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DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

MINING AND MILLING METHODS AT THE PILGRIM MINE, CHLORIDE, ARIZ.

BY

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By Earl F. Hastings[2/]

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INTRODUCTION

The Pilgrim mine, operated by the Pioneer Gold Mining Co., is 9-1/2 miles west of Chloride in the Pilgrim mining district on the eastern slope of the Black Mountains, or River Range, Mohave County, Ariz. In March 1937, 100 tons of gold ore per day were being mined and milled at the property by a total force of 54 men.

Climatic conditions allow year-round operations. Due to present activity in the county there is an ample supply of labor for the requirements of the mine.

The Pilgrim is the only producing property within a radius of 6 miles; recently, however, the South Pilgrim Mining Co. has launched a development campaign on claims that adjoin the Pioneer Co. property to the south.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6945."

2/ Manager, Pioneer Gold Mining Co., and one of the consulting engineers, U. S. Bureau of Mines.

Production from the Pilgrim mine was 26,000 dry tons in 1935 and 29,304 dry tons in 1936. Working time in 1936 was 19,381 man-shifts in 350 days.

ACKNOWLEDGMENTS

Acknowledgment is made to M. C. Richardson, mine superintendent, T. N. Slaughter, mill superintendent, and B. E. Charles, geologist, for their assistance in gathering special information requested by the writer.

HISTORY

The Pilgrim mine was located in 1903 by Dempsey and O'Dea; by 1907 the main shaft had been sunk to a depth of 360 feet, a small amount of drifting and crosscutting done, and a few tons of high-grade ore shipped. From 1907 to 1928 the mine was idle except for a little spasmodic development work. In 1928 M. C. Richardson, the present mine superintendent, took a lease and bond on the property and mined and milled a small tonnage of high grade ore. In 1933 the Pioneer Gold Mining Co., controlled by B. A. Laselle of Los Angeles, took a lease and bond on the property and developed it until 1934, when a flotation mill was built. Until November 1935 operations were irregular, owing to mechanical difficulties in the mill and power house. The installation of a new Diesel plant and complete revision of the mill flow-sheet solved both the mechanical and metallurgical troubles.

GEOLOGY

The River Range extends in a general north and south direction along the Colorado River for about 100 miles. Scattered mineralized areas containing commercial ore occur along its whole length, the most notable being the Oatman, Katherine, Pilgrim, and Klondyke districts.

Rocks in the vicinity of the Pilgrim mine consist of uptilted blocks of closely related volcanic flows and tuffs that dip toward the east at angles ranging from 10 to 20°. This series of flows and ash is intruded at irregular intervals by rhyolitic and basic dikes.

The veins so far developed follow a well-defined shear zone that runs along the center lines of the Mayflower, Pilgrim, and Plymouthrock claims; the strike is north 30 west and the dip 30° to the west. The hanging wall of the shear zone is a well-defined and persistent streak of red gouge that ranges in thickness from a few inches to 4 feet; beyond this efficient dam no mineralization has been found other than drag within the gouge itself. Soft rhyolitic flows and latite predominate behind the red wall. (Fig. 1.)

The foot wall of the shear zone is not well-defined. Roughly, it is andesite but has been influenced by a series of north-striking, steep-dipping, minor shear zones that cut through the foot wall and, in some places, filter on through to the hanging wall. These minor faults, at the intersection on acute angles with the major fault, show noticeable enrichment along the foot wall; and where filtration to the hanging wall occurs, a large, low-grade ore body is found from wall to wall. The foot wall is complicated further by rhyolite intrusions.

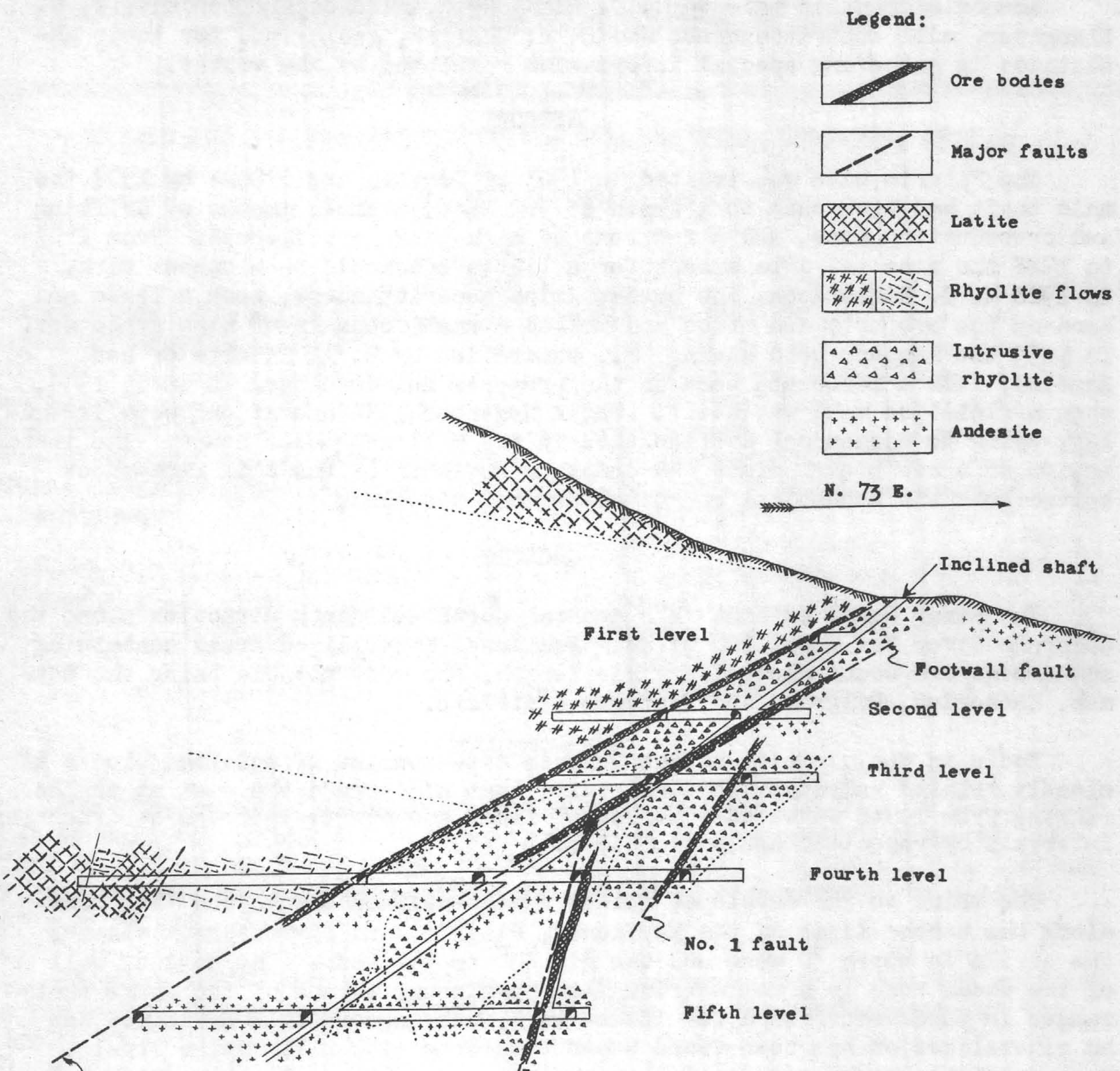

Figure 1.- Typical vertical cross-section at main shaft, Pilgrim mine, showing relation of ore bodies to structure. Scale, 1 inch equals 100 feet.

The north-striking, steeper-dipping minor shear zones are credited with mineralizing the main shear zone. Five of these feeders, as they are called, have been traced on the surface, and two developed by drifting underground. They are a series of rich quartz stringers with barren andesite or rhyolite between, making a width in all of 4 to 22 feet. The quartz stringers range from less than 1 inch to 24 inches in width and contain up to 1,000 ounces of gold per ton. Ore broken from stoping these stringers over an 8-foot width has averaged 1.75 ounces gold.

Vein material between walls is a breccia and fragments of the underlying tuffs. The distance between walls varies with depth but averages about 60 feet; commercial ore bodies 3 to 18 feet in width are found along each wall. Apparently, the wall rocks have little influence on ore deposition other than in their structural capacities as host rocks. At the present mining horizon commercial deposits are found more in the rhyolites than in the andesites, owing to the capacity of the latter to support open fissures or solution channels. Andesites in the Oatman district are much harder than those so far encountered here and are structurally suitable as host rocks for commercial ore bodies. It is expected that with more depth this will be true at the Pilgrim, also, as the andesites in both localities are similar in other respects. At the Pilgrim, however, about 1,800 feet of flows and tuffs must be penetrated before the geological horizon of the Oatman district can be reached.

Gold and silver are the only commercial metals found in the district, and occur in the approximate ratio of 1 ounce of gold to 1 ounce of silver. The gold is free; some of it is slightly tarnished. Small quantities of galena, pyrite, chalcopyrite, tetrahedrite, sphalerite, and marmatite can be identified by microscopic examination. Quartz is the predominating gangue mineral.

Three definite types of ore and variations of each are encountered in the mine; their occurrence is not limited to any given locality, and they do not always occur singly:

1. Quartz-rhyolite breccia, in which the quartz is of either the fourth or fifth stage,3/ being either waxy yellow, in the fourth stage, or waxy green, in the fifth stage, with primary values predominating but with secondary values in the cleavages of both quartz and rhyolite.

2. Quartz-rhyolite breccia, in which the quartz is of the fifth stage only and has primary values of very fine gold disseminated throughout the quartz. No secondary values are found. The fifth-stage mineralization, which is the richest, predominates in the breccia.

3/ Five stages of quartz have been identified by Ransome in the Oatman-Kingman district.
Ransome, F. L., Geology of the Oatman Gold District, Arizona. A preliminary Report: U. S. Geol. Survey Bull. 743, 1923, 58 pp.

3. Calcite-quartz breccia, with variations of calcite replaced by quartz and spar.

MINE-SURFACE EQUIPMENT

The mine has a complete surface plant. All equipment is in excellent mechanical condition and is well-housed.

The hoist is a single-drum Denver Engineering Co. unit, gear-driven by a 75-horsepower electric motor; it hoists a 1-1/2 ton skip at a speed of 450 feet per minute. The skip has sealed Hyatt bearings and an automatic dump.

Compressed air is supplied by an Ingersoll-Rand Imperial type 10, two-stage compressor with a capacity of 677 cubic feet per minute. A 104-horsepower motor supplies power through a short-center, endless leather belt. A second unit, consisting of a 325-cubic foot, electrically driven Chicago Pneumatic compressor, which is housed in the hoist house, is available in emergencies. A 350-cubic foot Chicago Pneumatic N-50 compressor driven by an oil engine, which was purchased before enough electrical power was available, is in the power house. It is connected with the main air lines in readiness for an emergency.

The blacksmith shop is 70 feet south of the main shaft. It is equipped with an Ingersoll-Rand tool sharpener, punch, oil furnace, and usual shop equipment. Approximately 200 crossbits are sharpened per day. Steel and tools are transported from the shaft collar to the shop on a small flat car with a minimum of time and effort.

A spacious change house equipped with hot and cold showers is 50 feet north of the shaft. Other buildings include a framing shed, warehouse, engineering office, and general office.

An electric-bell system is used in the shaft. The magnetic coil and control for the system, which was built by the plant electrician, are in the hoist room. This system is interesting in that it makes use of Ford starter buttons for push buttons on the levels. The starter buttons, as purchased, are dismantled and reassembled with heavier insulation to accommodate 110 volts. A three-wire transmission line of No. 12 rubber-covered wire is strung down the shaft; it serves both the station-lighting and signal systems. This unit is simple, cheap, and effective.

PROSPECTING AND EXPLORATION

Detailed geological maps recently completed do not permit orderly interpretation. Geological study indicates that ore deposition, structural conditions, or faulting are not systematic. Reliance cannot be placed upon projection of ore bodies or faults from one level to the next because of intervening local movements. More often than not successive vertical cross-sections at 50- or 100-foot intervals do not appear to be sections of the same mine.

This condition seriously complicates prospecting; the direction of each succeeding round in a heading is governed by the geological evidence disclosed in the previous one. Wherever possible the policy of "follow the ore" is adhered to. In general, prospecting is confined to the major fault zones along one of which a drift is run; then at intervals diamond-drill holes are used as crosscuts. The diamond drill is used solely for locating quartz, and little or no attention is paid to the assay of the core of sludge. A small drill giving a 3/4-inch core is used; all work is done on a contract basis at a cost of $1.65 per foot.

MINING METHODS

Development

The mine is developed by numerous pits and shafts along the 4,500 feet of surface exposure. The main shaft is in the center of the Pilgrim claim, and from it drifts have been run into the end line claims to both the north and south. This shaft has been sunk to a depth of 575 feet on an incline of 37°; it starts at the hanging wall on the surface but, having an inclination steeper than the vein, cuts the foot wall near the third level. Stations have been cut on the 50-, 150-, 263-, 354-, and 500-foot levels, with drifts along one or both walls on each level.

The main shear zone as well as a few of the minor ones have been exposed by 775 feet of shafts, 950 feet of raises, 1,875 feet of crosscuts, 5,100 feet of drifts, and 360 feet of winzes. All these workings are connected with the main shaft. In addition, a 100-foot shaft with a 25-foot crosscut was sunk to develop an indicated ore body on the Water Witch claim about 1,500 feet southeast of the main shaft. Drillings from a 6-inch water well assayed 1 ounce of gold to the ton. Crosscutting to this vein was in progress when all development was recessed awaiting a new power installation. It has not yet been resumed.

All drifts and crosscuts are 5 feet by 7 feet in section. Ingersoll-Rand L-74 and Chicago Pneumatic No. 5 drifters are used for drilling. The number of holes per round ranges from 9 to 15, depending upon the ground. An average advance of 4.2 feet is made per round. A miner and helper are employed in each heading; they do the shoveling, tramming, drilling, and blasting. When timbering is necessary these men are assisted by the shift boss; occasionally it is necessary to miss a drilling shift to permit placing the timber.

Winzes generally are sunk in the better ore and usually pay their own way; the dimensions vary with the width of the ore in which they are being sunk.

At least two and usually three development headings are in progress per shift. A crew of five miners and six muckers is employed for this purpose.

Costs

Drifting costs average $8.20 per foot and winzes $12.20 per foot. Expenditures for development are limited to approximately $1 for labor and supplies per ton of ore milled. The development cost per ton of ore milled is as follows:

Labor and supervision	$0.680
Supplies	.314
Power	.086
Assaying	.030
Development cost per ton of ore milled . .	1.11

Powder consumption is 0.32 pound per ton of ore milled, and lumber consumption is 2.6 board feet per ton of ore milled; these items are included in supplies of the above tabulation.

Stoping

The open-stope method of mining is used exclusively. However, owing to the various types of ore bodies, variations in dip of ore, thickness and hardness of the ore, as well as condition of walls, it has been found impossible to apply the same detailed methods in any but a few instances.

The hanging-wall vein is very flat (32°) and is composed of hard vein material or a breccia, upon which rests a mechanical clay under extreme pressure. The rock behind and above the clay or red gouge is soft. The problem, then, is to break hard or brecciated vein material from under a soft, flat, hanging wall with a minimum of dilution. This is accomplished best by raising through the ore between levels in two or more places. The number of raises depends upon the length of the ore body; usually in development of an ore body there is a raise every 60 feet. Both sides of the raise are then slabbed down with jackhammers, retreating from a common center. Slabbing at the top of the raise progresses faster and is graduated downward, giving a V-shaped opening. (Fig. 2.) The slabbing holes seldom are more than 2 feet from the face, although the holes usually are 4 feet deep. This type of drilling has three distinct advantages under the circumstances; first, it propels the ore downward toward the chute on the level below; second, it opens very little ground unsupported by timber; third, the downward hole has a tendency to undercut. In carrying a flat back stope the charge in an upward hole would tend to dislodge the hanging-wall gouge and dilute the ore. Rills and benches are avoided in these flat stopes, as slushing and mucking are accomplished better if the progressing face is straight.

In short, this method makes for cleaner ore, allows retreat in two directions from a common center with protected working faces, and minimizes slushing and mucking of ore.

The back is supported by 8- by 8-inch stulls cut to fit and with umbrella headboards when needed. The timbering is carried to within 4 feet of the face. The slabbing holes are blasted lightly with only one or two sticks of

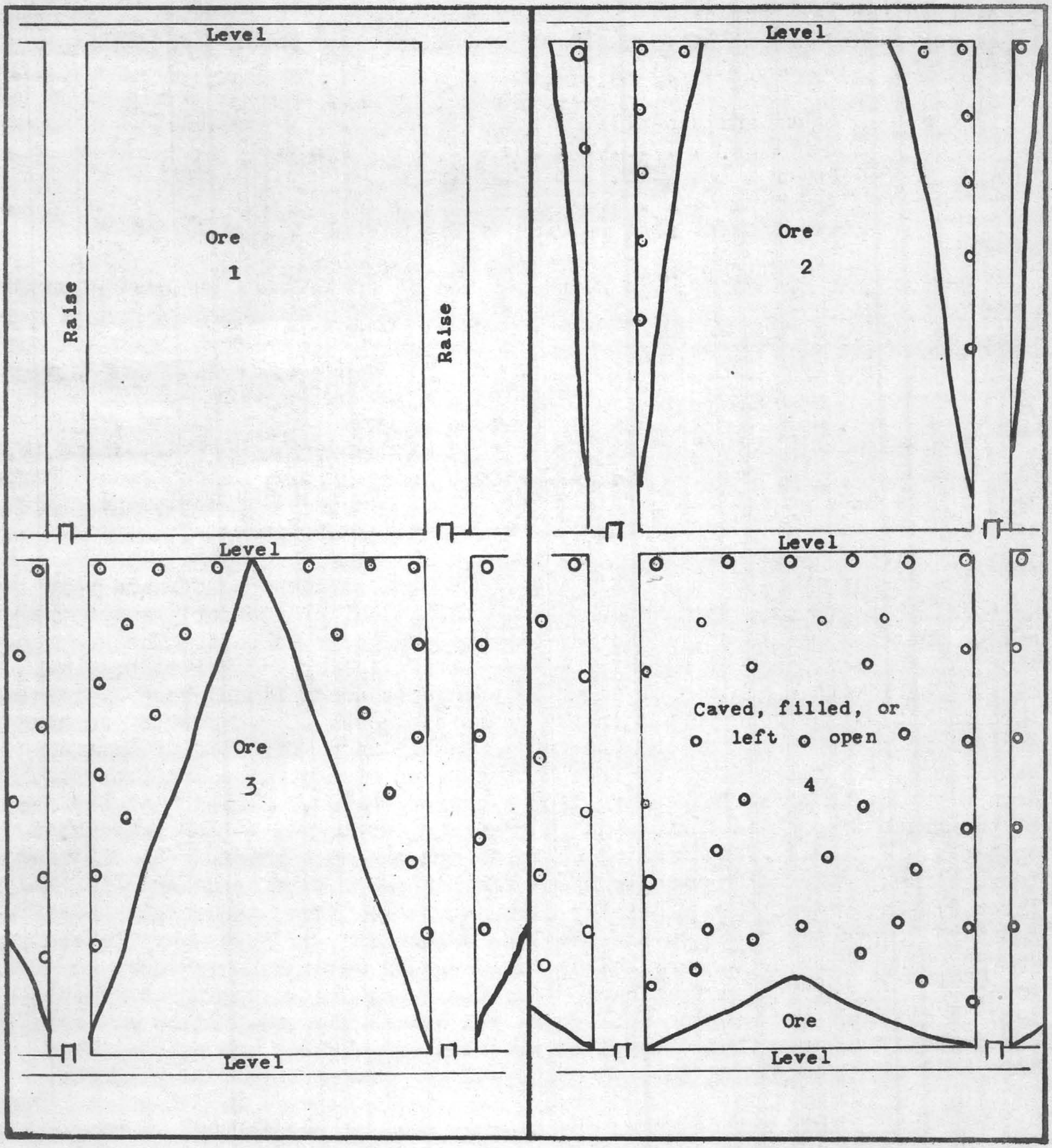

Figure 2.- Ideal progression of stoping on hanging-wall vein. Variable with extent of ore. Timbering at random.

explosive, hence the timber seldom is broken or loosened. The stulls afford ample protection for enough time to retreat; within a few weeks the headboards are forced down over the ends of the stulls, and the red gouge, having been exposed to the air, begins to slough. The mined-out stopes, except at the working faces, are soon, practically if not wholly, filled by the hanging wall and the loosened material behind it.

Broken ore is conveyed to loading chutes either by slusher hoists or by hand. If by the latter, shaker chutes of 12-inch split fan pipe are commonly used.

An underhand stoping method is used in steeper-dipping zones. Stoping begins at the top of a raise and is carried downward in a series of benches. The triangular banks of ore left between raises near the lower level are blasted down on shoveling sheets on the floor of the drift and shoveled into cars.

Where there is commercial ore from the foot wall to the hanging wall extreme care must be exercised in mining because of the vertical shearing of the vein material. This shearing produces vertical slabs of ore that weigh 20 to 150 tons each. The slabs are well-shattered and have been loosely cemented; when dropped they break easily. Each slab must be let down singly so that only a little ground will be opened at a time. The foot-wall ore proper is first mined, thereby undercutting the slabs; then a flat back stope is carried upward and the exposed slab, constituting the working face, is well-braced. Three short holes are drilled in the exposed slab and loaded with 1/3 to 1/2 stick of powder - just enough to jar it. The braces are loosened and the shots fired, which drops the ore. Wedges, blocking, and long stulls are convenient for immediate timbering of the hanging wall, and short, temporary braces are placed to support the newly exposed slab of ore. The broken ore is removed and the procedure repeated. So far this type of deposition has been encountered only in the narrower parts of the main shear zone where the distance from wall to wall averages from 12 to 20 feet. The rock behind the red-gouge hanging wall is harder in the vicinity of these stopes, and little trouble is encountered in keeping the stope open while mining is in progress, if it is properly timbered. When all ore has been removed the long stulls are removed gradually, and the stope is filled with waste from development headings, thereby saving the timber and eliminating the hoisting of waste.

In instances where the bottom of a drift is in ore that is known not to be continuous to the level below, the underhand stoping method for mining small ore bodies is employed. If the ore is not continuous downward, owing to a raking fault, which is often the case, an underhand stope is started at the intersection of the drift and the fault and is worked downward and in the direction of the rake. Slabbing holes are drilled with jackhammers along the progressing face, as in other stoping, and the method of timbering is identical with that first described. Ore is removed from the stope either by an air slusher or by laying track and lowering the regular mine cars into the stope by means of a small air hoist.

Thirty-percent-strength Hercules Gelatin is principally used for stoping, although a supply of 40-percent strength is kept on hand for harder formations. Where ventilation is exceptionally good and the rock is hard, the Hercules 40-percent-strength Extra Gelatin is used at a considerable saving in cost.

Round stulls of native pine are available from the mills near Flagstaff. The cost of an 8-inch minimum-diameter stull is 9 cents per running foot. This timber is quite satisfactory for stope timbering and temporary work. For permanent work, however, such as drift, shaft, or winze sets, only the best grade of Oregon fir lumber is used; this stock, delivered at the mine, costs $37.00 per thousand.

Costs

Stoping costs are distributed as follows:

Labor and supervision	1/	$1.74	$0.722
Supplies	2/	.51	.283
Power		.17	.147
Water		.01	.010
Assaying		.03	.050
Total stoping cost per ton of ore milled		2.46	1.212

1/ Includes compensation insurance at $8.99 per $100 of payroll and a 1-percent Federal payroll tax.

2/ Includes 2.2 pounds of powder and 3.1 board feet of lumber per ton of ore, as well as all other supplies.

MILLING METHODS

General

The original flotation mill, as completed in November 1934, had a capacity of 48 tons per 24-hour day. Mechanical troubles occurred to such an extent that the mill could be operated less than 60 percent of the time, and recoveries were low.

By means of various improvements, including the redesign of the ore feeder, reduction of ball-mill speed, discarding of some of the obsolete flotation equipment, revision of flow sheet, and complete overhaul of all mill equipment, the capacity of the mill was increased to an average of 88.8 dry tons per 24-hour day for the year 1936. Recovery has been increased to 91.9 percent and capacity increased about 30 tons per day by installing an improved-type classifier.

In consequence of the poor results achieved with the original mill, the question was raised as to whether or not it would be more economical to discontinue flotation entirely and install a countercurrent-decantation cyanide plant, such as is commonly used in this range. However, tests indicated that the ore was amenable to treatment by flotation and that relatively high recovery could be obtained by this treatment. Moreover, it was found that the cost of

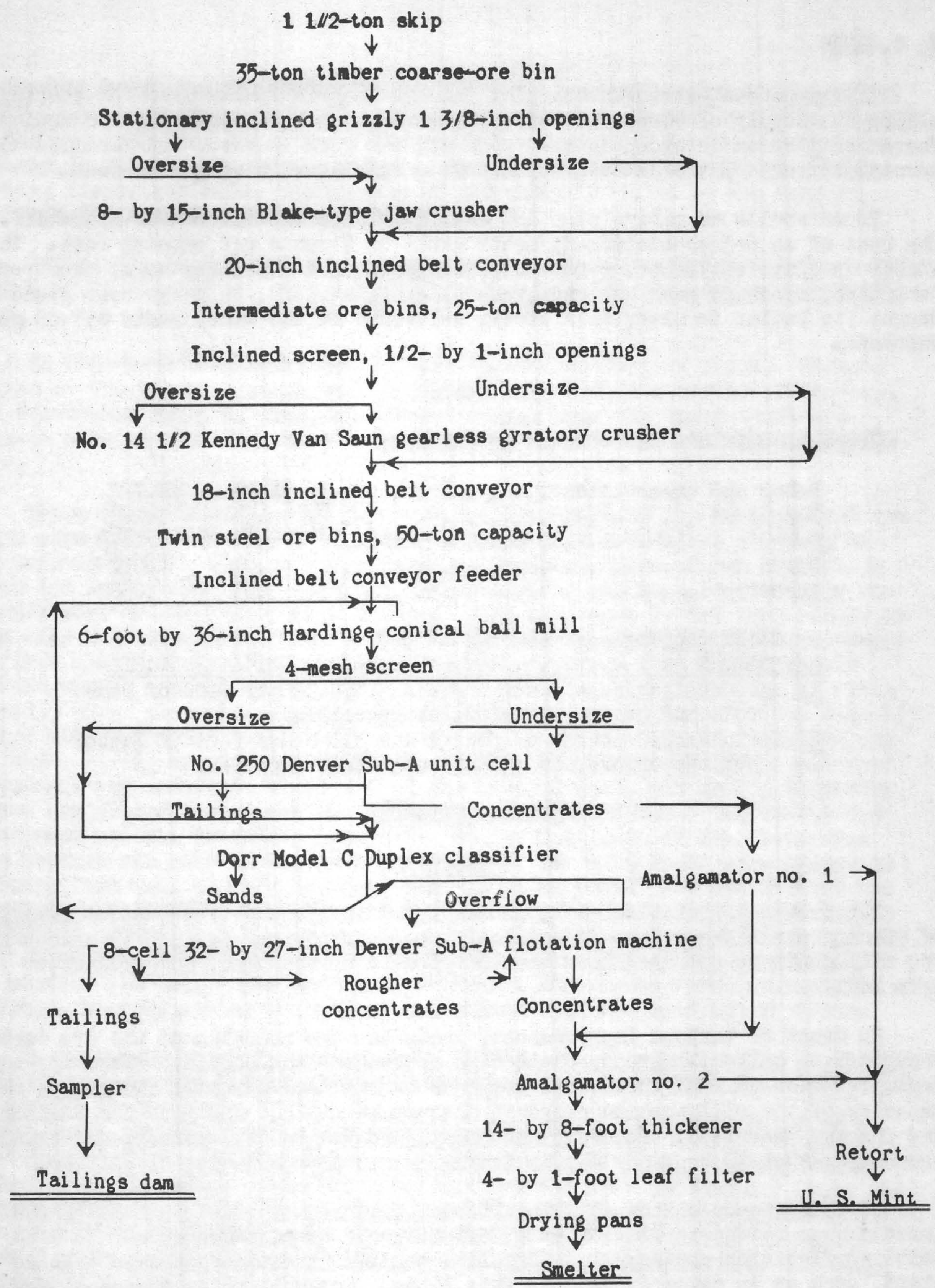

Figure 3.- Flow sheet, Pilgrim mill.

marketing concentrates, which usually is a serious item of expense at flotation plants, could be reduced materially by amalgamating the concentrate. It was decided that with so much of the material on hand straight flotation would be the cheaper method under the circumstances. Amalgamators were designed to suit the conditions and were installed, with immediately improved results.

The coarser particles of gold were removed from the concentrates by amalgamation, but there was still an appreciable amount of very fine gold to be shipped to a smelter in the form of concentrates. A small cyanide plant was built at which to treat amalgamation tailings, and, although it has not been perfected by any means, the results have been gratifying to the extent that 96.7 percent of the recovered gold has been reduced to sponge gold or bullion.

The cyanide plant, to be made entirely practical, required the expenditure of additional capital for equipment. Plans were made accordingly, but amalgamation recoveries arose to a new level, thereby leaving less gold to be extracted by cyanide. The difference in revenue between 1 ounce of gold shipped to the mint as against the smelter is $1.37; this saving applied to 4 to 8 percent of recovered gold was not sufficient to justify further capital expenditure and operating labor.

A small testing laboratory is a part of the mill equipment and is utilized almost daily by the mill superintendent. Equipment includes a Braun laboratory ball mill and a 2,000-gram Denver Sub-A laboratory flotation cell.

Breaking and Crushing

From the mine ore bin in the headframe the ore passes by gravity to a stationary, steeply inclined grizzly with 1-3/8-inch openings (fig. 3). The oversize is discharged into a New Century jaw crusher and reduced to minus 1-1/2 inches. Grizzly undersize and breaker product are conveyed by a 20-inch conveyor belt to a steel coarse-ore bin 60 feet away. The primary breaker is operated by the mine topmen intermittently throughout the 24-hour day; operation continues for less than 8 hours per day.

From the coarse-ore bin the ore gravitates over a rectangular stationary screen with 1- by 1/2-inch openings minus 1/2 inch in a no. 14-1/2 Kennedy-Van Saun gearless gyratory crusher.

Crusher product and screen undersize are conveyed to twin steel bins having a combined capacity of 70 tons.

The secondary crusher is operated principally on day shift, but topmen are well-instructed and often operate it for short periods during the night. It is operated about 6 hours a day.

Grinding

The ore from the fine-ore bins is conveyed to the ball mill on a rubber-belt conveyor-feeder. Flat gates with threaded stem and an adjustment wheel are used on the bins. The head pulley of the belt is driven through a reduction-gear system made from an old pump jack, which gives the belt a speed of about 10 feet per minute. A dry-reagent feeder is also driven from the gears. This gives a very close control over tonnage and ratio of reagent to the ore.

The ball mill is a 6-foot by 36-inch Hardinge conical mill powered by a 100-horsepower electric motor through a 9-belt Tex-rope drive; the speed of the ball mill is 29 r.p.m. The discharge end of the mill is fitted with a circular screen to classify the discharge into a plus and minus 4-mesh product. The plus 4-mesh product is delivered direct to the classifier by gravity. The minus 4-mesh product is delivered to a no. 250 Denver Sub-A unit cell. A density of 65 percent solids is maintained from the ball-mill discharge, but the additional water necessary for efficient screening lowers the unit cell feed to approximately 45 percent solids. Unit cell tailings are discharged into a 4-inch by 16-foot Dorr duplex classifier. Average screen sizes in the grinding circuit are:

Size, mesh	Ball-mill feed, percent	Ball-mill discharge, percent	Classifier sands, percent	Classifier overflow, percent
+ 4	30	2	2.0	--
-4 + 10	26	2	6.0	--
-10 + 20	15	3	8.0	--
-20 + 48	8	6	14.0	--
-48 + 100	12	16	55.0	9
-100 + 150	3	20	6.0	22
-150 + 200	2	14	3.0	16
- 200	4	37	6.0	53
	100	100	100	100

Flotation

The flotation combination consists of a Denver Sub-A unit cell in closed circuit with the ball mill and classifier; an 8-cell Sub-A unit, of which seven cells are used for roughing and one cell for cleaning.

The unit cell in the grinding circuit extracts 85 percent of the gold saved, with a concentration ratio averaging approximately 700 to 1. These concentrates are delivered by gravity to the No. 1 amalgamator (fig. 3).

The classifier overflow averages 29 to 31 percent solids and is delivered by gravity to the rougher cells. Concentrates from the five roughers assay 1-1/2 ounces gold per ton; they are cleaned in one cell of the Sub-A unit to 32 ounces per ton. The cleaned concentrates are delivered to the no. 2 amalgamator; the cleaner tailings are combined with the rougher heads. The products of this circuit are handled entirely by gravity.

Reagents used are:

Reagent	Where added	Pound per ton
Aerofloat 15	Ball-mill feed	0.11
Pentasol xanthate	Unit-cell feed	.09
Amyl xanthate	Classifier overflow	.06
Ethyl xanthate	Classifier overflow	.04
Frother B-23	Classifier overflow	.01
Soda ash 1/	Ball-mill feed	1/ .05

1/ Pound per ton of soda ash varies with the type of ore treated. A pH of 8.0 to 8.2 is maintained.

About 73 percent of the silver is recovered; this percentage can be increased by the use of ammonium phosphate but at the expense of a lower ratio of concentration.

Wet reagent feeders are made in the machine shop; they are of the common syphon type and have proved to be quite satisfactory.

Concentrate Treatment

Flotation concentrates are treated by a continuous amalgamation process that has almost eliminated the marketing of concentrates.

The amalgamators are of the barrel type, built according to specifications of the Pioneer Gold Mining Co. by the Cottrell Engineering Corporation of Los Angeles. The smaller amalgamator No. 1, is a barrel 18 inches in diameter and 34 inches in length driven by a gear-reduction motor at a speed of 26 r.p.m. A piece of cold-rolled shafting 5 inches in diameter rolls on the inside of the barrel; the action is not violent but is sufficient to burnish the particles of gold and accomplish some reduction in size of the concentrates upon which it acts. A charge of 30 pounds of mercury, enlivened with metallic sodium, is placed in the amalgamator.

A larger amalgamator, No. 2, is also in continuous operation. This barrel is 20 inches in diameter and 10 feet long but is divided into two 5-foot sections. One roll 5 inches in diameter and a 30-pound charge of enlivened mercury are used in each section.

The unit-cell concentrate is treated in No. 1 amalgamator. Sodium hydroxide is fed at the rate of 2 pounds per 24 hours, to attack grease and flotation reagents. This treatment yields a 96.1-percent recovery of gold in the unit-cell concentrates if clean-ups are made every second or third day.

The discharge of No. 1 amalgamator is combined with the cleaner concentrates as a feed for No. 2 amalgamator. A dilution is made at this point by the addition of 1/2 gallon of hot water per minute. Contact time within the amalgamator is approximately 50 minutes. Clean-up periods are bimonthly.

The success of the amalgamation treatment is due, in part, to:

1. The exceptionally high ratio of concentration in the flotation plant, which yields a very high grade product in small quantities. The barrels have a theoretical capacity far beyond the amount that is actually treated in them.

2. Pools of mercury seem to have a greater affinity for the gold in the concentrates than do plates, and enough mercury is added to the barrels to form a pool along the full length of the level bottom.

3. The gentle rolling action of the rods in the barrels burnishes the gold particles and keeps any coating of scum on the surface of the mercury broken so that the particles may come in direct contact with "live" quicksilver.

4. Hot water and sodium hydroxide help to clean the tarnish from gold and disperse the accumulated scum on the surface of the mercury.

5. Continuous amalgamation of fresh concentrates; the recovery of the gold in concentrates with stale reagents, which have been allowed to accumulate for batch treatment, is not over 50 percent.

Very little flouring of mercury has been noticed; most of the loss is retrieved in a centrifugal pump used to pump the concentrates into the drying pans. Total loss of mercury, including retorting, spillage, and flouring, amounts to approximately 0.3 percent.

Total power requirements for amalgamation amounts to 4 horsepower. Clean-ups are made at regular intervals by the mill superintendent.

The sponge produced from retorting the amalgam averages 680 fine gold and 305 fine silver; it is shipped to the United States mint.

Pulp from the amalgamators is thickened, filtered, and dried. The drying is accomplished by discharging the Diesel exhaust under eight 3- by 8-foot pans of light sheet iron.

Tailings Disposal

Tailings flow by gravity about 500 feet to a tailings dam 130 feet long. A launder is carried from 6 to 10 feet above and 6 to 8 feet inside the face of the dam. The launder bottom is drilled at 3-foot intervals with a 3/4-inch hole so that the tailings may be discharged at any point across the dam. The sands in the tailings settle close to the point where they are discharged, while the slimes tend to flow back from the dam; at the point farthest from the dam a pool of clear water is formed. This clear water flows through a 3-inch pipe to a 500-gallon tank situated below the dam. A 3 by 6 Triplex pump driven by a 10-horsepower motor pumps the water to the main mill supply tank. The motor is equipped with a float switch adjusted to the water level in the tank. Water recovery during the winter months averages about 78 percent of the fresh supply; this drops in the summer to about 52 percent.

The location of the dam is ideal, in that there is an ample drop from the mill for the gravity flow of the tailings and in that because of the small drainage area back of the dam there is little danger of washouts during the summer cloudburst season.

Sampling

All samples, except for a 24-hour tailings check sample, are cut by hand. The head samples are taken at 15-minute intervals at the ball-mill feed hopper. The entire product is taken over a period governed by the number of revolutions of one of the gears of the feeder-belt head pulley, which gives a quantity of sample proportional, at each cutting, to the tonnage being treated. Tonnage is computed on the same basis.

Tailings samples are cut at 15-minute intervals by hand at a point in the tailings launder arranged for the purpose. An automatic sampler cuts a tailings sample over a period of 24 hours as a check against the three shift samples. This sampler is operated by a flow of water that trips every 6 minutes.

Hand sampling of heads and tailings and computed tonnage of 26,000 tons in 1935 left 21 ounces of gold unaccounted for and in 1936 170 more ounces were produced than head samples indicated. Automatic check samples on tailings gave a slightly lower average than hand samples on the tailings.

As there is only one mill operator on a shift, the men are selected very carefully. A competitive spirit between operators is not promoted, and no personal credit is given to exceptionally good mill results, moreover, occasional poor results are not severely reprimanded. For these reasons there is little cause for deliberate cheating or neglect in sampling. Consequently, in spite of crude methods employed, it is considered that accurate results are obtained.

General metallurgical data, year 1936

Total milled	dry tons	29,304
Time operated	days	330
Man-shifts		1,901
Average output per 24 hours	dry tons	88.8
Concentrates produced	tons	69.5
Ratio of concentration		426.1
Average concentrates per 24 hours	tons	.21
Gold recovered	ounces	7,153.21
Recovery of gold	percent	91.9
Silver recovered	ounces	2,932.9
Recovery of silver	percent	73.0
Ounces per ton in concentrates before amalgamation		100.88
Ounces per ton in concentrates after amalgamation		8.60
Recovery by amalgamation	percent	91.4
Ball consumption per ton of ore (cast steel, 3-inch grinding balls)	pounds	2.637
Liner consumption per ton of ore (manganese liners)	pound	0.456

1/ Recovery by amalgamation for last quarter of 1936 was 95.3 percent of values saved.

Milling costs per ton

Last quarter 1936, 8,430 tons treated.

Labor	$0.310
Repairs and supplies	.355
Assaying	.060
Power	.233
Water	.091
Total cost	$1.049

The above labor cost includes six men per 24 hours - mill superintendent, three operators at $5 per shift each, one crusher man at $4.50, and one helper at $4.50 per shift. The mill superintendent's duties include personal attention to the amalgamator clean-up. The helper attends to tailings disposal and general cleanliness around the plant.

POWER

The present power plant, which, in November 1935 replaced an antiquated, 2-cylinder, 1907-model, semi-Diesel engine, is a modern 450-horsepower, 2-cycle, Fairbanks Morse Model 32 full Diesel, direct-connected to a 375 kv-a, 480-volt alternator and shunt-wound exciter. All equipment was purchased new.

The Diesel is equipped with a safety cut-out adjusted to cooling-water temperature and lubricating-oil level; a bell mounted on the switch-board gives a 2-minute warning signal if water reaches a temperature of 140° or if lubricating oil falls toonear the oil-pump intake level. If the trouble is not rectified within the 2-minute period the engine automatically stops.

Special equipment includes a Woodward governor, Vortox air cleaner mounted on the engine inside the building, and lubricating-oil filter and centrifuge. The indirect cooling method is used to prevent the deposition of excess scale in water jackets.

Centrifuged fuel is delivered from Los Angeles by a Diesel truck; lubricating oil is carried in stock at Kingman, Ariz. Both lubricating and fuel oil are of excellent grade, having low sulphur and carbon residue content; the merit of good oils is evident from the condition of exhaust ports and piston rings after 6 months of continuous operation.

March 1937, operating data include the following information:

Hours operated		744
Time lost		0
Efficiency in unit of time	percent	100
Fuel consumption	gallons	10,613
Lubricating-oil consumption	do.	162
Power produced	kw.-hr.	101,350
Cost per kw.-hr.		$0.01352

The plant is operated by one operator per 8-hour shift. A small machine shop for general work is located in the power house. Equipment includes an 18-inch lathe, power-drill press, pedestal guider, power saw, and milling machine. The chief engineer, who operates the plant on day shift, the two night operators, and the electrician are all competent machinists for general work.

WATER

During the early development of the Pilgrim mine, a well on the Water Witch claim supplied enough water for mining and camp purposes, but with the formulation of plans for the mill it became necessary to seek a larger supply. The Pioneer Co. purchased a ranch 9 miles south of the mine, on which are both Willow and Cottonwood Springs.

When the ranch was purchased the combined flow of the springs was 9 gallons per minute. Recent development by a diamond drill that made a 3/4-inch core increased the flow of the original spring from 9 to 22 gallons per minute; a sump, 20 by 10 by 14 feet deep, on Cottonwood Spring increased the flow from a few gallons per day to 21 gallons per minute. Both flows drop off in the summer, depending upon the rainfall, but it is considered that more water can be developed by vertical drill holes in the bottom of the Cottonwood Springs sump.

The original equipment included a combination 2- and 3-inch pipe line to the mine, a low-pressure pump, and a gasoline engine. This plant spasmodically delivered the 9-gallon-per-minute flow of the springs.

At present the waters of both springs flow by gravity to a 10,000-gallon storage tank 300 feet away from the closest spring and are thence pumped by a double-acting, Fairbanks Morse, 3- by 6- inch high-pressure pump against a head of 465 pounds per square inch. This pump is of the enclosed-gear type, having a crankcase capacity of 3 gallons of lubricating oil and no oil or grease cups to be attended. The pump is powered by a 2-cylinder, 20-horsepower, Fairbanks Morse Model 36 Diesel engine with clutch and endless leather belt drive. Safety equipment, as on the main power-plant Diesel, is installed. This pumping plant is a neat, compact, easily operated unit. A cottage is furnished for an attendant, who checks on fuel, lubricating oil, and cooling water twice daily. The plant operates throughout the night with no one in attendance.

Operating data last quarter 1936 were as follows:

Fuel consumption, gallon per hour	0.9
Lubricating oil consumption, gallon per 24-hour day	.5

Cost per ton ore milled (water used for all purposes):

Labor	$0.0709
Repairs	.01
Fuel	.018
Lubricating oil	.0021
Total cost per ton	$1.101

ASSAYING

A complete assay office, fully equipped for fire assaying but not for wet determinations, is convenient to both mine and mill. Although as many as 75 determinations have been made in a day, the usual average is approximately 36.

A full-time assayer is in charge; he is assisted by a mine sampler. The mine sample man takes charge of all mine samples from the time they are taken until pulverized and placed in the weighing room; he also bucks the mill samples.

The cost of sample preparation and assaying in March 1937 was $0.487 per determination.

GENERAL COSTS

Following is a complete cost sheet on a per-ton basis, last quarter 1936:

8,430 tons treated

Truck and automobile	$0.052
Cartage	.018
Insurance	.040
Legal expense	.029
Taxes	.025
Office and supervision	.177
Milling	.665
Stoping	1.01
Development	.994
Power	.466
Surface expense	.225
Pumping plant	.101
Assay office	.140
Miscellaneous	.101
Boarding house	.015
Warehouse supplies	.024
Total expenditures	$4.082

Redistributed it is as follows:

Mining	$1.217
Milling	1.049
General	.706
Direct costs	$2.972
Development	1.110
Total costs before depreciation.	$4.082

Crew for entire operation as of March 1937

General manager	1
Bookkeeper	1
Mine superintendent	1
Mine shift bosses	2
Hoist engineers	2
Topmen	2
Trammers	2
Blacksmith	1
Miners	13
Muckers	14
Mill superintendent	1
Mill operators	3
Crusher operator	1
Mill utility man	1
Chief Diesel engineer	1
Diesel engineers	2
Electrician (mechanic)	1
Carpenter (timber framer)	1
Pumpman	1
Assayer	1
Sampleman	1
General utility man	1
Total	54

Wage scale as of January 1, 1937

The following wage scale was in effect on January 1, 1937; the 1936 scale was $0.50 per shift lower:

Shift bosses - underground	$6.00 per shift
Miners	5.00
Shovelers	4.50
Timbermen	5.00
Shaftmen	5.50
Hoist engineers	5.50
Topmen	4.50
Trammers	4.50
Mill operators	5.50
Crusher operator	5.00
Diesel operators	5.50
Electrician	6.00
Mechanic	6.00

Labor statistics for entire operation as of March 1937

2,593 dry tons mined and milled

	Man-shifts
Mine development surface	36.2
Mine development underground	502.4
Buildings (new)	20.7
Machinery and equipment installation	46.6
Marketing (drying and sacking concentrates and retorting amalgam).	24.6
Mine extraction surface.	152.6
Mine extraction underground.	459.6
Milling - operating and repairs, including crushing and tailings disposal	160.1
Power - including motors and power lines	96.5
Laboratory	62.1
Pumping	38.0
General and overhead	89.7
Total manshifts	1,689.1

Note: Due to increasing detail included in bookkeeping during 1936 and 1937, it was deemed best, in order to give a true cross-section of operations, to give metallurgical data for entire year of 1936, as they were complete, costs of operation for the last quarter of 1936, as better detail was kept for this period, and man-shifts for March 1937, as that month was the innovation of that detail.

REPORT

on

PILGRIM MINE

To Mr. C. B. Schoenmehl,
Waterbury,
Connecticut.

I, R. K. HUMPHREY, a member of the Montana Society of Engineers, do hereby make and submit the following report, made from a personal examination of a certain mining property known as the PILGRIM Group of Mining Claims, located about 10 miles west of Chloride, Arizona, in the Pilgrim District, Mohave County, at an elevation of about 3600 feet:

The Group embraces an area of about 180 acres, or nine full claims 600 x 1500 feet, the names of the locations being as follows:

PILGRIM, MAYFLOWER and PLYMOUTH ROCK on the veins, and the MABLE, LUCILLE, BILLY BOY, CONTENTION, SIDE LINE and WATER WITCH being side claims.

See Map Sheet No. 1, Scale 1" equal 300 ft.

The description of these locations was furnished by Mr. Wm. Odea, the present owner.

IMPROVEMENTS

The improvements consist of hoist house, compressor house, blacksmith shop of corrugated iron, boarding house, small bunk houses, and a large bungalow or manager's residence.

EQUIPMENT

The mine is equipped with the following plant of machinery:

1 - 12 H.P. Gas Hoist and cable;
1 - 40 H.P. Gas Engine;
1 - 12 x 12 Compressor;
1 - Drill Sharpener and set of blacksmith tools;
3 - Jackhammers and steel;
2 - Jackhammer columns.

WATER SUPPLY

A drilled well 354 ft. in depth located about the center of the water witch claim, 1470 feet from the collar of the shaft, which is cased to a depth of 300 feet with 6 inch casing. It is equipped with a deep well pump 3" column, and 2½" cylinder and is said to furnish an abundance of water.

HISTORY

The property was located in March 1904, by Dempsey and O'Dea, who shipped several tons of high grade gold ore from shallow shafts and open cuts, said to have returned about $100.00 per ton. As there never has been a gold mill within reach of the mine, none of the ore has been stoped and milled.

While the claims are unpatented, the title appears to be clear at this time. The property has been incorporated into a Stock company of 1,500 shares, 500,000 shares being now in the treasury. 770,000 shares of the common stock is owned by Wm. O'Dea, and 230,000 shares by Wm. Loftus.

DESCRIPTION

The outcrop of one well defined vein is vividly traceable thru part of the PLYMOUTH ROCK Claim, through the entire length of the PILGRIM claim and for about five hundred feet into the MAYFLOWER claim, which is known in this report as the West or Hanging Wall vein. Its strike is N. 29 W. and dips at an angle of about 30° to the west, having an average thickness of from four to eight feet.

The development on this vein consists of the following openings:

At sta. 24 on the PLYMOUTH ROCK claim, a shaft about 15 ft. deep.

At Sta. 23 on the Pilgrim claim, shaft 110 ft. deep.

At Sta. 20 main working shaft 350 ft. deep, 5 x 8 in.
Timbered with 8 x 8 timber, dry and in good condition.

On the 150 ft. level the vein is exposed in a crosscut about 60 ft. from the shaft where it is about 8 ft. in thickness.

On the 222 ft. level it is again exposed in a crosscut about 84 ft. from the shaft and is about 8 ft. thick. At this point a drift has been run to the south to a distance of about 50 ft. and to the north about 65 ft.

On the 350 ft. level a crosscut has been driven to the vein but is not accessible at this time.

At Sta. 21 north of the shaft is an open cut about 20 ft. long by 6 feet deep.

At Sta. 22 north, a shaft 35 feet deep.

At Sta. 23 north, a shaft 60 feet deep.

300 ft. North Sta. 25 an open cut.

On the MAYFLOWER CLAIM at

Sta. 27, 10 ft. shaft

Sta. 28, 10 ft. shaft

EAST OR FOOT WALL VEIN

About thirty feet east of the west or Hanging Wall vein, there is another vein which runs almost parallel and dips at an angle of about 36 and 38" and is known in this report as the "east or footwall vein." It is developed on the surface at Sta. 22 south by a 10 ft. shaft. At this point the vein is about 15 ft. wide and about 300 ft. north of working shaft, 10 ft. hole. Again at Station 29 on the MAYFLOWER claim by a 10 ft. shaft.

Near the 150 ft. level, this vein entered the main shaft where a small amount of drifting was done and continued along near the roof of the shaft for a distance of about 120 ft. where it dipped at a steeper angle and left the shaft going down into the footwall again where it was again cut in a crosscut, about 37 ft., back from the shaft on the 350 ft. level, where some drifting was done. (See Plan of Workings.)

On the 22 ft. level, a drift was driven on this vein about 140 ft. to the north and was cut in a 25 ft. drift to the south.

SAMPLING

A total of 50 assays were taken from the surface and the different levels, showing the returns to be as follows:

The sampling of four dumps shows that there is about 1316 tons of milling ore averaging $12.26 per ton. $16,140 gross value on the dumps.

150 Level East Vein

Samples from breast of N. drift on 150 ft. level shows 3 ft. of ore averaging $8.86 per ton.

222 Level East Vein

Samples taken for a distance of about 140 ft. north of shaft on east vein shows the average width of ore to be about 4 ft. Average of all assays on ore shows the value to be $14.82 per ton. By eliminating one high assay leaves the average at $9.45 per ton.

Shaft

The average up and down , a portion of the face of vein at the shaft for a distance of about 120 ft. gave an average of $6.33 per ton, showing a body of ore 120 ft. high about 4 ft. thick and 140 ft. wide, making about 4800 tons, having an average value of $7.89 per ton, or $37,872 gross, or $13872 net.

West Vein on 222 ft. level

(Taken from the N. drift -- on account of caving the south drift could not be sampled)

3 Samples taken from the N. drift having an average width of 8 ft. making a general average of about 5 ft. of ore 60 ft. long north of the crosscut, gives an average value of $11.62 per ton, assuming that this ore body extends 20 ft. up and down would equal 857 tons, or $9, 958 gross, or $5,673 net.

350 ft. level

On account of the drifting on this level not being on either vein the east vein being badly faulted and irregular, only three small samples were taken. One taken for a width of 4 ft. returned $2.00 per ton. The other samples were all country rock. The crosscut to the west vein in this level not being open, the west vein could not be sampled.

Estimated ore opened on two sides.
220 ft. level.

				Gross
East Vein,	4800 tons,	assay value	$7.89 per ton	$37,872.00
West Vein,	857 tons,	" "	11.62 " "	9,958.00
Dumps,	1316 tons,	" "	12.26 " "	16,140.00
				$63,970.00
		Less Mining and Treatment		34,865.00
			Net	$29,105.00

GEOLOGY

The country rock for several miles in the vicinity of the PILGRIM vein is principally andesite, Rhyolite and Basalt. To the west of the apex of the west vein a dyke of later andesite or "Trachytic Rhyolite" (see Schrader Bulletin 397) rises to a height of 500 or 600 ft. the vein being in the contact and dipping under this dyke at an angle of about 30°. The hanging wall next to the ore is a soft brown and gray talcose gouge about 8 ft. in thickness, of a flakey texture, all surfaces being highly polished.

The vein filling is calcite with some quartz and adularia. All these ores are oxidized and very free milling, carrying about 3 oz. silver to 1 oz. gold per ton. The gold is rather fine and would be easily recovered by amalgamation or cyanide. This vein lies on a foot wall of brecciated Rhyolite, which carries about $2.40 per ton to a thickness of 4 to 8 ft.

In fact the whole formation back to the east vein carries values and might be considered vein matter in which bodies of payable ore might be found.

The East or foot wall vein is a fissure lying about 80 ft. east running nearly parallel to the main vein, appearing to make a junction near Sta. 23, about 275 ft. south of the working shaft where there seems to be a body of quartz 20 to 30 ft. in thickness, then it appears to follow the west vein to near Sta. 24 south, where the cropping is massive and heavily impregnated with quartz, which should make large ore bodies at depth.

As the east vein is fissure in the old brown andesite and is more or less irregular, now showing very plain on the surface, it is not easily traced. There are only a few openings on the surface, hense only few surface samples were taken from this vein.

CONCLUSIONS

From all data available, we would conclude that the main or west PILGRIM vein should continue to considerably greater depth, that the calcite should gradually become more silicified as depth is gained, where large bodies of payable ore should be developed; that the east vein should diminish as it gets further away from the main vein and this enrichment should be finally found in or near the main contact vein; that not far below the 250 ft. level a similar vein should be found, within 50 or 60 ft. east of the contact vein, carrying good gold values and dipping away gradually from the contact, which, should take the place of the present east vein as to values and position.

We are of the opinion that the main working shaft was located on one of the weakest portions of the vein system where neither of the veins showed on the surface. Hence, we believe that to properly develop the mine, a drift should be run to the north about 300 ft., from the crosscut on the 350 ft. level to a point near the 60 ft. shaft at Sta. 23 and connected with shaft by a raise of 300 ft. Then the north drift on the west vein on the 220 ft. level should be run about 250 ft. to connect with the raise, a total of about 850 ft. of development.

This development should cost not to exceed $20,000.00 Ore from this work should almost pay for the development.

As the workings are all very dry, there is danger of fire, hence the above connections are necessary for safety as well as for the development of the ore reserves. By cross cutting from Sta. 317 on 350 ft. level to the west vein 20 ft. and drifting south 100 ft. a point in line with 110 ft. shaft on Sta. 23, would be reached, where a raise should be driven about 150 ft. to connect with the bottom of the shaft. This development would no doubt open up a large body of payable ore and might be done first instead of drifting to the north.

Surface indications indicate several large shoots of ore as follows: One at Sta. 23, and at Sta. 24 south of the shaft; One extending from Sta. 21 to Sta. 23, north of the shaft one or two smaller shoots near the end lines of the PILGRIM AND MAYFLOWER.

For ore of this character, we would recommend the most simple crushing and cyaniding plant that can be designed. viz. dry crushing with rolls and leaching in vats, because it is in our experience the cheapest, using a minimum of power and solutions. The plant should have a daily capacity of not less than 50 tons.

The accompanying sketches are as follows:

Sheet No. 1 -- Is a map of the claims showing the vein croppings, etc.

Sheet No. 2 -- A cross section looking south showing the vein system with main shaft and various crosscuts, with assays where the east vein shows in the shaft.

Sheet No. 3 -- A longitudinal section along the vein system showing the various openings with assays.

Sheet No. 4 -- A plan view of the development from the main shaft with assays.

Legend (Vein croppings on surface
(
(
(----- Position of veins on
(----- 150 ft. level
(
(----- Position of vein on
(----- 222 level
(yellow stain -- 222 ft. level
(
(Position of veins on
(350 ft. level.

ASSAYS

List of Assays from PILGRIM mine, June 28 to July 14, inclusive.

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150 ft. level

No. 14,	1 ft. ore, N., breast of drift. one east vein	$16.48
No. 15,	2 " quartz porphyry over No. 14 Average of E. vein $8.86 per ton	1.24
No. 31,	4 ft. ore West vein, in west crosscut, next to gouge	2.40
No. 37,	4 " " " " " below No. 31 8' ore average $3.00 per ton.	3.60

220 ft. level (East of Footwall vein)

No. 1,	Corner at junction of shaft and north drift floor to roof	3.20
No. 2,	6 ft. ore W. Side drift 11 ft. N. Sta. 201	8.84
No. 3,	" " " " 20 " " sample No. 2	4.60
No. 4,	6 ft. " " " 10 " " sample No. 3	51.16
No. 21,	check sample on No. 4, same place, cuts 6 in. apart	114.96
No. 5,	7 ft. of ore 10 ft. N. of No. 4	4.60
No. 6,	Across top of drift, 6 ft, and 4 ft. down E. Side same place in drift that sample 4 was taken	15.10
No. 7,	8 ft. long, cut, top and east side of drift at Sta. 305	9.10
No. 8,	Across roof of drift, 4 ft. at Sta. 208,	2.80
No. 9,	Both sides and roof 12 ft. N. at Sta. 208,	4.00
No. 10,	4 ft. N. Sta. 210, above slip, 4 ft. ore	2.00
No. 11,	Across ground below slip along N. side E. crosscut at Sta. 10,	1.60

Note: An average composite assay of first 11 samples averages $10.00

No. 12,	One foot ore, top of incline winze at Sta. 205,	5.20
No. 13,	2 ft. ore above No. 12, in breast of incline	5.20

Note: by eliminating one high assay, an average width of 4 1/3 ft. assays $9.45 per ton.

<u>220 foot level</u> (From West or Hanging Wall Vein)

No. 16,	4 ft. of ore, or footwall half of vein at crosscut	5.00
No. 17,	4 " " " or hanging wall half of vein "	12.08
No. 18,	6 " " " hanging wall half of vein, 15 ft. N. of No. 17,	20.88
No. 19,	4 ft. of ore hanging wall half of vein, 27 ft. N. of No. 17,	8.60
No. 20,	4 ft. Rhyolite footwall in N. breast, N. drift	2.40

Note: By eliminating No. 20 which was country rock, the average is $11.64 per ton.

<u>Shaft</u> (Taken from North side of Man way)

No. 22,	2 ft. of ore near roof, 15 ft. below 220' level	8.40
No. 23,	2 " " " " 15 ft. below No. 22 sample	5.20
No. 24,	5 " " " " 4 ft. above 282 level	5.60
No. 25,	5 " " " " 20 ft. above No. 24 sample	8.70
No. 26,	6 " " " " 15 ft. above No. 25 sample	6.40
No. 27,	5 " " " " 10 ft. above No. 26 sample	3.70

Average per ton $6.33

<u>350 ft. Level</u> (Not on either vein)

No. 28,	1 ft. of ore 10 ft. from breast of O'Dea drift. about 170 ft. south of main shaft, approximately 20 ft. in footwall from west vein,	.80
No. 29,	2 ft. of ore 6 ft. west of Sta. 1, n. side crosscut	Trace
No. 30,	4 " " 26 ft. " " " 2, nr. floor of north drift.	2.00

<u>Surface and shafts south of working shaft</u>

No. 32,	2 ft. of ore S. side of 10 ft. hole, near footwall at Sta. 22 on east vein.	1.00
No. 33,	7 ft. of ore N. side 110 ft. shaft, 44 ft. from surface, near Sta. 23.	6.10
No. 34,	7 ft. ore S. side 110 ft. shaft, 20 ft. below sample 33.	1.35

No. 35,	8 ft. ore S. side 15 ft. below No. 34	5.60
No. 36,	5 ft. ore N. " 120 ft. shaft, 20 ft. above 33	8.80
No. 56,	Across 30 ft. of croppings on the surface boulders at Sta. 24.	.80
No. 57,	From dump 15 ft. shaft Sta. 25, S. on Plymouth Rock	2.40
No. 58,	Dump of shaft at Sta. 23, about 200 tons	4.80

Surface and shafts north of working shaft, on west or hanging wall vein

No. 38,	Dump of about 200 tons from Sta. 23	2.40
No. 39,	10 ft. of ore in 60 ft. shaft, Sta. 23, south side 4 ft. from breast	3.40
No. 40,	16 ft. above No. 39, 6 ft. of ore	3.20
No. 41,	7 ft. of ore 60 ft. shaft at surface	1.20
No. 42,	N. side of shaft at Sta. 22, 6 ft. of ore	1.60
No. 43,	N. side of shaft at Sta. 22, 4 ft. of ore bottom of open cut	14.40
No. 44,	Dumps of open cut, Sta. 21, 100 ft. N. of shaft 15 tons.	15.36
No. 45,	2 ft. of ore on surface, 30 ft. south of Sta. 23	204,20
No. 46,	2 ft. of ore on surface, Sta. 34 N.	3.60
No. 47,	6 ft. of ore in cuts S. side in gulch, 300 ft. N. of Sta. 25	3.20
No. 48,	2 ft. hanging wall in 10 ft., hole on MAYFLOWER claim, Sta. 7	5.60
No. 49,	3 ft. wide at MAYFLOWER location on E. Vein	.80
No. 50,	4 " of ore " " Sta. 29, 10 ft. hole on west vein.	5.20
No. 51,	From ore dump 10 ft. hole on E. vein about 300 ft. north of working shaft, about 50 tons	24.80
No. 52,	Ore dump at shaft, at PILGRIM location, 152 ft. north of working shaft, about 50 tons	15.70
No. 53,	2 ft. streak, 20 ft. N. Pilgrim location, open cut	22.20
No. 54,	2 ft. of ore 2 ft. from hanging wall at Sta. 21. N 100 ft. from working shaft, open cut	14.80
No. 55,	Dump of main shaft taken from drifts, etc. about 1250 tons	11.60

Note: By eliminating one assay of 2 ft. highgrade, $204.20 makes an average of 12 assays of $6.61 per ton.

Respectfully submitted,

Member Montana Society of Engineers.

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Pilgrim

REPORT

on

PILGRIM MINE

To Mr. C. B. SCHOENMEHL,
Waterbury,
Connecticut.

I, R.K. HUMPHREY, a member of the Montana Society of Engineers, do hereby make and submit the following report, made from a personal examination of a certain mining property known as the PILGRIM Group of Mining Claims, located about 10 miles west of Chloride, Arizona, in the Pilgrim District, Mohave County, at an elevation of about 3600 feet:

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See Map Sheet No. 1, Scale 1" dqual 300 ft.

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A drilled well 354 ft. in depth located about the center of the water witch claim, 1470 feet from the collar of the shaft, which is cased to a depth of 300 feet with 6 inch casing. It is equipped with a deep well pump

3" column, and $2\frac{1}{2}$" cylinder and is said to furnish an abundance of water.

HISTORY

The property was located in March 1904, by Dempsey and O'Dea, who shipped several tons of high grade gold ore from shallow shafts and open cuts, said to have returned about $100.00 per ton. As there never has been a gold mill within reach of the mine, none of the ore has been stoped and milled.

While the claims are unpatented, the title appears to be clear at this time. The property has been incorporated into a stock company of 1,500,000 shares, 500,000 shares being now in the treasury. 770,000 shares of the common stock is owned by WM O'Dea, and 230,000 shares by WM LOFTUS.

DESCRIPTION

The outcrop of one well defined vein is vividly traceable through part of the PLYMOUTH ROCK Claim, through the entire length of the PILGRIM claim and for about five hundred feet into the MAYFLOWER claim, which is known in this report as the West or Hanging Wall vein. Its strike is N. 29 W. and dips at an angle of about 30° to the west, having an average thickness of from four to eight feet.

The development on this vein consists of the following openings:

At sta. 24 on the PLYMOUTH ROCK calim, a shaft about 15 ft. deep.

At Sta. 23 on the PILGRIM claim, shaft 110 ft. deep.

At Sta. 20 main working shaft 350 ft. deep, 5 x 8 in. Timbered with 8 x 8 timber, dry and in good condition.

On the 150 ft. level the vein is exposed in a crosscut about 60 ft. from the shaft where it is about 8 ft. in thickness.

On the 222 ft. level it is again exposed in a crosscut about 84 ft. from the shaft and is about 8 ft. thick. At this point a drift has been run to the south to a distance of about 50 ft. and to the north about 65 ft.

On the 350 ft. level a crosscut has been driven to the vein but is not accessible at this time.

At Sta. 21 north of the shaft is an open cut about 20 ft. long by 6 feet deep.

At Sta. 22 north, a shaft 35 feet deep.

At Sta 23 north, a shaft 60 feet deep.

~~300~~ 300 ft. North Sta. 25 an open cut.

On the MAYFLOWER CLAIM at

Sta. 27, 10 ft. shaft

Sta. 28, 10 ft. shaft.

EAST OR FOOT WALL VEIN

About thirty feet east of the west or Hanging Wall vein, there is another vein which runs almost parallel and dips at an angle of about 36 and 38" and is known in this report as the "east or footwall vein." It is developed on the surface at Sta. 22 south by a 10 ft. shaft. At this point the vein is about 15 ft. wide and about 300 ft. north of working shaft, 10 ft. hole. Again at Station 29 on the MAYFLOWER claim by a 10 ft. shaft.

Near the 150 ft. level, this vein entered the main shaft where a small amount of drifting was done and continued along near the roof of the shaft for a distance of about 120 ft. where it dipped at a steeper angle and left the shaft going down into the footwall again where it was again cut in a crosscut, about 37 ft., back from the shaft on the 350 ft. level, where some drifting was done. (See Plan of Workings.)

On the 22 ft. level, a drift was driven on this vein about 140 ft. to the north and was cut in a 25 ft. drift to the south.

SAMPLING

A total of 50 assays were taken from the surface and the different levels, showing the returns to be as follows:

The sampling of four dumps shows that there is about 1316 tons of milling ore averaging $12.26 per ton. $16,140 gross value on the dumps.

150 Level East Vein
Samples from breast of N. drift on 150 ft. level shows 3 ft. of ore averaging $8.86 per ton.

222 Level East Vein
Samples taken for a distance of about 140 ft. north of shaft on east vein shows the average width of ore to be about 4 ft. Average of all assays on ore shows the value to be $14.82 per ton. By eliminating one high assay leaves the average at $9.45 per ton.

Shaft
The average up and down, a portion of the face of vein at the shaft for a distance of about 120 ft. gave an average of $6.33 per ton, showing a body of ore 120 ft. high about 4 ft. thick and 140 ft. wide, making about 4800 tons, having an average value of $7.89 per ton, or $37,872. gross, or $13872. net.

West Vein on 222 ft. level
(Taken from the N. drift -- on account of caving the south drift could not be sampled)

3 samples taken from the N. drift having an average width of 8 ft. making a general average of about 5 ft. of ore 60 ft. long north of the crosscut, gives an average value of $11.62 per ton, assuming that this ore body extends 20 ft. up and down would equal 857 tons, or $9,958. gross, or $5,673. net.

350 ft. level

On account of the drifting on this level not being on either vein the east vein being badly faulted and irregular, only three small samples were taken. One taken for a width of 4 ft. returned $2.00 per ton. The other samples were all country rock. The crosscut to the west vein in this level not being open, the west vein could not be sampled.

Estimated ore opened on two sides.
220 ft. level.

				Gross
East Vein,	4800 tons,	assay value	$7.89 per ton	$37,872.00
West Vein,	857 tons,	" "	11.62 " "	9,958.00
Dumps,	1316 tons,	" "	12.26 " "	16,140.00
				$63,970.00
		Less Mining & Treatment- - -		34,865.00
			Net - - - - -	$29,105.00

GEOLOGY

The country rock for several miles in the vicinity of the PILGRIM vein is principally andesite, Rhyolite and Basalt. To the west of the apex of the west vein a dyke of later andesite or "Trachytic Rhyolite" (see Schrader Bulletin 397) rises to a height of 500 or 600 ft. the vein being in the contact and dipping under this dyke at an angle of about 30°. The hanging wall next to the ore is a soft brown and gray talcose gouge about 8ft.in thickness, of a flakey texture, all surfaces being highly polished.

The vein filling is calcite with some quartz and adularia. All these ores are oxidized and very free milling, carrying about 3 oz. silver to 1 oz. gold per ton. The gold is rather fine and would be easily recovered by amalgamation or cyanide. This vein lies on a foot wall of brecciated Rhyolite, which carries about $2.40 pen ton to a thickness of 4 to 8 ft.

adinole = light gray or green clay slate, metamorphosed with qtz etc

In fact the whole formation back to the east vein carries values and might be considered vein matter in which bodies of payable ore might be found.

The East or foot wall vein is a fissure lying about 80 ft. east running nearly parallel to the main vein, appearing to make a juction near Sta. 23, about 275 ft. south of the working shaft where there seems to be a body of quartz 20 to 30 ft. in thickness, then it appears to follow the west vein to near Sta. 24 south, where the cropping is massive and heavily impregnated with quartz, which should make large ore bodies at depth.

As the east vein is fissure in the old brown andesite and is more to less irregular, not showing very plain on the surface, it is not easily traced. There are only a few openings on the surface, hense only few surface samples were taken from this vein.

CONCLUSIONS

From all data available, we would conclude that the main or west PILGRIM vein should continue to considerably greater depth, that the calcite should gradually become more silicified as depth is gained, where large bodies of payable ore should be developed; that the east vein should diminish as it gets further away from the main vein and this enrichment should be finally found in or near the main contact vein; that not far below the 350 ft. level a similar vein should be found, within 50 or 60 ft. east of the contact vein, carrying good gold values and dipping away gradually from the contact, which, should take the place of the present east vein as to values and position.

We are of the opinion that the main working shaft was located on one of the weakest portions of the vein system, where neither of the veins showed on the surface. Hense, we believe that to properly develop the mine, a drift should be run to the north about 300 ft, from the crosscut on the 350 ft. level to a point near the 60 ft. shaft at Sta. 23 and connected with shaft by a raise of 300 ft. Then the north drift on the west vein on the 220 ft. level should be run about 250 ft. to connect with the raise, a total of about 850 ft. of development.

This development should cost not to exceed $20,000.00. Ore from this work should almost pay for the development.

As the workings are all very dry, there is danger of fire, hence the above connections are necessary for safety as well as for the development of the ore reserves. By cross cutting from Sta. 317 on 350 ft. level to the west vein 20 ft. and drifting south 100 ft. a point in line with 110 ft. shaft on Sta. 23, would be reached, where a raise should be driven about 150 ft. to connect with the bottom of the shaft. This development would no doubt open up a large body of payable ore and might be done first instead of drifting to the north.

Surface indications indicate several large shoots of ore as follows: One at Sta. 23, and at Sta. 24 south of the shaft; One extending from Sta. 21 to Sta. 23, north of the shaft one or two smaller shoots near the end lines of the PILGRIM and MAYFLOWER.

For ore of this character, we would recommend the most simple crushing and cyaniding plant that can be designed. viz. dry crushing with rolls and leaching in vats, because it is in our experience the cheapest, using a minimum of power and solutions. The plant should have a daily capacity of not less than 50 tons.

The accompanying sketches are as follows:

Sheet No. 1-- Is a map of the claims showing the vein croppings, etc.

Sheet No. 2-- A cross section looking south showing the vein system with main shaft and various crosscuts, with assays where the east vein shows in the shaft.

Sheet No. 3-- A loggitudinal section along the vein system showing the various openings with assays.

Sheet No. 4-- A plan view of the development from the main shaft with assays.

Legend (Vein croppings on surface
(
(
(------ Position of veins on
(------ 150 ft. level
(
(------ Position of vein on
("----- 222 level
(yellow stain -- 222 ft. level
(Q
(Position of veins on
(350 ft. level.

ASSAYS

List of Assays from PILGRIM mine, June 28 to July 14, inclusive.

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150 ft. level

No. 14,	1 ft. ore, N., breast of drift. one east vein	$ 16.48
No. 15,	2 " quartz porphyry over No. 14	1.24
	Average of E. vein $8.86 per ton	

No. 31,	4 ft. ore West vein, in west crosscut, next to gouge	2.40
No. 37,	" " " " " " " below No. 31	3.60
	8' ore average $3.00 per ton.	

220 ft. level (East of Footwall vein)

No. 1,	Corner at junction of shaft and north drift floor to roof	3.20
No. 2,	6 ft. ore W. Side drift 11 ft. N. Sta. 201	8.84
No. 3,	" " " " 20 " " sample No. 2	4.60
No. 4,	6 ft. " " " 10 " " sample No. 3,	51.16
No. 21,	check sample on No. 4, same place, cuts 6 in. apart	114.96
No. 5,	7 ft. of ore 10 ft. N. of No. 4	4.60
No. 6,	Across top of drift, 6 ft, and 4 ft. down E. Side same place in drift that sample 4 was taken	15.10
No. 7,	8 ft. long, cut, top and east side of drift at Sta. 305,	9.10
No. 8,	Across roof of drift, 4 ft. at Sta. 208,	2.80
No. 9,	Both sides and roof 12 ft. N. at Sta. 208,	4.00
No. 10,	4 ft. N. Sta. 210, above slip, 4 ft. ore	2.00
No. 11,	Across ground below slip along N. side E. crosscut at Sta. 10,	1.60

Note: An average composite assay of first 11 samples averages $10.00

No. 12,	One foot ore, top of incline winze at Sta. 205,	5.20
No. 13,	2 ft. ore above No. 12, in breast of incline	5.20

Note: by elmiminating one high assay. an average width of 4 1/3 ft. assays $9.45 per ton.

220 foot level (From West or Hanging Wall Vein)

No. 16,	4 ft. of ore, or footwall half of vein at crosscut	5.00
No. 17,	4 " " " or hanging wall half of vein " "	12.08
No. 18,	6 " " " hanging wall half of vein, 15 ft. N. of No. 17,	20.88
No. 19,	4 ft. of ore hanging wall half of vein, 27 ft. N. of No. 17,	8.60
No. 20,	4 ft. Ryholite footwall in N. breast, N. drift	2.40

Note: By eliminating No. 20 which was country rock, the average is $11.64 per ton.

Shaft (Taken from North side of Man way.)

No. 22,	2 ft. of ore near roof, 15 ft. below 220' level	8.40
No. 23,	2 " " " " 15 ft. below No. 22 sample	5.20
No. 24,	5 " " " " 4 ft. above 282 level	5.60
No. 25,	5 " " " " 20 ft. above No. 24 sample	8.70
No. 26,	6 " " " " 15 ft. above No. 25 sample	6.40
No. 27,	5 " " " " 10 ft. above No. 26 Sample	3.70
	Average per ton $6.33	

350 ft. Level (Not on either vein)

No. 28,	1 ft. of ore 10 ft. from breast of O'dea drift. about 170 ft. south of main shaft, approximately 20 ft. in footwall from west vein,	.80
No. 29,	2 ft. of ore 6 ft. west of Sta. 1, n. side crosscut	Trace
No. 30,	4 " " 26 ft. " " " 2 nr. floor of north drift.	2.00

Surface and shafts south of working shaft

No. 32,	2 ft. of ore s. side of 10 ft. hole, near footwall at Sta. 22 on east vein	1.00
No. 33,	7 ft. of ore N. side 110 ft. shaft, 44 ft. from surface, near Sta. 23.	6.10
No. 34,	7 ft. ore S. side 110 ft. shaft, 20 ft. below sample 33.	1.35
No. 35	8 ft. ore S. side 15 ft. below No. 34	5.60
No. 36	5 ft. ore N. " 120 ft. shaft, 20 ft. above 33	8.80
No. 56,	Across 30 ft. of croppings on the surface boulders at Sta. 24.	.80
No. 57,	From dump 15 ft. shaft Sta. 25, S. on Plymouth Rock	2.40
No. 58,	Dump of shaft at Sta. 23, about 200 tons	4.80

Surface and shafts north of working shaft, on west or hanging wall vein

No. 38,	Dump of about 200 tons from Sta. 23	2.40
No. 39,	10 ft. of ore in 60 ft. shaft, Sta. 23, south side 4 ft. from breast	3.40
No. 40,	16 ft. above No. 39, 6 ft. of ore	3.20
No. 41	7 ft. of ore 60 ft. shaft at surface	1.20
No. 42	N. side of shaft at Sta. 22, 6 ft. of ore	1.60
No.43	N. side of shaft at Sta. 22, 4 ft. of ore bottom of open cut	14.40
No. 44,	Dumps of open cut, Sta. 21, 100 ft. N. of shaft 15 tons.	15.36
No. 45,	2 ft. of ore on surface, 30 ft. south of Sta. 23	204.20
No. 46,	2 ft. of ore on surface, Sta. 34 n.	3.60
No. 47,	6 ft. of ore in cut S. side in gulch, 300 ft. N. of Sta. 25	3.20
No. 48,	2 ft. hanging wall in 10 ft., hole on MAYFLOWER claim, Sta. 7	5.60
No. 49,	3 ft. wide at MAYFLOWER location on E. Vein	.80
No. 50,	4 " of ore " " Sta. 29, 10 ft. hole on west vein	5.20
No. 51.	From ore dump 10 ft. hole on E. vein about 300 ft. north of working shaft, about 50 tons	24.80
No. 52,	Ore dump at shaft, at PILGRIM location, 152 ft north of working shaft, about 50 tons	15.70
No. 53,	2 ft. streak, 20 ft. N. Pilgrim location, open cut	22.20
No. 54,	2 ft. of ore 2 ft. from hanging wall at Sta. 21. N 100 ft. from working shaft, open cut	14.80
No. 55,	Dump of main shaft taken from drifts, etc. about 1250 tons	11.60

Note: By eliminating one assay of 2 ft. highgrade, $204.20 makes an average of 12 assays of $6.61 per ton.

Respectfully submitted

Member Montana Society of Engineers.

Name of Engineer lacking.