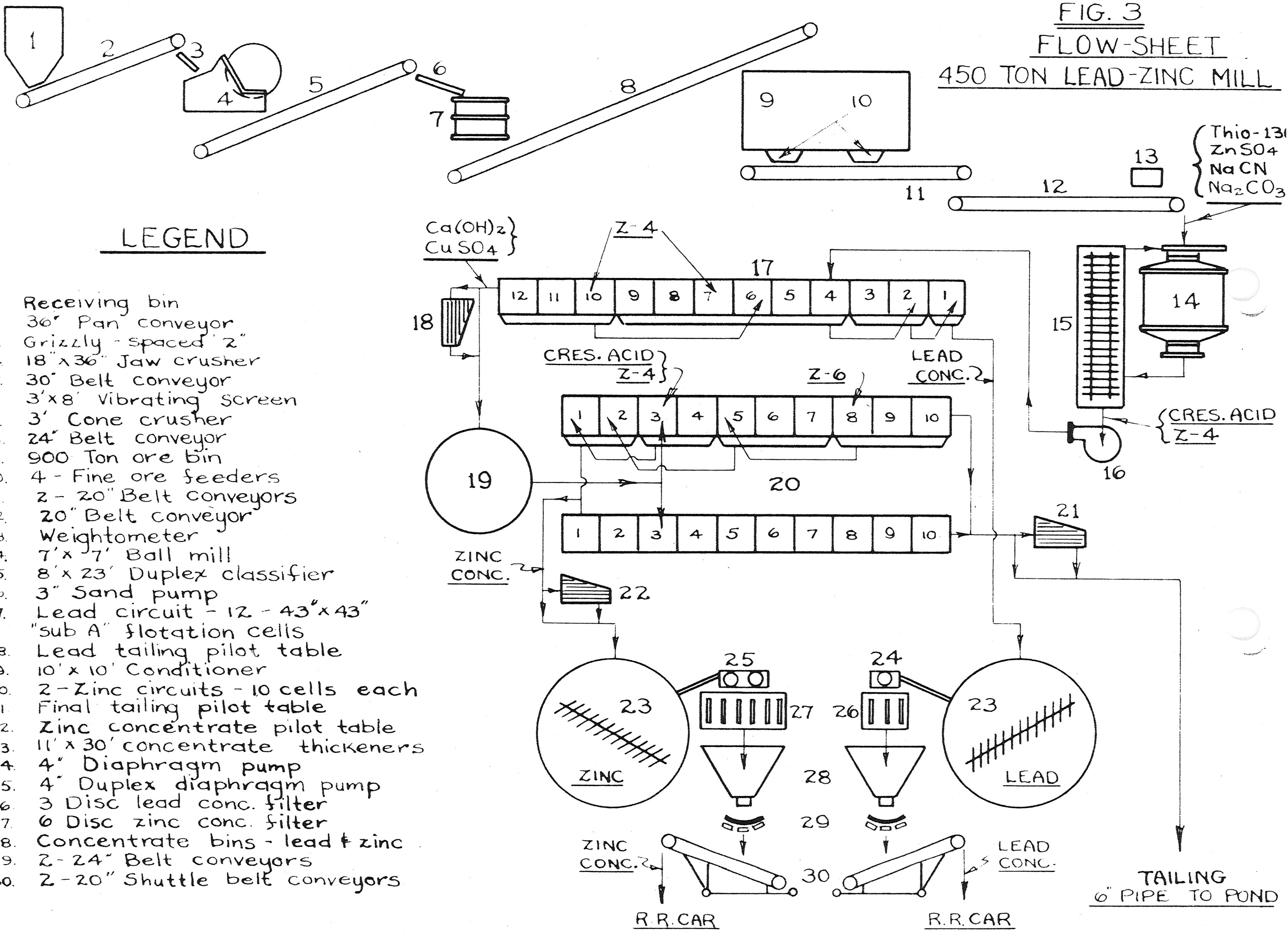
FIG. 3
FLOW-SHEET
450 TON LEAD-ZINC MILL



LEGEND

1. Receiving bin
2. 36" Pan conveyor
3. Grizzly - spaced 2"
4. 18"x36" Jaw crusher
5. 30" Belt conveyor
6. 3'x8' Vibrating screen
7. 3' Cone crusher
8. 24" Belt conveyor
9. 900 Ton ore bin
10. 4 - Fine ore feeders
11. 2 - 20" Belt conveyors
12. 20" Belt conveyor
13. Weightometer
14. 7'x 7' Ball mill
15. 8'x 23' Duplex classifier
16. 3" Sand pump
17. Lead circuit - 12 - 43"x43" "sub A" flotation cells
18. Lead tailing pilot table
19. 10'x 10' Conditioner
20. 2 - Zinc circuits - 10 cells each
21. Final tailing pilot table
22. Zinc concentrate pilot table
23. 11'x 30' concentrate thickeners
24. 4" Diaphragm pump
25. 4" Duplex diaphragm pump
26. 3 Disc lead conc. filter
27. 6 Disc zinc conc. filter
28. Concentrate bins - lead & zinc
29. 2 - 24" Belt conveyors
30. 2 - 20" Shuttle belt conveyors

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is added as a saturated solution, overflowing from a lead-lined box. Lime is added in amounts to give a pH of 9.8 to 10.0 in the zinc circuit feed. To accomplish this, 2 to 6 pounds are added per ton of original feed, with the average about 3 pounds. Cresylic acid and sodium ethyl xanthate are added to the pulp from the conditioner as it enters the No. 3 cell in the zinc circuit. Potassium amyl xanthate is added to the No. 6 cell and at times cresylic acid is added to the No. 6 or 7 cell.

Three laboratory concentrating tables are incorporated in the flotation flow sheet; one on lead circuit tailing, one on finished zinc concentrate, and one on flotation final tailing. These pilot tables present at all times a reliable picture of the performance of the different circuits and enable the operator to quickly note changing conditions throughout the plant.

Reagents are mixed on the day shift in quantities adequate for 24 hours of operation. The mixing tanks are piped for air agitation, and are connected to smaller drums located on the feeding floor below. Single and double stainless steel feeders are used for all wet reagent feeding and 18-inch cone type feeders for the dry reagents. Facilities for close measuring and distributing of the various reagents are very good. Table 4 summarizes reagent information for operations to date.

Concentrate Handling

Finished lead and zinc concentrates flow by gravity to two 29 ft., 6 in. x 11 ft. thickeners which were constructed by installing mechanisms and superstructures on two Blaisdell tanks. The underflows from the thickeners, at from 55 to 65% solids, are delivered by diaphragm pumps to a 3-disc filter for lead and a 6-disc filter for zinc concentrate. The lead filter discharges the concentrate directly into a storage bin of 200 tons capacity. The lead concentrate is conveyed from the bin by means of two connecting conveyors which deliver the material to a car loader when loading box cars, or over the side of gondola cars when that type car is supplied. The zinc filter discharges concentrate onto an 18-inch conveyor which delivers the material to a bin for box-car loading, or to a cross conveyor which discharges directly into a gondola car when that type of car is being loaded. The zinc concentrate storage bin has a capacity of 400 tons. Vacuum for filtering is normally maintained around 21 inches of mercury by a 16 x 12 inch single cylinder vacuum pump, converted from an air compressor. The overflows of the concentrate thickeners pass to settling ponds which in turn produce waste overflows.

Shortly after the start of operations and at the request of the smelters, the method of shipping concentrates was changed from box car to gondola shipments. This was the cause of some concern for it was thought that excessive losses in transit would result from the use of open gondola cars, especially in the case of the zinc concentrate where the distance to the smelter is 1100 miles.

Gondola shipments have proved entirely satisfactory, however, and now the gondola type of car is actually preferred to the box car for concentrate shipping. Of a total of 232 cars of zinc concentrate shipped to date, 191 were gondola cars, and the average loss in transit for all shipments was 189 pounds per car.

Tailings Disposal

Approximately two-thirds of the total tonnage milled leaves the plant as waste tailing. This tailing pulp flows by gravity through a 6-inch line constructed of transite and wood pipe to an old tailing pond 3200 feet from the mill. The pond has an area of approximately 2,500,000 square feet and a border about 10 feet in height encloses the pond. No further border building will be required for a long period of time, so the mill tailing is simply spilled into the pond and a clear overflow drawn off through a weir. No tailing water is recovered.

Sampling and Testing

A sample of the crushed ore as it discharges into the storage bin is manually collected at 30 minute intervals. This sample is dried and the moisture content applied to the mine shipment scale weights for the dry ore receipt figure. Weightometer tonnage, as ball mill daily feed, is adjusted to meet the ore receipts. Shift samples are taken at 30 minute intervals, of the flotation feed, lead concentrate, zinc concentrate, and mill tailing. The feed and tailing samples are cut by electrically controlled automatic samplers and the concentrate samples are cut by hand. Lead and zinc concentrate shipment samples and car weights supply metal statistics for use in checking mill production calculations. All sample pulps are prepared for assaying in the sample preparation room at the mill and the sacked pulps are sent to the mine assay office for assay.

Dust Control

Every effort is made to provide working conditions that are not injurious to the health of the employees. All chutes, junction boxes, and crushing equipment are enclosed and connected to the dust collecting system. Equipment for the control of dust created in crushing operations consists of a 6500 C.F.M. exhauster and a 7-foot dry collector. Dust settled in the collector is intermittently discharged onto the stream of ore conveyed to the fine ore bin. In filtering lead concentrate, it developed that a small amount of the material, as fine dust, was carried back into the mill by an uprising current of air through the lead concentrate storage bin. A hooded enclosure around the filter, with connections to an exhauster, has proved effective in controlling this condition.

The nature of the concentrate loading operations is such that some dust escapes around the loading area, and makes the matter of clean-up of great importance in minimizing the hazard of lead contamination. In this connection, the frequent washing of floors is insisted upon, also the wearing of rubberized gloves in performing certain concentrate loading operations.

Approved respirators are furnished men engaged in ore unloading, crushing and concentrate loading.

MILL DATA

In the following tables are shown data covering results obtained to date, from November 17, 1945 to March 19, 1946. Since this four months' period represents the break-in period of a new mill, subsequent operations will no doubt result in additional improvements in metallurgy.

Table 1 summarizes metallurgical data for the mill operations to date.

Table 2 presents screen analysis of mill feed and grinding products.

Table 3 presents screen assay analyses of a monthly composite of mill products.

Table 4 shows the reagent consumption for operations to date.

Table 5 shows the power distribution for operations to date.

Table 1

Metallurgical Data
November 17, 1945 - March 19, 1946

Wet tons of ore milled	52878.00
Percent of moisture	1.82
Dry tons of ore milled	51916.19
Operating days	121
Dry tons milled per operating day	429.06
Dry tons milled per 24 hours of running time	466.45
K.W.H. per ton of ore milled	20.37
Ball consumption per ton of ore milled	1.23
Dry tons lead concentrate produced (Smelter Receipts)	6409.029
Dry tons zinc concentrate produced (Smelter Receipts)	11614.830
Percent moisture lead filter product	8.0
Percent moisture zinc filter product	9.2
Ratio of concentration lead concentrate	8.1005
Ratio of concentration zinc concentrate	4.4698

Product Analysis

	Ounces per ton		Percent			
	Gold	Silver	Copper	Lead	Zinc	Iron
Heads	0.017	3.42	0.82	6.41	13.60	9.93
Lead Concentrate	0.059	15.72	4.49	43.46	9.64	12.99
Zinc Concentrate	0.014	3.84	0.79	2.75	50.96	6.28
Tailing	0.010	0.95	0.13	0.66	1.55	10.60

Product Recovery

	Gold	Silver	Copper	Lead	Zinc	Iron
Lead Concentrate	43.62	56.79	67.77	83.70	8.75	16.15
Zinc Concentrate	18.73	25.14	21.61	9.60	83.82	14.15
Tailing	37.65	18.07	10.62	6.70	7.43	69.70

Table 2

Typical Screen Analyses of Mill Feed and Grinding Products

Percent of Total Weight

Mesh	Mill Feed	Mill Discharge	Classifier Sand	Classifier Overflow
.525 in	13.75	-	-	-
4 mesh	43.62	12.94	-	-
6	7.46	1.93	-	-
8	4.79	1.40	-	-
10	3.74	1.24	-	-
14	3.76	1.74	-	-
20	2.93	2.19	30.85	-
28	2.15	2.98	4.87	-
35	1.91	4.98	7.22	-
48	1.65	7.80	9.95	-
65	1.56	10.97	12.70	4.01
100	1.48	11.95	12.41	6.58
150	1.38	8.47	8.08	10.19
200	1.36	6.47	4.58	12.04
-200	8.46	24.94	9.44	67.18

Table 3

Screen Assay Analyses - Month Composite

Screen Size Mesh	Percent Weight	Assays - Percent Copper	Lead	Zinc	Metal - Percent of Total Copper	Lead	Zinc
Heads							
on 65	3.40	0.22	0.74	3.91	1.03	0.40	1.07
on 100	6.80	0.29	1.37	6.53	2.70	1.47	3.60
on 150	10.30	0.53	2.05	10.90	7.48	3.34	9.11
on 200	13.00	0.54	2.68	14.20	9.53	5.50	14.97
thru 200	66.50	0.87	8.50	13.21	79.26	89.29	71.25
	100.00				100.00	100.00	100.00

Screen Size Mesh	Percent Weight	Assays - Percent Copper	Lead	Zinc	Metal - Percent of Total Copper	Lead	Zinc
Lead Concentrate							
on 65	0.10	15.33	8.85	3.50	0.32	0.03	0.05
on 100	0.30	15.33	8.85	3.50	0.97	0.05	0.15
on 150	1.60	10.96	24.00	5.45	3.69	0.72	1.27
on 200	5.40	7.08	36.29	6.37	8.05	3.66	5.01
thru 200	92.60	4.46	55.27	6.94	86.97	95.55	93.52
	100.00				100.00	100.00	100.00
Zinc Concentrate							
on 65	0.60	1.65	7.88	39.58	1.32	1.70	0.45
on 100	2.80	0.82	3.35	47.40	3.07	3.38	2.51
on 150	9.20	0.80	3.45	48.63	9.81	11.44	8.46
on 200	14.40	0.92	2.96	49.76	17.67	15.36	13.54
thru 200	73.00	0.70	2.59	54.39	68.13	68.12	75.04
	100.00				100.00	100.00	100.00
Tailing							
on 65	4.40	0.15	0.94	2.47	3.93	4.29	8.89
on 100	8.00	0.19	0.99	1.49	9.05	8.21	9.75
on 150	11.10	0.24	1.18	1.23	15.83	13.58	11.16
on 200	12.00	0.19	0.89	1.08	13.57	11.07	10.60
thru 200	64.50	0.15	0.94	1.13	57.62	62.85	59.60
	100.00				100.00	100.00	100.00

Table 4

Reagent Consumption

	Pounds per ton of ore milled	Point of addition
Sodium cyanide	0.67	Ball Mill
Zinc sulphate (25.5%)	1.85	Ball Mill
Sodium carbonate	0.36	Ball Mill
Thiocarbanalid	0.12	Ball Mill
Lime (Hydrate)	3.24	Lead circuit tailing box
Copper sulphate	1.78	Lead circuit tailing box
Cresylic acid	0.24	Classifier overflow Zinc circuit cells 3, 7
Sodium ethyl xanthate	0.12	Classifier overflow Lead circuit cells 7, 10 Zinc circuit cell 3
Potassium amyl xanthate	0.05	Zinc circuit cell 6

Table 5

Power Distribution

	K.W.H. Per Ton of Ore Milled	Percent of Total
Unloading and crushing	1.42	6.95
Grinding	9.79	48.07
Flotation	6.60	32.38
Concentrate handling	0.95	4.68
Miscellaneous	1.61	7.92
Total	20.37	100.00

Supplement

Since the preparation of this article, the mill capacity has been doubled and the flow sheet for the zinc circuits changed. Doubling the capacity was effected through duplication of the original installation for fine grinding and flotation, addition of a 75-foot thickener and an 8-foot, 6-inch, 6-disc filter for zinc concentrate handling, and changes in the lead concentrate handling scheme to provide for the greater tonnage of lead concentrate produced. Operations to date with the increased tonnage, indicate a slight improvement in metallurgy. Recoveries of all metals are approximately the same as shown in the preceding data and grade of the zinc concentrate has been raised.

"BLOCK CAVING AND GLORY HOLE MINING OPERATIONS AT THE COPPER QUEEN MINES"

By

F. W. NELSON

Presented at Supervisory Staff Dinner of
Phelps Dodge Corporation, Copper Queen Branch
September 19, 1929.

Block Caving and Glory Hole Mining Operations
at the Copper Queen Mines.

The Location and Extent of the Porphyry Ore Bodies

The outlines of the Porphyry ore bodies, as to vertical and horizontal extent, have been determined principally by churn drilling and to a lesser extent by underground prospecting. In a number of cases this underground work has been used largely to check the accuracy of the churn drill hole sampling. The recoveries in steam shoveling Sacramento Hill and also in the underground caving have tended further to show the accuracy of this churn drill sampling.

As far as known, the Copper Queen has four Porphyry ore bodies. All of these ore bodies are related to the so-called Sacramento Stock. The porphyry ore bodies are designated by their position with reference to the East Ore Body, which is cut by the Sacramento Shaft and which extends northward under the remaining high portion of Sacramento Hill. This ore body extends vertically from about the 200 level for a depth of about 250 feet. A portion of this ore body cannot be mined without jeopardizing the Sacramento Shaft and the surface plant. Those caving blocks which are now laid out are far enough from the shaft to make it secure during the life of these blocks.

Extending from the lower portion of the East Ore Body in the direction of the Lowell Depot is the Southeast Extension ore body. This body extends from about the 500 level to the 1200 level. The ore body is about 100 feet in width and averages 450 feet in length down to the 950 level. The dip is at about 70° to this depth. Near the 950 level the ore body tends to flatten out into a large inverted cone shaped mass which as far as known, has its apex near the 1200 level.

The third ore body lies to the northeast of the Sacramento Shaft and extends from about the 600 to the 800 levels. No underground development has been done on this ore body, known as the Northeast Extension of the East Ore Body.

The best known of the Porphyry ore bodies is the Sacramento Pit or West Ore Body. This ore extends to the 400 level or about 135 feet below the bottom of the present lower bench of the Pit.

The grade of the ore in these ore bodies is not uniform. Included in the milling ore are irregular masses of high grade material. There are also horses, or bodies of waste.

Extent of Stoping Operations in the Porphyry ore bodies prior to the Steam Shovel and Caving Operations

Before it was decided to steam shovel the West Ore Body, a stoping section had been laid out on the 400 level, but this was abandoned.

On the 200 and 300 levels of the East Ore Body, a small amount of material was mined underground during the operation of the pilot mill for the Warren Concentrator.

The most extensive stoping operations were above the 1000 level in the Southeast Extension, where a large tonnage of the higher grade porphyry ore was top sliced and a lessor amount extracted by square setting. The lower grade material around these timbered stopes will probably be caved with difficulty.

The General Principles and Practice in Block Caving

Caving was probably discovered when the first miner encountered heavy ground. Later, caving was practiced in some cases where the cry for tonnage became too urgent upon the Boss and he resorted to "pulling the Lagging". Caving as a mining method has a number of variations, but time limitation prevents a discussion here of any other than Block Caving.

Block caving, by which most of the low grade copper ore in Arizona is being mined, is a development since about 1910. In this system the extent of the ore body must be determined in advance. The development program does not permit prospecting but merely serves as a check on conclusions already drawn. If the ore body is very large it is mined in a large number of sections or blocks. The horizontal and vertical extent of a given block is determined by the character of the ground to be caved and also by the length of time permissible for the development of the block. The vertical heights of caving blocks have been increased very materially in recent years where the shape of ore bodies has permitted such an increase. This is due in part to the elimination of hand tramming and also to standardization of the caving systems to suit local conditions. In the Southeast Extension ore body the height of the caving blocks has been increased from 60 feet on the 600 level to 125 feet on the present 850 caving level. The lift below the 1000 level will probably be about 150 feet. Except for the earlier stopes the heights of the blocks have been limited by the position of the ore. In the East Ore Body the lifts will be such as to take in the entire thickness of the orebody. The maximum height will be about 250 feet. In any caving operation where the height of the caving block to be taken is being considered, the possibility of greatly increased repair costs because of the longer life of the stope must be compared with the lower initial development cost.

The horizontal dimensions of a caving block may in part be determined by the size and shape of the orebody. It is also advisable to have the production coming from a number of sources rather than a very few. This will give a more uniform tonnage production and also a more uniform grade of ore. In any ore body the size of a caving block must be determined by trial and error. Very few errors are allowed. In the Copper Queen the approximate size of the caving blocks has been 100 feet by 125 feet.

When the size and position of the caving blocks have been determined, the layout of the haulage level must be decided upon. In some cases the haulage level position may be such as to utilize a level already opened up in prospecting or in mining other ore bodies. If such a level is utilized, the expense and time of running a new shaft pocket may be saved. Ordinarily, old mine levels are not designed to handle the large tonnages and at best considerable alteration is necessary. Most of our limestone drifts are about 6 feet by 8 feet in cross-section. The porphyry haulage drifts average about 8 feet by 9 feet in cross section. In the Copper Queen caving operations, a raise of less than 60° pitch has been found unsatisfactory. Knowing the elevation of our grizzly level, the spacing of the haulage drifts can be determined. When the haulage drifts are complete those haulage raises to be used in the first caving block are run to the grizzly level. The grizzly level may have been driven before the raises are complete in order to save time. When the haulage raises and the grizzly levels are completed, the work of driving the fingers or openings from which the caved ore is to be extracted is begun. The first fingers run are those around the margins of the caving block, because these openings are to be extended upward through the entire thickness of the ore. These openings are not timbered any more than is necessary and are ordinarily stull raises. All of these vertical raises are connected by horizontal openings at regular intervals.

In some districts, such as at Ray, shrinkage stopes are run through the entire thickness of the ore around the margins of the block. In such practice the broken ore must not pack and the ground must be of such character as to permit long openings. Because the shrinkage stopes are empty when undercutting is started the entire mass of ore may drop as a unit. In such cases, the grizzly level is completely caved in and redevelopment becomes necessary under very trying conditions. This possibility of the block coming in as a unit has necessitated the present practice of weakening rather than removing the side support of the caving block. Moreover, the cost in such practice is less.

When the cutoff or boundary stopes are complete the work of undercutting or removing the support beneath the block is begun. The undercutting level is ordinarily about 16 feet above the grizzly level timber. This feature makes it possible to have a support of solid ground above the place from which men will later extract the caved ore. The support beneath a caving block is not removed at one time. Small units are removed as rapidly as advisable, starting from one corner of the block. The determination of the rate at which undercutting should be done is based on judgment. If undercut too rapidly, the block does not take weight and cause a crushing action above the area undercut. If not undercut rapidly enough, the weight in the undercut portion may be transferred into the remaining pillars and make their removal dangerous or impossible. Where solid pillars are left on the undercut level, a support is given to the ground to be caved and the weight is transferred to the grizzly level and these openings are apt to be closed. It is essential that all support must be removed from beneath any caving block.

When the block has been prepared as described above, the solid ground in the block should start to cave due to its own weight and lack of support. The tendency is to form a dome having its maximum height near the center of the block. Any dome is dependant upon the support around its base for its stability. In the caving block we have deliberately weakened this support and the successive domes collapse as they tend to form. This collapse or sloughing is slow at first but accelerates until the block as a unit collapses, and breaks up. The coarseness of this broken material effects the cost of drawing off of the ore in the block. In some cases the material is fine enough to pass readily through the fingers and on to the grizzlies. In other cases where the material is very coarse, a great deal of blasting is required to get the boulders through the fingers. When such material reaches the grizzlies, further blasting of the boulders may be required or at least a double jack massage.

As I have already stated, the amount and the grade of the ore in a given caving block is determined by prior prospecting. After the block is caved, the problem of recovering the estimated pounds of copper in the block presents itself.

The glass model which I have here may be used to illustrate the method used in recovering the estimated pounds of copper in a caving block. I realize that the conditions and the character of the material mined from underground are not exactly comparable to those of the model. In spite of this, however, the fundamental principles involved in removing material from this model should be the same as those used in removing ore from an underground caving block.

Let us assume that someone has a number of diamonds which they wish to give to us in a rather novel manner, under the following conditions:

The diamonds are mixed with 100 pounds of white sand and the mixture placed in the bottom of this model. On top of this material red sand is placed so as to fill the box. We are to be allowed to remove only 100 pounds of material through the numerous openings in the bottom of the box and the diamonds in this 100 pounds of material are to be ours. Of course, we have no desire to include any of the red sand in our 100 pounds of material and before long we conclude that the best method of recovering only the white sand is by removing a small amount of material from each opening in regular sequence.

In a caving block, the openings in the bottom of the model compare with the fingers of the caving block above the grizzly level. The white sand is the ore and the red sand the capping or barren material. The diamonds are the copper. If in the above illustration no limit were placed on the amount of material we could remove, we could have taken all the material in the box and have recovered all of the diamonds. The unfortunate thing is that in the illustration and also in block caving the amount of material to be removed is limited.

In block caving, records of the amount of ore drawn from each finger are kept and also a record of the grade of this tonnage. Each finger has a certain tonnage and amount of copper to be recovered. In the case of individual fingers, the pounds of copper recovered may be above or below the estimate. For an entire block, however, the recoveries as a rule approximate that of the estimate. In each stope there is a man whose sole duty is to keep a record of the amount of ore drawn from each finger. He also takes samples of the ore drawn on his shift. From these data records of the stope production are kept and those places from which ore is to be drawn are designated.

A block is not abandoned until the grade of the ore becomes so low as to make its extraction unprofitable.

Block caving operations at the Copper Queen were started during the Spring of 1925 on the 600 level of the Southeast Extension ore body. Since that time caving levels have been worked out on the 650 and 725 levels. The present 850 level will be worked out early in 1930.

On the 600 and 650 levels, the Morenci timbered slide was used with moderate success. On the 725 level these slides were discarded and a layout devised locally employed. On the 850 level the spacing of the fingers was increased and this inovation has tended to materially decrease repair costs. The 850 level will also see the complete elimination of hand tramming. During the development of the present caving practice the heights of the caving blocks have been increased as much as permissible by the pitch of the ore body. The first blocks were 50 feet in height and the present 850 level has blocks 125 feet in height. On the 950 caving level the layout to be used will be identical to the East Ore Body stope which I will later describe. It should also be mentioned that the tonnage production of the Southeast Extension ore bodies has shown a gradual increase without any material increase in the size of the stoping areas. Also, the production at the present time is much more uniform than in the earlier operations.

In order to clarify some of the points that I have tried to bring out, it may be well for us to inspect the layout of one of the caving blocks of the East Ore Body. In this block work has been done toward its development on 4 mine levels, the 500, 400, 300 and 200. The motor haulage is on the 500 level which will serve all of the present blocks of the East Ore Body as well as the West Ore Body Glory Hole operations. All of the workings on the 500 level are new and have been driven in the past two years. The shaft pocket into which the ore is dumped from this level is new and has a capacity of nearly 1,500 tons of ore. The ore is loaded into the skips on the 600 level. The grizzly level is on the 400 and some of the old workings have been utilized to advantage. To facilitate operations, the cutoff or boundary stopes of the block have been driven from the 300 and 200 levels. In other words, these cutoff stopes were driven upward from the 400 to the 300 level and then from the 300 level to the 200. On the north side of the block, these openings extend above the 200 level a short distance. On the 400 level our grizzly drifts are spaced at 40 foot centers and are run at right angles to the haulage drifts below and directly above the haulage raises, which are also at 40 foot centers along the haulage drifts.

Openings into the grizzly drifts from below are spaced at 20foot centers along the drift. These openings are covered by rails spaced at 16" centers so as to prevent large boulders from entering the haulage raise. From the grizzly openings are driven upward at right angles to the grizzly drift so that they will intersect the undercut level at a point 8 feet above the grizzly drift timber, on 20 foot centers. These openings are known as "fingers" and are the holes through which all of the broken ore is extracted. These fingers are connected on the undercut level by horizontal openings. The blocks of solid ore remaining on the undercut level after the fingers have been connected are known as "pillars". These pillars must be removed before the block will cave. Into each of the pillars a "stub" drift is driven in order to facilitate the removal of the pillar. When the cutoff stopes were complete the "shooting" in of the block was started as shown. Two pillars have been shot in each three days by a gang of four men. Holes are drilled outward into the pillars from the various openings and also upward. The holes are then loaded and all of those in the two-pillars are blasted simultaneously with electric detonators.

After the pillars are shot in as much broken material as possible is drawn out from the fingers beneath the pillars, if this is not done, no room remains for more rock to break up and weight may be transferred to the grizzly drift below. After this swell has been drawn off, it may be some time before more ore can be drawn unless the block has started to crush. This general caving does not begin until about two-thirds of the block has been shot in, except in cases where the ground is very heavy.

It should be borne in mind that after a pillar is shot in, that portion of the undercut level is abandoned and all further mining is done from the grizzly level. In undercutting, manways are always provided ahead of the pillars to be blasted as a means of escape, These means of escape are very important and greatly facilitate the handling of supplies. The last pillars in a caving block are usually the most difficult to bring in, especially if the block is taking excessive weight. Here speed is essential,

On the undercut level as little timber as possible is used. Any timber that is used must be blasted out when the pillars are shot in.

After the block has been entirely undercut and shot in, the task of drawing the broken material onto the grizzly level and also keeping the various openings in repair presents itself. The goal is then to recover all of the estimated pounds of copper in the block without any excess tonnage.

The glory Hole Operation

The mining of the lower portion of the Sacramento Hill or West Ore Body by steam shoveling presented a number of haulage problems which probably would give an increase in mining costs and also a decreasing production. For this reason it was decided to recover this ore from underground by means of Glory Holes. To recover this remaining ore it was calculated that six glory holes could be so spaced that most of the ore could be drawn out on the 400 level. When mined out these glory holes will leave a crater with slopes of about 45°. From the 400 level to the 500 level, large haulage pockets have been brought up directly beneath each glory hole from a large loop drift. The vertical distance from the 500 haulage level to the 400 is 90 feet; from the 400 to the bottom of the Pit is 135 feet.

The haulage drift to serve the glory holes was driven from the 500 level of the Sacramento Shaft over 2000 feet. When the loop was completed, the six haulage raises were driven. These raises have five compartments for the first 50 feet above the haulage level. Above this height the size of the raise was cut down to about 8 by 5 feet. These raises are raw, but the chute fronts on the 500 level are heavily timbered and reinforced with rails and boiler plate, and the collars below the 400 level are lined with reinforced concrete. On this concrete collar 80 pound double rails were placed so as to leave 20" openings between them. Above the grizzlies a heavily timbered opening was made measuring about 10 feet in width by 15 feet in length and 8 feet in height. This timber is supplemented by steel rails and flat cable. This protection is needed because of the large boulders which may fall from the Pit and also because of the large amount of blasting needed to break the boulders. In front of the grizzlies and bulldozing chambers, an L shaped square set, 8 post raise was run up a distance of 90 feet above the level. These raises could not be holed to the surface because the steam shovels were still operating above them. In these raises 5 foot posts were used. To remove the timber from the completed raise one-half of the posts on each floor were drilled, but in no case were posts above each other drilled on adjoining floors. All of the drilled posts were then blasted, using a number of delay electric primers, the upper timber being shot first. The broken timber was then withdrawn from the 400 level without any material difficulty. The hole so stripped was over 10 feet in diameter and came within 40 feet of the surface. Churn drill holes were then sunk from the Pit above the top of each underground opening. Several of these churn drill holes were blasted simultaneously and an opening made to the surface.

When production started a number of churn drill holes were spaced around the openings and were blasted. The resulting boulders were very large and those which could be reached on the surface were drilled and blasted with pluggers by Mexicans working on ropes in life belts. As the craters opened up the slopes became less steep and more opportunity was given to break the large boulders before they entered the glory holes. Also, a larger proportion of the ground was broken with the pneumatic drills. As the glory holing operation progresses, the use of the churn drills in breaking ground will diminish because of the increasing amount of ground toward the bottom of the holes.

We should bear in mind that in the Glory Hole operation, all of the ore is broken by drilling and blasting where as in the caving operations, the bulk of the ore is broken and crushed by its own weight in caving.

Motor Haulage and Porphyry Production

Our present motor haulage levels are on the 500 and 1000 levels of the Sacramento. The 500 level haulage will serve the entire East Ore Body and also the Glory Hole operations of the West Ore Body. Three 6-ton trolley motors are used on this level and each pulls 60 tons of ore in 4-ton cars. On the 1000 level $3\frac{1}{2}$ ton cars are employed and are pulled by storage battery motors.

When the block now being caved in the East Ore Body reaches full production, its tonnage output should be 1000 tons per day. The production of the Glory Holes will be 2000 tons per day and the production from the 1000 level of the Southeast Extension, 1500 tons per day. It seem probable that these tonnages will be increased as the operation develops.

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Mine Ventilation

Delivered before The Copper Queen Club
May 28, 1930 by H.N. Walcott.

The ventilation of mine workings possesses three primary objectives; viz; supplying fresh air to working faces, reducing underground temperature to a point within the limits of human comfort, and the removal of foul air, dust or harmful gasses to the surface. These three objectives are sufficiently obvious as to need no explanation except with respect to that which deals with the lowering of temperatures. ~~The cooling power of a current of moderately dry cool air upon the human body is of course known to be the major premise of~~ In underground ventilation, the movement of air through hot workings exerts a cooling effect not only upon the workmen with whom it comes in contact, but upon the workings themselves. The importance of the last named function is liable to be overlooked or at least regarded rather lightly. ~~[illegible]~~, There has never been any extensive or systematic study made of the effect of ventilation upon rock temperatures over a long period of time. However from the data available, and from logical reasoning it must be realized that over a period of many months or years the total cooling effect exerted by the absorption of heat from the rock into the air stream ~~must be~~ is of no little importance. This cooling is not confined to the rock surfaces, but extends for varying distances into the virgin rock, depending of course upon the difference in temperature between the rock and the air stream. Lacking some source of heat in close proximity, such as crushed sulphide ore bodies, decaying timbers or mine fires, the ultimate effect of the circulation of cool air is a lowering of temperatures throughout the workings, with resultant cooler ventilating air and more comfortable working conditions.

In some localities where rock temperatures are abnormally high and the character of the ore bodies will permit, it is considered advantageous to do some of the preliminary development work a year or so in advance of the time when it will be actually needed in order to allow the block to be cooled somewhat before stoping operations begin.

Natural Ventilation

Underground ventilation is accomplished either by mechanical means, natural drafts, or a combination of both. Under certain conditions, where workings are shallow or not too extensive, where rock temperatures are not abnormally high, and where topographic irregularities make for air columns of different weights at the various surface openings, the natural movement of air may furnish satisfactory ventilation for most of the working faces. However, even under ideal conditions natural air movement is subject to seasonal variations and other factors which make it unsatisfactory for the majority of mines. It therefore becomes necessary to supplement or entirely supplant it with mechanical air movers which are subject to human control and are not affected by factors which seriously impair the action of natural draft.

In the mines of the Copper Queen ~~[illegible]~~ there is one area, the Southwest-Czar-Holbrook division, which depends mainly upon natural draft for ventilation. During the winter months when there is a great difference between surface and mine temperatures, there is a comparatively large volume of air in circulation through the workings. However, even under these conditions there are several local areas in which it is necessary to use auxiliary mechanical means to produce temperatures which are bearable, but by no means comfortable. (~~[illegible]~~)

During the warmer months of the year the natural drafts are greatly diminished. The direction of air flow in many of the openings is reversed, and an entirely new set of difficulties is presented toward efforts to cool the workings. The great number of openings to the surface in this area is actually a hindrance rather than a help in promoting natural drafts, the effect being similar to that of removing the lids of a coal range and thereby checking the draft through the grates.

The effective control of natural air movement throughout this area would necessitate the construction of numerous doors and brattices, an expensive process. Many of these stoppings would be used for part of the year and then removed when seasonal changes caused a reversal and consequent re-distribution of air currents. It is obvious that such a procedure is bound to have serious drawbacks, and that some portion of the workings in this area will always be unsatisfactorily ventilated.

It appears logical to assume therefore, that mechanical ventilation which would be vastly easier to control in this area will be the ultimate solution of the difficulties now experienced. The cost of installation and operation of fans should be more than offset by increased working efficiency and the safeguard against sudden dangerous reversals of air currents in case of an underground fire.

Mechanical Ventilation

With the exception of the divisions just mentioned, all of the active underground workings are mechanically ventilated. Seven primary fan units ~~and one booster unit~~ are the source of air movement. The location of these units and the workings which they ventilate are shown on the accompanying drawing, the original of which was prepared by Mr. G.E. McElroy, Mining Engineer with the U.S. Bureau of Mines. Reference to this drawing will render it easy to follow the subsequent descriptions.

Four of the primary units are used to ventilate the various portions of the porphyry caving blocks and the grizzly chambers under the Sacramento pit glory holes. The Spray 600 level fan intakes from the Spray Shaft, forcing the air up drainage raises to the 500 level loop. Here the air is split, part of it going up manways to the 400 level grizzly chambers and out to the Gardner Shaft or up the glory hole raises to the surface. The balance of the air goes over on the 500 level, and up to ventilate the grizzly lines on the 450 and 400 levels of the East ore body. ~~This air joins with~~

~~fresh air from the Sacramento shaft,~~

The exhaust air from the East ore body is pulled up from the 400 level through a series of raises to the Sacramento tunnel level by an exhaust fan which is installed at the portal of the tunnel. ~~This~~ ~~might be regarded as a booster, but~~ ~~exhaust fan~~ is ~~classified~~ as a primary fan ~~because of the fact that it derives a portion of it's air from a source which is independent of any other fan.~~

The third and fourth primary units used for ventilating the porphyry mines are the 1000 level and 1200 level Sacramento fans. These take fresh air from the Sacramento shaft and force it up through various raises to the 1100, 950 and 850 levels from where it exhausts on the 800 and 600 levels to the Gardner and Silver Bear shafts.

The limestone workings of Division 7 in the Sacramento are ventilated by two primary blowers, those on the 1500 and 1800 levels. These fans force air from the Sacramento shaft through the various stopes in the Hardscrabble claim, and those in the 14-10 country. A portion of the exhaust air from these fans goes to the Lowell shaft on the 1500 and 1600 levels, some to the Hoatson shaft through old side-line connections, and the balance to the Lowell and Gardner on the 1400 level.

Fresh air for the Lowell and Gardner stopes is supplied by the 1400 Dallas fan. Part of the air from this fan goes over on the 1400 level and up through the 14-44 and 13-34 country and on up to the 1100 level where it ventilates ~~is picked up by a large booster unit and forced through~~ the 11-40 country on it's way out to the surface through the Silver Bear. The balance of the air from the Dallas blower goes up through raises to ventilate the Lowell workings as high as the 1000 level. Here most of it goes to the surface via the Gardner shaft and Lowell air shaft, although a considerable quantity escapes through contiguous C&A workings along the Oliver side line (~~[illegible] Claim~~).

General data on some of the primary fans appear in the following table:

SIZE AND TYPE OF FAN	LOCATION	VOLUME AIR DELIVERED	RESISTANCE IN INCHES W.G.	H.P. CONSUMED	MECHANICAL EFFIC.
Sturtevant Multivane, Size 12 Design 3 - Double Inlet, double width.	1800 SAC	40,000 c.f.m.	2 5/8	61.1	27.0 %
Sturtevant Multivane, Size 11, Design 2 - Single Inlet, single width	1500 SAC	34,000 c.f.m.	2 9/16	25.7	53.3 %
Sturtevant Multivane, Size 11, Design 3 - Double Inlet, double width.	1400 DALLAS	80,000 c.f.m.	1 3/4	57.9	38.0 %
American Sirocco, No. 7 Single inlet, single width	1000 SAC	25,000 c.f.m.	2 7/16	40.2	24.1 %
Sturtevant Multivane, Size 11 Design 2 - Double Inlet, double width,	600 SPRAY	28,000 c.f.m.	Not	Obtainable	at Present
Sturtevant Multivane, Size 7, Design 3 - Double inlet, single width.	SAC TUNNEL	25,000 c.f.m.	"	"	" "

It should be understood that the above data are only approximate, since ~~we have~~ at present we lack the means and equipment for obtaining strictly accurate test results. The tests which are made should therefore not be used as a basis for comparison with the fan performances given in the manufacturers ratings. Measurements of the volume of air delivered and of the pressure at which it is delivered cannot be accurately obtained various speeds and maximum pr...

at any underground fan installation. However, the tests which we do make are of value in determining the relative performance of any given fan from time to time.

Characteristics of Copper Queen System

~~As previously stated, ventilation of the underground mines of the Copper Queen is accomplished by the use of underground fans to blow air through the various openings. The majority of the larger mines throughout the country use exhaust fans on the surface to draw air through the workings.~~ ~~Both systems have their advantages, but the~~ ~~. Local conditions in Bisbee have influenced the adherence to a system which is not favored elsewhere in Arizona, viz; the presence of a number of sealed off ~~ff.~~ mine fires in the active working areas.~~

The location of primary fans underground as in Bisbee makes for higher resistances and consequently higher pressures since the flow of air is restricted to fewer channels than is the case where surface exhaust fans are used. General mining practice favors the latter method of ventilation, and the Copper Queen and C+A are the only large mines in Arizona which adhere to the practice of underground installations of primary fans. The factor which has caused the departure from general practice in Bisbee has been the presence of mine fires in the active workings of both companies. Because of side line connections at various points, each company has felt the effects of these fires when they occurred; and since it is almost impossible to ~~effect~~ maintain absolutely air-tight seals around a fire area, there is the ever present necessity [illegible] combatting the ga[illegible] which emanate from the various fires. The natural tendency under such conditions has been to raise the pressure of the air in the workings outside of the bulkheaded fire zones and thereby prevent the escape of gas through or around the bulkheads. To a certain point this theory is undoubtedly sound, and anyone can ~~[illegible]~~ answer objections to it by pointing out the fact that it has worked satisfactorily and accomplished the results which were expected of it. However, if we are to use this theory as an argument against the lower pressure exhaust system which is used elsewhere, we must follow it a few steps further.

Since it is obvious that neither the Copper Queen nor the C&A could ~~change~~ decrease their pressures to any great extent unless both companies acted accordingly and simultaneously, let us assume for the sake of argument that the mines of the Copper Queen are isolated and not connected with any other workings. In view of the fact that the perfect sealing of any fire is practically impossible, it follows that any difference in pressure on the two sides of a bulkhead is bound to cause the air to move toward the side of lower pressure through cracks in the bulkhead or around the edges of the bulkhead. The greater the difference in pressures, the larger will be the flow of air. ~~In the case of an underground blower such as ours, the direction of flow will be away from the blower where the pressure is higher. With an exhaust fan the flow would be toward the fan.~~ In ~~either~~ any case ~~[illegible]~~ fresh air is caused to enter the sealed area and thereby furnish the fuel for continued combustion therein.

And the higher the pressure which is maintained, the more fuel is being added to the fire. If it happens that the air which is forced through the fire area issues at points from which it is exhausted without passing through active workings, then it probably causes little or no trouble because of it's gas content. Naturally the Copper Queen ventilation scheme is predicated upon such an arrangement.

Continuing upon our assumption that the Copper Queen mines are not connected in any way with neighboring workings, what would be the effect if surface exhaust fans were to be substituted for the present underground blowers? The result of such a move would of course be to drop the underground pressures from a point considerably above atmospheric pressure to a point somewhere below atmospheric pressure. We would still be causing air to move through the fire areas, but certainly in no greater quantity than at present. And presupposing of course that our exhaust fans were placed at inactive openings, why should the gasses bother us any more than they do now? So long as bulkheads or brattices are subject to leakage, our fire zones will continue to receive some supply of fresh air; and it is hard to see the difference between pushing the air through these zones with an underground blower, or pulling it through with a surface exhauster. Where then does the advantage of underground fan installations lie?

In the case of the Copper Queen and C&A, each company must hold a pressure to offset that being applied by the other, and whenever these pressures are unequal the company which carries the lower pressure gets the benefit of the gas or exhaust air from the company with the higher pressure. Normally the difference in pressures between workings along either side of the property lines is so small that the air flow one way or another is of little consequence, and fire gasses are sufficiently diluted so as to cause no trouble.

If however for some reason, such as shutting down a blower, the pressure on one side drops considerably below that in the adjoining workings, the flow of gas toward the side of low pressure becomes immediately apparent; and if the condition is allowed to continue for any length of time, it becomes necessary to abandon all workings through which the gas laden air currents are passing.

In view of the fact that both the Copper Queen and C&A have already made numerous underground blower installations, it is not to be supposed that both companies would make a simultaneous change to surface exhaust systems. For future installations however, in areas which are not and will not be affected by any of the present active fires along property boundaries, the matter of a surface exhaust system might be well worth consideration. The only portion of the Copper Queen workings which really necessitates the continuance of our higher pressure blowers is that in Division 7 which is contiguous to the C&A fire area along the East side of the Hardscrabble Claim. Other fire areas such as the 14-10 and Lowell unit in all probability cause no more trouble with a low pressure exhaust system than they do under present conditions.

The strongest arguments in favor of surface exhaust systems may be summed up as follows:

(1) Accessibility in case of underground fires.

(5) Lower pressures throughout the ventilating system and consequently a greater volume of air in circulation for any given power consumption.

(3) Short circuiting of air currents results only in the leakage of fresh air into the exhaust air streams, whereas in the case of underground blowers the reverse is true, and any short circuit results in the contamination of fresh air by exhaust air.

(4) Fewer regulating doors are necessary, and the blower station air locks along main haulage ways are eliminated. This means a reduction in the chances for accidents which doors operating against high pressures necessarily present.

(2) The hazard of high voltage power cables in shafts and underground workings is done away with.

(6) Electrical equipment is not subjected to dust and moisture as is the case in underground installations.

(7) Fewer fan units are necessary, and for a given volume of air circulated, the surface installation is considerably cheaper.

Air leaving a fan under pressure will of course follow the path of least resistance, and in order to cause it to go where it is needed we can do either of two things, viz: decrease the resistance along the path we wish it to follow, or increase the resistance along the course which we do not want it to take. Since the size of the airway limits the decrease which we can make, we must necessarily resort to increasing the resistance where we do not need a flow of air. This resistance is effected by doors, brattices, or other similar stoppings, and by this means we can reduce or entirely check the movement of air.

The standard ventilation door used on main haulage drifts at the Copper Queen is 4½ by 7 feet in size. It is constructed of two plies of one-inch lumber the boards being laid vertically and horizontally. The door is swung on three heavy iron strap hinges which are bolted through the door to an iron strap on the opposite side. The door frames are built of 10 x 10 timber which is sealed to the rock by concrete. The doors are opened by a compressed air cylinder and plunger, and closed by a counterweight and pulley. Two or more levers and steel cables on either side of the door at varying distances operate the compressed air and open or close the door. ~~for the passage of~~. In most cases, these doors are installed in pairs to provide an air lock and thereby reduce the loss of air during the passage of trains or men. The two doors of the air lock are usually ~~enough apart~~ two or three hundred feet apart, although in some cases they are much closer together.

There are two factors which tend to nullify the effects ~~for~~ which doors are expected to produce. The first of these, and the most serious is the tendency of workmen, especially motor crews, to leave doors open after passing through. Frequently the entire benefit of a fan is wiped out by this carelessness or indifference. And in nearly every case, it is not the offender who suffers by his act; but rather the men in some remote stope who are dependent upon the fan for their supply of fresh air. It is probably because of this fact that the evil effects of leaving doors open are not felt by those who do it, that such practices continue. ~~They [illegible] not realize what they are inflicting upon their fellows, and if telling them fails to impress the fact upon their minds, then other measures should be resorted to. If the compressed air mechanism for operating a door fails to function, the fact should be reported to the foreman or boss immediately, and orders given to those responsible for maintenance to make the necessary adjustments or repairs.~~ These things are purely a matter of habit and discipline, and workmen can be made to adhere to proper practice just as closely as they follow safety rules or other regulations which are more rigidly enforced.

The other factor which impairs the value of a door is leakage. Few of us realize the volume of air which even a good door allows to leak through or around it. Experiments have shown that a two-ply door, stripped or battened with canvas, will leak a volume of from 2000 to 3000 c.f.m. when under a pressure of only one inch water gauge. Under higher pressures the volume of leakage increases, and there have been doors on some of our main haulage ways which were actually passing as much as 6000 or 7000 c.f.m. This quantity of air in a large motor tunnel is barely noticeable, and it is not strange that we fail to realize the importance of such a leak. However if we have two or three such doors on any one ventilating circuit it is easy to understand why we do not get the proper quantity of air at the working faces.

Canvas strips, sill blocks, water seals in the drainage ditches and minimum openings for pipe and trolley lines should be maintained on every one of the doors which control primary ventilation circuits. In other words, every door which requires compressed air for it's operation should be so arranged. As stated before, even under the most favorable conditions, door leakage is a persistent enemy of proper ventilation, and if not constantly watched can ruin an otherwise effective ventilation system. In order to better emphasize the ease with which a comparatively large leakage can escape notice, attention is directed to the following facts;

An air flow of 2000 c.f.m. in a 5'x7' drift means a velocity of only slightly over one foot per second, or a movement which we ~~[illegible]~~ ordinarily would not notice unless we were looking for it. A flow of 5000 c.f.m. in the same drift would mean a velocity of 2½ ft. per second) and would be regarded as only a moderate draft. In any of our large motor haulage drifts (say a 7x9) even a volume of 10,000 c.f.m. would not be particularly noticeable and it can thereby be understood why every effort should be made to keep all main line doors in proper condition even though it is impossible to feel any air movement through the drifts in which they are installed.

Auxiliary Ventilation

In remote portions of the mine where it is impossible or impractical to lead the main air currents from the primary fans, recourse must be had to other means for supplying a circulation of air. This is accomplished in some instances by small booster fans of various types, and in others by a compressed air venturi jet. The use of compressed air jets is not general in the United States, but is common in the deep mines of Africa. The United Verde is the only other large mine in Arizona which follows this practice to any extent.

Because of it's low cost of construction and installation, ~~the~~ air jet can be used to good advantage in places where the installation is only temporary; and if it's use is strictly confined to this purpose it has a definite place in the scheme of ventilation. If however it's use is continued for any length of time in one location, the continued cost of operation will defeat the purpose for which it is designed. Compressed air is an expensive product, and it's widespread use for ventilation is a practice which should and could be greatly reduced. Under certain conditions it is doubtless justifiable to open an air valve and "let 'er blow" for a short time. When however this practice is carried to extremes, the results are immediately seen in the cost sheets.

It is interesting to estimate in dollars + cents what it costs to use compressed air for blowing out working faces. When we crack a valve for this purpose, we are using from 60 to 150 c.f.m. of air. Say the average is 100 c.f.m. In one hours time we will therefore use 6000 cu.ft. of air for one such blower. Suppose there are twenty places in the mine (and that is a low estimate) where the air blows in this manner for seven hours a day. The air consumption for the twenty places would amount to 840,000 cu.ft. At a cost of 2½ cents per M cu.ft. this would mean \$21.00 per day or \$525.00 per month spent for compressed air ventilation. Over a period of a year this will amount to \$6300.00, a sum which would purchase a good mechanical blower for each of the twenty places.

For extremely long development headings where it is necessary to carry air over a distance of 1000 feet or more in ventilation pipes we have the Roots Positive Pressure Blower. In reality it is a positive quantity blower. Operating like a pump, it handles a given quantity of air at each revolution, regardless of the resistance which it has to overcome, a blower of this type is comparatively expensive, heavy to handle, and because of the limited quantity of air it delivers is not suitable for general mine ventilation where large volumes of air are essential. For the purpose mentioned above however, it gives excellent results which an ordinary multivane fan could not begin to duplicate.

The most generally used fans for auxiliary ventilation are either small multivanes ~~type~~ or paddle wheel fans. Several of each type are in use in various locations, and both types are remarkably efficient. These fans are commonly connected to canvas vent-tube or galvanized iron piping, and for short distances they are satisfactory. Here again however we must face our old enemy, air leakage. Improperly installed pipe or tubing will necessarily allow air to escape at each joint, and it needs only a few such joints to destroy the effectiveness of a small fan. Good practice would therefore demand that in any but a very temporary installation, all joints (metal pipe now being standard) should be well wrapped and cemented.

Pipes should be carried as close as possible to the working face in order to derive the maximum benefit from the fan. It is not at all uncommon to find a mucker stewing in his own juice, compressed air blowing, and the end of the ventilation pipe two hundred feet or more back of the face. So long as it is possible to exist most men seem to prefer to endure the discomfort of foul hot air rather than to spend the time necessary to hang a canvas blasting section. To educate miners into a different point of view is no small task; but it is one which in the long run should prove to be well worth the trouble. In other words, satisfactory ventilation cannot be attained by the efforts of any single individual or any small group of individuals. It can be accomplished only by the concerted and serious attention of everyone whose duties take him at any time into the underground mines.

ESTIMATED COST OF INSTALLING EXHAUST BLOWER IN SACRAMENTO TUNNEL

Cutting Blower Station	$ 400.00
Cleaning out approximately 600 feet of drift	200.00
Installing 4 doors and 3 tight brattices (Including Blower Bulkhead and Air Locks)	400.00
Repairs on 300 Level and Slabbing on 300 and Sacramento Tunnel Levels	100.00
6 Power Poles - Sacramento Collar to Portal of Tunnel	240.00
Wire and Cable	620.00
Cut-Outs	60.00
New 2200 Volt Starter	305.00
Oil Switch	75.00
Labor - Electricians	150.00
Labor - Mechanical	150.00
Contingencies	135.00
Total	$2,835.00

There will be no expense for blower or motor on above job, since it is proposed to utilize some which are already on hand.

Ma foot...

LEAD SMELTING PRACTICE AT COPPER QUEEN SMELTER

Paper Delivered by H.L. Humes, before a meeting of the Copper Queen Smelter Club, October 17th, 1929.

The Lead Smelter consists essentially of an automatic ore sampling and crushing plant, storage bins, one Dwight-Lloyd sintering machine, one nine hearth Queen type roaster, one blast furnace, two 50-ton lead drossing kettles, lead bullion casting equipment, furnace dust flues, and bag house for collection of lead fume.

The ore and concentrates are obtained principally from lease and custom shippers, and to some extent from the Company's own mines at Bisbee. Limerock, copper converter slag, scrap iron, pyrite, and a copper irony-limey ore from Bisbee are used as fluxes on the furnace charge to obtain a fluid slag. Coke is burned in the blast furnace to accomplish the smelting and reduction of the blast furnace charge.

All ore to be handled through the crushing and sampling plant is dumped directly from railroad cars into a 50-ton steel hopper, with a pan feeder discharge, which feeds the ore onto a Duplex conveyor belt. This belt delivers the ore to a #7½ Gates gyratory, set to crush to 2½ inches maximum. The crushed ore is then elevated to the top of the sample mill by a bucket elevator, and is discharged into a 60 inch Snyder cutter, which automatically cuts out 1/5 of the lot of ore being sampled. This is the main sample. This 1/5 cut passes over a shaking feeder to a #32 Telsmith crusher set to crush to ¾ inch maximum, then to a 42-inch Snyder cutter, where a 1/5 cut of the main sample is made. This 1/25 or 4% of the original lot of ore is then passed over a shaking feeder to a set of 24" by 12" rolls, set to crush to 1/8-inch, and then drops through a bank of riffle samplers, which can be adjusted to take cuts from $\frac{1}{2}$ to 1/32 of the 4% samplo. This final samplo is in duplicate; the first sample is sent to the bucking room, and the floor sample is held pending settlement with shipper. The rejects from the riffle sampler, drops to the belt leading to the sinter mixes.

The sample mill is so arranged that the ore may be handled in four ways:

1. The entire lot, coarse and fines, spread over coarse ore beds.
2. All fines screened out for the Dwight-Lloyd sinter mixes, and coarse, to ore beds.
3. Loaded in railroad cars for stock. The entire lot of ore, or either the coarse or fines separately can be loaded into cars.
4. Crushed to 3/8 inch for Dwight-Lloyd sinter mixes, or into railroad cars.

After crushing to $2\frac{1}{2}$ inches in the gyratory, oxidized lead ores pass over a 4' by 5' Hummer electrically vibrated screen, the oversize from which is conveyed to a shuttle conveyor above the coarse ore bins, where the ore is spread evenly over the full length of the bed. There are usually three coarse ore beds of about 500 tons capacity each. The undersize from the screen, which is minus $\frac{1}{2}$ inch, is discharged onto a conveyor belt running over the sinter mix bins, and is spread evenly over the full length of bed by a tripper. A small cross conveyor is provided to convey material from the tripper to railroad cars. There are two beds of 500 tons capacity each for Dwight-Lloyd sinter mixes.

If the ore is a sulphide, the oversize from the screening operation is crushed in a Symons vertical disc crusher set to $\frac{1}{2}$ inch, and both crushed product and screen undersize, discharge onto the conveyor leading to the sinter mixes.

Concentrates are hand sampled and shoveled into a hopper with a screw feeder discharge, and drops onto the conveyor leading to the sinter mixes, where it is bedded in mixes of correct analysis for sintering. Another small hopper is also provided to handle dust, bag house cinder and other reverts which go to the sinter mixes. The crushing plant is also used to crush Dwight-Lloyd sinter, return foul slag or settler barrings, matte and uncrushed flux. If flux is already crushed, it is hand sampled and dumped in a 50-ton hopper, which discharges by means of a pan feeder onto conveyor belts leading to the bins over the sinter machine and blast furnace. Coke is also handled through the same hopper.

The correct mixing of the various constituents in the coarse ore and sinter mixes must be carefully done to obtain good smelting results. Conveyors run in tunnels underneath these two sets of bins, and the mixes are carefully discharged through slots in the bottom of the bins, in a manner to prevent segregation. As an additional precaution, a Stedman disintegrator is provided on the sinter mix side. The disintegrator consists of two cages with manganese steel bars, revolving in opposite directions, which breaks up lumpy concentrates and makes a more uniform and thorough mix. The coarse ore mix, sinter feed mix, flux, coke, etc., all discharge onto an underground reclaiming conveyor, being of course, handled separately, which delivers to a bucket elevator taking the material to the top of the smelter building proper. By a system of conveyors, equipped with trippers, each material is delivered to its proper bin, that for the sinter feed mix being located on the east side of the building above the Dwight-Lloyd sintering machine, while the other constituents of the charge are delivered to one of the six bins above the last furnace. The capacity of these bins are 30-50 tons depending on the material.

The purpose of the sinter machine is to eliminate sulphur from the charge, and to sinter or agglomorate the fines to make a coarse, porous cake for blast furnace charge. It is necessary to eliminate some of the sulphur in the concentrates so that an excessive matte fall is not made at the blast furnace. Again, if concentrates and fine ore are fed direct to the furnace, there would be a large amount of dust made, and would also give trouble in furnace operation, by offering considerable resistance to the ascention of the air blast through the shaft full of charge.

The feed hopper for sinter mix is constructed with vertical partitions part way down the bin to lessen segregation. The feed discharges from the hopper onto a revolving distributing table, with adjustable plows to regulate feed, and fixed rabbles set to discharge the mix into a swinging spout. Water is added to the charge on the table.

This swinging spout distributes a 4 inch bed of charge on the pallets of the machine, in such a way as to have coarsest material on the bottom for grate dressing, and the fines on top.

The machine itself consists of a pair of endless track circuits, in which run pallets or truck-like elements, 24 inches long and 42 inches wide. These pallets are provided with four wheels, which engage with the tracks at all parts of the circuit. Two large sprocket wheels raise the pallets from the return track underneath to the upper track, where the pallets are pushed onward tangentially from the top of the sprocket wheels. The pallets have herringbone type grates on which the charge is laid down by the swinging spout. The pallets then pass under the muffle oil furnace where the top of the charge is ignited, and at the same time, the pallets, with ignited charge, come under the influence of the downward moving current of air in the suction box, induced by the suction draft of a 75 H.P. fan. This suction box is 22 feet long, by 42 inches wide. The speed of the machine is so regulated that the sintering and roasting action is carried progressively downward, until it reaches the grates, by the time the end of the suction box is reached. The pallets discharge the sinter cake into a hopper, and then descend by gravity on the lower track, to again be picked up by the sprocket wheels to begin a new cycle.

The fine ore in the charge offers considerable resistance to the suction. To give the charge more porosity, some sinter and suction box cleanings are returned to the next mix. The addition of water to the charge also gives more porosity, voids being formed as the downward heat evaporates the water.

The sulphur in the feed will vary from 11 to 13%, and the sulphur in the sinter produced will average about 3½%.

The sinter machine has a capacity of from 75-200 tons per 24 hours, with a speed of from 12-30 inches of pallet length per minute. The wide variation is caused by fluctuations of the physical character of the charge.

The sinter machine produces about 20,000 cubic feet of gas per minute at 200-350 degrees Fahrenheit. This gas is discharged by the fan into the main header flue, and mixes with blast furnace gas. It is necessary to mix these two gases, as the basic blast furnace gas neutralizes any acid in the Dwight-Lloyd gas, and prevents the rotting of bags in the bag house.

The sinter discharges by gravity through a hopper into an electrically operated skip hoist. The skip hoist is connected so that when six pallets have discharged into skip, it is raised automatically to top of blast furnace bins, and sinter is dumped into bin, and skip drops back to its original position. The skip can be raised, and sinter discharged into cars on track below machine. Charge which goes through grates is collected in hoppers below the machine, and is loaded into cars and returned to next sinter mix.

The blast furnace charge is made up from six feed bins of about 40 tons capacity each. These bins are discharged by means of pan feeders under close control of the feed floor operator, into an Atlas weigh car, each constituent of the 8000 pound charge being weighed to the nearest 10 pounds. The charge car is fifteen feet long, motor driven, with body of car suspended on knife edges, and connected to a dial scale weighing up to 10,000 pounds. The car runs over tracks above top of blast furnace, into which it is discharged, after opening the 30 inch doors on top of furnace shaft.

The charge consists of eight components:

1.	Dwight-Lloyd Sinter	-	43%
2.	Coarse oxide ore	-	31%
3.	Limerock		17%
4.	Settler barrings or foul slag	-	$2\frac{1}{2}$%
5.	Converter slag (only used occasionally)		
6.	Scrap iron	-	$2\frac{1}{2}$%
7.	Dross	-	4%
8.	Coke (as per cent of ore plus flux charged)	-	$14\frac{1}{2}$%

Note: The above represents an average, and is varied from day to day, depending upon the composition of the mix.

Scrap iron and dross are weighed on a platform scale, situated on feed floor, and charged direct into the furnace. The furnace has a capacity of 250-300 tons of charge per day.

The blast furnace consists of a shaft $25\frac{1}{2}$ feet high, from the top of the crucible to the feed floor. The bottom six feet of the shaft is composed of ten steel water jackets. The side jackets rise vertically for two feet from bottom, and then slope outward to form a bosh of ten inches from the vertical, measured at the top of the four feet of slope. Above the jackets fire brick is used to form the shaft of the furnace, and is sloped outward to a point 16 feet from the crucible, at which point, the shaft is 80 inches wide. The furnace is 45 inches wide and 16 feet long, measured at the tuyeres. The ends of the furnace are vertical for their entire height.

There are four side jackets four feet wide to each side of the furnace, with a total of 32 - 4 inch diameter tuyeres, the center line being 15 inches above bottom of jackets. Air blast is supplied to the tuyeres through pipes leading to bustle pipes, and air main, connected to a 200 cubic foot Connersville blower at the Power House.

The shaft rests on a crucible which consists of a concrete block with a depression in the top. This depression is lined with fire brick. The crucible measures 30 inches deep, 38 inches wide and 15 feet long. This crucible block is covered on the sides and bottom with a riveted steel shell to prevent the seepage of lead.

In operation, the ore column is carried at about 10 feet from feed floor, making about 15½ feet of charge in the furnace. The pressure of the blast for this ore column varies from 20 to 40 ounces, depending on the rate of driving (or speed) of the furnace, on the tightness (or fineness) of the charge, or on furnace irregularities such as high fire, or scaffolded charge, or unbalanced slag analysis.

The gas is drawn from the top of the furnace by an offtake at the side, just under the feed floor, and passes into the main header flue to mix with the Dwight-Lloyd gas. The temperature of the gases from the top of furnace is 150-200 degrees Fahrenheit.

The charge is dumped from the charge car onto a "V" type deflector, just below the feed floor, in the furnace, which throws the charge against the side walls. The coarse rebounds to the center of the furnace, and the fines drop along the sides of the shaft. The charge is, therefore, loose along the center line of furnace, and offers less resistance to the penetration of the air blast.

The operation of the furnace is controlled by careful observation of the condition of tuyeres, temperature of jacket cooling water, regularity of descent of charge in the shaft of furnace, and the appearance and composition of the matte and slag.

The metallurgy of the lead blast furnace is to accomplish the reduction of lead compounds into lead bullion, which collects the gold and silver. The earthy or gangue constituents of the charge are so proportioned in the mixes, that, by the addition of fluxes in the charge car, a fluid clean slag is obtained. A matte is formed consisting of artificial sulphides of copper, lead and iron. The matte fall is maintained above 6-7 per cent, since lower than this figure will give trouble inside the furnace, as the copper will dissolve in the lead, and choke the crucible.

The furnace column may be divided into three different zones. The upper zone is the preheating zone, in which water is driven from the charge. The middle zone is the reduction zone, in which carbon and carbon monoxide gas acts on lead compounds to reduce them to metallic lead. In the upper part of this zone, lead sulphide reacts with lead sulphate to give lead oxide and sulphur dioxide

$$PbS + 3PbSO_4 = 4PbO + 4SO_2$$

This lead oxide reacts with additional lead sulphide to form metallic lead and sulphur dioxide.

$$PbS + 2PbO = 3Pb + SO_2$$

Some lead oxide also reacts with carbon from the coke to form metallic lead and carbon dioxide $2PbO + C = 2Pb + CO$. At this stage, carbonates, and some sulphates begin to dissociate.

At a bright red heat in the lower part of the reduction zone, carbon monoxide reacts with lead oxide to form metallic lead, and carbon dioxide, $PbO + CO = CO_2$. Carbon monoxide also reacts with hematite, iron oxide, to form a lower oxide of iron and carbon dioxide, $Fe_2O_3 + CO = 2FeO + CO_2$. The oxide of iron, FeO, unites with carbon to form metallic iron and carbon monoxide. The metallic iron so formed combines with Galena (lead sulphide), forming metallic lead and iron sulphide, which is a constituent of the matte.

The carbon monoxide so formed is available for the reduction of additional metallic oxides.

As may be seen, from the above reactions, metallic lead trickles down through the charge over the entire furnace, and in so doing, collects the gold and silver present in the charge.

Iron oxide, FeO, formed in this zone, in the presence of silica, starts to form slag of low melting point, which when it trickles through the charge, combines with the more refractory slag forming compounds to make the final slag to be tapped.

Copper oxide unites with iron sulphide, liberating iron, and forming copper sulphide, another constituent of matte. Any remaining lead sulphide goes to form the other artificial sulphide of the matte.

The lower or smelting zone of the furnace is marked by very high temperature, and begins at about the top of the six foot jackets. Lead silicates are not decomposed by carbon or carbon monoxide, but at high temperatures, are fluxed by iron oxide, FeO, and lime, CaO, which combine with the silica to form slag and liberate lead oxide.

A bed of coke about one to two feet deep must be maintained at all time at the tuyere line. The incoming air meets this bed of incandescent coke and burns the carbon of the coke to carbon monoxide. Any carbon dioxide formed at this point, combines with additional carbon to form carbon monoxide.

As may be seen, iron plays an important part in the chemistry of the furnace. If the coke balance is low in the bottom of the furnace, there is insufficient iron oxides reduced to precipitate metallic lead. This results in high lead in mattes and slags, and is termed poor reduction.

Lead settles through the slag and matte, collecting gold and silver from the matte and slag, and accumulates in the crucible. Matte settles through the slag removing lead, copper and silver, for which it has a higher affinity, than has slag.

The lead, matte and slag accumulate in layers in the 15 inch space between tuyeres and taphole, and is tapped out at regular intervals. The slag and matte are tapped from the furnace through a three inch taphole in one end of the furnace at the bottom of the front end jacket, and 15 inches below the center line of tuyeres. Ordinarily, the lead is tapped from furnace through a lead well at one side of furnace, three feet from front end. This lead well is a 9" x 9" channel, extending from top of crucible shell outside of jackets, down through crucible wall into bottom of crucible. The crucible is kept filled with molten lead at all times. The lead rises in the well from the crucible, due to the weight of charge and the pressure of the blast exerted on the lead in crucible, and either overflows, or is tapped into half ton ladles on trucks. These trucks are trammed to the 50-ton lead drossing kettles and dumped.

At the present time, the lead well is not used, the lead passing out into the first settler with the matte and slag. This is due primarily to the amount of copper on the charge. The copper in the charge unites with sulphur to form a matte, but some of the copper dissolves in the lead. When the lead cools from a red heat, this copper-lead mixture separates out of solution, and forms what is called dross. When enough copper is present in the charge to form a matte assaying over 25 per cent copper, a large amount of dross is formed. This dross deposits on the sides of lead well, and in the crucible, eventually closing them up entirely. To obviate this evil, the lead well is closed off.

Separation of slag, matte, and lead is effected by the difference in specific gravity of the three products. The specific gravity of lead bullion is 11.0 - 11.3, matte 4.5 - 5.5, and slag 3.2 - 3.5. The lead, being the heaviest settles to the bottom of the settler and is tapped through a $1\frac{1}{2}$ inch hole at the bottom of the side of the settler. The matte settles out on top of the lead and is tapped through a $1\frac{1}{2}$ inch hole at a higher level on the settler side than the lead top hole. The slag overflows at the end of the settler. A secondary settler is provided to settle any matte and lead not caught by the first settler. The slag overflows from the secondary settler into slag pots of 12 fon capacity, from which it is poured on the dump. The average life of the 4' x 4' x 9' brick lined first settler is six weeks, and that of the secondary settler three days. When it becomes necessary to change settlers, they are drained, and the shell dumped. This shell, called settler barrings, is broken up and resmelted.

The matte and lead are tapped separately, from the first settler into saucer shaped pans, which hold about one ton. The lead is dumped, still molten, into the lead kettles. The matte is allowed to solidify, and is borken through a 7 inch grizzly into railroad cars to ship to the copper converter department, or to return to the lead plant for further concentration of its copper content.

The composition of the ideal slag for a lead furnace is silica, (SiO_2) 30%, iron and manganese oxides (FeO - MnO) 40%, and lime (CaO) 20%, the balance of 10% being zinc, alumina, magnesia, sulphur, lead, copper, etc. This smelter receives ores which are for the most part siliceous. Therefore, it would be necessary to add large quantities of flux, both iron and lime, to approximate this ideal slag. This is not feasible from an economic standpoint, therefore in practice, the silica and lime are raised and the iron is lowered.

Lead in the slag and matte is maintained at as low an amount as possible. One per cent and under of lead in the slag, and 11 per cent and under of lead in the matte is considered as good work. The per cent of coke on the charge is the main factor governing clean slag and matte, but other factors affect good reduction, such as poor furnace handling, unbalanced slag, too high or too low air blast, and too high or too low furnace charge column.

Typical metallurgical results are indicated for the month of July, 1929:

Element	Slag	Bullion	Matte
Gold Oz. per ton	0.002	0.78	0.02
Silver " " "	0.43	269.01	67.74
Copper %	0.26	0.12	30.5
Lead %	1.00	97.22	10.7
Silica %	32.00		
Alumina %	4.7		
Iron - Manganese oxides %	34.5		Iron % 28.3
Lime %	19.3		
Magnesia %	0.8		
Sulphur %	1.0		22.0
Zinc %	4.0		3.5
Antimony %		1.52	
Bismuth		0.16	

Certain elements contained in lead ores cause considerable trouble and expense in operating a lead furnace. Zinc is eliminated, finally, as zinc oxide in the slag, and to properly flux this zinc oxide, expensive iron flux must be added. Zinc oxide is very refractory, and increases the fluidity temperature of the slag. When present in large quantities as sulphide, zincy ores must be double or triple roasted, as zinc sulphide is most difficult to oxidize. Zinc makes the specific gravity of slags higher, and that of matte lower, resulting in poorer separation of slag and matte, with increases in metal losses. The formation of scaffolds or "hangs" in the furnace shaft is greatly accelerated when high zinc is on the charge.

The presence of any appreciable amount of copper is undesirable. It requires more coke per ton of charge to obtain clean slag, and also requires roasting, sintering, smelting and converting of the copper lead matte produced.

Alumina, when present over 5 per cent, tends to make the slag thick, and it also increases the formation temperature of the slag.

The presence of barite results in very fluid slags, and would be a desirable flux, were it not for its high specific gravity, which makes the slags heavy, and thus hinders the separation of slag and matte.

Arsenic and cadmium make a very combustible fume in the bag house. Antimony and to some extent arsenic, collect in the bullion, increasing the refining cost. Bismuth collects in the lead bullion practically altogether, and greatly increases the refining cost.

The drossing and lead casting equipment consists of two 50-ton cast iron kettles, set on cylindrical brick lined, oil fired furnaces, a small cast steel kettle to sweat the lead out of the dross, steel baskets to drain the lead out of the dross, siphon pipes, and moulds, etc.

The lead is heated to about 900 degrees Fahrenheit. The dross is removed from the surface of the liquid lead by a large paddle and dumped into the steel basket with perforated bottom. When full of dross, the basket is set in the small hot sweating kettle, where the entrained lead is drained. The dross is dumped, loaded into a buggy, and transferred to the feed floor. An analysis of dross is as follows: Gold 2.10%; Silver 163.34%; Copper 18.80%; and Lead 57.6%.

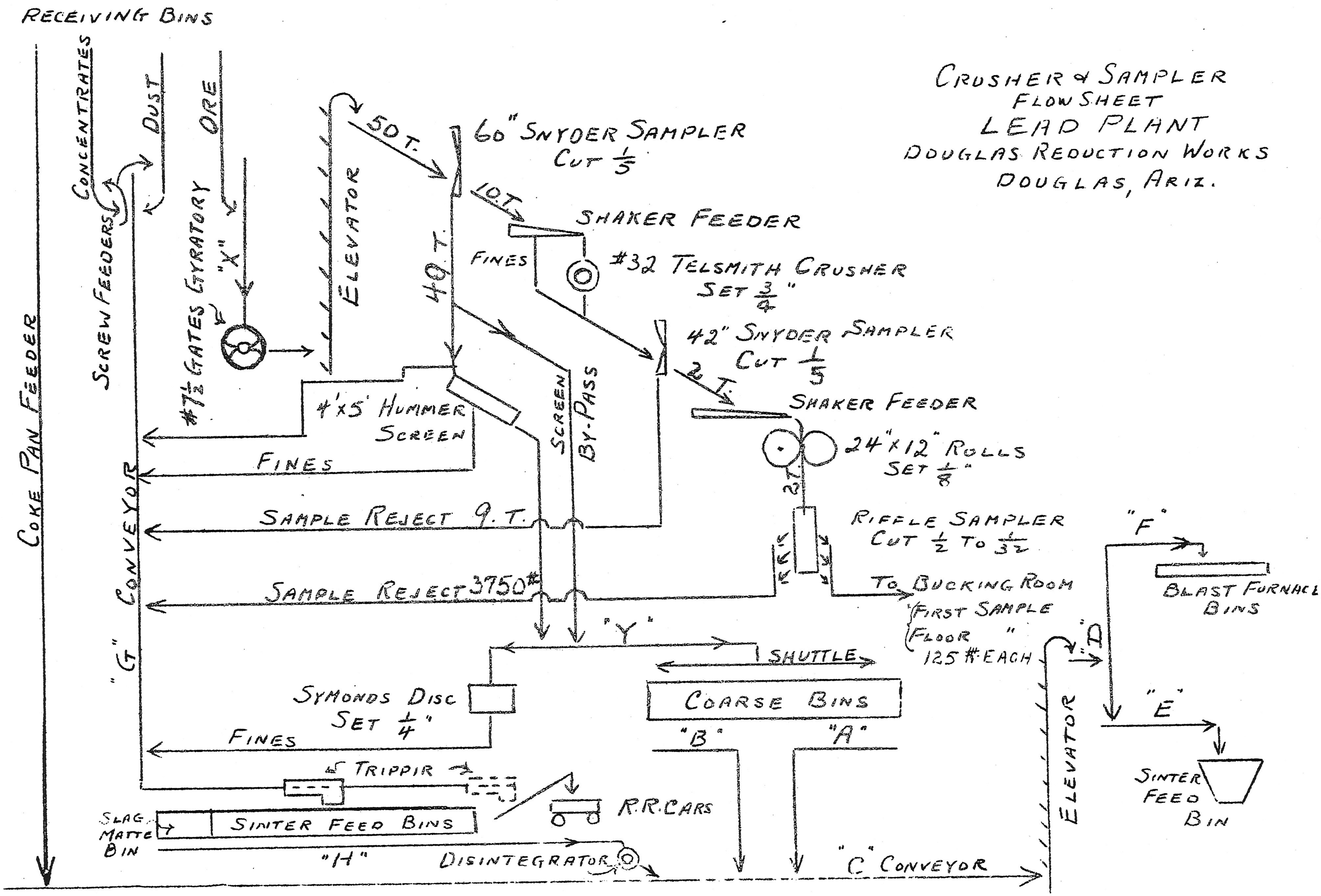
The lead is agitated with air to cool to about 650 degrees F. The air oxidizes copper, zinc, and some of the other impurities, which rise to the surface. Additional dross also rises, with the lowered temperature. This final dross is removed as before, and the lead is reheated to about 700 degrees F., preparatory to casting. Small conical samples are taken while the lead is stirred. The siphon pipe is then inserted in the lead, and the pipe filled. The valve is closed, and the valve end thrown over the side of the kettle. A distributing pipe conveys the lead from the siphon pipe to the moulds. The rate of flow is regulated by the siphon valve. Pigs of lead are cast weighing 95 pounds each, cooled with water, and loaded into railroad cars for shipment to the refinery.

The gases from the furnace and sintering machine carry a considerable amount of lead and silver in dust and fume. In order that dust may be settled from the gas, large settling flues are provided. Fume will not settle out of a current of moving gas, so that bag filtration of all gases is necessary before entering the stack. The bag house is built up of four sections with two cellars under each section. This permits the operator to work on one section while the others are operating. The entire bag house has 1008 sixteen inch diameter, 28 feet long, wool bags, hung vertically, and especially woven for this service. The bags are tied on the bottom, around thimbles 5 inches high in the steel floor between the bag section and the cellars. Two 30 H.P. fans are provided to maintain a pressure of 0.2 - 1.5 inches of water in the cellars of the bag house. The gases filter through the bags and pass to the stack. The fume collects on the bags. The bags are mechanically shaken when the pressure reaches 1.5 inches of water, to drop the fume into the bottom of the cellars.

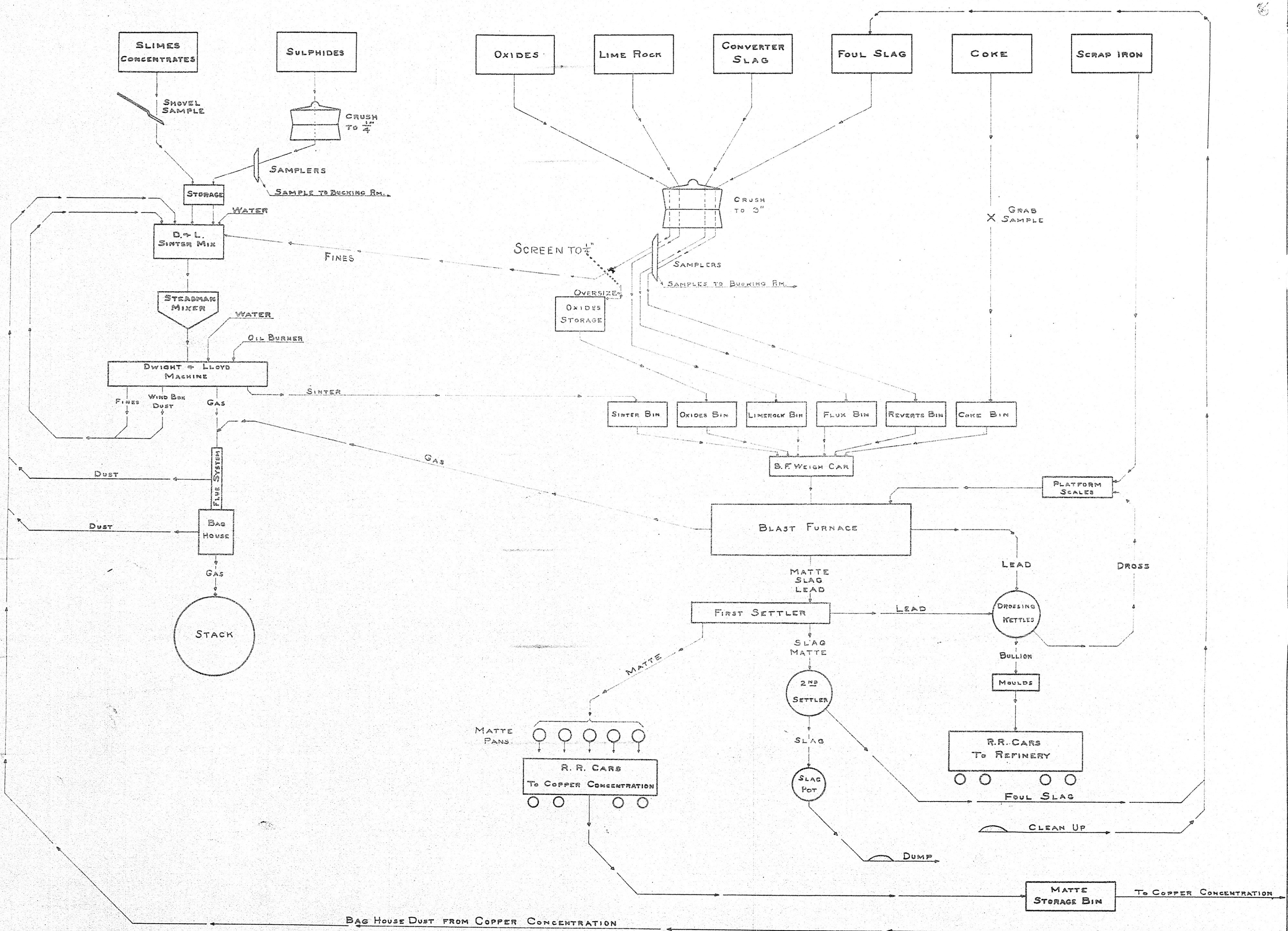
The fume has a very low specific gravity, weighing about 17 pounds per cubic foot. To handle this kind of material would be dangerous to the health of the operators. Fortunately, the fume is combustible, due to the carbon soot from the coke, and when burned there is formed a cinder of about $\frac{1}{4}$ the original volume, or 70 pounds per cubic foot. This cinder is porous and can be dampened before loading. A typical analysis of bag house cinder is: Gold 0.033 oz. per ton; Silver 28.98 oz. per ton; Copper 0.43%; Lead 41.4%; Sulphur 8.0%; Zinc 8.0%; Arsenic 1.00%; Antimony 1.30%. Cadmium over 15 per cent makes a fume which cannot be handled in a bag house, due to the danger of spontaneous combustion, and resultant loss of bags. Therefore, fume of this character is withdrawn from the lead plant, and sold to a buyer of cadmium bearing material.

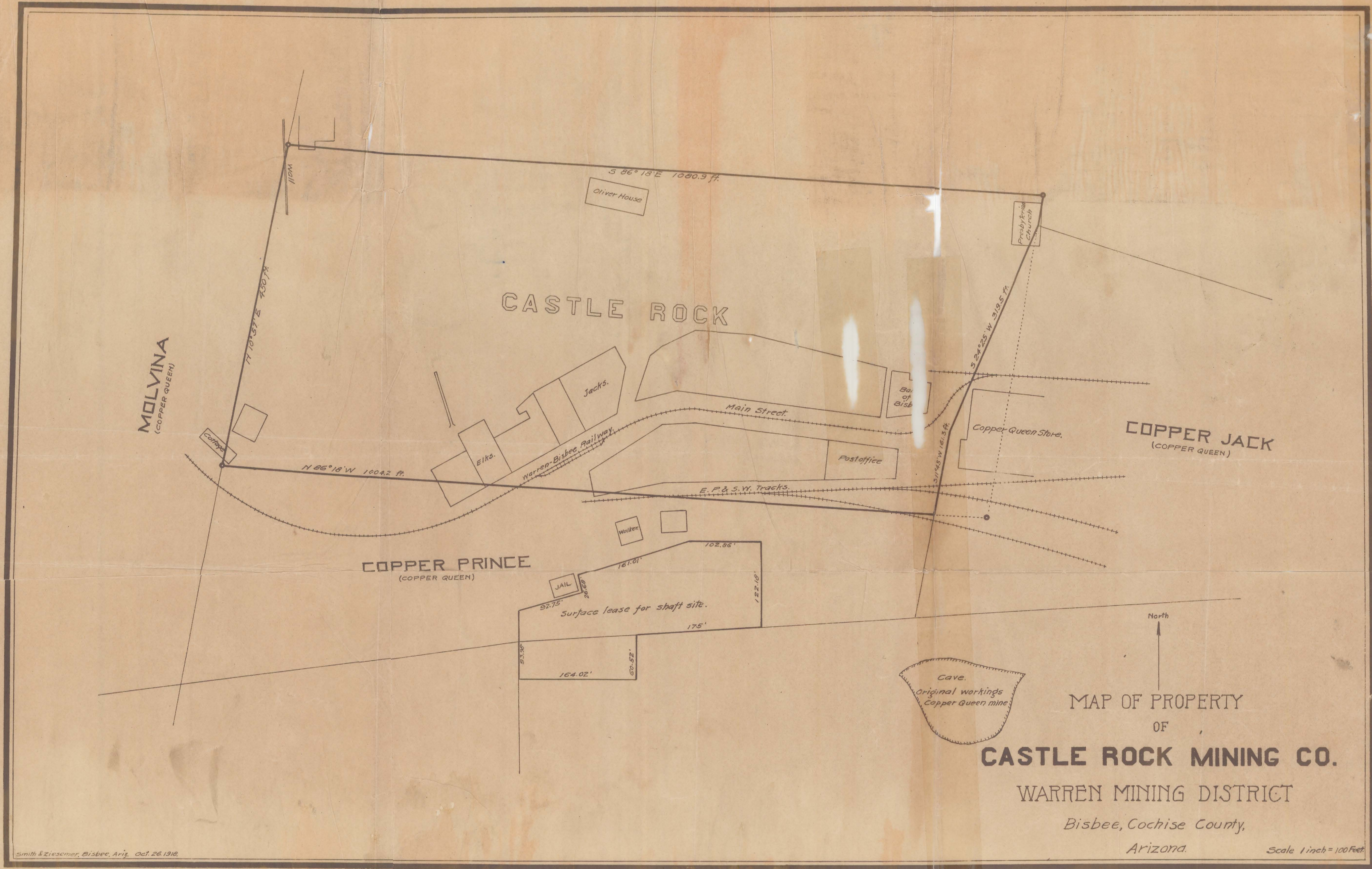
The safety and health of employees is a very important factor in the lead plant. All parts of the plant where fume is present are carefully hooded and connected with a suction system maintained by a 25 H.P. fan. Men must utilize special change rooms and lunch rooms. Monthly physical examinations are made, and the results carefully recorded, so that treatment can be given at the first sign of lead absorption, before any actual poisoning begins.
C

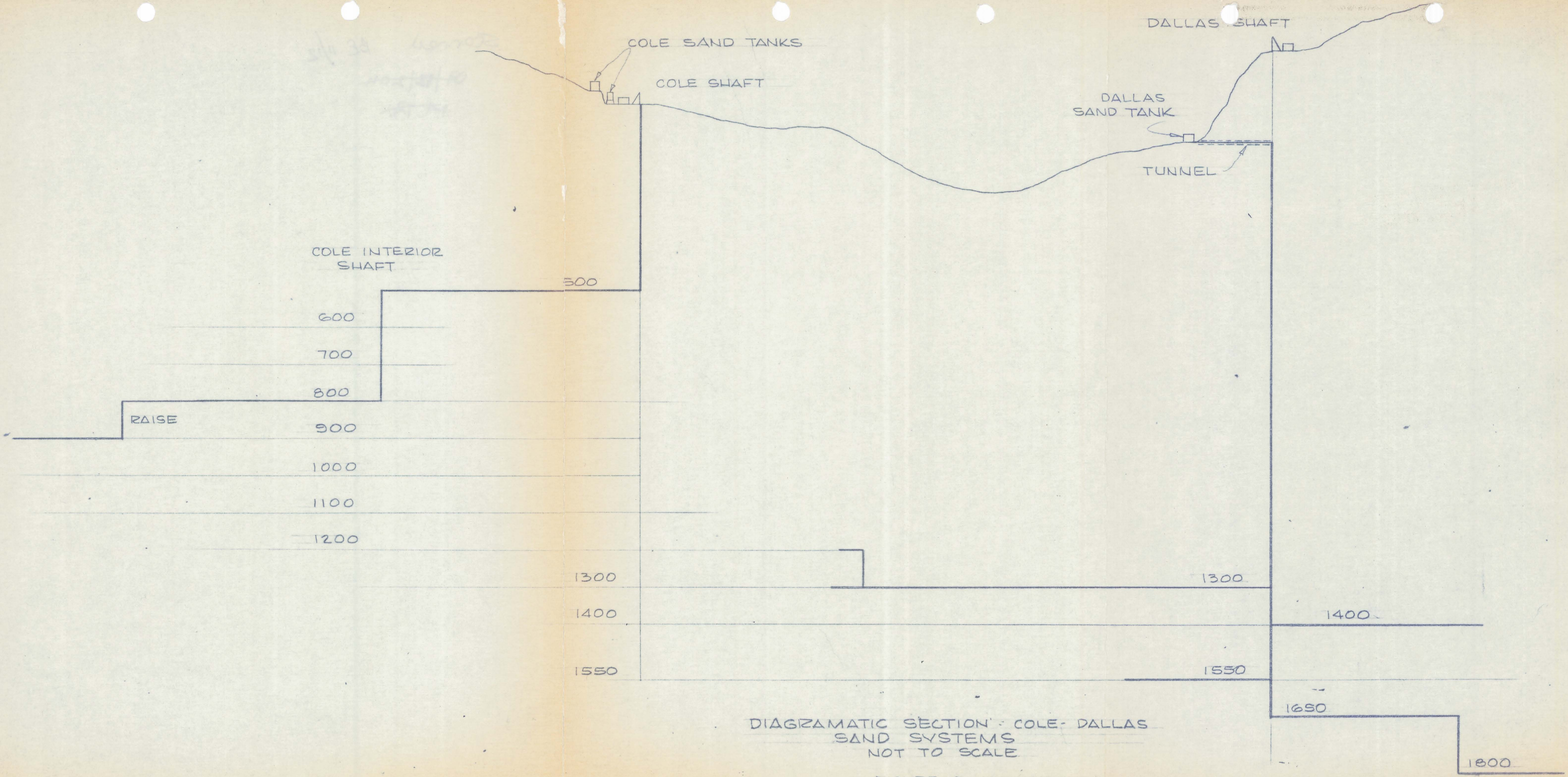
Crusher & Sampler
Flow Sheet
Lead Plant
Douglas Reduction Works
Douglas, Ariz.



FLOW SHEET LEAD SMELTING




MAP OF PROPERTY
OF
CASTLE ROCK MINING CO.
WARREN MINING DISTRICT
Bisbee, Cochise County,
Arizona.
Scale 1 inch = 100 Feet.
Smith & Ziesemer, Bisbee, Ariz. Oct. 26. 1918.
North
CASTLE ROCK
MOLVINA
(COPPER QUEEN)
COPPER PRINCE
(COPPER QUEEN)
COPPER JACK
(COPPER QUEEN)
S 86° 18' E 1080.9 ft.
N 10° 57' E 450 ft.
N 86° 18' W 1004.2 ft.
S 24° 25' W 319.5 ft.
S 11° 45' W 161.3 ft.
Wall
Oliver House
Presbyterian Church
Cottage
Jacks.
Elks.
Warren-Bisbee Railway.
Main Street.
Copper Queen Store.
Postoffice
E. P. & S. W. Tracks.
Walker.
JAIL
102.86'
161.01'
26.68'
122.18'
92.75'
Surface lease for shaft site.
175'
93.98'
60.52'
164.02'
Cave.
Original workings
Copper Queen mine



DIAGRAMATIC SECTION · COLE- DALLAS
SAND SYSTEMS
NOT TO SCALE

FIGURE 2