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OPEN-PIT MINING AND MILLING METHODS AND COSTS AT THE YELLOW ASTER MINE, RANDSBURG, CALIF.

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By A. W. Frolli[2]

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2/ One of the consulting engineers, Bureau of Mines, and manager, Anglo American Mining Corporation, Ltd.

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INTRODUCTION

This paper, describing the open-cut mining, milling, and cyaniding methods and costs at the Yellow Aster gold mine, is one of a series being prepared by the Bureau of Mines on similar practices in the United States. It is presented primarily to show methods and costs of mining and treating low-grade gold ore on a large scale.

The Yellow Aster mine, mill and cyanide plant are situated at Randsburg, Kern County, Calif., 42 miles northeast of Mojave and 11 miles from Searles, the nearest railroad station. Most of the supplies are hauled 147 miles by truck from Los Angeles.

The property lies at the northeast end of the Rand Mountains at an altitude of about 4,000 feet. A semiarid climate permits operation without difficulty all year.

ACKNOWLEDGMENTS

Acknowledgment is made to the executives of the Anglo American Mining Corporation, Ltd., for permission to publish this paper, and to members of the company staff for their assistance in its preparation.

HISTORY

The properties of the Yellow Aster mine are owned by the Yellow Aster Mining & Milling Co., of Los Angeles. Early in 1933 the Anglo American Mining Corporation, Ltd., of San Francisco, obtained a lease and option to purchase these properties.

Gold was first discovered on ground near the present glory hole in 1895 by three prospectors - Singleton, Burcham, and Mooers. The mine was named "Yellow Aster" after a novel that one of the prospectors was reading at the time. Rich float was abundant, and dry washing was conducted on Olympus Hill with profit soon after the discovery. The first development was on Olympus Hill and in the old Trilby workings.

Ore was first treated in small mills at Garlock 8 miles distant, where water was available. Later it was shipped to Barstow and treated in a custom stamp mill there. In the latter part of 1898 a 30-stamp mill was erected near the mine, and in 1901 a 100-stamp mill was built on the opposite side of the gulch.

From the time of discovery to 1905 most of the ore was mined from the vein system. Ore was mined principally from open stopes with pillar or stull support and from square-set stopes. In 1905 glory-hole mining was begun, and most of the tonnage was produced by this method until the latter part of 1918, when the mine was closed. About 1916 the 30-stamp mill was abandoned. Early in 1918 a new crushing and screening plant was built, and the fines were sent to the 100-stamp mill, but after about 4 months the mine was shut down. A few years later the crushing plant was destroyed by fire.

In 1921 the mine was reopened under the leasing system, and the ore treated in the company mill. At this time the 50 stamps on the east side of the mill were abandoned, leaving only those on the west side in operation. In 1926 the company began mining and milling in conjunction with the lessees and continued until October 1933, when the property was taken over by the Anglo American Mining Corporation, Ltd.

During this period inside battery-and-plate amalgamation was used for recovering the gold. Most of the ore passed through 40-mesh battery screens, and the amalgamation-plate tailings were deposited on the hillside below the mill.

As soon as the present operators took over the property the mill was closed for remodeling. A new crusher, vibrating screen, conveyors, and other equipment were installed and the 50 idle stamps repaired. Flotation equipment was added to treat the amalgamation tailings, and the plant resumed operation in January 1934.

The flotation of the amalgamation-plate tailings was not very successful, and after about 14 months the flotation plant was closed, amalgamation only being used as before.

In the fall of 1934 the Anglo American Mining Corporation, Ltd., acquired control of the old stamp-mill tailings and the following year built a sand and slime cyanide plant having a combined capacity of 1,100 tons per day. Current stamp-mill tailings were sent to this plant for further treatment.

In September 1936 it was decided to rebuild the crushing and screening plant so that two-stage crushing and screening could be done and a smaller undersize product sent to the stamp mill. This plant was operated in conjunction with the stamp mill from November 1, 1936, to March 1, 1937. Results from the mill runs and tests indicated that a larger tonnage would have to be handled to make the plant profitable. Mining and milling were suspended in March 1937 and plans made for larger-scale operations.

Since January 1934 production has originated from selective mining in veins, from glory holes, and from old square-set stope fills, pillars, and walls. Before crushing, most of the ore was passed over grizzly bars spaced 3 to 4 inches apart, and the oversize was discarded as waste.

PRODUCTION

From 1895 to May 1, 1939, the records show that slightly more than 3,400,000 tons of ore were milled, which yielded about 500,000 ounces of gold valued at $12,000,000. Virtually all of the gold was recovered by amalgamation, with only a minor amount by flotation.

Impounded and current mill tailings treated by cyanidation to date amount to 1,700,000 tons with 41,000 ounces of gold recovered.

GEOLOGY AND ORE DEPOSITS

The mine lies in a granite-schist complex near the summit of the Rand Mountains. The ore bodies occur in granite that has intruded the schists. The granite includes small to large islands of schist that have been broken off and dragged in from the parent rock. Some of the flat-dipping faults that characterize the area immediately surrounding the mine may have had considerable movement. Three of these faults mainly limit the ore deposits;

one called the Jupiter forms the hanging wall; another, the Hercules, forms the northeast side; and a third, the Security, forms the foot wall. The Jupiter fault dips 35° to 45° to the north, and the other two dip at about the same angles to the northeast. The ore-bearing area within the triangular-shaped fault block roughly gives an over-all length of 1,200 feet and a maximum width of 800 feet. Intruding these rocks is a series of faulted rhyolite dikes that are closely associated with deposition of the gold. Flat-dipping schists on the east, north, and northwest overlie both the granite and the ore bodies.

Although the gold occurs mainly in the granites, minor deposits have been found in the schists. Three main types of ore bodies are found: (1) Deposits along fault zones, (2) stockworks in granites, and (3) fissure veins containing some quartz. Virtually all of the ore bodies have a northwesterly strike and a vertical or northeasterly dip. They occur at intervals of 20 to 100 feet or more. Most of them are only a few feet wide, but some are over 50 feet wide. The bulk of the ore mined has come from the oxidized zone.

The gold which is generally free is accompanied by a little pyrite and arsenopyrite and some scheelite. Quartz and calcite are scarce. Some of the gold is coarse. High-grade ore is found in pockets and some of the smaller stringers. The gold is distributed erratically, making accurate sampling difficult. Silver occurs in the ore in the ratio of 1 ounce of silver to 3 1/2 ounces of gold.

Nearly all of the ore-bearing granites are oxidized and somewhat soft; the gold generally lies in the small fractures and joint planes of the rock. When blasted or crushed, the rock breaks along the planes of weakness and releases the gold with the fines, while the harder, unaltered granite breaks into larger pieces carrying minor quantities of gold. These rock characteristics make it possible to raise the gold content in the undersize by stage crushing and screening and discarding the oversize. This process is economical only with low-grade ore.

POWER

Electric power is purchased from the Nevada-California Power Corporation. This power, which is 3-phase and 60-cycle, is transmitted to the Randsburg substation at 90,000 volts from hydroelectric plants near Bishop and Mono Lake. From the Randsburg substation it is transmitted two-thirds mile at 33,000 volts to the 1,500-kv.-a. Yellow Aster substation, where it is stepped down and delivered to the cyanide plant and mill at 2,400 volts. At the cyanide plant and mill it is again stepped down to 480 volts by a 450-kv.-a. and a 300-kv.-a. transformer, respectively. At the mill and mine all motors of 150 horsepower or larger use 2,400-volt current. Transformers for reducing voltage from 480 to 120 for lights and other accessories are situated at convenient places.

Power for the Goler water system comes from a 33,000-volt line and is stepped down at each pump station to 440 volts.

For the year ended May 1, 1939 the total actual power for all operations averaged 1,443 horsepower for a connected load of about 2,000 horsepower. Energy consumption averaged 548,855 kilowatt-hours per month, giving a load factor of 70 percent. The demand and energy cost for this period averaged 9.1 mills per kilowatt-hour.

WATER SUPPLY

Water for mining and milling is pumped from wells near Goler, 7 1/2 miles distant, and just southeast of the Garlock fault. The main well called "New Well" is drilled and cased to a depth of about 500 feet. A 29-bowl, 10-inch L. C. Pomona deep-well turbine, direct-connected to a 50-horsepower 1,160-r. p. m. motor and having a capacity of 375 gallons per minute, delivers water to a 21,000-gallon tank. From this tank it is pumped in four relays through 6.9 miles of 6-inch casing to a 600,000-gallon reservoir at Randsburg by high-speed Cameron centrifugal pumps direct-connected to motors.

Data from tests made on these relay pumps, with discharge valves wide open, follow:

Data on relay pumps, July 23, 1937

Location	Type	Stages	Speed, r. p. m.	Capacity, g. p. m.	Pressure, pounds	Input, hp.	Motor, hp.
New Well......	2 GT	2	3,540	335	284	86.0	75
No. 1 relay...	2RVH-20	1	3,450	313	68	17.8	20
	2MRV-50	2	3,450		193	54.7	50
No. 2 relay...	2 GT	2	3,450	329	264	76.0	75
No. 3 relay...	2MRV-50	2	3,450	327	184	54.4	50

The pumps are throttled down to a capacity of 300 gallons per minute, and each relay pumps from a steel tank. A float switch for automatic start-and-stop control is used on all relay pumps and the turbine.

Water from the Randsburg reservoir is pumped through one-half mile of 8- and 6-inch casing to another reservoir at the mine level holding 86,000 gallons. For this lift a two-stage, high-speed, Cameron centrifugal pump direct-connected to a 75-horsepower motor is used. Water for the various operations is distributed from the mine reservoir.

The distance pumped totals 7.4 miles with a static head of 1,470 feet and a friction head of 1,240 feet (a total head of 2,710 feet). For the year ended April 30, 1939, an average of 9,265,000 gallons per month was consumed by all Yellow Aster operations. All pumping plants are under the care of one pumpman.

Water for domestic use is purchased from the Randsburg Water Co., which pumps from wells 5 1/2 miles northeast of Randsburg.

PROSPECTING AND DEVELOPMENT

The mine has been developed to a depth of 800 feet below the highest point of the outcrop (altitude 4,425 feet) by about 15 miles of adits, shafts, drifts, crosscuts, raises, and winzes. No standard level interval has been used; the greatest is 135 feet. The Rand level, on the floor of the glory hole at an altitude of 3,965 feet, is the main haulage level. The workings below the haulage level are reached through two incline shafts called the Rand and Hercules and through the Rand Vertical shaft.

The rim of the big glory hole is about one-fourth mile long by one-eighth mile wide, and the maximum height above the floor is 400 feet. Many of the old underground workings are exposed in the face, and some have long since been removed by the glory hole.

Diamond drilling has been done to prospect for new ore bodies below the glory-hole floor. About 50 holes were drilled in 1907 and 1908 and 16 holes in 1934 and 1935. This work did not reveal any important ore bodies.

Development in the lowest workings indicates that the present ore shoots have been bottomed and that there is a slight increase in sulfides. Further development down the northerly dip of the granite intrusive may expose other ore shoots.

SAMPLING AND ORE ESTIMATION

Various methods of sampling and estimating ore have been used in the past. However, the spotty character of the ore makes accurate sampling difficult. In development and stoping cut and grab samples were panned or assayed, or both, one serving as a check on the other. Panning was a quick, practical means of determining roughly the grade of the ore. Samples for panning were pounded in mortars with old air-operated piston drills strapped vertically to posts. Grab samples generally were taken of the coarse and fine material in the stopes, glory hole, and cars. Several thousand cut samples were taken in the development workings and stopes. Sampling by diamond drills did not prove satisfactory. Sorting by sight can be relied upon only for removing barren schist and hard-gray granite.

The difficulty of obtaining accurate samples precludes accurate estimation of ore tonnages. A face of good ore may become noncommercial in a round or two and vice versa. Generally the ore must be followed as closely as possible with panned and assayed samples as guides. For selective mining considerable development was required to maintain the ore reserves.

The most reliable method of determining the value of large sections of very low grade ore was a mill run followed by a clean-up. At various times since the fall of 1933 the present company has tested sections of the mine above and below the floor of the glory hole by this method. Comparatively large tonnages were mined or drawn from mill holes, old stope walls, pillars, and fills and sent to the crushing plant. The rock was crushed, and the screened oversize and undersize were weighed and thoroughly sampled. The ore in place, or value before crushing and screening, was calculated from the weights and assays of the two screened products. Any oversize rock discarded at the mine grizzlies was considered in the calculation. The oversize was discarded as waste, and the final screen undersize was sent to the stamp mill where a check was obtained on the undersize-product sampling by cleaning up the amalgam and assaying the amalgamation tailings for a certain number of tons milled. The results checked closely, although the mill runs showed a slightly higher value than assays of the undersize.

After many thousands of tons of rock were tested by mill runs, the value and tonnage of certain sections of the mine above and below the floor of the glory hole were estimated. The estimates indicated several million tons of ore averaging about 0.020 ounce of gold per ton in place, the screened fines from which would assay up to 0.061 ounce of gold per ton.

The possibility of stripping the barren overburden to mine the underlying ore has been considered, and surveys and calculations have been made.

Old mine dumps west of the mill contain about 1,300,000 tons of material. Thorough sampling of the surface shows an average grade of 0.024 ounce of gold per ton. A higher-grade product also can be screened from these dumps.

SELECTION OF MINING AND MILLING METHODS

Selective mining had reached a point where no profit could be made on the remaining ore, and unless new ore bodies were found this method of mining had to be discontinued. Mill holing as practiced in the past or any system of underground mining would be too costly even on a large-tonnage basis. Tests of stage crushing and screening indicated that a profit could be made if a large tonnage were handled and mining costs were very cheap.

It was decided that open-pit mining with power shovels and truck haulage would be the most flexible and cheapest method. Plans were made to begin mining on the present floor of the glory hole and to mine first all ore above the floor that was not covered by barren overburden. Large islands of schist would be mined separately when possible. This method would necessitate a bank or single bench with a maximum height of about 400 feet and the use of toe holes and coyote blasting.

Before mining could be done below the floor of the glory hole, waste overlying other ore would have to be stripped. The large mine dumps could be mined by a shovel at any convenient time.

Some screening of low-grade ore has been practiced at the Yellow Aster mine for a long time. When the early operators were mining in the glory hole, quarry forks, with tines about 1 inch apart, were used to separate the fines from the coarse material; the latter was discarded as waste. On the transfer

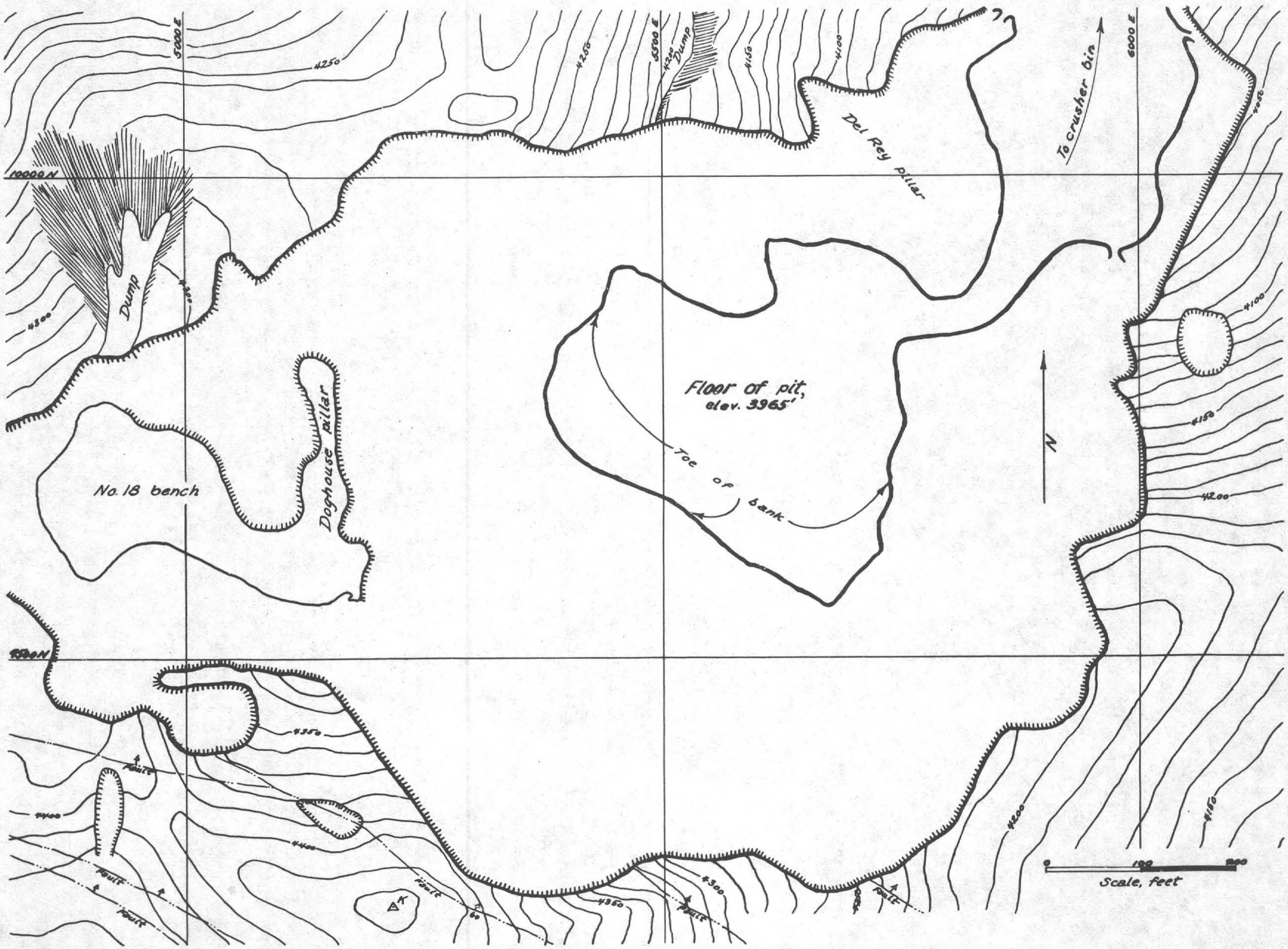

Figure 1.- Plan of Yellow Aster pit.

levels just below the mill holes, heavy grizzly bars were set 6 inches apart and coarse hard granite and schist sorted out. At times, ore hoisted from underground workings was passed over grizzly bars placed over the shaft bin. These bars were set 3 to 4 inches apart, and the oversize was discarded. Laboratory screening tests, as well as tests in the large crushing and screening plant built in 1918, showed that stage crushing and screening of low-grade ore were feasible.

Part of the time from January 1934 to September 1936 single-stage crushing and screening were practiced on low-grade rock, and the oversize from a screen with 1 1/2-inch square openings was discarded as waste; the undersize was delivered to the stamp mill. Approximately 128,000 tons of rock were crushed and screened during these test runs; the results showed that two-stage crushing and screening and a smaller undersize product were necessary.

A two-stage crushing and screening plant was built, and from November 1936 to February 1937, inclusive, 48,000 tons of rock were tested, the smaller undersize material being delivered to the stamp mill as before. These tests indicated that the secondary or final screen should have either 1/4- or 3/8-inch square openings to bring the undersize up to the desired grade.

Requirements at the cyanide plant called for a minimum of slimes, and to accomplish this it would be necessary to pass the fine ore through the classifier first and overflow as much material as possible before it reached the ball mill.

At times coarse gold is found that requires long contact with cyanide solution for proper extraction. This situation could be handled by removing the coarse gold by jigs and tables before delivering the pulp to the cyanide plant and by amalgamating the concentrates.

MINING

General

Mining is done on contract by the Macco Construction Co., of Clearwater, Calif. The contractor mines and delivers ore to the crusher bin and waste to the mine dump with his own equipment.

Open-cut mining was begun in January 1938, but before any work could be done in the glory hole itself it was necessary to cut a roadway through the Del Rey pillar to bring equipment into the pit. About 25 percent of the rock excavated from this cut was hard gray granite, which was hauled to the mine dump as waste. Figure 1 shows a surface plan of the pit.

After the floor of the pit and old slides were cleaned, toe-hole blasting was begun on the exposed banks. The soft character of the rock, the slips paralleling the face, and the weight of the high banks cause slow-moving slides to settle into the pit. This condition is continuous, and sometimes a slide can be started by digging into the toe with a shovel. When necessary, coyote blasts are used to break large tonnages. Large boulders are block-holed.

Conditions are favorable for low drilling and blasting costs and for hauling, as the floor of the pit is on a level with the top of the crushing plant.

Drilling and blasting

Compressed air for drilling is furnished by two portable Gardner-Denver model W. B. G. compressors driven by 2,007 Caterpillar-Diesel, 75-horsepower engines using 27°-plus Diesel oil. Each compressor has a capacity of 360 cubic feet of free air per minute and maintains a pressure of 90 to 100 pounds. The auxiliary gasoline engine for starting is run 2 to 10 minutes. These compressors are mounted on trailers equipped with pneumatic tires and air brakes. When only toe holes and boulders are being drilled one compressor is used, but when coyote drifts and crosscuts are also being driven two compressors are required.

Toe holes are drilled with a model 17D Gardner-Denver drifter mounted on a type WDA-10 Cleveland wagon drill. The wagon drill has an air feed with a 10-foot travel. Drill steel (1 1/4-inch hollow-round) ranges in length from 6 feet for starters to 26 feet for the longest finishers. Timken detachable cross bits are used with 3 1/2-inch-gage starters and 1 1/2-inch-gage finishers. Coyote drifts and crosscuts are driven and boulders plugged with unmounted Gardner-Denver S55 jackhammers. One-inch, hollow, hexagonal drill steel is used with Timken detachable Carr bits. All drilling is done dry.

Bits are sharpened with an Ingersoll- Rand J. A. 4 grinder, using a flat and a V-faced emery wheel. A bit can be sharpened an average of three times.

Toe holes up to 30 feet deep are drilled in the part of the face that offers the best opportunity for drilling and breaking. Usually drilling is confined to ridges and noses of hard material. Breaking the key sections brings down large tonnages from above and on either side. Overhanging points in the face that appear dangerous are broken by placing explosives in cracks or at other advantageous points.

Usually toe holes are sprung and blown out three or four times to form a chamber. Springing is done with 1 1/8- by 8-inch, 60-percent-strength Gelamite No. 1 and 1 1/4- by 12-inch, 45-percent-strength Gelamite No. 2, using No. 6 instantaneous electric detonators fired with blasting machines.

After the holes are chambered the pocket is loaded with Herculite 20-percent-strength bag powder by a powder loader. The loader comprises a copper funnel attached to 3/4-inch standard brass pipe in 6- and 12-foot lengths. (See fig. 2.) A jet of compressed air forces the powder through the pipe, the end of which rests a short distance from the bottom of the hole; as the pocket fills the pipe is gradually withdrawn. The load is set off by a primer made of a stick of Gelamite No. 2 and a No. 6 instantaneous electric detonator. Occasional holes, inclined upward, are loaded with Gelamite No. 2.

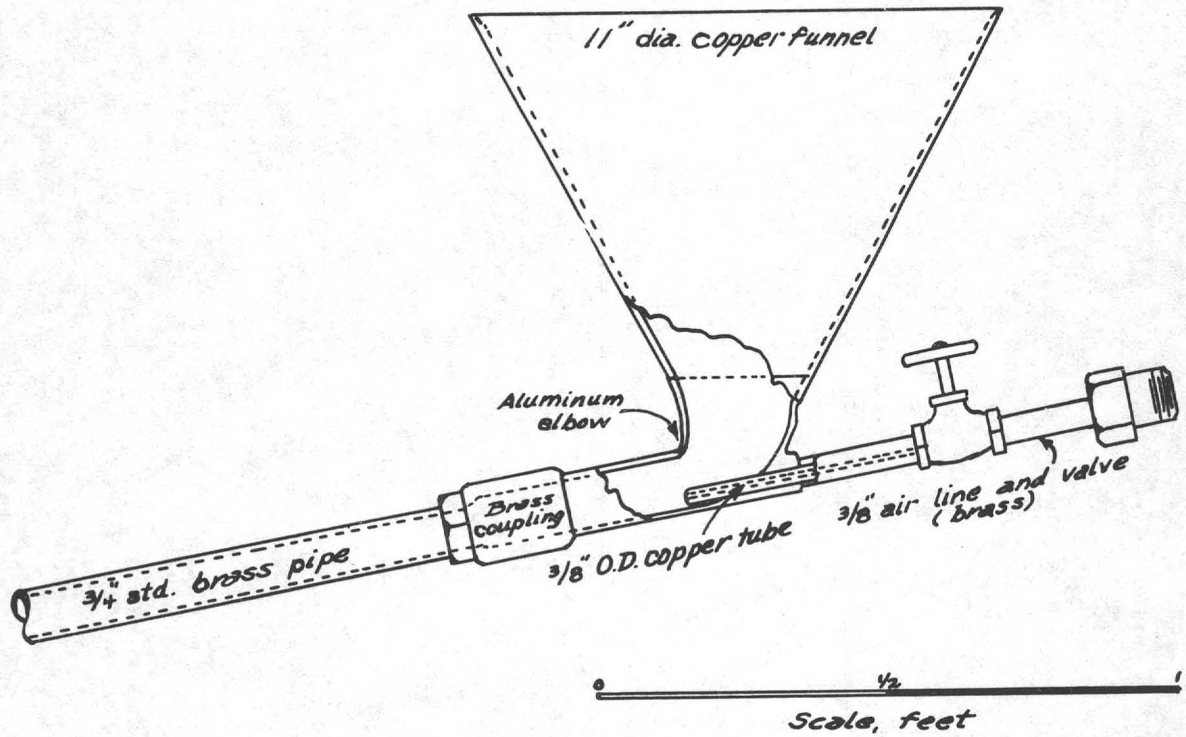

Figure 2.- Powder loader.

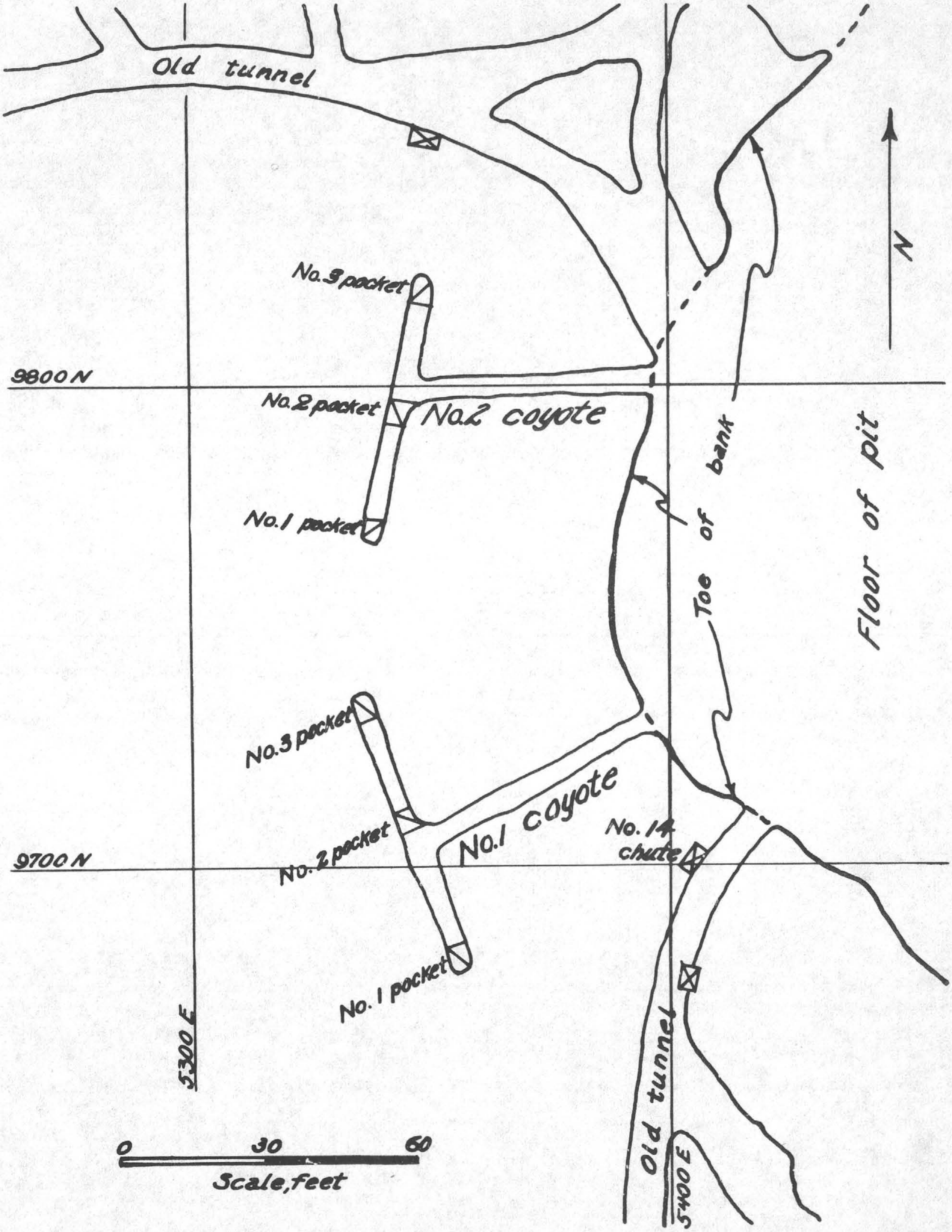

Figure 3.- Plan of coyote blasts Nos. 1 and 2.

Coyote blasting is done in sections of the pit where the bank is too high for toe holes and no sliding occurs. The work comprises driving adits and crosscuts 4 feet high by 3 1/2 feet wide with floor pockets where desired. Figure 3 shows a plan of coyote blasts 1 and 2 which were shot together. Each coyote had three floor pockets 4 feet wide, 5 feet long, and 4 feet deep. The pockets were large enough so that the entire charge could be loaded below the floor of the crosscuts. The adits averaged about 54 feet long, and the crosscuts were about 25 feet in one direction and 32 feet in the other from the center of the tunnel.

Drilling in comparatively soft oxidized granite was done with unmounted jackhammers. A crew of two men on each 8-hour shift drilled, blasted, and mucked out a full round or more per shift. Generally seven holes were drilled per round. These usually broke clear to a depth of 4 to 5 feet. Blasting was done at any time during the shift with Gelamite No. 2, using fuse and No. 6 blasting caps. The spoil was hauled and dumped near the portal of the adits by wheelbarrows.

When the adits, crosscuts, and pockets were completed preparations were made for loading. No. 12 rubber-covered wire fastened to pegs and nails in the roof of the adits and crosscuts extended down into the pockets. A primer was placed in the bottom of each pocket. The primer comprised three sticks of Gelamite No. 2, each of which contained one No. 6 instantaneous electric cap wired in parallel. The sticks were made into a bundle and placed in a box of Gelamite No. 2. The leg wires of the electric caps were cut to 30 inches and attached to the No. 12 wires. One end of the box was notched to allow the wires to protrude and the cover nailed on tightly. Champion 5-percent bag powder and Herco black blasting powder were loaded loose around and on top of the primer box and the workings backfilled with spoil to within 10 feet of the tunnel portal. Both coyotes were wired in parallel and shot simultaneously with 440-volt electric power from a switch at the machine shop.

The two coyote blasts contained 300 pounds of Gelamite No. 2, 10,000 pounds of Champion bag powder, and 27,750 pounds of Herco black powder, or a total of 38,050 pounds of explosives. Approximately 120,000 tons of rock were broken, or 3.2 tons per pound of explosive. The shot was loaded heavily because of its proximity to old adits and stopes. The material was well thrown out and broken. Table 1 gives data on the coyote blasts.

Boulders in the pit vary in size and occasionally weigh as much as 150 tons. A large part of the drilling time is spent in blockholing boulders that are 30 inches or more in dimension. Sometimes as many as three jackhammers are used on this work. The depth of the holes drilled varies with the size of the boulder. Shooting is done with Gelamite No. 2, using fuse and No. 6 blasting caps.

TABLE 1. - Coyote blasting data

Pocket No.	Interval between pockets, feet	Vertical height above pocket, feet	Distance pocket to face, feet	Powder load, pounds			
				Gelamite No. 2	Champion bag	Black blasting	Total explosives
Coyote 1:							
1........		125	62	50	1,000	7,000	8,050
2........	31	115	55	50	1,000	5,000	6,050
3........	25	118	55	50	1,000	7,000	8,050
Coyote 2:							
1........		85	50	50	5,000	--	5,050
2........	25	85	50	50	1,000	3,750	4,800
3........	26	100	52	50	1,000	5,000	6,050
Total...........................				300	10,000	27,750	38,050

Excavation

After blasting a bulldozer cleans the floor of the pit and pushes the loose scattered material against the toe of the bank. It is also used to clean up around the shovel and maintain roads. The bulldozer operates about 1 1/2 hours per shift. It is a Caterpillar-Diesel tractor, RDS, equipped with a 12-foot angle-dozer blade and driven by a 90-horsepower engine using 27°-plus Diesel oil. An auxiliary gasoline engine is run from 2 to 10 minutes at each start, depending on atmospheric temperature.

A Model-80 Northwest shovel with a 2 1/2-cubic-yard dipper is used for loading. It is operated by a 110-horsepower engine burning 35°-plus stove oil; the engine is started on gasoline. The dipper is equipped with four teeth having manganese steel shanks and removable "hi-carbon-steel" points. Worn shanks are built up with alternate layers of stainless and "hi-carbon-steel" and coated with cast iron. Shanks require building up every 2 or 3 months. The life of a set of steel points is about 15 days. They are discarded when worn out. A gasoline-engine-driven, 1,500-watt, 110-volt, Kohler electric plant provides power for the two 250-watt headlights and other lights on the shovel. The actual number of hours the dipper operates is recorded by a Model-K Servis recorder attached to the side of the shovel cab.

Several gallons of gasoline are used for starting, and 50 gallons of stove oil per 8-hour shift are used to run the shovel engine.

The shovel faces the bank and loads the trucks alternately on each side. As one truck pulls away another is already in place. In the softer places considerable digging can be done by the shovel without blasting.

Boulders of schist and gray granite are considered as waste and at first were dumped into an old glory hole in the floor of the pit. When the hole was filled the waste was hauled to a waste dump nearly one-fourth mile distant. Waste boulders are set aside by the shovel. When too large for loading into trucks, they are block-holed. The quantity of waste handled in the pit has ranged from less than 100 to more than 6,000 tons per month. Surface boulders of yellow granite are blasted ahead of the shovel; others are set aside by the shovel for block holing. Rock smaller than 30 inches in maximum dimension is sent to the crusher; some blasting, however, was required at the receiving grizzly. Later the grizzly was abandoned.

Transportation

Hauling is done in four remodeled, 3 1/2-ton, six-wheel, International, gasoline-driven, end-dump trucks. The original frame was built up and the gear ratio cut down so that the trucks now carry 9- to 10-ton loads. Seven trucks are on the job; three are ready as spares or are under repair. The trucks are old and have been amortized on previous jobs; however, they serve the purpose as the haul here is short and virtually level.

The average hauling distance for the ore is 1,350 feet and for waste, 1,150 feet; the roadbed is maintained with the bulldozer.

Trucks are driven at a relatively high speed; a round trip, exclusive of loading, requires about 3 minutes. Occasionally, when four trucks are in use, 10 loads can be hauled in 15 minutes; the average without interruptions is 10 loads in 19 minutes. From November 1938 to January 1939, when two 8-hour shifts per day were worked, an average of 2,975 dry tons of rock was mined per working day. Of this amount, 2,900 tons were hauled to the crusher and 75 tons to the waste dump. When the mill bin is filled with fines, ore hauling and crushing are stopped; when the mill bin is low, the mining and crushing crew work overtime.

A truck uses 2 1/2 gallons of gasoline per hour, and a set of tires lasts about 6 months.

Weighing

All trucks, whether loaded with ore or waste, are weighed on 30-ton, type-S, Fairbanks Morse autotruck scales equipped with dial and printomatic weigher and a bank of nine designating numbers. The scale platform is 10 feet wide by 28 feet long. Trucks do not come to a complete stop when being weighed. Just before they arrive at the platform the scaleman presses a designating-number button corresponding to the number on the truck. When the front wheels of the truck are on the platform operation of the dial begins, and the printomatic button is pressed when all wheels are on the platform. The truck number and weight are printed on a tape. Tare weight of the trucks is taken once a shift, or when desired.

The weigher has a printed form, which he makes out in triplicate, showing the load and truck number; gross, tare, and net weight; time; and remarks. Every tenth truck load is subtotaled. The weigher's wages are paid equally by the contractor and the company.

Pit and road lights

The pit is lighted by two 1,000-watt 120-volt lights attached to a pipe frame 13 feet high mounted on skids. The frame is dragged to the desired location by the bulldozer and pulled back to a safe place before blasting. The lights have 26-inch-diameter aluminum reflectors. Two-hundred-watt, 120-volt lights are placed at intervals along the road and at the ore bin. The face in front of the shovel is lighted by the shovel lights.

Pit labor

An operating crew for two 8-hour shifts per day comprises the following men:

Scalemen	2
Shovel operators	2
Shovel oilers	2
Truck drivers	8
Drillers	2
Powderman	1
Coyote miners	6
Grizzlymen	2
Mechanics	2
Foreman	1
Total	28

Grizzlymen were not used after the grizzlies were discarded. Coyote miners are employed only when coyote blasting is done. The bulldozer is operated by any available number of the crew.

PLANT CONSTRUCTION AND COST

After all data from mill tests were tabulated a flow sheet was made and plans were drawn for a 300-ton-per-hour-maximum-capacity crushing and screening plant and a 550-ton-per-24-hour-capacity grinding and classifying unit. Later, the capacity of the grinding unit was increased to 1,000 tons by putting the ball mill in open circuit with the classifier.

The contract for construction of the plant was given to the Western-Knapp Engineering Co., of San Francisco, on a cost-plus basis. Most of the purchased equipment was new, with the exception of the large jaw crusher and motor, ball mill, and some belt-conveyor parts. A few large and small motors, some conveyor parts, the concentrating table, and the thickeners, which had been used in the old mill, were installed in the new plant.

Buildings for housing the crushers, screens, and some of the conveyors were constructed of timber and red corrugated iron. The new grinding and classifying unit was installed in the old stamp-mill building and the old ore bin used.

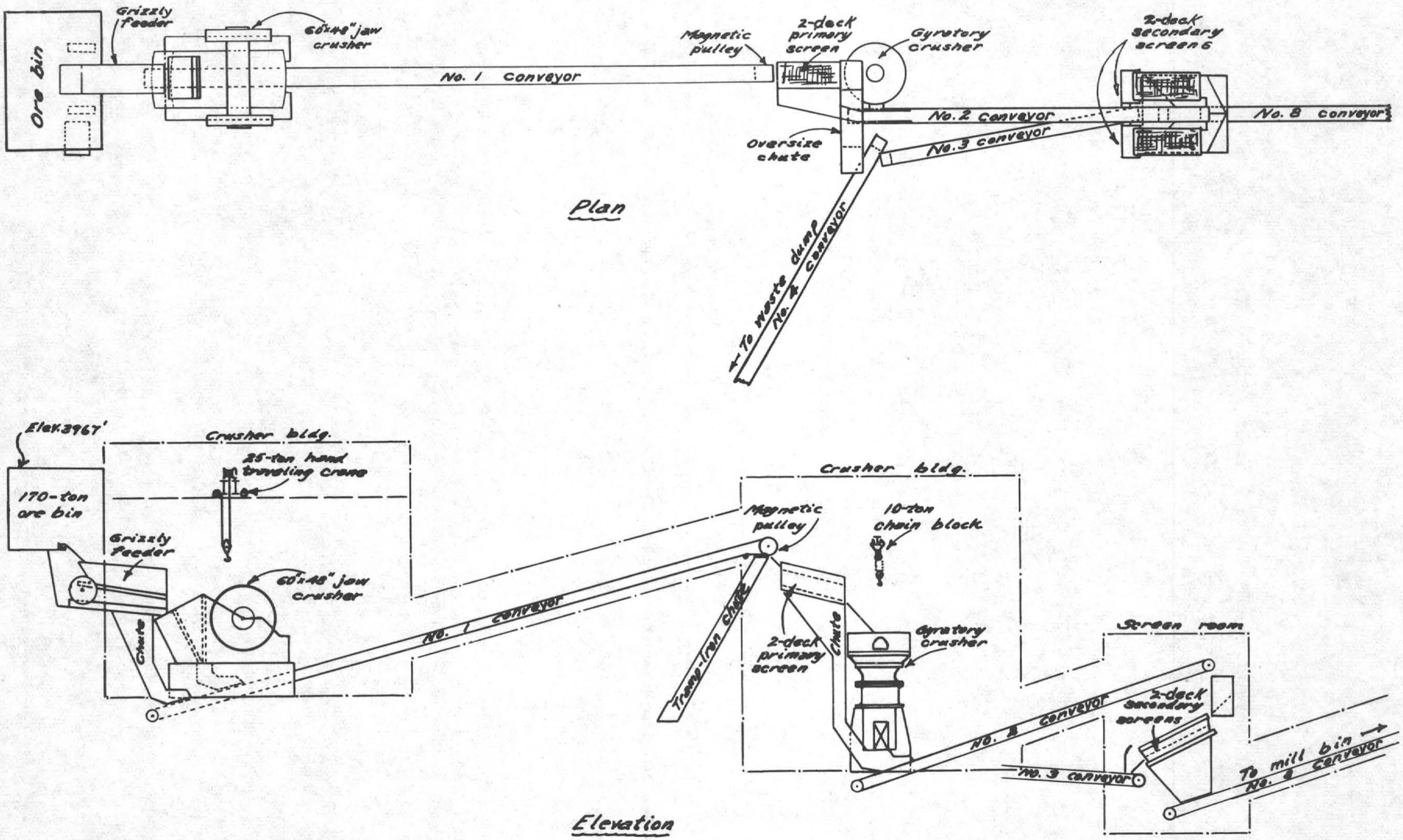

Figure 4.- General arrangement, crushing and screening plant.

Mine pit

Power shovel (2 1/2-cubic-yard capacity)

Autotrucks (9- to 10-ton capacity)

Autotruck scale (30-ton capacity)

Grizzly (30-inch square openings)

Ore bin (170 tons full capacity)

Sheridan grizzly feeder (4- by 10-foot, 5- by 7-inch openings)

Oversize | Undersize

Jaw crusher (60- by 48-inch)

No. 1 belt conveyor (30 inches wide)

Magnetic pulley (30- by 32-inch)

Double-deck vibrating screen (4- by 10-foot)

Upper deck (3-inch square openings)

Oversize | Undersize

Lower deck (1-inch square openings)

Oversize | Undersize

Gyratory crusher (4-foot, type-TY)

No. 2 belt conveyor (24 inches wide)

2 double-deck vibrating screens (4- by 10-foot)

Oversize | Undersize

No. 3 belt conveyor (24 inches wide) | No. 8 belt conveyor (18 inches wide)

No. 4 belt conveyor (24 inches wide) | Mill ore bin (1,800 tons full capacity)

No. 5 belt conveyor (24 inches wide)

No. 9 belt conveyor (24 inches wide)

No. 6 shuttle belt conveyor (24 inches wide)

No. 7 stacker belt conveyor (24 inches wide)

Waste dump

Figure 5.- Flow sheet, crushing and screening unit.

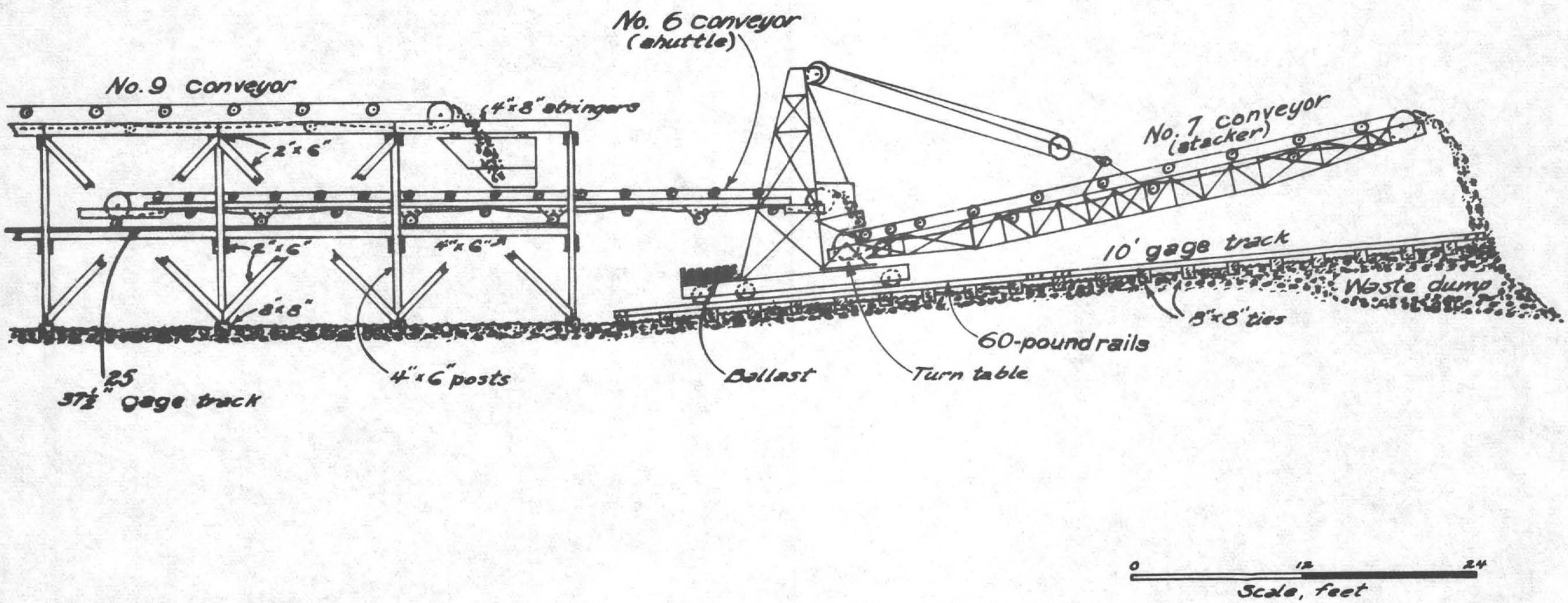

Figure 6.- Waste-distributor system.

Construction work was begun on October 1, 1937, and the plants were completed about January 15, 1938. The total cost of the crushing, screening, and grinding plants was $120,000, including the contractor's charges.

CRUSHING AND SCREENING PLANT

General

Figures 4, 5, and 6 show the general arrangement and flow sheet of the crushing and screening plant.

Operations were begun on a one-shift basis January 19, 1938, and after a few minor changes the capacity of the plant was soon increased to 54,000 dry tons per month.

In the latter part of October of the same year the plant began operating on a two-shift basis, and the tonnage was increased to a maximum of 92,000 dry tons per month and an average of 85,600 tons per month over several months.

To May 1, 1939, 70 percent of all material delivered to the crushing and screening plant from the open pit proper was discarded as waste, and 30 percent was fines delivered to the mill ore bin.

The plant has given the desired screening results, and the capacity, upkeep, operating efficiency, and costs have equaled or exceeded expectations.

Primary crushing

At first a flat grizzly made of 110-pound inverted-T rails with 30-inch square openings was used on top of the crusher bin. Owing to frequent breaking of the rails it was decided to abandon the grizzly and depend on the shovel operator to load rock smaller than 30 inches. This arrangement worked very satisfactorily, especially if the bin was kept partly full.

The crusher bin of heavy timber, is 20 feet long, 14 feet wide, and 12 feet deep on the inside and has a capacity of 170 tons. It is long enough for two trucks to back up to the edge and dump at the same time. A section of the front and bottom of the bin is cut out and a steel hopper 4 feet wide inserted, which allows the ore to enter the Sheridan grizzly feeder.

In order to operate a plant of this type successfully the feed to the primary crusher must be as uniform as possible. After several types of feeders had been investigated the 4- by 10-foot Sheridan grizzly feeder was selected. This feeder has balanced-screen manganese plates with 5- by 7-inch openings, of which 75 percent was blinded, as too much of the undersize was passing through. It is set at a down slope of seven-eighths-inch per foot. A 25-horsepower, 900-r. p. m., variable-speed motor connected by a flexible coupling drives the feeder through a speed reducer. By the use of constant-duty grids and drum controller the feeder can be operated constantly at any speed required to give the desired feed.

The coarse material is moved along the feeder into the jaw crusher, while the fines drop through the openings and combine with the crushed product on the belt conveyor (conveyor No. 1) below.

The primary-crushing unit is operated by one man who is seated at the side of the No. 1 conveyor in front of the crusher so that he can see into the feeder. He regulates the feed by an extension of the drum controller and sorts sticks and large rock slabs from the belt. The sticks and slabs drop through an inclined chute into 1-ton ore cars which a roustabout trams to a nearby waste dump.

Primary crushing is done by an Allis-Chalmers 60- by 48-inch semisteel Blake-type crusher equipped with manganese-steel jaw and side plates. The crusher is operated at a speed of 145 r. p. m. and is driven by a 250-horsepower, 690-r. p. m., variable-speed motor through an endless, 20-inch, two-ply, special-heavy, waterproof leather belt. The crusher pulley is 120 inches in diameter and the motor pulley 24 inches in diameter. An idler pulley 20 1/2 inches in diameter fitted with a floating bronze bushing keeps the belt tight. The bushing is loose between the bore and the shaft and is grooved for lubrication by Alemite fittings. An angle-iron frame hinged to two vertical channel-iron posts supports the pulley shaft. An extra grooved bushing is kept as a spare, but after 15 months of operation no replacement has been necessary. A little neat's-foot oil is applied to the outer side of the belt every day as a preservative, and castor oil is applied to the inner side when needed to prevent slipping. Occasionally, caked dust has to be scraped off before the dressings are applied.

Forced lubrication, using Keystone KV medium oil for the pitman bearing, is provided by an oil pump driven from one end of the eccentric shaft; the oiling system comprises pump reservoir, strainer, filter, and necessary piping for connecting it to the crusher. Cooling water circulates through copper tubes in the pitman bearing. The pitman shaft bearings in the frame are lubricated with Keystone No. 96 wool-yarn elastic grease. The swing-jaw shaft does not move in the frame bearings; instead, a thick brass bushing 2 feet long is set in each end of the swing-jaw bore and allowed to move on the shaft. The upper halves of the bushings are grooved and lubricated through Alemite fittings with Gargoyle grease AA No. 2 in winter and AA No. 4 in summer. Toggle seats are lubricated with whatever used oil may be available. Felt pads 1/2 inch thick are placed on top the toggles to help keep out dust and to saturate the seats with oil.

Jaws are set with a maximum opening of 5 1/2 inches. A larger opening allows too many large slabs to pass through; these slabs interfere with transfer at the conveyor chutes.

A 25-ton, hand-operated, two-speed, traveling-bridge crane with a 28-foot span is used for handling crusher parts.

Primary screening

The minus 5-inch material from the primary crushing unit is fed by conveyor No. 1 to the primary screen. This is a Tyler-Niagara, type-600, double-deck, 4- by 10-foot vibrating screen suspended by four cables and coil compression springs and set at an angle of 19°. The springs minimize vibration of the building, especially when the screen is starting and stopping. A 7 1/2-horsepower, 1,160-r. p. m. motor drives the screen at 650-r. p. m. through a V-belt drive, using 3-B belts.

A 3-inch-square-opening screen with 5/8-inch-diameter wires is set on the top deck and a 1-inch-square-opening screen with 1/4-inch-diameter wires on the lower deck. Both screens have hi-manganese-spring-carbon-steel wires with a flat-top-crimp weave. This weave gives longer wear than the double-crimp or lock-crimp weave.

The plus 3-inch material, which comprises about 20 percent of the total feed, is discarded as waste and drops onto conveyor No. 4 through an inclined steel chute lined with 25-pound T rails. The minus-3-inch rock is delivered to the secondary crusher and the minus-1-inch material to conveyor No. 2, where it is combined with the crusher discharge and taken to the secondary screens.

Secondary crushing

Secondary crushing is done with a 4-foot Traylor, type-TY, gyratory crusher which is set to discharge a product having a maximum size of 1 1/2 inches. It is driven by a 150-horsepower, 580-r. p. m., variable-speed motor through a V-belt drive using 9-E belts which give a crusher pulley speed of 590 r. p. m. The crusher has a feed opening of 10 inches and a 5/8-inch head motion set at 3/4 inch (closed side) with head in middle position. The bell-head mantle and curved concaves are of manganese steel.

Gargoyle D. T. E. extra-heavy oil is used in the circulating system, and Gargoyle 600 W. on the suspension or head bearing. A water-circulating system in the oil reservoir cools the circulating oil.

Feed to the crusher is distributed through three chutes to obtain even wear on the concaves. The original mantle and concaves are still in use after 15 months. On May 1, 1939, approximately 235,000 tons of rock had passed through the crusher.

A 10-ton, hand-operated, self-alining, geared, I-beam trolley with a 10-ton chain hoist is used for handling crusher parts.

Secondary screening

Secondary screening is done in parallel with two Tyler-Niagara, type-300, single-deck, 4- by 10-foot vibrating screens also suspended by cables and coil compression springs but set at an angle of 21°. Later these screens were changed by adding another deck below, thereby increasing the screening capacity 20 to 25 percent. Five-horsepower, 1,750-r. p. m. motors are used to drive each screen at a speed of 1,200 r. p. m. through V-belt drives using 3-B belts.

The upper screen is used as a scalper and has 3/4-inch square openings of No. 8 (0.162 inch in diameter) wire with a flat-top-crimp weave. The lower screen has 1/4- by 1/2-inch rectangular openings with the long dimension across the screen and uses a No. 11 (0.120 inch in diameter) wire with a double-crimp weave. Hi-manganese-spring-steel wire is used on both upper and lower screens.

Fines from the lower screens drop onto conveyor No. 8 through a hopper and are taken to the mill bin. The oversize from both decks goes to conveyor No. 3 as waste.

Conveying and waste disposal

Mine belt conveyors are required to transport the rock to the secondary crusher, screens, waste dump, and mill bin. Figures 4, 5, and 6 show the lay-out of the system, and table 2 gives details of the conveyor equipment and other data.

All conveyors except No. 8 are driven by geared-head motors through sprocket-and-chain drives, and all inclined conveyors except No. 8 are equipped with ratchet-and-pawl hold-backs to prevent loaded belts from running backwards if the sprocket chain breaks or the motor stops.

Oil- and dust-tight, guard-type casings are used on all sprocket-and-chain drives, and all moving parts are fully enclosed. The casings are made of sheet steel and are split crosswise to facilitate installation and inspection. The lower run of the chain drive is in contact with the oil, and an indicator shows the oil level in the casing.

Belts on conveyors Nos. 1, 2, 3, 4, 6, and 8 are driven by head pulleys and those on conveyors Nos. 7 and 9 by tail pulleys. The belt on conveyor No. 5 is driven by an intermediate pulley. All belts are kept tight by screw-type take-up bearings.

Throughing idlers and return rolls have roller bearings and are lubricated by Alemite fittings. Positive, self-alining three-roll troughing idlers are placed about 125 feet apart on conveyors Nos. 5 and 9.

Feed chutes for both coarse and fine material are built with vertical sides that flare in front and slope toward the direction of belt travel in back. Steel baffle plates at transfer chutes are lined with 25-pound T rails. One-half-inch rubber, backed with two-ply duck, has proved very satisfactory as a liner for feed chutes and skirt boards.

TABLE 2. - Conveyor data

Conveyor No.	1	2	3	4	5	6	7	8	9
Conveyor:									
Length center to center between pulleys........................ feet	89	57	38	90	286	48	38	160	226
Slope........................ degrees	+15	+18	+15	+15	0	0	+12±	12, 0	0
Belt:									
Width........................ inches.	30	24	24	24	24	24	24	18	24
Ply..................................	6	5	5	5	5	5	5	5	5
Speed................ feet per minute	308	325	325	325	368	385	370	365	368
Weight of duck.................. ounces.	28	28	28	28	28	28	28	28	28
Top-cover thickness............. inch..	3/16	1/8	1/8	1/8	1/8	1/8	1/8	1/8	1/8
Bottom-cover thickness........... do...	1/16	1/16	1/16	1/16	1/16	1/16	1/16	1/16	1/16
Pulleys:									
Head-pulley diameter........... inches.	30	24	24	24	20	20	18	30	20
Tail-pulley diameter............. do...	20	20	20	20	18	18	20	20	20
Take-up length.................. do...	24	24	18	18	24	18	18	36	30
Drive, type..........................	1/H	1/H	1/H	1/H	2/I	1/H	3/T	1/H	3/T
Rollers:									
Trough type No........................	4/31	4/31	4/31	4/31	5/70	4/32	4/32	5/70	5/70
Diameter....................... inches.	6	6	6	6	5	5	5	5	5
Spacing, center to center......... feet.	4	4	4	4	4, 4 1/2	3	4 1/2	5 1/2	4, 4 1/2
Return idlers:									
Type No..............................	4/S1	4/S1	4/S1	4/S1	5/71	4/S1	4/S1	5/71	5/ 71
Diameter....................... inches.	5 1/2	5	5	5	5	5	5	5	5
Spacing, center to center......... feet.	9 1/2	8	10	10	12	10	10	10	12
Drive:									
Sprocket-and-chain size No............	6/120	6/80	6/80	6/100	6/100	6/60	6/80	(7/)	6/100
Geared-head motor.......... horsepower.	15	7 1/2	5	10	10	3	5	8/15	7 1/2

1/ H - head pulley.
2/ I - intermediate pulley, 30-inch diameter.
3/ T - tail pulley.
4/ Rex-Sterns.
5/ Link-Belt.
6/ Roller chain.
7/ Pulley-and-gear.
8/ Not geared-head.

All conveyor trestles are built of timber and have a 2-foot platform with a guard rail on one side of the belt. Electric lights are installed above the belts and transfer chutes for night work. One- by twelve-inch boards on one side of the belts keep the wind from blowing the belts off the troughing idlers when the conveyor is running empty.

Rock spillage below transfer chutes is removed by a small, gasoline-driven dragline hoist and scraper.

Conveyor No. 1, which conveys the fines from the feeder and the discharge from the jaw crusher to the primary screen, is 89 feet long between centers, slopes 15° upward, and has a 30-inch belt which travels at a speed of 308 feet per minute. (See table 2.)

A Dings magnetic pulley is the head pulley for conveyor No. 1. It is 30 inches in diameter and 32 inches long and has a speed of 40 r. p. m. A motor-generator set supplies direct current for the pulley. The generator, which has a capacity of 3 kilowatts at 125 volts, is direct-connected to a 5-horsepower, 440-volt, 1,725-r. p. m. motor. The tramp iron drops into a pocket below the head pulley and slides down an inclined chute into an ore car with a capacity of 1 ton and thence goes to the waste dump.

Conveyor No. 2, which transports the fines from the primary screen and the discharge from the gyratory crusher to the secondary screens, is 57 feet long, slopes 18° upward, and has a 24-inch belt that travels at a speed of 325 feet per minute.

Conveyor No. 3 takes the oversize (waste) from the secondary screens and carries it to conveyor No. 4. It is 38 feet long, has an upward slope of 15°, and travels at a speed of 325 feet per minute. In other respects it is about the same as conveyor No. 2.

Conveyor No. 4 is set at an angle of 145° to the right to conveyor No. 3 and extends toward the waste dump. Besides receiving the waste from conveyor No. 3 it receives the plus-3-inch discard from the primary screen. It is 90 feet long, slopes 15° upward, and has the same belt speed as conveyor No. 3. It discharges onto conveyor No. 5.

Conveyor No. 5 extends in a direction across the canyon and is set at an angle of 135° to the right of conveyor No. 4. It is 292 feet long, lies about horizontal, and has a belt speed of 368 feet per minute. The drive, which comprises a 30-inch-diameter drive pulley and two snub pulleys, is beyond the center of the conveyor. The conveyor trestle, which ranges from 11 to 14 feet in height, has bents spaced at 12-foot centers.

Conveyor No. 9 receives the feed from conveyor No. 5 and extends down the center of the canyon at an angle of 123° to the right of conveyor No. 5. It is 226 feet long at present and has the same belt speed and other characteristics as conveyor No. 5 except that it has a tail pulley driven by a 7 1/2-horsepower motor. The conveyor trestle is 13 feet 8 inches high and has bents spaced at 12-foot centers.

Conveyor No. 6 is a "shuttle" conveyor that moves ahead on a track laid below conveyor No. 9. (See fig. 6.) The track is 37 1/2-inch gage with 25-pound rails and is laid on stringers held by cross timbers bolted to the bents. The conveyor is made of structual steel, and the front end is attached to the four-post frame of conveyor No. 7 (stacker). As the "stacker" conveyor moves ahead, additional trestle bents are added to conveyor No. 9, and the track is extended. When the shuttle conveyor moves ahead to the limit, enough belt is added to conveyor No. 9 to extend it 36 feet. The shuttle conveyor is 48 feet long, and the belt travels 385 feet per minute.

Conveyor No. 7 is made of structual steel and consists of a four-post frame 15 feet high mounted on a platform car 11 feet 8 inches wide by 15 feet 6 inches long. (See fig. 6.) A 38-foot boom is pivoted to a turntable fastened to the top of the car by a kingbolt. The outer end of the boom is raised or lowered by a sheave and wire rope fastened to the top of the tower and operated by a worm-drive hand winch attached to the side of the feed hopper. Several tons of iron ballast are placed on the rear end of the car. The track is 10-foot gage with 60-pound rails and is laid on 8- by 8-inch by 14-foot ties spaced at 3-foot centers. As the car moves forward the rails and ties are taken up and relayed, except that every fourth tie is left as a sill for the trestle bent. The boom generally slopes 12° upward, and the track is laid on about a 15-percent upgrade. In order to have conveyor No. 9 level it is necessary to maintain this grade to compensate for settling of the end of the waste dump.

A step jack held in place by a bumper clamped to the rail back of the rear wheel pushes the car ahead. Usually the car is moved forward 6 feet at a setting. Generally not more than 40 feet of rail is kept ahead of the car.

The boom can distribute waste over an arc of 180° and is pulled around with rope blocks by one man.

The belt is driven by the tail pulley and travels at a speed of 385 feet per minute.

The difference in elevation between the brow and the toe of the waste dump is 175 feet, and the angle of repose of the rock is 35°.

Conveyor No. 8 takes the fines from a small bin below the secondary screens and distributes them in the mill ore bin by an automatic friction-driven tripper. The conveyor is 160 feet, slopes 17° upward to the top of the bin, and then extends horizontally to the end of the bin. The belt, which is 18 inches wide, travels at a speed of 365 feet per minute and is driven by a 15-horsepower motor through a belt-driven pulley-and-gear mechanism.

Operating control signal system

For smooth operation it was necessary to install a signal system to control the feed to the primary crusher. With the large tonnage being handled there would be congestion of rock from plugged chutes, broken belts, burned-out motors, or other break-downs if the feed were not cut off immediately.

To indicate to the crusherman whether or not all motors were running a red-light signal system was installed. This comprises a Jeffries Type SS, 50-volt-ampere, 440- to 12-volt, single-phase transformer connected between each motor and its corresponding magnetic switch. A board carrying red lights was placed by the side of the crusherman's seat, and insulated wires were run from each transformer to a red light, above which was printed the name of the equipment operated by the motor. As long as the motor is running the red light shows, but when the motor stops the light goes out and the crusherman immediately stops the feed. When the motor is started after each shut-down, no rock is fed until all red lights show.

A "hooter" also was set up near the crusherman and connected to push buttons in the secondary crusher and screening buildings and on nearly all conveyors so that any operator could signal the crusherman by "hoots" as follows: 1 - stop feed; 2 - start feed, 3 - increase feed, 4 - decrease feed.

The hooter signal system has a 110- to 12-volt transformer on the electric-light circuit, and the push-button contact is on 12 volts.

Dust control

Dust in the plant is controlled as much as possible by spray nozzles above the crusher bin, grizzly feeder, jaw crusher, conveyor No. 1, primary screen, and gyratory crusher and at other places where necessary.

Labor

Crushing is done on two shifts from 7:30 a.m. to 4:00 p.m. and from 7:30 p.m. to 4:00 a.m. with a half-hour shut-down each shift for lunch. This arrangement prevents the ore in the mill bin from getting too low between shifts.

The operating and repair crew for a day of two 8-hour shifts comprises the following men:

Crusher operators	3
Screen operators	3
Waste-stacker operators	3
Oilers and roustabouts	2
Mechanic (repairs and maintenance)	2/3
Electrician (repairs and maintenance)	2/3
Machinist (repairs and maintenance)	1/4
Truck driver (miscellaneous)	1/3
Foreman (supervision)	1/2

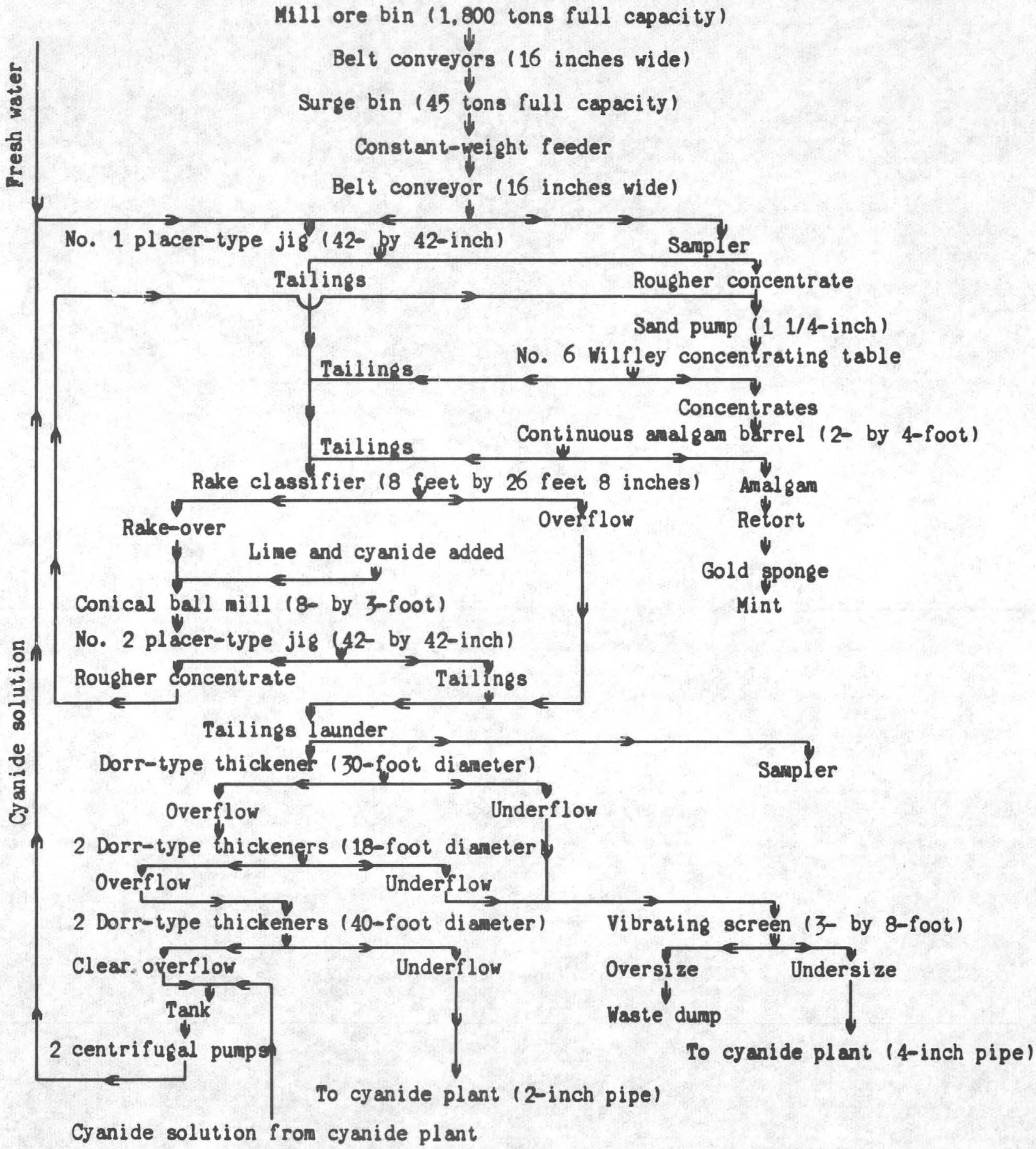

Figure 7.- Flow sheet, grinding and classifying unit.

The crusher, screen, and stacker operators spend part of their time on repair or other work. The crusherman has charge of the shift and looks after the direct operation of the feeder, jaw crusher, and conveyor No. 1. When inspecting the other operations, he leaves a man in his place temporarily. The screenman looks after the gyratory crusher, all screens, and conveyors Nos. 2, 3, 4, and 8, and the wasteman takes care of conveyors Nos. 5, 6, 7, and 9. The mechanic, electrician, and oilers work on odd-hour shifts so that they can work part of their time when the plant is shut down between shifts. About a third of the mechanic's and electrician's time is spent on repairs to the grinding unit.

Between shifts the crew lays track for conveyor No. 7, moves conveyors Nos. 6 and 9 forward when necessary, drags rock spillage away from beneath transfer chutes, and does general repair work.

The foreman supervises crushing, screening, and grinding.

MILLING

General

The grinding and classifying unit began operation on January 28, 1938 and was capable of milling 600 tons per 24 hours. Later it became necessary to increase the tonnage and deliver a larger proportion of sands to the cyanide plant. Consequently, in the following October, the ball mill, which in the original flow sheet was in closed circuit with the classifier, was put in open circuit, and another jig was installed. This increased capacity to 1,000 tons. Figure 7 shows the present flow sheet of the plant.

As already stated, the grinding and classifying unit was installed in the old stamp-mill building and the old ore bin used. This building is made of timber and galvanized corrugated iron. The floor is at an elevation of 3,894 feet.

Ore bins and feeders

The ore bin, which is constructed of timber, is 87 feet long by 15 1/2 feet wide by 27 feet high (inside dimensions) and has a maximum capacity of 1,800 tons. The top of the bin is at an elevation of 3,937 feet. Owing to its fineness, the ore "chimneys" at the chutes when the moisture content is over 5 percent and must be barred down. When the moisture content is below 5 percent the live capacity is about 1,000 tons.

There are eight chutes with rack-and-pinion gates on either side of the bin which feed onto slow-speed, 16-inch belt conveyors. A cross belt conveyor carries the feed from the west to the east conveyor, which in turn delivers the ore to a 45-ton-capacity surge bin built like an inverted pyramid. Attached to the bottom of this bin is a type 3-C, Hardinge, standard, constant-weight feeder equipped with a 26-inch belt and operated by a 1/4-horsepower, 440-volt motor through a sprocket-and-chain drive. A counter records the number of revolutions of the belt, and a no-load cut-off and bell signal actuated by mercoid tubes are attached to the feeder.

The constant-weight feeder delivers the ore to a 16-inch belt conveyor 51 feet long, which slopes 15° upward. The conveyor is driven at a speed of 151 feet per minute by a 3-horsepower, geared-head motor through a sprocket-and-chain drive. This conveyor feeds the ore through an inclined launder to the No. 1 jig where it first comes in contact with cyanide solution. As the dry ore drops off the belt, samples are cut automatically at regular intervals.

Classifying and grinding

The No. 1 jig makes a low-grade concentrate, and the jig tailings go to a model BH, Wemco duplex rake classifier 8 feet wide and 26 feet 8 inches long, which slopes 3 1/4 inches per foot and operates at a speed of 27 strokes per minute. A 10-horsepower, 1,200-r. p. m. motor drives the mechanism through a V-belt drive with 8-B belts. One grooved sheave is varipitched to change the speed of the classifier rakes when necessary.

Classifier overflow goes to the tailings launder and the raked product to the ball-mill scoop box. The large amount of water used in the No. 1 jig maintains the density of the classifier overflow at a maximum of 13 percent solids.

Typical screen analyses of the mill heads and classifier rake-over and overflow are given in tables 3, 4, and 5, respectively.

TABLE 3. - Screen analysis of mill heads

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 3...	0.71	0.71	Plus 48..	4.62	75.64
4...	7.06	7.77	65..	3.96	79.60
6...	11.09	18.86	100..	3.10	82.70
8...	11.62	30.48	150..	2.52	85.22
10...	10.11	40.59	200..	2.05	87.27
14...	9.69	50.28	Minus 200 (sand)..	1.36	88.63
20...	8.45	58.73			
35...	12.29	71.02	Minus 200 (slime)..	11.37	100.00

TABLE 4. - Screen analysis of classifier-rake product

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 3...	0.85	0.85	Plus 48..	3.53	91.10
4...	12.18	13.03	65..	2.46	93.56
6...	16.94	29.97	100..	1.59	95.15
8...	14.58	44.55	150..	.97	96.12
10...	12.16	56.71	200..	.64	96.76
14...	10.73	67.44			
20...	8.71	76.15	Minus 200 (sand)..	.56	97.32
35...	11.42	87.57	Minus 200 (slime).	2.68	100.00

TABLE 5. - Screen analysis of classifier overflow

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 20..	--	--	Plus 150.	9.73	22.83
35..	0.44	0.44	200.	11.02	33.85
48..	1.25	1.69	Minus 200 (sand)..	6.79	40.64
65..	4.28	5.97			
100..	7.13	13.10	Minus 200 (slime).	59.36	100.00

New cyanide solution and lime are added to the feed launder leading to the scoop box. Cakes of cyanide are dissolved in a 7-gallon tank into which a small stream of water runs, and the solution is allowed to overflow through a pipe. Lime, crushed to minus 1/4 inch, is fed by a 6-inch slow-moving belt conveyor driven by the classifier shaft through a belt-and-gear reduction unit. The belt moves under a V-shaped, 1/2-ton-capacity steel hopper which has a 3- by 1 1/2-inch opening on the bottom.

The ball mill is a Hardinge conical mill 8 feet in diameter by 3 feet in length with a standard scoop feeder 5 feet in radius and a cylindrical discharge 16 1/2 inches in inside diameter. It is driven at a speed of 21 r. p. m. by a 150-horsepower, 580-r. p. m. variable-speed motor through a V-flat drive of 11-E belts and counter shaft. The mill itself weighs 24,000 pounds and the liners 20,500 pounds; the normal operating load is 30,000 pounds of balls.

Wedge-bar-type manganese-steel liners are used. A set of liners lasts about 10 1/2 months and mills an average of 240,000 tons of ore. Liner consumption is 0.085 pound per ton of ore.

Relining takes about 32 hours and requires 144 man-hours. The job comcomprises removing the balls, taking out the old liners, scraping the shell, installing new liners, and refilling with balls.

Four-inch manganese-steel balls are used, and ball consumption is 0.574 pound per ton of ore.

Tables 6 and 7 show typical screen analyses of the ball-mill discharge and general mill tailings.

Concentration

Two jigs and a concentrating table are used principally to recover coarse gold. The No. 1 jig, which receives the mill feed after it comes in contact with cyanide solution, is a single-cell, Pan-American, placer-type jig driven at a speed of 144 strokes per minute by a 3-horsepower, geared-head motor through a V-belt drive. It is 42 inches square and is bedded with 300 pounds of 3/16-inch steel shot and 125 pounds of 1/4-inch shot. The jig tailings are laundered to the classifier and the low-grade concentrates pumped to a 9-cubic-foot surge box which feeds the concentrating table.

TABLE 6. - Screen analysis of ball-mill discharge

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 3..	--	--	Plus 48..	8.69	59.88
4..	0.67	0.67	65..	7.92	67.80
6..	1.60	2.27	100..	5.03	72.83
8..	2.88	5.15	150..	4.40	77.23
10..	5.33	10.48	200..	3.55	80.78
14..	8.11	18.59	Minus 200 (sand)..	2.79	83.57
20..	10.93	29.52			
35..	21.67	51.19	Minus 200 (slime).	16.43	100.00

TABLE 7. - Screen analysis of mill tailings

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 3..	--	--	Plus 48..	8.59	58.38
4..	0.33	0.33	65..	7.36	65.74
6..	1.31	1.64	100..	5.14	70.88
8..	2.59	4.23	150..	4.09	74.97
10..	5.01	9.24	200..	3.53	78.50
14..	8.21	17.45	Minus 200 (sand)..	2.57	81.07
20..	10.92	28.37			
35..	21.42	49.79	Minus 200 (slime).	18.93	100.00

The No. 2 jig receives the discharge from the ball mill and is the same size and type as the other. It is driven at a speed of 125 strokes per minute.

A good feed distribution to the No. 2 jig is obtained by sloping bars that radiate from a center at the ball-mill discharge. (See fig. 8.)

Tailings from No. 2 jig flow to the tailings launder and combine with the classifier overflow; concentrates are pumped to the surge box at the head of the table.

The two jigs in 24 hours produce a total of 40 to 50 tons of concentrates which assay $15.00 to $20.00 in gold per ton (gold at $35.00 per ounce).

A No. 6 Wilfley concentrating table, driven at a speed of 250 strokes per minute by a 1-horsepower motor, is used to raise the grade of the concentrates. The table cover is made of rubber 1/16 inch thick backed with 1-ply heavy duck. Approximately 1,500 pounds of concentrates are produced per 24 hours which assay $700 to $800 in gold per ton. The concentrates contain some scheelite, but the quantity is too small to warrant further treatment in order to recover it. The tailings, which assay about $5.00 per ton, are piped to the classifier.

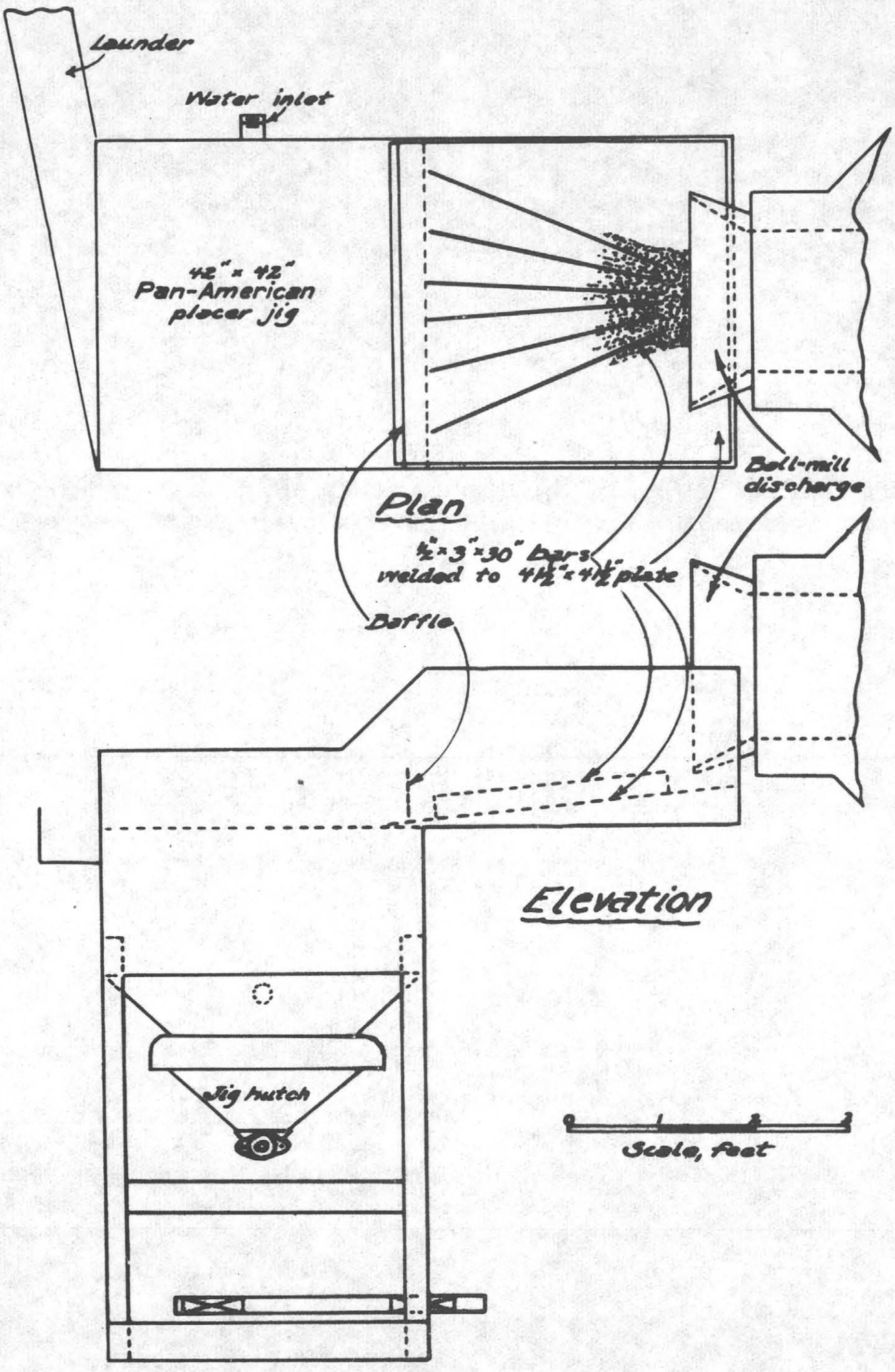

Figure 8.- Pulp distributor to jig.

Amalgamation

The table concentrates are amalgamated in a continuous-type amalgamating barrel 2 feet in diameter and 4 feet in length which is made of a 1 1/4-inch rolled-plate shell and 1-inch boilerplate heads and lined with rubber. The barrel is mounted on trunnions through which the concentrate pulp enters and the thin sludge leaves. There are two 2-inch threaded plugs on one side near each end and a manhole in the center on the opposite side. The barrel is driven at a speed of 20 r. p. m. by a 3-horsepower, geared-head motor through a 60-RC chain-and-sprocket drive. About 600 pounds of 1 1/2- to 2 1/2-inch-diameter steel balls are used as an operating load. The tailings, which leave through the discharge trunnion, assay about $15 per ton in gold. They are piped to the classifier after passing over a mercury trap.

Clean-ups are made about every 3 days. The barrel is shut down, and the balls are taken out through the manhole. The amalgam and partly pulverized concentrate are sluiced out into a square box set on wheels. The box is then wheeled into the clean-up room and hoisted from the frame onto a stand at the head of a 33- by 81-inch inclined amalgamation plate. After the amalgam is cleaned it is put through a screen to remove amalgamated pieces of copper blasting caps and copper wire; this "copper-gold amalgam" is retorted and shipped to the smelter. The screened gold amalgam is put through an amalgam press, retorted, and shipped to the mint.

Generally a charge of 650 troy ounces of mercury is fed into the barrel after each clean-up; however, the amount varies with the richness of the concentrate.

The squeezed gold amalgam has an average fineness of 255 parts of gold, or a value of approximately $9 per troy ounce, and the gold sponge has an average fineness of 737 parts of gold and 190 parts of silver. Mercury loss is 20 to 25 pounds per month.

Gold recovery by concentration and amalgamation is approximately 30 percent of the mill-head value.

Mill tailings to cyanide plant

Mill tailings, which comprise the classifier overflow and the No. 2 jig tailings, have a density of 40 percent solids and are laundered to a set of thickeners which dewater the pulp to 45 to 50 percent solids and reclaim the mill solution. The wooden tailings launder, which is 11 inches deep and 8 inches wide on the inside, is lined with 7/32-inch rubber backed with one-ply duck and set on a grade of 1 inch to the foot.

All of the thickeners, which were used in the old mill, have a hi-head Dorr-type mechanism in steel tanks 8 feet deep. The first one is 30 feet in diameter and overflows slimes to two thickeners 18 feet in diameter and set in parallel; the underflow is piped to a hopper feeding a vibrating screen.

The overflow from the 18-foot thickeners goes to two 40-foot thickeners set in parallel and the underflow to the screen hopper. The 40-foot thickeners overflow a clear solution into a sump tank from which it is pumped back to the mill-solution tanks for reuse in the mill. Colloidal slimes from the underflow have a density of 25 to 30 percent solids and flow through a 2-inch pipe to the 100-foot, center-pier thickener at the cyanide plant.

A 12-gage steel pipe line 4 inches in inside diameter now conveys the pulp from the underflow of the 30-foot thickener and the two 18-foot thickeners to the cyanide plant. To prevent the pipe line from plugging, the pulp density must be maintained between 45 and 50 percent solids and the plus 6-mesh material removed with a vibrating screen.

This is a 3- by 8-foot Link-Belt, type UP238, double-deck vibrating screen, inclined 24° driven at a speed of 1,650 r. p. m. The top deck has a short 1/2-inch-square-opening screen to take the wear and break the fall of the pulp onto the lower deck, which has a 5/32-inch-square-opening screen.

Oversize material falls into a hopper at the end of the screen and is fed to a slow-moving 12-inch-wide upward-sloping belt conveyor which stacks it on the hillside as waste. Approximately 1.4 percent of the total tonnage milled (or 14 tons per day having an assay value of $1 per ton) is discarded.

The screen undersize drops into a hopper which feeds it into the 4-inch pipe line. This is 2,740 feet long and slopes downward 250 feet between the intake and the discharge.

Steel pipe has worn better than redwood pipe and is easier to repair.

Intermediate (low-value) cyanide solution is pumped through an 8-inch pipe line from the cyanide plant to the sump tank at the 40-foot thickeners, where it is repumped through a 5-inch pipe line to the mill tanks. This solution is tapped off the 8-inch line and used to sluice the old mill tailings.

Reagents

Water. - Water used for milling contains rather high percentages of sodium and magnesium sulfates and calcium and magnesium bicarbonates and small amounts of sodium chloride, sodium carbonate, silica, and aluminum oxide. These salts have no known deleterious effects chemically, and the water, which has a pH of 8.1, acts as an alkali and slime settler.

The only fresh water in the mill circuit is added on the Wilfley concentrating table and where there is a shortage of return cyanide solution.

Cyanide. - Cyanide (white sodium cyanide, 91 percent NaCN) is added just ahead of the ball mill to increase the strength of the circuit. Solution samples, which are taken daily from the 30-foot thickener overflow, show a free cyanide content of about 0.016 percent (in terms of KCN).

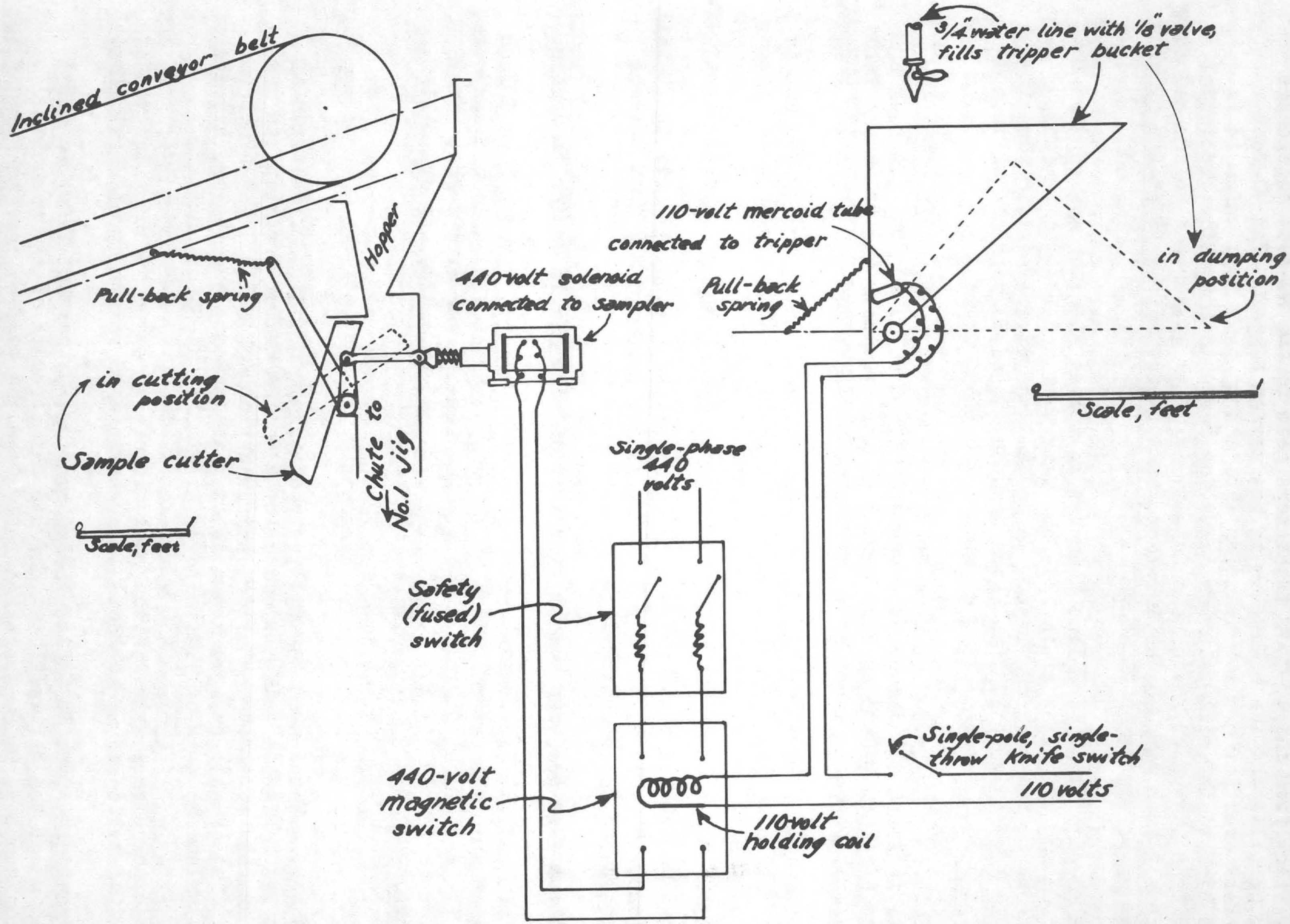

Figure 9.- Automatic ore sampler with wiring diagram.

Lime. - Lime is also added just ahead of the ball mill. The daily solution samples from the 30-foot thickener show an alkalinity of about 0.007 percent CaO.

The lime is purchased from the West End Chemical Co., at Searles Lake, Calif., which burns limestone to obtain CO_2 and sells the lime as a byproduct. It is crushed to minus 1/4-inch mesh and delivered in carbide cans holding 100 pounds. An average analysis is as follows:

	Percent
Available calcium oxide........	93.0
Aluminum and iron oxide........	.6
Magnesium oxide................	1.0
Silica.........................	3.0
Calcium hydroxide..............	1.0
Ignition loss..................	1.4

Mill control

Tonnages. - The volume of ore left in the mill bin on the first of each month is measured and the tonnage calculated on a basis of 20 cubic feet per dry ton.

The tonnage milled per day is measured by the constant-weight feeder which is calibrated each morning. Samples for determining moisture are taken at the time of calibration.

The tonnage of screen-oversize mill tailings is computed from the time required to fill the hopper, the weight per cubic foot, and the moisture content.

As mentioned previously, the ore delivered to the crushing plant is weighed on the platform scales and the moisture content estimated.

At the end of each month the tonnage of ore delivered to the mill bin and the tonnage of waste conveyed to the dump are calculated from the total tonnage crushed and the quantity of ore milled. All tonnage figures are given in dry tons of 2,000 pounds.

Sampling. - Mill ore (mill heads) is sampled ahead of No. 1 jig by a water-operated electrically controlled sampler. (See fig. 9.) The system comprises two circuits - one of 440 volts which energizes the solenoid operating the sampler and one of 110 volts which includes the timing device and energizes the holding coil of the magnetic switch in the 440-volt circuit. When the water in the trip bucket rises to a certain level the bucket dumps, discharging the water and closing the 110-volt circuit in the mercoid tube. After the water is completely discharged the bucket returns to its normal position, where it is held by a spring, and the 110-volt circuit opens again.

The frequency of taking sample cuts is governed by the length of time necessary to fill the trip bucket to the discharging level. This sampler is set to cut a sample every 16 minutes.

Sample cuttings drop through a splitter set on top of a two-compartment box; each half goes into a compartment having a sloping bottom. At the end of the mill day (7:30 a.m.) the sample from each compartment, which weighs about 350 pounds, is split to 20 pounds and dried. When dried, the two 20-pound samples are pulverized to about 16-mesh, combined into one, and thoroughly mixed. This sample is then split into two samples of 10 pounds each, designated A and B. These are finely pulverized and mixed by rolling. Three 1-pound samples are taken from each. Two of these are sent to the assay office, and the third is kept for use if an umpire assay should be required.

Mill-tailings samples, when desired, are taken by a device similar to that used for taking head samples; the timing and control mechanism operates a vertical-slotted cutter which moves across the pulp stream at the end of the launder.

Samples of the incoming mill solution are taken daily. The gold content ranges from 15 to 35 cents per ton.

Densities. - Operators take samples for density at frequent intervals; these are recorded on the daily mill report. The average densities of the pulps are as follows:

	Solids, percent
Ball-mill discharge...........	60
Classifier overflow...........	13
Mill tailings before dewatering	40
Mill tailings after dewatering.	45 to 50

Screen analysis. - Screen analyses are made, when desired, of the mill feed, classifier overflow and rake over, ball-mill discharge, and mill tailings.

Titrations. - Solutions are tested daily for free cyanide and protective alkalinity.

Reports. - Daily reports show the tonnage of ore delivered to the crusher bin, tonnage milled, running and lost time, cause of lost time, percentage of moisture, pulp densities, cyanide and lime in solutions, amount of squeeze amalgam produced on barrel clean-ups, and other items of interest.

Labor

The operating and repair crew for a day of three 8-hour shifts comprises the following men:

Labor

The operating and repair crew for a day of three 8-hour shifts comprises the following men:

Mill operators	3
Roustabout	1
Mechanic (repairs and maintenance)	1/3
Electrician (repairs and maintenance)	1/3
Machinist (repairs and maintenance)	1/6
Truck driver (miscellaneous)	1/3
Foreman (supervision)	1/2

CYANIDATION

General

Details of the earlier operations of the cyanide plant are given in a previous publication.3/

The present paper deals principally with the present flow sheet and operation of the plant.

The cyanide plant is situated about one-half mile northwest of the mill at an elevation of 3,560 feet, or 335 feet below the mill level. It was built principally to treat the 2,300,000 tons of old impounded stamp-mill tailings and a small amount of current mill tailings. Screening tests indicated that 46 percent of the impounded tailings were minus 200-mesh (classed as slimes) and 54 percent minus 20- plus 200-mesh (classed as sands). The slime section was built originally to treat 450 tons of slimes and the sand section 650 tons of sands, a total of 1,100 tons per 24 hours.

Construction of the plant was begun about the middle of April 1935 and operation on September 1 of the same year.

During 1936 and 1937 the tonnage treated per day averaged 1,200 tons, but to treat this quantity it was necessary to avoid the more slimy or colloidal material and treat a larger proportion of the plus 200-mesh material than contemplated in the original plans. By the summer of 1938 the slimes (minus 200-mesh) had increased to 62 percent and the sands (minus 20- plus 200-mesh) had decreased to 38 percent of the total tonnage left in the deposit. The proportion of colloidal slimes also increased. As a result the capacity of the plant was considerably decreased.

To keep the plant on a balanced tonnage it was necessary to increase the capacity of the slime section. This was done early in 1939 by converting a primary thickener into a hydroseparator and installing a 100-foot-diameter Dorr Torq-type (center-pier) thickener and other equipment.

3/ Cooper, Corwin L., Mining and Milling Methods and Costs at the Yellow Aster Mine, Randsburg, Calif.: Bureau of Mines Inf. Circ. 6900, August 1936, 21 pp.

At present the daily capacities of the slime and sand sections are approximately 650 tons each. Of the total, 350 tons are impounded tailings and 950 tons current mill tailings.

To date the entire cost of the plant as outlined in the flow sheet (fig. 10) is $225,000.00.

Mining tailings

Formerly the tailings were sluiced to the cyanide plant with water, but since April 1, 1938, 2 months after operation of the new mill was begun, cyanide solution has been used.

As the plant is slightly below the toe of the impounded tailings, the pumping head required to deliver the pulp to the head of the plant is not great.

Intermediate cyanide solution (intermediate in value between pregnant and barren) is pumped from the cyanide plant in two stages through an 8-inch pipe line to a 21,000-gallon surge tank which is situated above the deposits and slightly below the mill level. Solution from this tank is pumped to the mill. For sluicing tailings, laterals of 4-inch, 12-gage, slip-joint pipe are taken off the 8-inch line where it is desired to mine. Outlets are provided along the 4-inch line for connecting short sections of 2- or 3- inch pipe to which 2-inch rubber-covered hoses are attached. As the solution is drawn from the main 8-inch pipe line the tank acts mostly as a medium for maintaining a constant head.

Attached to the hoses are 5/8- or 3/4-inch nozzles which direct streams of solution against the bank of material being mined. Each nozzle usually is attached to a swivel set on a tripod to provide flexibility, although in some installations the nozzle is merely tied to the tripod. With very slimy material it is sometimes necessary for the hoseman himself to handle the nozzle to give a cutting action to the stream. Pressure at the nozzles varies; the maximum is about 90 pounds per square inch. The banks of sand and slime range in height from a few feet to 100 feet.

The slope of the ground is very favorable for sluicing and allows the pulp to flow readily to bulkheads near the toe of the deposit. These bulkheads are built of timber or stone and prevent surges of pulp at the mining pump. A punched plate with 1-inch-diameter holes is used in the ditch above the bulkhead to remove rocks, weeds, and sticks. From the bulkhead the pulp flows by gravity through about 500 feet of redwood stave pipe 6 inches in inside diameter to the mining-pump sump or suction box. A three-sided grizzly with vertical wooden bars spaced 1 inch apart is placed in front of the pipe intake to catch foreign material.

The main-pump sump, which is made of concrete, has an overflow outlet on one side connected to a ditch. A howler and light are connected to electric contacts at the overflow to warn the hoseman of trouble at the pump.

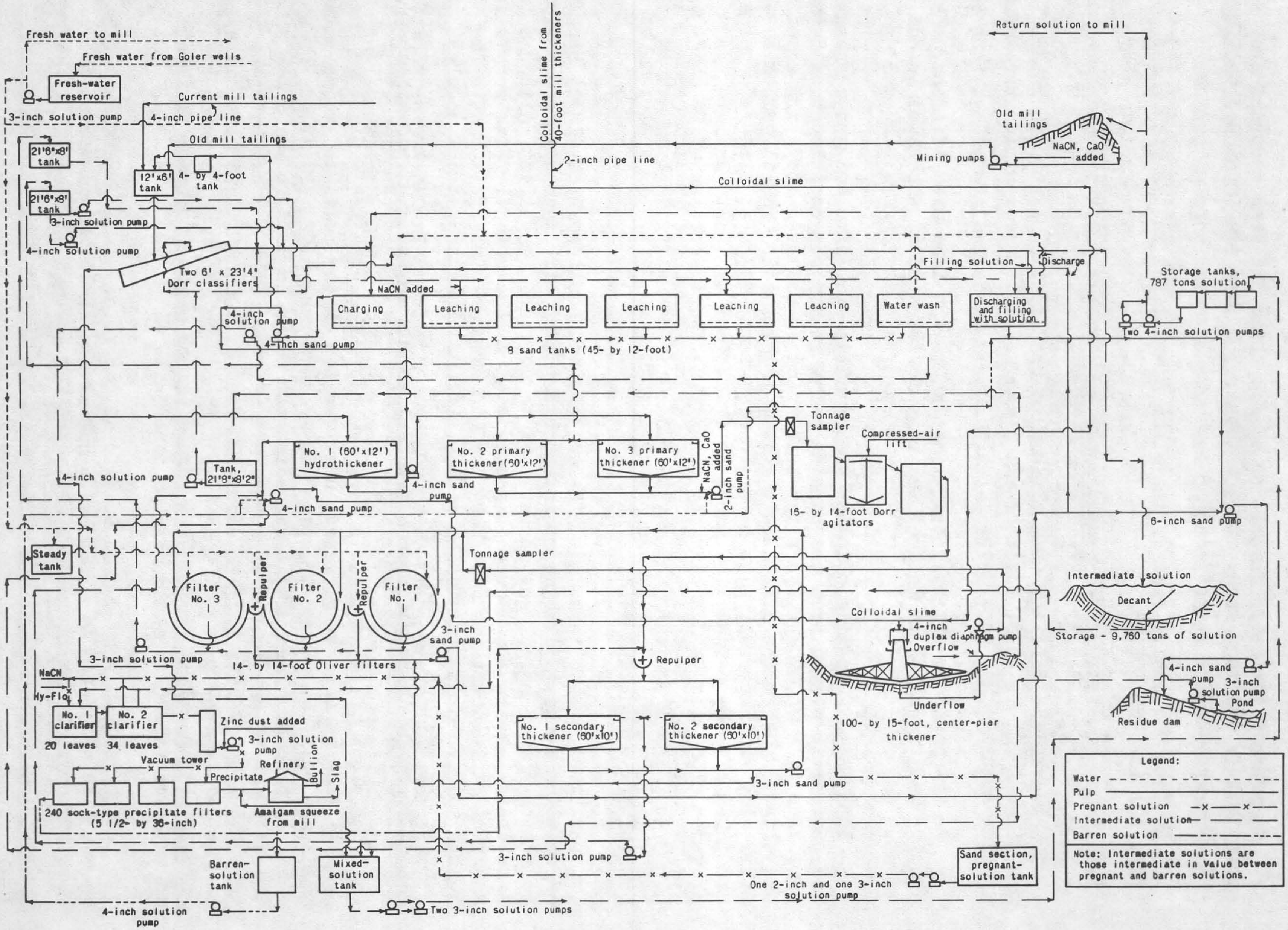

Figure 10.- Flow sheet, Yellow Aster cyanide plant.

A steel plate punched with 1/2-inch-diameter holes is placed on top of the sump to collect pebbles and other foreign matter. Solution is piped to the sump when it is needed to dilute the pulp or wash the screen.

A 6-inch Wilfley sand pump driven at a speed of 1,270 r. p. m. by a 75-horsepower motor through a V-belt drive draws the pulp from the sump and delivers it to the cyanide plant through 850 feet of 5-inch, 12-gage steel pipe. The static head is 88 feet. Valves are placed at intervals along the pipe line to drain it in case of "plug-ups," and 1 1/2-inch nipples are welded onto the line to flush it with solution or water.

Sodium cyanide (91 percent NaCN) and lime are added to the pulp at the pump sump. Cyanide cakes are dissolved by the solution used to make the milk of lime.

The pulp is delivered to the cyanide plant at an average density of 30 percent solids by weight.

In the more isolated sections of the deposit tailings are sluiced to a smaller sand pump which delivers the pulp to the main-pump sump.

Mining is done 100 to 2,000 feet from the mining pumps. No estimate can be made of the loss by seepage and spillage, but it is not great.

The plant is operated on three 8-hour shifts. For night work 750-watt lights are used.

Telephones, howlers, and colored lights are used for intercommunication and signals.

One sluicing boss and four hosemen do the mining. Close cooperation is required between the hosemen and the head operator on shift at the plant.

Classification

The mined tailings and the current mill tailings are delivered into a 12- by 6-foot redwood surge tank above the classifier. (See fig. 10.) The combined pulps then flow to two 6- by 23-foot, Model D rake classifiers which are arranged in parallel, set at a slope of 2 1/16 inches per foot, and operated at a speed of 14 strokes per minute.

Intermediate cyanide solution is added to the classifiers to dilute the pulps and is used to sluice the sand or raked product through launders to the sand tanks for leaching. Slimes overflowing the classifiers have 20 to 22 percent solids and flow through a launder into a hydroseparator where the coarse and fine (colloidal) slimes are partly separated.

The hydroseparator, originally used as a primary thickener, has a 60- by 12-foot redwood tank with the rake mechanism speeded up to a rake-tip speed of 100 feet per minute.

Fine slimes are drawn off by five 4-inch decant pipes inserted 4 1/2 feet below the top of the tank and connected to a 6-inch collecting line which partly encircles the tank. These slimes, which have a density of 5 to 6 percent solids, flow to a 4-inch sand pump which delivers them to a 100-foot-diameter Dorr Torq-type (center-pier) thickener. The underflow of coarse slimes from the hydroseparator has a density of 31 percent solids and is pumped to two primary thickeners set in parallel.

Just before th hydroseparator was installed the slimes treated in the three original primary thickeners required 24 square feet of settling area per dry ton.

Typical screen analyses of the sand, slime, and combined plant heads are shown in tables 8, 9, and 10, respectively, and an analysis of the hydroseparator overflow is given in table 11.

TABLE 8. - Screen analysis of sand-plant heads

Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative	Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative
Plus 6......	0.52	0.52	Plus 65.....	10.61	82.27
8......	2.53	3.05	100.....	6.85	89.12
10......	4.86	7.91	150.....	4.45	93.57
14......	8.83	16.74	200.....	2.40	95.97
20......	11.92	28.66	Minus 200 (sand)......	1.09	97.06
35......	27.87	56.53	Minus 200 (slime).....	2.94	100.00
48......	15.13	71.66			

TABLE 9. - Screen analysis of slime-plant heads

Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative	Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative
Plus 35.......	0.66	0.66	Plus 150.....	18.00	29.26
48.......	1.30	1.96	200.....	7.85	37.11
65.......	3.16	5.12	Minus 200(sand)	4.12	41.23
100.......	6.14	11.26	Minus 200(slime)	58.77	100.00

TABLE 10. - Screen analysis of combined plant heads

Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative	Mesh	Weight retained, percent: This mesh	Weight retained, percent: Cumulative
Plus 6.......	0.28	0.28	Plus 65.....	7.18	46.73
8.......	1.36	1.64	100.....	6.52	53.25
10.......	2.62	4.26	150.....	10.69	63.94
14.......	4.76	9.02	200.....	4.92	68.86
20.......	6.43	15.45	Minus 200 (sand)	2.49	71.35
35.......	15.34	30.79	Minus 200 (slime).....	28.65	100.00
48.......	8.76	39.55			

TABLE 11. - Screen analysis of hydroseparator overflow

Mesh	Weight retained, percent		Mesh	Weight retained, percent	
	This mesh	Cumulative		This mesh	Cumulative
Plus 20.....	--	--	Plus 150..	3.11	7.34
35.....	0.13	0.13	200..	5.13	12.47
48.....	.52	.65	Minus 200 (sand)...	2.02	14.49
65.....	1.37	2.02	Minus 200 (slimes).	85.51	100.00
100.....	2.21	4.23			

Sand treatment

The sand plant comprises eight 45- by 12-foot redwood tanks arranged in two rows of four tanks each. The tanks have false bottoms of 1-inch square strips of wood spaced 1 inch apart and set on tapered joists to give a slope of 1 inch per foot toward the center. Coco matting is placed on the strips and covered with 12-ounce duck. Each tank has a normal capacity of 650 tons of dry sand. A main rubber-lined wooden launder placed above and between the two rows of tanks delivers the sand feed to lateral launders leading to the center of each tank. Feed and solutions are distributed by a two-arm revolving launder. A 6-inch overflow pipe is placed a few inches below the top of each tank to draw off slimy solution which is replaced with sand when filling. Tanks are drained through a 3-inch pipe set in the floor near the wall. The slimy overflow and drain pipes lead out between the rows of tanks where they connect to respective 6-inch pipe mains laid along the ground. Sands are sluiced through an 8-inch-diameter, central discharge opening which has a gate that seats on the under side of the discharge flange. A shaft connected to the gate and guided by a spider in the flange passes through the sand to supports above the tank. The upper end of the shaft has a screw thread and handwheel which opens and closes the gate. A 4-inch pipe line is laid under the center of each row of tanks to draw away the sluiced-out sands. High-pressure water and barren solution are tapped into these lines to clear any "plug-ups."

The tanks are set high enough for the pipe lines to carry solutions and sand residues to the pumps at the lower end of the plant.

Tanks are charged in rotation, and usually one is filled every 24 hours.

First they are filled with intermediate solution, which is replaced by the sands as the tank is charged. The overflowing solution, which is quite slimy, is pumped back to the classifiers and primary thickeners. When a tank is filled with sand any remaining solution is drained out and a head sample taken with a ship's auger. About 25 pounds of cyanide cakes are then laid on top of the charge, which is flooded with full-strength intermediate solution continuously for about 2 1/2 days without any aeration period. Next weaker solution is added, followed by a water wash, and the sands are drained and bored for a tailings sample. The drain valve is then closed and fresh water added to make the sands mobile for sluicing, which is done by opening the discharge gate and allowing water and slime residues to flow onto the charge through the revolving launder and wash out the sands. The 4-inch sand-residue lines terminate at a suction box of the 6-inch Wilfley residue pump where the sands combine with the residues from the slime plant.

Pregnant solution drained from the charges of sand flows to a sump tank and is pumped to the clarification and precipitation unit.

A summary of the cycle of operation in the sand unit is as follows:

		Hours
(1)	Fill tank with solution........................	4 to 5
(2)	Fill tank with sands........................	24
(3)	Drain and sample........................	8
(4)	Treat with full-strength solution........................	60
(5)	Treat with low-strength solution........................	36 to 48
(6)	Wash with fresh water........................	16
(7)	Drain and sample........................	9 to 12
(8)	Saturate with water for sluicing........................	4
(9)	Sluice out with slimes and water........................	12
(10)	Interval of lost time........................	Variable

On a basis of one tank charged with sand every 24 hours, the time of actual treatment or contact with cyanide solution is about 5 1/2 days.

One man on each 8-hour shift has charge of classification and sand treatment. He is also the head operator of the entire plant during his shift.

Slime treatment

As stated before, the fine or colloidal slimes from the hydroseparator overflow are pumped to a 100-foot-diameter, center-pier thickener; where colloids from the 40-foot thickeners at the mill are also treated. The reservoir for the thickener mechanism was built of sand and slime residues which were pumped to a ravine above the plant and deposited as a dam. Colloidal slimes made the reservoir virtually impervious and in a short time filled in the irregularities and formed a basin about 120 feet in diameter at the top. The mechanism is supported by a 42-inch-diameter steel cylinder set on a concrete base and connected to the bank of the reservoir by a steel bridge and trestle. A circular-feed well surrounds the cylinder and rotates with the two rake arms. A 3-horsepower, geared-head motor drives the rakes at a tip speed of 16 feet per minute.

The clear solution overflows through a steel launder, hung from the side of the trestle, to a 6-inch pipe line which leads to a sump tank. The solution is then pumped to the classifiers for dilution, to the sand tanks for leaching, and finally to a 9,700-ton-capacity storage reservoir built of residues deposited in a ravine above the plant.

The thickened underflow, which contains 24 percent solids, is drawn off the bottom through a 4-inch pipe laid along the sloping bottom of the reservoir and connected to a 4-inch, duplex, Dorr-type diaphragm pump set on the dam.

From this point the pulp flows by gravity through a 4-inch pipe to the filter unit where tonnage measurements are taken and the pulp is distributed equally to the three filters.

Approximately 200 tons of colloidal slimes are treated per day in this thickener. Caustic-starch solution usually is added to aid settling.

The underflow from the hydroseparator is pumped to two primary 60- by 12-foot redwood-tank thickeners set in parallel. These thickeners have a hi-head Dorr-type mechanism and are driven at a rake-tip speed of 20 feet per minute by 2-horsepower, geared-head motors. Each thickener is equipped with an annular overflow launder.

As most of the colloidal slimes have been removed the settling rate in these thickeners is considerably faster than formerly.

The clear overflow solution is collected in a tank and pumped to the head of the sand-plant launder to flush the sands into the tanks; it is also used to leach the sands.

The underflows from each of the two thickeners, which average 50 percent solids, are combined in a junction box where fresh cyanide and lime are added and the pulp is diluted to 42 percent solids with barren solution. A 2-inch sand pump then delivers the pulp to the first of three agitators set in series. Tonnage is measured ahead of the agitators.

The agitators consist of 16- by 14-foot redwood tanks equipped with Dorr-type agitating mechanism belt-driven at 3 r. p. m. by a 3-horsepower motor through a jack-and-line shaft. Air under pressure of 9 to 10 pounds per square inch at the receiver is used in the air lifts.

Pulp is treated 5 to 6 hours in the agitators.

After leaving the third agitator, the pulp is diluted with more barren solution, and caustic-starch solution is added to aid settling in the two secondary washing thickeners to which it flows.

These thickeners have 60- by 10-foot redwood tanks set in parallel and are equipped with mechanism of the same type as the primary thickeners.

The clear overflow solution (low value) is pumped to storage tanks for use at the mill or for mining old impounded tailings.

The underflow, which has an average density of 50 percent solids, is either pumped to the filters for further treatment or pumped with the repulped filter cake to the suction box of the 6-inch Wilfley residue pump. When the capacity of the filters permits, the entire underflow is taken, but usually only 70 percent can be treated in the filters.

Normally 450 tons are treated per day through this section of the slime unit.

Slime washing is done with three 14- by 14-foot Oliver filters equipped with oscillating agitators to keep the coarser slimes in suspension. Each is driven by a 3-horsepower motor at a speed of 4 r. p. m. divided into a cycle to allow 1 1/2 minutes for caking, 1/2 minute for drying, 1 1/2 minutes for washing, and 1/2 minute for blowing and recovering. An average vacuum of 22 inches is maintained at the filter valve, and cake is blown off at a pressure of 5 to 7 pounds. Scraper blades are tipped with a rubber strip 3 inches wide and 1/8 inch thick, backed with 2-ply duck. This strip is held to the steel blade by 1-inch iron strips welded 4 inches apart so that the rubber can extend about an inch beyond the blade. Filters are covered with twill and wound with No. 12 gage wire spaced 1 3/4 inches apart. Cloths have an average life of 3 months. They are not given a hydrochloric-acid wash, as was done formerly.

All-iron Oliver spray nozzles are used for washing the cake with fresh water.

The washed cake is blown or scraped off the filter and drops into blade-type repulpers running at 75 r. p. m. where it is diluted with fresh water to a density of about 30 percent solids. This pulp is either pumped to the sand unit for sluicing out the sand tanks or delivered to the suction box of the 6-inch Wilfley residue pump where it combines with the sand-plant residues.

Filtrate is collected in a receiver and pumped to a surge tank, whence it flows by gravity to the head of the main sand-plant launder.

One man on each 8-hour shift looks after the slime treatment and the clarification and precipitation sections.

Residue disposal

All residues are combined in the suction box of a 6-inch Wilfley sand pump running at 1,050 r. p. m. and driven through a V-belt by a 60-horsepower motor. The pulp is pumped through 1,150 feet of 5-inch, 12-gage, slip-joint pipe to the suction box of a 4-inch Wilfley sand pump which in turn discharges it through 270 feet of 4-inch pipe to a 4-inch header pipe extending the full length of the top of the dam. The header pipe is connected in 75-foot sections by rubber hose and is supported on 2- by 4-inch posts which project about 10 feet above the sand when first set in. Spikes and wire are used to hold the pipe to the posts. When the face of the dam is built up to the top of the posts new ones are set about 15 feet back. At 15- to 20-foot intervals 1 1/4-inch holes are burned in the header pipe, and saddles with 1 1/4-inch short nipples are clamped on. Rubber-covered flexible hoses 1 1/2 inches in diameter and about 15 feet long are used to distribute the pulp along the face. Several are used at one time, and they are quickly moved from one saddle to another.

The density of the pulp ranges from 30 to 45 percent solids, depending on whether sands are being discharged from the tanks. The pulp must be dilute enough to drop the coarser material at the face of the dam and allow the slimes and solution to flow back to the low point where the solution collects in a pool. As the solution in the pool clears it is pumped back to the plant for further use.

The face of the dam is built up by setting 1- by 12-inch rough boards end to end and supporting them in front with 2- by 2-inch stakes. Some sand is shoveled against the back of the boards to hold them in place, and the hoses distribute the coarse material, building it up to the top of the boards. At each lift the boards are removed and set back about 15 inches, which gives the face a slope of about 35° after settling.

At present the face of the dam is 1,000 feet long and 115 feet high.

A small sump built in the ground at the toe of the dam collects some seepage solution which is pumped back to the plant.

The average life of casings, runners, and follower plates on the 6-inch residue pump is as follows:

	Days	Tons of dry solids
Rubber casings........................	91	118,000
Iron casings.........................	36	47,000
Runners..............................	12 1/2	16,000
Follower plates......................	12 1/2	16,000

Two men take care of the residue dam and are assisted by the repair crew when the header is moved.

Clarification and precipitation

As much solution as possible is passed through the sand charges before going to the clarification and precipitation unit. At present an average of 2,750 tons of solution is precipitated daily. Of this quantity, 2,250 tons passes through the sand tanks and the remaining 500 tons is drawn direct from storage in the 9,700-ton-capacity reservoir.

Pregnant solutions are first clarified by passage through two tanks of Butters-type leaves. One tank has twenty 81- by 52-inch leaves and the other thirty-four 82- by 72-inch leaves. The clarifying medium is 12-ounce burlap covered with Pequot sheeting. Hy-Flo supercel (diatomaceous earth) is added to the incoming solutions; it coats the leaves and aids filtration. On an average leaves are washed once every 6 days. Some cyanide is added to the incoming solutions.

A Crowe deaerating tower is connected to the leaves through manifolds, and a vacuum of 23 to 24 inches is maintained.

Solution is drawn from the Crowe tower by a submerged 4-inch horizontal centrifugal pump and forced through a precipitate-collecting system comprising four tanks of 60 bags each. The bags, which are 5 1/2 inches in diameter and 36 inches in length are tied to bosses that have internal threads and screw onto nipples in the manifolds. Each bag comprises an outer bag of 8-ounce duck and an inner bag of bleached muslin.

Zinc dust (Merrillite) is fed through a cone into the suction pipe of the pump by a Merrill belt feeder. A small quantity of solution, tapped from the pump discharge, is fed into the cone with the zinc dust.

Barren solutions assay a trace to about 1 cent in gold per ton of solution and flow by gravity from the bag tanks into a barren storage tank.

The pressure on the bags ranges from 1 to 4 pounds per square inch; the normal pressure is 1 to 3 pounds.

Precipitate treatment and refining

The 240 bags are cleaned up every 10 days, at which time the pressure has built up to approximately 4 pounds per square inch. In the meantime clean bags are tied to extra bosses to replace the old ones. When all new bags are screwed onto the manifolds, about 12 pounds of Hy-Flo supercel and 15 pounds of zinc dust are put into the cone and pumped into the bags to precoat the inside. Thereafter the zinc is fed uniformly by the belt. After precipitation begins the barren solution is circulated through the clarifier tanks for one-half hour before it is allowed to flow to the barren tank.

The precipitate in the bags is emptied into a lead-lined redwood tank equipped with a two-blade propeller for stirring. Bags are turned inside out, and the adhering precipitate is washed with water into the tank.

In order to change and empty the bags with as little delay as possible, the clean-up man requires the help of four men for about 4 hours.

Commercial sulfuric acid is allowed to run gradually into the tank, which already contains some water and the precipitate. Enough acid is added to dissolve the zinc and still maintain a certain dilution. The pulp is agitated by the propeller, and when chemical action stops the zinc sulfate solution is decanted by siphoning. Normally, one or two acid treatments and 100 to 150 pounds of acid are required for each clean-up. After decantation the precipitate is washed three or four times with hot water, being stirred and decanted each time.

The precipitate is finally washed into a redwood-tank filter below, which is connected to the main vacuum system. Siphoned solutions also are piped to the filtering tank. This tank has a wooden grate near the top covered with canvas upon which filter paper and unbleached muslin are laid. About 3/16 inch of Hy-Flo is sprinkled over the muslin before filtering begins.

When filtered and partly dried the precipitate is taken up with the muslin and filter paper, put into pans, and dried in an electric drier in the refinery. After being in the drier about 36 hours, it is broken up and fluxed.

The precipitate is refined in a No. 125 clay-lined graphite crucible which is placed in a circular firebrick furnace heated by a medium-pressure burner using stove oil. The crucible is handled by heavy tongs attached to a windlass on a revolving boom.

Normally 100 pounds of precipitate is obtained from each 10-day clean-up A typical flux for 100 pounds of dry precipitate is as follows:

	Pounds
Soda ash..................	15
Borax glass...............	25 to 35
Silica[1].................	35
Niter.....................	10 to 20
Manganese dioxide.........	3
Calcium fluorspar.........	1

[1] Deduct silica (Hy-Flo) added as a coating in the precipitation bags and on the precipitate filter from the 35 pounds

The bar is poured by tilting the crucible on a steel rack so that the contents will flow into a rectangular mold. Slag overflows into a container which is chilled with water.

Bars have a fineness of 740 in gold and 210 in silver and, after being sampled by drilling, are shipped to the mint. After each melt the slag is crushed and sampled, and the accumulated lots are shipped to the smelter twice a year. The slag has an average value of 85 cents per pound, avoirdupois.

With the exception of the four extra men who help for 4 hours on the first day of the clean-up, the clean-up man does all routine work pertaining to precipitation and refining.

Usually 3 days are required to clean-up and melt.

Reagents and supply consumption

Water. - Goler water is used in the following places in the plant: (1) Final wash in sand charges; (2) sluicing out sand charges in conjunction with slimes; (3) Oliver-filter sprays; (4) repulping filter cake; and (5) washing clarifier leaves.

Cyanide. - White sodium cyanide (91 percent NaCN) is added at the main mining-pump sump, at the leaching tanks, to the primary-thickener underflow, and at the clarifying tanks. A strength equivalent to 0.012 to 0.014 percent KCN is maintained. An average of 2,750 tons of barren solution is released from the precipitating plant daily. Of this quantity, approximately 1,800 tons are used as wash in the secondary thickeners, 700 tons are added to the 100-foot center-pier thickener, 235 tons are used for dilution in the agitators, and 15 tons are used for sluicing the sand-tank discharges.

Lime. - Lime is added at the main mining-pump sump and to the primary-thickener underflow. A strength of 0.003 to 0.006 percent CaO is maintained throughout the plant. Lime is used principally to settle slimes, as there are virtually no cyanicides in the ore.

Zinc dust. - Merrillite zinc dust is used. It is fed at the rate of 0.05 pound per ton of solution precipitated. No sodium plumbite is added in the clarifier tanks.

Supply consumption. - Consumption of major supplies in the operations is as follows:

Cyanide (91 percent pure NaCN), pound per ton of ore treated..	0.205
Lime, pounds per ton of ore treated..........................	4.265
Zinc dust, pound per ton of ore treated......................	.114
Zinc dust, pound per ton of solution precipitated............	.052
Zinc dust, pounds per ounce of gold bullion..................	3.719
Hy-Flo supercel, pound per ton of ore treated................	.105
Hy-Flo supercel, pound per ton of solution precipitated......	.046
Power, kilowatt-hours per ton of ore treated.................	6.864
Water, tons per ton ore treated[1]...........................	1.137

[1] Includes water used for milling and for mining old tailings. No solutions are discarded on account of fouling. Water losses are explained by seepage and spillage in hydraulic-mining operations, evaporation in the tanks and residue dam, and absorption by the residue dam.

Plant control

Tonnages. - Sand volumes are determined by measuring the level of the sand below the top of the tank staves. Sand tonnages are calculated by dividing the volume by 23 (the number of cubic feet per dry ton of sand).

Slime tonnages are calculated using as factors the time required to fill a container to a certain depth and the specific gravity of a sample. Determinations are made every 2 hours and averaged for the 24-hour day.

Tonnages of solution precipitated are determined by measuring the pressure head at an orifice discharge from a tank.

Sampling. - Sand heads and tailings are sampled by a ship's auger; the holes are spaced on a spiral.

Slime-plant head and tailings samples are taken every 2 hours by hand (automatic samplers are being installed). Determinations are made of the total residue, undissolved and dissolved value losses of the general slime tailings, and undissolved and dissolved value losses of the secondary-thickener underflows and the filter cake. A little 5-percent potassium permanganate solution is placed in the container for dissolved- and undissolved-value-loss samples to prevent dissolution by any cyanide solution while the sample is being taken.

Samples of the precipitation-bag heads and tailings are taken by a drip sampler. Other solution samples usually are grab samples taken by hand.

Densities. - Density samples are taken of classifier and hydroseparator overflows, hydroseparator and primary- and secondary-thickener underflows, and third-agitator discharge. Specific gravity of the general plant feed is 2.78.

Screen analysis. - Screen analyses are made from composites of the daily samples of the sand- and slime-plant heads.

Titrations. - Titrations for free cyanide and protective alkalinity are made daily on samples taken from the primary- and secondary-thickener overflows, third-agitator discharge, and clarifiers.

Assaying. - The cyanide plant has a well-equipped assay office and testing room where the assaying and testing are done for the mine, mill, and cyanide plant. Four melts each of 2 assay tons of pulp are made on the mill heads and the sand heads and tailings, and three melts of the same amount of pulp each are made on the slime heads and tailings. Normally 17 1/2 assay tons of solution are used in assaying solutions by the Chiddy method.

Reports. - Detailed daily and monthly reports are made on the ore tonnages treated, solution tonnages precipitated, assays, densities, titrations, reagents used, bag pressure, lost time, and other data of interest.

Labor

The operating and repair crew for three 8-hour shifts comprises the following men:

Sluicing boss	1
Hosemen	4
Sand-treatment operators	3
Slime-treatment operators	3
Residue-disposal men	2
Clean-up and utility man	1
Mechanic-electrician (repairs and maintenance)	1
Welder (repairs and maintenance)	1
Machinist (repairs and maintenance)	1/2
Roustabouts	2
Sampler	1
Foreman	1
Superintendent	1

SAFETY METHODS AND FIRST-AID ORGANIZATION

A physical examination is required of all men before they are employed, and x-ray chest plates are made of those who are to work under dusty conditions. Examinations are made by the local doctor at Randsburg. All employees are given first-aid instruction every year by a safety engineer of the Federal Bureau of Mines. A safety committee of five employees meets once a month and makes periodic inspections of the plants.

Complete tabulated records are made of all injuries, their causes, and the cost to the insurance carrier.

FIRE PROTECTION

Water tanks are placed on the hill above the plants for fire protection.

Fire hydrants with 1 1/2-inch hoses, 50-gallon barrels with buckets, and soda-acid and pyrene fire extinguishers are distributed at appropriate locations.

COST OF MAJOR SUPPLIES

The following is the cost of major supplies, delivered at Randsburg:

Cyanide, 5-ton lots	per pound	$0.1328
Lime, 10-ton lots	do	.0052
Zinc dust, 5-ton lots	do	.0939
Hy-Flo, 5-ton lots	do	.0253
Balls (Mn), 4-inch, 10-ton lots	do	.0341
Ball-mill liners (Mn), complete set	do	.1293
Sand-pump casing (rubber-lined), 6-inch	each	262.15
Sand-pump casing (iron), 6-inch	do	76.50
Sand-pump runner, 6-inch	do	32.70
Sand-pump follower plates, 6-inch	do	15.13

SUMMARY OF OPERATING COSTS

All costs are figured on the basis of dry tons of material.

Table 12 summarizes the cost of mining per ton mined, including contractors overhead and profit.

Tables 13, 14, and 15 summarize the direct operating costs of crushing, milling, and cyanidation.

Table 16 summarizes the combined costs of mining, crushing, milling, and cyanidation per ton milled.

Randsburg-office overhead, property insurance and taxes, and assessment work on mining claims are not included in any of these costs.

TABLE 12. - Mining costs, per ton of rock mined

From November 1938 to January 1939, inclusive, 249,747 tons of ore and 6,790 tons of waste were mined.

Classification	Costs
Drilling and blasting..............................	$0.0510
Loading..	.0397
Hauling..	.0547
Supervision, overhead and contractor's profit.	.0372
Total[1/].................................	0.1826

1/ Total cost per ton of ore mined, including waste handled, is $0.1876.

The foregoing drilling, loading, and hauling costs include rental charges for contractor's equipment used on the job.

TABLE 13. - Crushing costs, per ton of ore crushed

From November 1938 to January 1939, inclusive, 249,747 tons of ore were crushed.

Labor	Supervision	Power	Supplies	Compensation and insurance[1/]	Total
$0.0225	$0.0012	$0.0095	$0.0139	$0.0016	$0.0487

1/ Accident compensation and social-security insurance.

These costs include crushing, screening, and waste disposal. Of the 249,747 tons crushed, 31.4 percent was sent to the mill bin and 68.6 percent discarded as waste. Cost per ton milled is $0.1550.

TABLE 14. - Milling costs, per ton of ore milled

From November 1938 to January 1939, inclusive, 78,411 tons of ore were milled.

Classification	Labor	Superfision	Power	Water	Supplies	Compensation and insurance[1/]	Miscellaneous	Total
Milling..	$0.0293	$0.0037	$0.0251	$0.0379	$0.0412	$0.0026	--	$0.1398
Marketing bullion.	---	---	---	---	---	---	$0.0033	.0033
Assaying.	---	---	---	---	---	---	.0018	.0018
Total..	0.0293	0.0037	0.0251	0.0379	0.0412	0.0026	0.0051	0.1449

1/ Accident compensation and social-security insurance.

Of the water used for milling and cyanidation, one-half is arbitrarily charged to each department. Cyanide and lime consumption is charged to the cyanidation department.

TABLE 15. - Hydraulic mining and cyanidation costs, per ton of ore treated

From November 1938 to January 1939, inclusive, 78,700 tons of ore were treated.

Classification	Labor	Supervision	Power	Water	Supplies	Reagents[1]	Compensation and insurance[2]	Miscellaneous	Total
Mining.........	$0.0213	$0.0019	$0.0173	$0.0044	$0.0091	---	$0.0015	---	$0.0555
Treatment......	.0438	.0038	.0222	.0252	.0317	$0.0612	.0030	---	.1909
Residue disposal........	.0101	.0019	.0208	---	.0156	---	.0007	---	.0491
Marketing bullion..........	---	---	---	---	---	---	---	$0.0061	.0061
Assaying.......	---	---	---	---	---	---	---	.0076	.0076
Total.....	0.0752	0.0076	0.0603	0.0296	0.0564	0.0612	0.0052	0.0137	0.3092

1/ Reagents were cyanide, lime, and zinc dust.
2/ Accident compensation and social-security insurance.

TABLE 16. - Summary of costs, per ton of ore milled[1]

Classification	Costs
Mining..................................	$0.5975
Crushing................................	.1550
Milling.................................	.1449
Cyanidation[2].........................	.2537
Total............................	1.1511

1/ Based on 78,420 tons of ore.
2/ Does not include hydraulic mining.

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Geology of the Yellow Aster Mine, California

Remarkable Ore Deposits in Granite at Randsburg, Kern County, That Have Produced about $6,000,000 in Gold. Discovered in 1895

BY WILLIAM H. STORMS*

Randsburg, Cal.

The consolidation of mining claims owned by the Yellow Aster Mining Company, of Los Angeles, Cal., constitutes one of the most important gold-mining properties of the West. The mines are situated in the Mojave desert, Kern county, California, about one-half mile south of Randsburg.

The discovery of gold in the district was made in 1895 by C. A. Burcham, Fred M. Mooers and John Singleton, who came into the region from the dry placer camp, at Goler, where they had met with indifferent success. The first discovery of gold was made in the dry sand of a gulch which cut the outcrop of the gold-bearing veins of the Yellow Aster hill. The dirt was concentrated in a dry-blowing machine. Soon after this the veins were discovered and development proceeded as rapidly as the isolated situation of the property would permit.

The output of high-grade ore from the various mines about Randsburg early attracted much attention. For the most part, the veins proved to be rather short and narrow fissures in micaceous and hornblendic schists, and the first few years of active mining rapidly depleted the smaller veins on the ridge, north of town, known as Butte hill. The group of claims to the south, on Yellow Aster hill, developed year by year until the consolidated claims, known as the Yellow Aster mine, became an important and productive gold mine.

A Region of Horizontal Schists

For several miles about the village of Randsburg, the rocks consist chiefly of a series of granitoid schists, comprising both micaceous and hornblendic varieties. In some parts of the region typical amphibolite schist occurs. The hornblende schists are often fissile at and near the surface, having distinct schistose structure. These typical schists are found interbedded with layers of granite rock 4 to 5 ft. in thickness, possessing little or no trace of schistosity. These bedded schists extend in a northeast-southwest direction, a southwestern extension of the Slate range, although wholly detached from the latter range of mountains.

Origin of the Flat Schists

That horizontal schists extending over any considerable area are unusual, will be readily admitted. Several theories to account for those about Randsburg have been advanced by prominent geologists. Among these theories is the possibility that the rocks comprising the schists were, at one time, buried so deeply that the tremendous pressure of the superincumbent mass induced a condition of schistosity.

Another theory, and a more probable one is that schistose conditions have been produced by the proximity of a huge batholith (probably granite) that may underlie the region. Still another theory conceives that the schistose and semi-schistose structure is due to the rocks having originally been deposited as a series of flows. This latter supposition seems improbable, as none of the rocks are of volcanic type, but of granitic character.

The only remaining tenable theory is that the schistose structure may, to some extent, represent original sedimentation; and, that these, now crystalline schists, were originally horizontal sedimentary beds, which have been brought to an extreme state of alteration by pressure, still preserving traces of their original horizontality.

Personally I favor the theory of the batholith, which, in my opinion, lies at no very great depth beneath the present surface. To some extent the topography of these hills is influenced by the character of the rocks, the slopes of some of the hills conforming to the dip of the schists.

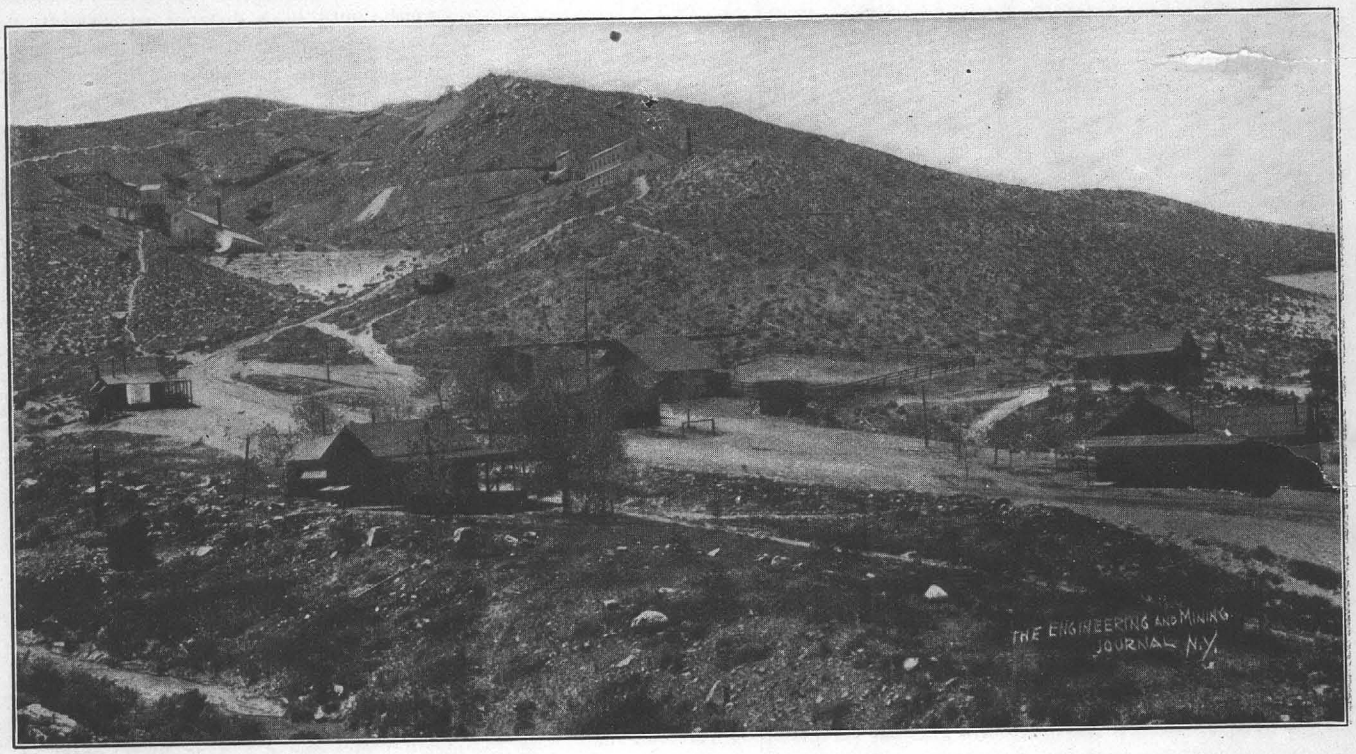

FIG. 1. YELLOW ASTER HILL FROM THE NORTH; 100-STAMP MILL AT THE RIGHT; 30-STAMP MILL AT THE LEFT

*Mining engineer, Los Angeles, Cal.

The Surrounding Region

Northeast of Randsburg, 3 or 4 miles, is an eminence known as Red mountain, which consists entirely of volcanic rocks—rhyolite, tufa and tuffaceous-breccia, but so far as known, no ore deposits of any description exist in this volcanic area. Less than a mile east of Randsburg, are large intrusions of granite, in which are included several irregular masses of limestone. These were probably torn from an older formation lying beneath, but which is unknown elsewhere in the vicinity of Randsburg. There are limited occurrences of quartzite and dense jasperoid rocks nearer town, particularly on the north slope of the ridge known as Butte hill, which, with the limestones, indicate sedimentary origin. Four miles east of Randsburg, dikes of pegmatite cut the schists. These dikes have, among other constituents, ores of tungsten—notably scheelite, which has been profitably mined in the Atolia mine.

West of the Randsburg district is a desert valley about six miles in width, on the farther side of which is Goler, the noted dry-placer camp. Several deep wells sunk here, on what is presumed to be an ancient gravel channel, furnish the principal water supply for the Yellow Aster mine; a further supply being obtained from other deep wells sunk six miles northeast of Randsburg. Goler lies at the mouth of a cañon cutting a north and south range of rugged hills, which consist mainly of schist. These schists are of a different type from those at Randsburg and vicinity, being evidently the result of the alteration of various greenstones. Igneous intrusions of huge dimensions are not uncommon in these hills.

South of Randsburg schistose rocks extend for many miles, and in them are known to be numerous gold-bearing veins, some of which have been profitably exploited.

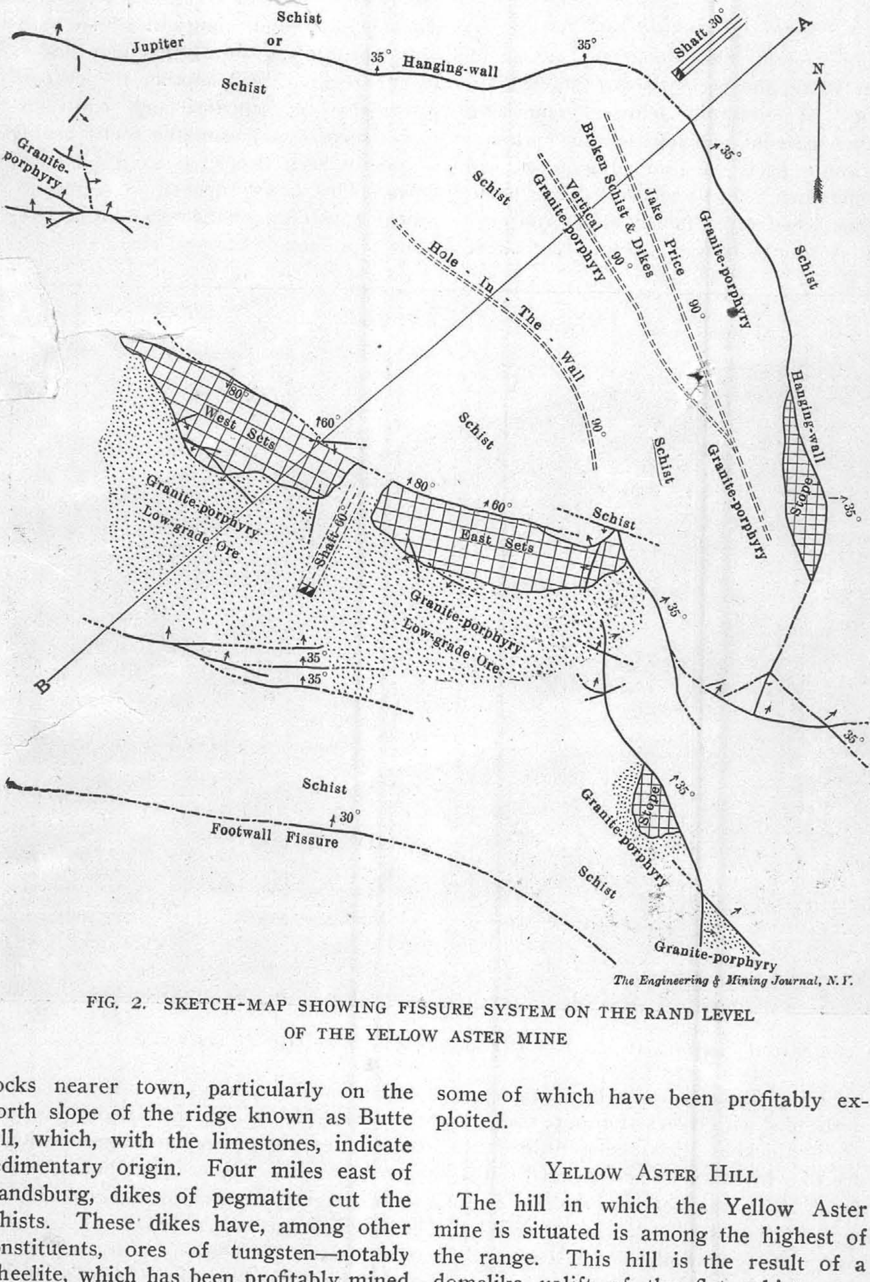

FIG. 2. SKETCH-MAP SHOWING FISSURE SYSTEM ON THE RAND LEVEL OF THE YELLOW ASTER MINE

Yellow Aster Hill

The hill in which the Yellow Aster mine is situated is among the highest of the range. This hill is the result of a domelike uplift of the flat schists previously described. The schistose rocks, originally nearly horizontal, have been intruded by numerous dikes, large and small, and these intrusions have caused the older rocks to be lifted into a dome, the effect extending for thousands of feet in all directions from the center of disturbance. Not only have the intrusive masses lifted the schists from their flat position into a dome, but they have thrown them into a series of sharp folds, some of which appear near the northern base of the hill, east of Main street, and others in the cañon west and southwest of Randsburg. These folds afford additional relief from the tremendous compressive stress resulting from the repeated intrusion by a series of remarkable fissures occurring in Yellow Aster hill, along all of which a greater or less amount of movement has taken place.

The accompanying sketch map, Fig. 2, of the Rand level, which is at present the main working level of the mine, shows the distribution of the most of these fissures, and the relations to one another, and also, to some extent, to the various orebodies.

The footwall fissure—seemingly the only one that is absolutely continuous, so far as known—at the west end of the property in Olympus claim, has a strike a few degrees north of west, dipping about 30 deg. to the north. It runs easterly, gradually turning in a broad, sweeping curve to the south, until in a distance of 3000 ft. it strikes directly south, dipping 35 deg. east. This fissure is indicated on the map, and is found at several points in the mine workings and in surface cuts. It is generally accompanied by one or two feet of tough, putty-like gouge. The sweeping curve of its strike from nearly east-west to due south is somewhat unusual.

There are other fissures in the southern area of the property, the more important of which are indicated on the map, but these fissures seem to be a series of fractures following a common direction and overlapping each other, rather than continuous fractures.

What is commonly called the hanging-wall fissure is not such in reality, for orebodies occur both above and below. This fissure, known as the Jupiter, is also indicated on the map. In the west end of the mine the strike is nearly east-west, almost parallel with the footwall fissure, dipping north 20 to 25 deg., but becoming somewhat steeper toward the east, and with depth. Its dip is 35 deg. on the Rand level. This fissure and the footwall fissure are not over 30 ft. apart at the surface, in the west end of the mine. They separate rapidly eastward.

Proceeding along the Jupiter fissure, there may be observed numerous subsidiary fissures, which are part of the system of fractures due to the stress that produced the main fissures. Going eastward the Jupiter fissure also swings around to the southward, finally taking a more abrupt bend than the footwall fissure, eventually having a strike somewhat west of south, and uniting with some of the more important fissures in the interior portion of the area. The dip of this fissure is almost uniformly 35 deg. on the Rand level, but is somewhat less both above and below. At 400 to 500 ft., on the incline, below the Rand level, the dip is 20 deg. In this respect it is similar to some important fissures elsewhere, notably the footwall fissure of the Comstock lode.[1]

Intrusive Rocks

The several dikes which cut Yellow Aster hill vary somewhat in strike, but all approach an east-west direction. About midway between the mine and the town a dike of greenstone is seen intersected by one of quartz-porphyry, at an angle of about 20 deg. No ore-deposits are known to accompany either of these dikes. In and about the mine itself the dikes dip from vertical to 60 deg., except where locally disturbed by movement subsequent to their intrusion.

These dikes comprise a variety of rocks, nearly all of which are acidic. They probably represent a differentiation of the granitic batholith, presumed to underlie the region. One of the most prominent dikes of the vicinity extends from the east side of the dome easterly a distance to have any direct relation to the ore deposits below. Near this large outcrop of syenite is found a dark-greenish, much altered boss, which was probably originally diabase.

There are other large granitic dikes occurring at various places within the limits of the domal uplift. These rocks also, along crushed zones, are gold-bearing, being as a class somewhat better than the felsites, but not sufficiently rich to form ore of commercial value.

The Granite Porphyry

The main ore-bearing rock is granite porphyry. Ore occurs in it principally in three forms: As distinct vein-like masses usually adjacent to a fracture or fissure; as impregnations of the rock mass itself, and, more rarely, in cracks or seams traversing the granite. The first type has resulted from a more or less complete silicification of the granite, forming a dense quartz rock resembling dirty, yel- as well as to the complex geological conditions in the region inclosed within the triangular section between the footwall and hanging-walls. Large masses of schist are found torn from their original beds, which are twisted and shattered between great irregular dikes of igneous rock. Not only this, but large sections of older dike rocks have also been broken from the earlier intrusions, and these are now found mingled with the schist in a chaotic mass between the later dikes. Moreover, the line of demarkation between schist and igneous rock is not always clearly defined, even where both rocks are exposed to daylight inspection in the great open cuts. They merge, one into the other, by a gradual transition.

All of the great orebodies of the mine have resulted from the infiltration of gold-bearing solutions into masses of crushed granite-porphyry. This is also true of many of the smaller, and sometimes isolated orebodies, but all of the granite-

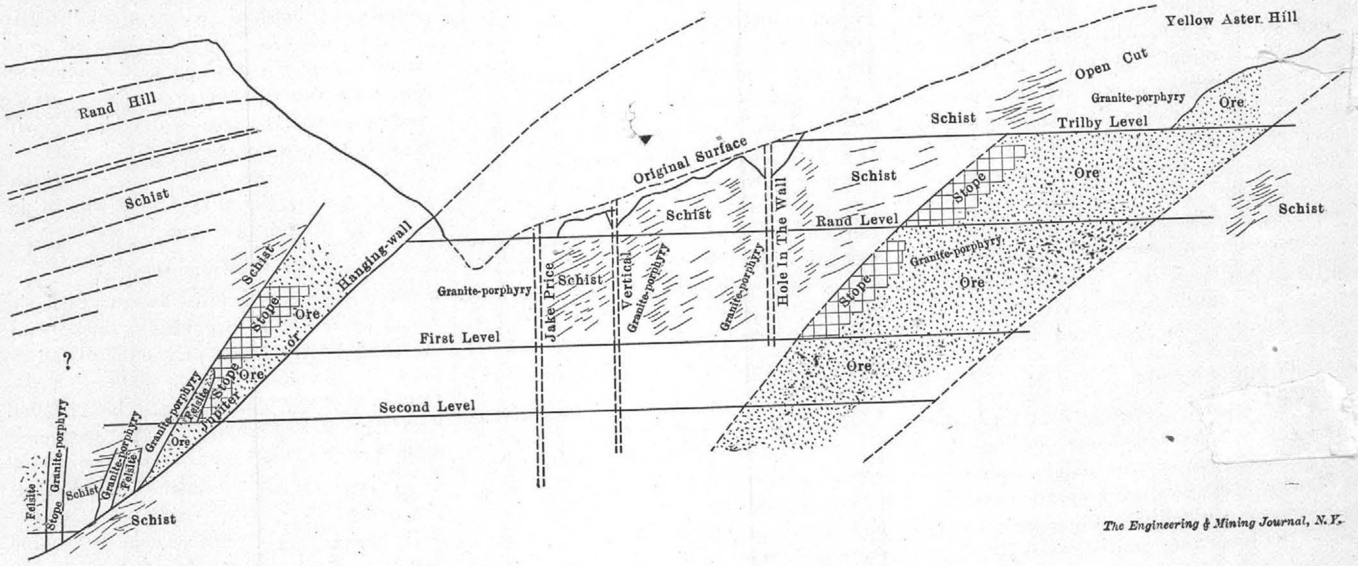

FIG. 3. SECTION THROUGH YELLOW ASTER MINE

of two miles. It varies from 4 to 40 ft. in width and is nearly vertical. At the east end it spreads into a great mass of quartz-felsite 100 ft. or more in diameter. Near the Yellow Aster hill this dike is a typical quartz porphyry. At one place only in the mine workings was a dike found which appeared to be identical with this particular intrusion, but as it is not known to be in any manner associated with ore deposition, little exploration has taken place in its vicinity.

Among the other intrusions are several large dikes of felsite which are exposed both on the surface and in the mine workings. Some of these felsite intrusions are more than 100 ft. wide and several hundred feet in length. Where crushed and altered the felsite is usually gold-bearing, but always, as far as known, low grade.

Syenite occurs on a hill north of the summit of the dome, but it is not known

[1]George F. Becker, *Geology of the Comstock Lode*, Monograph No. III, U. S. Geol. Surv.

lowish-gray hornstone. The second is merely the result of impregnation of masses of crushed granite porphyry by gold-bearing solutions. This rock presents the appearance of a partly decomposed, medium-textured granite, slightly iron-stained, with occasional evidence of the deposition of secondary quartz. The third type is merely a series of irregular cracks, following a common direction through the granite-porphyry, and in these cracks or seams coarse gold is found.

The granite-porphyry occurs, like most of the other intrusions here, in great dikes having a strike approximately north 85 deg. west, dipping north. Several of these granite-porphyry dikes may be seen on the surface, cutting the nearly flat schists. They can be readily traced to the Jupiter fissure; but to indentify any particular dike below that plane of movement is most difficult, if not impossible, owing to the lack of definite knowledge of the amount of displacement that has occurred, porphyry is not gold-bearing. Nor is it always, or even often, possible to distinguish, by its physical appearance, the pay rock from that which is poor. One of the chief duties of the shift bosses is to take samples of new faces of rock as fast as they are exposed. The ore occurs in zones of greatly varying width, from 2 or 3 ft. up to masses more than 200 ft. thick, and 40 to over 600 ft. long.

Striations along the Jupiter fissure indicate that the movement has not been uniform; they show that some blocks have moved horizontally, whereas others have moved in various directions, and in some instances the same rock has been shifted first one way and then another.

The largest stope in the mine has been opened on the Rand level. It is more than 600 ft. long, over 100 ft. wide in places, and nearly 100 ft. high. These dimensions do not indicate the total extent of this great mass of ore; but at the time it was mined they practically outlined the pay ore. Great stopes have been opened on the extension of this orebody, both above and below the Rand level.

About 100 ft. west from the west end of the stope just referred to, begins another large orebody, but belonging to an entirely different series of oreshoots. Huge stopes have been opened on the Rand level, also above and below it. In all of these stopes the limitation of stoping has generally been determined by commercial considerations, excepting where the ore limit has been sharply defined by a fissure or abrupt change in the character of the rock, clearly discernible by the eye. The east stope on the Rand level, for instance, is sharply defined on the hanging side by a wall, although this fissure is one showing little or no movement. Over most of the area covered by the exposure of this wall there is little or no gouge, while on the footwall side the gold contents stops with no apparent change in the appearance of the rock. Throughout the mine there still remain large blocks of ore in the immediate vicinity of the principal stopes which will admit of little or no profit if mined by stoping and square-setting, timber costing over $40 per M as delivered, and over $50 per M set up in the mine. However, there is little doubt that much of this ore could be profitably mined if the open-cut system were applied to it. Development is resulting in the discovery of new orebodies in various parts of the mine, the diamond drill having been found particularly useful in searching for new masses of ore.

Distribution of the Ore

To the casual observer the distribution of ore in the Yellow Aster mine appears to be extremely erratic and haphazard. Yet when the general map of the mine workings is studied it is found that nearly all of the most important orebodies belong to one of several systems or groups, each following an independent trend. Then there are orebodies apparently isolated, but which may later be found to belong to separate systems, the relations of which will be better understood by more extensive development, although there are already over 16 miles of underground workings in the mine.

Of the two important oreshoots mentioned, the most extensively mined is that lying to the east of a line striking about north 25 deg. east and approximately through the two shafts indicated on the sketch map, Fig. 2. This oreshoot, or more properly, series or succession of oreshoots, outcrops at the surface on the south slope of Yellow Aster hill near the west end of the property, where it forms a large body of low-grade ore. In its downward dip and trend it lies east of the line mentioned, its trend being about north 60 deg. east. Much of the ore mined from this succession of orebodies was very profitable.

The other large orebody appears, or did appear, at the surface near that just described, but the upper portion has since been removed in the open cut or "glory hole." This orebody has a trend about north 15 deg. east, so that with depth these two great ore shoots became widely separated. In many respects these two orebodies are similar, particularly as to their size, form and manner of occurrence.

At least three other separate series of orebodies have been developed in the vicinity of the Jupiter wall. These are known as the Midway, the Buffalo and the Homestake. Most of the ore in the Midway series lies above the Jupiter wall, though some of it occurs underneath. At the Buffalo the greater part of the ore thus far developed lies underneath the fissure. In the Homestake-Hunt series is the Christmas stope above the Jupiter wall, but near the surface, on the Rand level, a rich shoot was found under and in close proximity to the wall. This shoot is in direct line with the Homestake and Christmas and may very properly be considered as belonging to the same series. It should be understood that none of the orebodies mentioned as connected with the Jupiter wall have any relation to, or connection with, the large masses of ore in the central portion of the mine.

In the southeast part of the mine and entirely separate from the orebodies already mentioned, are several other shoots, some of which have been very profitable. In this part of the mine, and also toward the west end are extensive bodies of low-grade ore which undoubtedly will be mined at a profit at some time in the future.

An interesting fact in connection with this mine is that if a cross-section were taken every 100 ft., no two of the sections would be alike, even those at adjoining stations. Most remarkable changes take place often within a few feet.

In the northeast part of the mine, and about 300 ft. below the surface is a stope known as the Hobo. On the 200 level the ore is about 4 ft. in width. The hanging side is defined by a good wall; the footwall is irregular. The hanging-wall is nearly vertical. The footwall slopes back with varying angle, until, 60 ft. above the level, the stope is about 20 ft. wide. Just above this the schist swings over the granite-porphyry from the hanging-wall, and the ore following it, rapidly thins out and becomes a mere seam a few feet above the next level.

Vertical Zones of Impregnation

In addition to the large ore-deposits already referred to, there also occurs, in the granite-porphyry, a series of nearly vertical zones of gold-bearing rock which have no walls. These are indicated on the map, Fig. 2, and section, Fig. 3, by the name—the Price, Vertical and Hole-in-the-Wall. These peculiar occurrences are in no sense veins or orebodies of definite form, being merely a series of reticulated cracks or seams occurring along a common direction, such as may be observed in granite. A slight brownish stain of iron oxide is about the only indication of mineralization noticeable in this rock, and much of it shows little sign of oxidation. Occasionally a little secondary quartz and sometimes pyrite is present. The granite in these zones presents no evidence of movement or shearing, but coarse gold occurs in these zones of fracture and impregnation, and they have proved very profitable, the Price and Vertical being particularly high grade. These veins, as they are called, have been stoped in the ordinary manner, from 4 to 12 ft. wide, and square sets placed in the wider portions.

Ore in the Schist

Ore rarely occurs in the schist. In almost every instance where the gold-bearing granite-porphyry is interrupted or cut off by schist the gold disappears. A remarkable exception occurred near the summit of the dome where a small fissure cut the schist. This vein carried much limonite and was rich in gold. The rock was soft and decomposed at the surface, and was run through a dry-blower with most satisfactory results. This place is known by the peculiar name "Instinct," the discoverer claiming that it was by instinct he found the orebody.

A Problem in Mining

The orebodies do not always reach the surface, some of the larger ones being overlaid by the horizontal schist from 20 to 80 ft. in thickness.

Some of the older stopes have caved and the ore remaining can be recovered only by stripping the overburden which is, generally speaking, on the south side of the cut.

By doing this much ore would become available at a cost which will admit of a profit on the operation. Late accounts from the mine state that stripping operations have actually been commenced, and the great cut will consequently be broadened and later deepened.

China is showing much interest in mining affairs, according to the *Far Eastern Review*. The government at Peking has directed the governor of Chinese Turkestan to investigate and report on the copper deposits of Kucha and Paichang with a view to developing them along modern lines.

The output of vanadium in the United States, according to statistics of THE MINERAL INDUSTRY, increased last year. In 1908 the production was 64,800 lb. ($162,000), while in 1907 it amounted to 41,000 lb. ($102,500).

The production of tin in Queensland during 1908 (*Queensland Gov. Min. Journ.*, April 15, 1909) was 4825 tons, valued at £342,191. This is a decrease of 315 tons, as compared with the production of 1907.