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Ore Treatment at the Combination Mine, Goldfield, Nevada.

Written for the MINING AND SCIENTIFIC PRESS
By FRANCIS L. BOSQUI.

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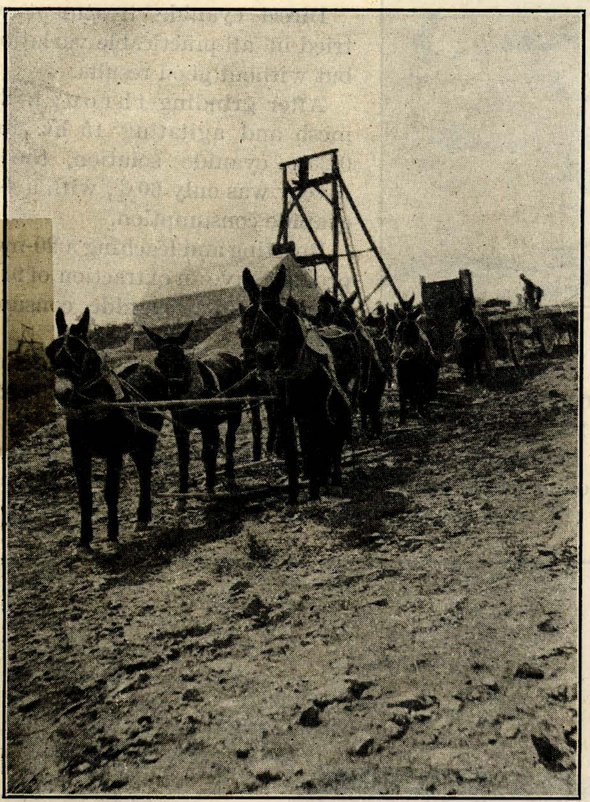
I. METALLURGY.

The Combination mine, situated about one-half mile northeast of the town of Goldfield, consists of ten full claims and three fractions, aggregating 200 acres. The original discovery on Combination ground was the first of any importance made in the now famous camp. The property was acquired from prospectors in 1903 by the representatives of two Eastern exploration companies, who, after a visit to one of the outlying districts, happened to be passing through the present site of Goldfield on the way to Tonopah; and thus, at the very outset of its career as a gold producer, the Combination was



Hauling Ore to the Railway.

blessed by the happiest accident that can befall a mine—it passed into good hands. The property has since occupied a unique place among the mines of southern Nevada. It has been well administered; it has had the benefit of the most approved and practical methods in mining and metallurgy; and its development has not been hampered by stock manipulations. Consequently, though the most interesting property in the district, whether we consider its varied metallurgical problems, or its ratio of output to small mill-capacity and small development, it is the least advertised and the least discussed.

The first shipments from the Combination were made in December, 1903. The gross output of the mine from the commencement of operations to April 1, 1906, is as follows:

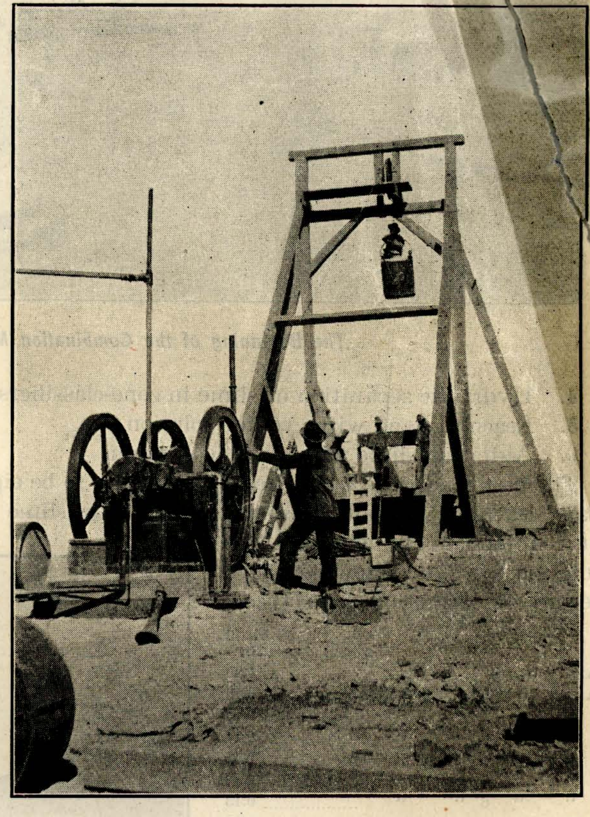
		Value	Total Value.
Shipping ore (tons)	1,773	$438.24	$776,992.84
Stamp bullion (oz.)	13,584	19.48	264,506.30
Concentrate (tons)	230	352.05	80,972.07
Cyanide precipitate (lb.)	734	45.77	33,594.18
Cyanide bullion (oz.)	4,401	16.44	72,346.51
			$1,228,411.90

The property was a shipper from the grass-roots. Almost any grade of ore could be segregated by rough screening and sorting, the practice being to reserve the milling ore ($25 to $100) in graded dumps until the completion of a mill. In shipping it was at first necessary to haul the ore to the railroad, a distance of 60 miles. The costs of transportation and smelter treatment were so high as to emphasize the importance of treatment on the ground, and an investigation was at once commenced with a view to installing a reduction plant.

29 items

98 pages Total

The ore has been described as a highly silicified dacite occurring in zones of fissuring in the decomposed dacite constituting the country rock. In the more shattered portions of the orebody the dacite is almost entirely altered into quartz, with stringers and patches of kaolinized material. During the progress of the preliminary tests, the ore showed certain freakish variations which made it difficult to decide upon a method of treatment. The fineness of the gold, and the almost entire absence of concentratable material in the upper levels indicated dry-



Original Combination Shaft. Man in Bucket.
(Site Now Occupied by Large Open-Cut.)

crushing, and a testing plant for dry-crushing was installed. But even after a long leaching with cyanide solution, it was still found possible to pan an appreciable quantity of gold from the residue. These tests had scarcely begun before the gold in the mine became coarser, and the proportion of sulphides increased. A series of tests by amalgamation, concentration, and cyanidation, gave decidedly promising results, showing an average saving of 45% by the first, 5% by the second, and about 40% by cyaniding the residual sand and slime—a total of about 90%. Later, it developed that certain portions of the oxidized ore carried a disquieting amount of acid—free sulphuric and ferrous sulphate, with here and there enough alum to make the rock astringent to the taste. In some of the tests the ore was so acid as to require 50 lb. of lime per ton as a neutralizer. But this very acid ore was not found to be in sufficient quantity to affect the general treatment seriously; and in subsequent milling tests an average sample of all the accessible oxidized ore was taken, and the acid condition met by using from 10 to 12 lb. lime per ton.

The results obtained in ore-tests made at Goldfield in

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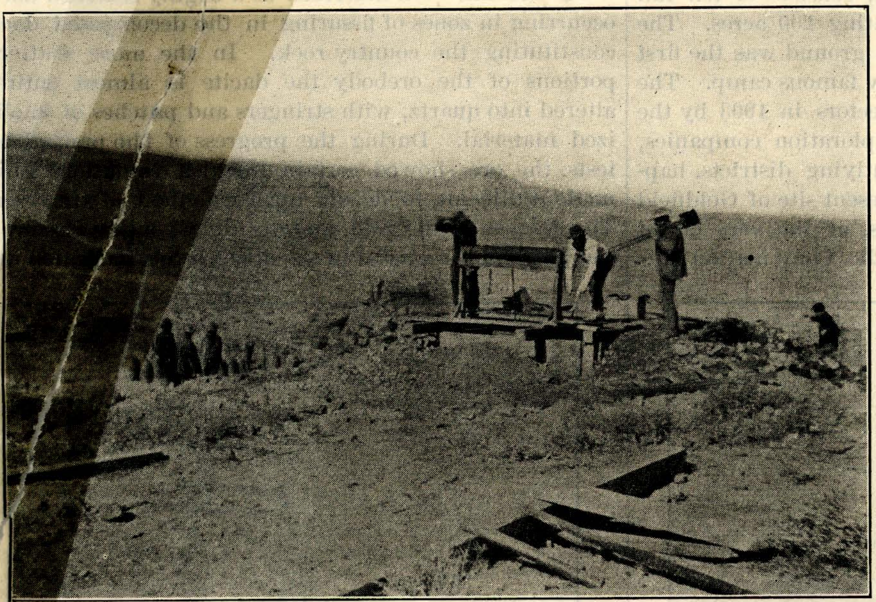
the early months of 1904 were confirmed during the summer by mill-runs made at an ore-testing plant in San Francisco. The representative test, which gave the best results, was made as follows:

1. Crushing through 40-mesh wire screen.
2. Plate amalgamation.
3. Concentration on a Frue vanner.



The Beginning of the Combination Mine.

4. Hydraulic separation of slime in cone-classifiers.
5. Leaching sand with cyanide solution.
6. Agitating slime with cyanide solution.

The best conditions for the sand were found to be eight days' leaching with a 0.2% solution; while the slime required four hours with a 0.15% solution. The following is a record of extraction from slime:

	Gold. oz.
Assay heads	1.20
Assay after 1 hr. agitation	0.46
" " 2 " "	0.20
" " 3 " "	0.14
" " 4 " "	0.12
" " 5 " "	0.12
" " 6 " "	0.13
" " 7 " "	0.12
" " 8 " "	0.12

The amalgamation plate and zinc-box were cleaned up and the following results obtained from the whole test:

Indicated extraction by cyanide, 83.4%.

Actual extraction by cyanide, 77.9%.

Indicated total extraction by all processes, 93%.

Actual total extraction by all processes, 91%.

On a small scale, better results were obtained by crushing to 50 mesh, and it was found that by sliming the whole product, a still higher recovery might be made. But the unproved efficiency of American tube-mills at the time the tests were made and the high cost of power and labor at Goldfield, left the advantage in favor of sliming too small to justify the experiment.

The mill was originally designed to treat oxidized ore only, although in places in the oxidized zone there were found small quantities of sulphide ore which resisted mill treatment by ordinary methods. But as there was no indication of the development of a large amount of sulphide ore at the time construction commenced on the mill, the installation was allowed to proceed. It was while the mill was being built that large shoots of sulphide ore were opened up as the limit of the oxidized zone was reached, and before the end of the year an extensive dump had accumulated with an average content of about 3 oz. gold. This was reserved for special treatment, and samples taken for investigation.

The sulphide in the 'sulphide ore' of the lower levels is a simple iron pyrite, for the most part finely disseminated. The following is an analysis of the ore:

Silica, 70.4%; alumina, 17.0%; sulphur, 4.2%; iron, 8.5%; and copper, trace.

Direct cyanide treatment was tried in all practicable variations, but without good results.

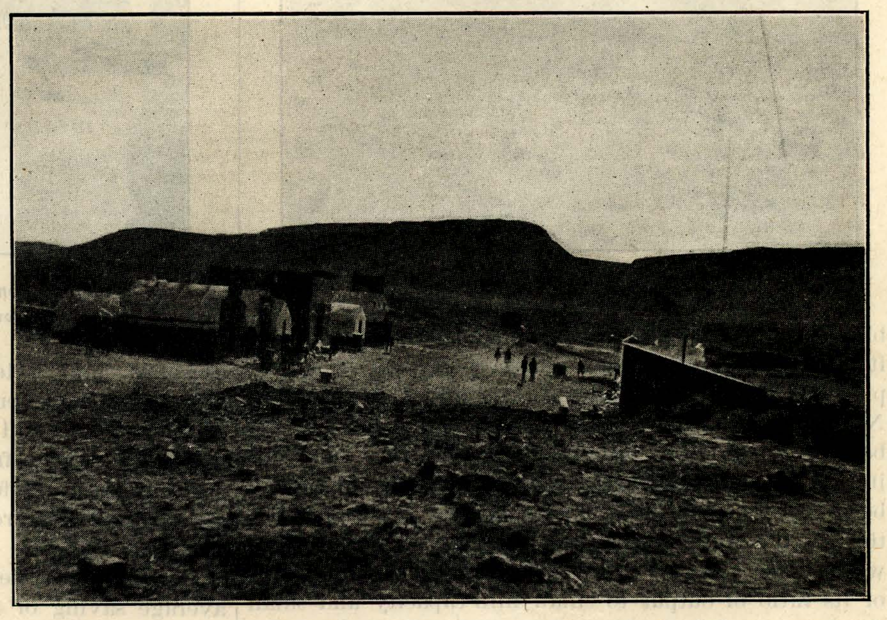
After grinding through 200 mesh and agitating 15 hr. in a 0.25% cyanide solution, the recovery was only 60%, with a 4-lb. cyanide consumption.

Roasting and leaching a 20-mesh product gave an extraction of 91%, with a 7.2-lb. cyanide consumption.

Roasting a 20-mesh product, regrinding to 200-mesh, and cyaniding by agitation in a 0.25% solution gave 93% extraction.

Pan amalgamation of the roasted ore gave an extraction of 54 per cent.

Oil concentration (Elmore process) of raw ore, with agitation of tailing in cyanide solution, gave 90% extraction with a consumption of 7½ lb. cyanide per ton.



Goldfield, at Sunset, in 1904.

Oil concentration followed by cyanogen bromide treatment of the tailing gave an extraction of 96% from heads assaying 3.48 oz. gold.

Chlorination by leaching, using an aqueous solution of chlorine produced by bringing together an 0.8% sulphuric acid solution, and a 0.7% chloride of lime solution, and leaching 36 hr., gave a recovery of 40%.

Chlorination by the barrel process, after four hours' treatment, yielded 78 per cent.

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These various methods were tried before wet concentration, because the aim was to treat all products on the ground and avoid shipping. A combination of concentrating and cyaniding, however, was ultimately considered the most suitable to Goldfield conditions, and was adopted.

The ordinary concentration of 30 or 40-mesh product was found ineffective. It did not make a close saving of the fine sulphide, which had to be removed on account of the poor cyanide recovery from the raw sulphide. It was necessary to evolve some closer method of recovery which would leave nothing for cyanide treatment except the finest particles that might elude the most efficient concentrating machinery. The only way to accomplish this was by a series of reductions, and by following each stage by appropriate concentration. The sulphide freed at each stage of grinding was at once removed before the ore passed to the next and finer stage of grinding, and thus an unnecessary comminution was avoided.

The following mill test forms the basis of the method adopted in practice. The ore, assaying 3.01 oz. gold, was crushed in stamps to 30 mesh and passed over a small Wilfley concentrator. This yielded 6% (by weight) of concentrate, assaying 27.4 oz. gold. The tailing was re-ground through 60 mesh, and re-concentrated; yielding 2.11% concentrate, assaying 19.8 oz. gold. The 60-mesh tailing was re-ground through 100 mesh and concentrated, yielding 0.4% concentrate, assaying 20.4 oz. gold. The residue was then ground to 200 mesh, and passed over a canvas table, yielding 10% of silicious concentrate, assaying 3.97 oz. gold. The final tailing from the above operation assayed 0.52 oz. gold, showing an extraction by concentration of 80%. The extraction by canvas alone was 13%, showing the marked adaptability of canvas to ore of this character containing so much extremely fine sulphide. Cyanide treatment of the slime resulting from this series of successive reductions and concentrations reduced the tailing to 0.22 oz. gold, making the total extraction 93 per cent.

This method was adopted because the required plant could be conveniently added to the mill already installed, and the same system of crushing used. Though a little complicated, the process was the least so of any of the methods considered. The high cost of fuel and supplies in Goldfield barred roasting. Besides, so long as the ore concentrated well, the advantage gained by segregating the roastable portion of the ore in small bulk was obvious.

The concentrate from the sulphide ore is now being shipped. It may later be treated by chlorination on the ground. The concentrate from the oxidized ore is about to be treated in the mill by fine grinding and prolonged cyanidation.

THE SEARCH FOR DIAMONDS.—Never before in the history of the United States has there been such a demand for diamonds as there was in 1905. Large quantities were imported, but the country produced none. In 1903 it produced diamonds to the value of $50, in 1901 it had an output worth $100, in 1900 its production was valued at $150 and in 1899 the country boasted native diamonds to the value of $300. Diamonds have been discovered in the United States in four different regions, but their actual place of origin is in every case unknown. All have been found in loose and superficial deposits, and all accidentally. It is not at all improbable, however, that some day the original sources of this queen of gems may be discovered.

GEOLOGY is every day assuming a greater importance in mining, but among its fundamental conceptions there must be no confusion between what is certain and what is more or less probable.

Coal Production.

According to the report of Edward W. Parker, Statistician of the United States Geological Survey, which is now in press, the production of coal in 1905 amounted to 392,919,341 short tons, having a value at the mines of $476,756,963, surpassing in both quantity and value all previous records in the history of the country. Compared with 1904, when the production amounted to 351,816,398 short tons, valued at $444,371,021, the output in 1905 exhibits an increase of 41,102,943 short tons, or 11.7% in quantity, and of $32,385,942, or 7.3% in value. Prior to 1905 the maximum output of coal was obtained in 1903, when the production amounted to 357,356,416 short tons, valued at $503,724,381, compared with which the record for 1905 shows an increase in production of 35,562,925 short tons, and of $26,967,418. The high value recorded in the statistics for 1903 was due to the somewhat abnormal inflation of prices, caused by the shortage of fuel supplies, which resulted from the strike in the anthracite region of Pennsylvania in the preceding year. The lower values in 1904 as compared with 1903 were simply a return to normal conditions, but the decline in 1905 was the result of a production in excess of market requirements, unusually large as they were.

Of the total production in 1905, 69,339,152 long tons (equivalent to 77,659,850 short tons), were Pennsylvania anthracite, with a value at the mines of $141,879,000. The total production of bituminous coal and lignite was 315,259,491 short tons, valued at $334,377,963. The production of anthracite coal in Pennsylvania in 1905 was 4,020,662 long tons (or 4,503,151 short tons) more than that of 1904, while the increase in the production of bituminous coal and lignite was 36,599,882 short tons. A portion of these increases in both anthracite and bituminous production was due to the efforts of operating companies to provide a supply of fuel in anticipation of a suspension of mining in April, 1906, when the wage scale agreements in the organized coal producing States and the award of the Strike Commission in the anthracite region of Pennsylvania would terminate.

Of the total amount of bituminous coal produced in 1905, 103,396,452 short tons were mined by the use of mining machines, as compared with a machine-mined product in 1904 of 78,606,997 short tons. The number of mining machines in use increased from 7,663 in 1904 to 9,184 in 1905.

The total number of men and boys employed last year in the coal mines of the United States was 626,174, against 593,693 in 1904. Of the total number employed in 1905, 165,406 were in the anthracite mines of Pennsylvania, and 460,768 were employed in the bituminous coal mines.

The larger part of the increased production in 1905 was due to the great activity in the iron industry, as is shown by the fact that the amount of coal made into coke increased from 31,278,537 short tons to 42,412,328 short tons, and that the larger increases were in the coking coal producing States and those which furnished fuel to the iron furnaces.

A BOULDER 17 by 12 by 7 ft., weighing 60 tons, has been dislodged from the bed of the Nile by the powerful current issuing from the barrier at the Assouan dam and hurled against the masonry. The tremendous force of rapidly moving water is not generally recognized. It is estimated that the weight of solids which can be moved by a stream increases as the sixth power of the velocity; thus, a stream moving at 10 miles per hour can move 64 times the mass which can be moved by a stream at 5 miles per hour.

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II. The Mill.

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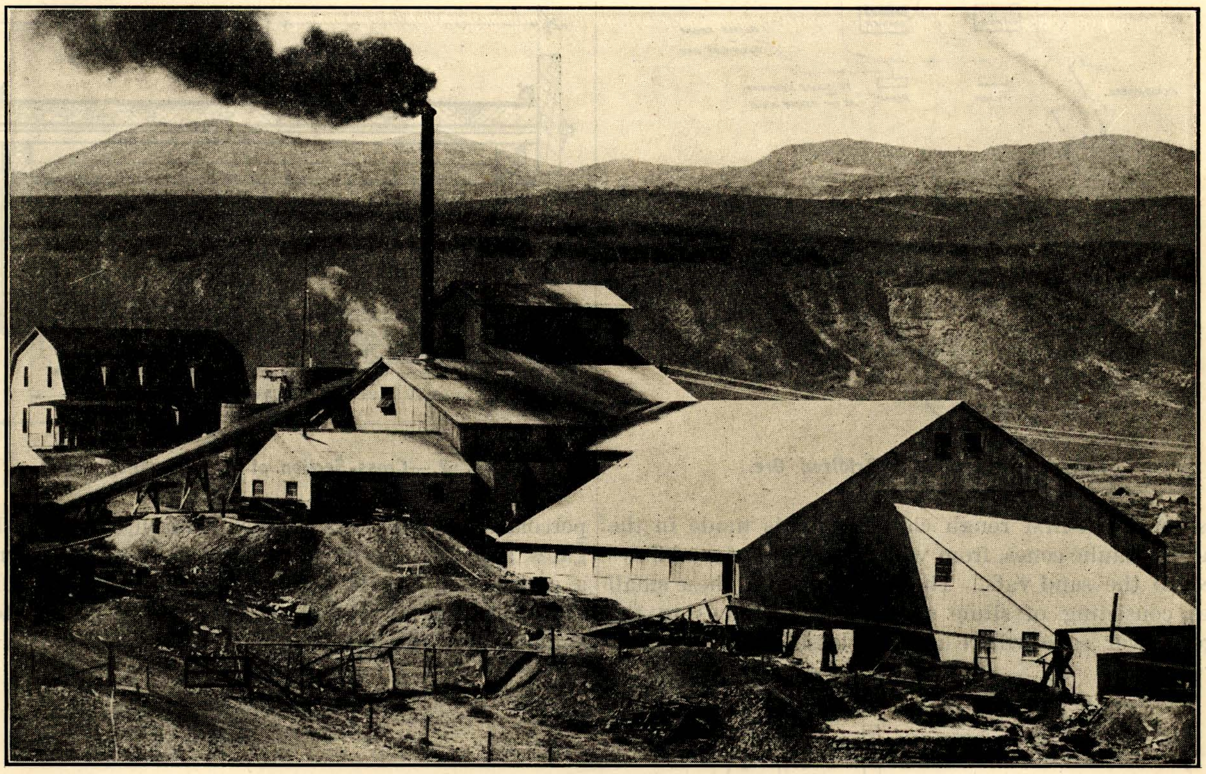
In considering a mill for the combination mine, it was thought that the size of the property would hardly justify the initial installation of more than ten stamps. The plant, as completed, is unusually large for such small capacity. An extensive and costly equipment was made necessary by the elaborate process required for the best recovery, and by the decision of the management to reserve one-half of the mill for the treatment of custom ore. The latter required one complete and separate unit; thus the whole plant was spread over a greater area than would otherwise have been necessary. The outlay, however, has been amply justified by the efficiency of the machinery, and the high recovery obtained.

The lack of grade for millsite introduced several problems into the construction which have necessarily affected the cost of treatment, namely, the elevating of ore to the bins, and the elevating of pulp to classifiers. The water problem, however, was solved at the start. The water is obtained from springs situated about ten miles west of the mine, and is pumped to the mill against a head of about 800 ft. at a cost of 0.1c. per gal. It is hot as it comes to the surface and carries about 0.1% sodium sulphate and a trace of sodium chloride. Though slightly brackish, the water is potable; and for milling purposes the sodium sulphate is beneficial in assisting the settlement of the slime.

The ore is trammed from the shaft to storage bins, from which it is delivered to a platform above a 10 by 16 Sturtevant roll-jaw crusher, where it is mixed with the required amount of lime, varying from 5 to 10 lb. per ton. The crusher reduces the ore to about one-half inch size. The lack of fall made the interposition of a sorting grizzley impossible. Everything passes to the crusher and is delivered direct to a 12-in. belt-conveyor set at an angle of 20°, which elevates the ore to the mill-bins. There are four of these bins for the four units of five stamps each. (Ten more stamps have recently been added for treating sulphide ore.) The point of discharge at the top of the mill is shifted by means of the usual form of adjustable carriage. The ore is fed from hanging feeders, attached to the bins, into the low mortars of the Boss cantilever battery system. This type of battery frame and mortar was described in the MINING AND SCIENTIFIC PRESS, of February 17, 1906.

In the following description of the milling system, I shall first take up the treatment of the oxidized ore, which is carried on by the original ten stamps and cyanide annex. As already explained, the mill was divided into two separate units for the purpose of treating custom ore. Very little custom ore, however, has been received; and recent favorable developments in the mine have decided the management to devote the entire mill to their own ore. In the following account, a complete five-stamp system will be considered. The scheme of treatment is exhibited diagramatically in Fig. 1.



Mill of the Combination Mines Co., at Goldfield, Nevada.

Inside amalgamation is carried on by means of a curved plate screwed to the chuck-block. The ore is crushed through 12-mesh wire screen. Outside the screen a splash-plate is used. From this plate the pulp falls to a lip-plate about 12 in. wide, with the front edge slightly bent down, giving the pulp a gentle drop to the apron-plate. There are three plates to each mortar, arranged in steps, giving an amalgamating surface 53 in. wide and about 12½ ft. long. The whole tray, by means of wheels and track, can be shifted during the clean-up of the battery, as shown in Fig. 2 and 3.

At the bottom of each tray is a small cone hydraulic classifier, which separates the coarse mill-pulp into two products: (1) Fine sand and slime, which passes to the outer discharge lip of the Bryan mill and thence direct to the concentrators; (2) coarse sand, which passes to the Bryan mill for re-grinding.

The ore being extremely hard and tough, is crushed with 1,350-lb. stamps, falling 100 times per minute, with a 6-in. drop. In spite of this, however, the stamp-duty is only 3½ tons, using a 12-mesh screen. One of the 5-ft. Bryan mills, running at half speed, takes all the coarse

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sand, from 10 stamps (approximately 20 tons per day), and crushes it through a No. 9 slotted screen, equivalent to 40 mesh. The final product from the Bryan is passed over two 6-ft. Frue vanners and two 6-ft. Triumph tables. From these concentrators the pulp is raised by two 54-in. Frenier sand-pumps to two sets of cone classifiers. This system is a modification of that introduced by Mr. Merrill at the Homestake cyanide plants. The top cone takes the intermittent discharge from the sand-pump and is so adjusted by valves that it sends a fairly uniform flow of pulp to the two smaller cones. The top cone is not a

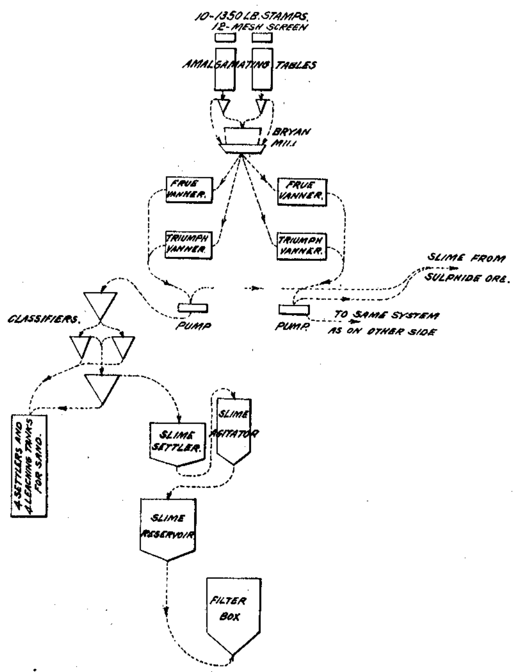

Fig. 1.—Treatment of Oxidized Ore.

classifier. The first rough classification is made in the small hydraulic cones, from which a stream of sand flows direct to the sand vats. The overflow from these small cones, consisting of slime and fine sand, flows to the larger lower cone, where a closer classification is made.

good (over 95%) since the introduction of the Butters-Cassel filter. A contemplated re-arrangement of the cones is expected to improve the extraction from slime.

In a small mill, classification requires constant attention. Slight interruptions, the suspension of one battery unit, or any variation from normal operating conditions, at once affect the flow of pulp in the classifiers, which are dependent upon nice adjustment for their efficiency. Obviously, the larger the mill, the smoother will be the operation of this system, and the less attention will it require.

From the cones, the sand flows to a pipe distributer and thence to a settling vat, of which there are four on each side of the mill. The fourth vat was added after the completion of the mill. These vats were at first provided with an overflow lip and a circular launder to carry off the surplus water. It was found, however, that occasional irregularities in classification resulted in the settlement of slime in these vats, which interfered with

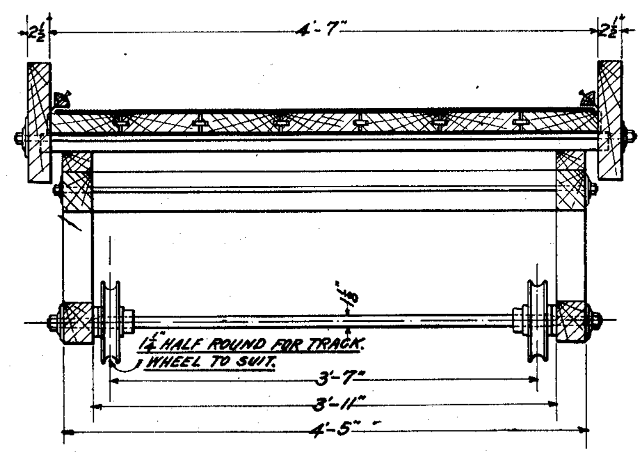

Fig. 2.—Cross-Section of Amalgamation Table.

percolation. The settlers were then fitted with slime-gates, and the overflow from the sand-settlers, carrying a certain amount of slime, now runs to a centrifugal pump at the lower end of the mill, to be sent to the slime-settlers for clarification. This is an awkward

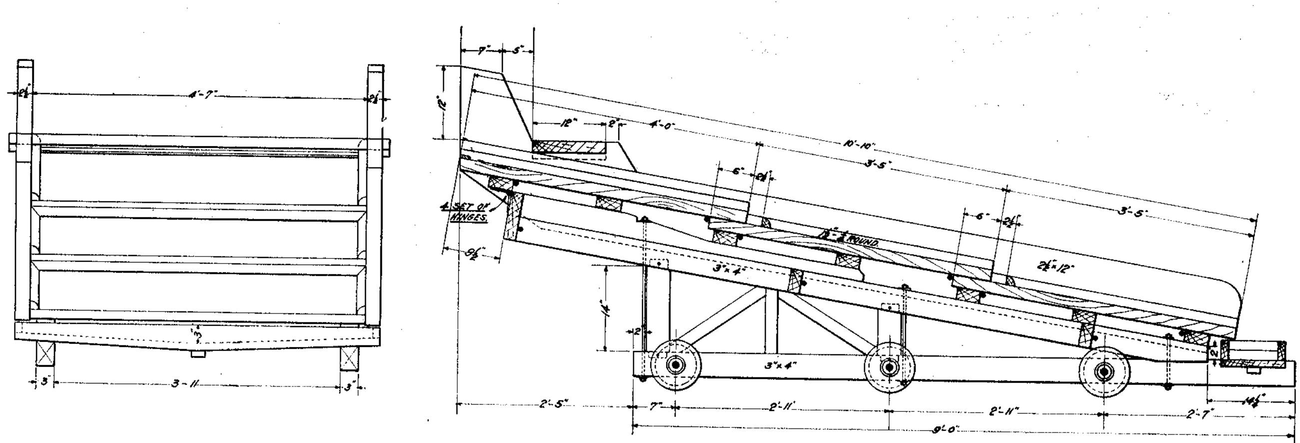

Fig. 3.—Movable Amalgamation Table Used in the Combination Mill.

The stream from the bottom of the latter also flows to the leaching vats. The overflow from this large lower cone passes to the slime settlers.

The pulp from the Bryan mill may, therefore, be said to consist of two products, namely:

1. Sand { Retained on 100 mesh............41% / Passing 100, retained on 200...35% / Passing 200 mesh..................24% } 67% of total.
2. Slime... Passing 200 mesh..................90%...33% of total.

This is not considered an ideal separation, inasmuch as the slime carries a large amount of fine sand. In spite of this, however, the recovery from the slime has been very

arrangement, but was unavoidable owing to the small gradient of the millsite.

When the sand settler is filled, the surplus moisture is removed by a Gould vacuum pump, and the charge shoveled to the treatment vat below. Here the charge is given an eight days' treatment with a 0.1% and a 0.2% cyanide solution, and the residue discharged by sluicing through a central bottom-discharge door.

The slime is delivered to the centre of a conical-bottom settler, provided with a rim overflow. There are two of these settlers on each side of the mill. Each is alternately allowed to fill and overflow for 12 hours, and allowed 12

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hours for settling. A pipe decanter carries off the surplus water, leaving the slime with about 50% moisture. Sufficient strong cyanide solution is added to the charge of slime to make a solution of from 0.15 to 0.2% cyanide, and to give the pulp a consistency of three parts solution to one of slime. By means of a centrifugal pump, the pulp is transferred to an agitator, with a steep cone-bottom. By means of valves, the same machine (a 3-in. Krogh slime-pump) is applied to the agitation, taking the slime from the bottom of the vat and throwing it back at the top. A supplementary agitation is given by means of a mechanical stirrer revolving slowly. The pulp and stirrer give an ideal agitation, being sufficiently complete for the best results. The pump, which has a lift of about 10 ft., is run at 375 rev. per min. Thus, at short intervals, the whole content of the vat passes through the pump, where it is aerated by means of a pet-cock on the suction pipe. The stirrer prevents the settlement of fine sand at the junction of the cone bottom with the staves, and keeps up the agitation during pump repairs. The only repair required in the pump is the replacement of the shaft about once per month. The objection urged against the centrifugal pump as an agitator—that it is apt to clog with slime during stoppages—is not a sound one. The 'slime' product at the Combination mill carries a large amount of fine sand, and the pump has repeatedly been started up after stoppages of several hours, without the least trouble.

After agitation, lasting from 12 to 18 hours, the pulp is discharged into a slime reservoir, a vat of large capacity provided with a mechanical stirrer from which it is drawn, as required, for filtration in the Butters-Cassel filter.

A filter-press was at first used for filtering the slime. This was an American machine consisting of fifty 42-in. plates and 2-in. distance frames, and had a capacity of 2½ tons of (dry) slime. It was evidently made of poor material, as it was the source of exasperating trouble through breakage. The plates were continually cracking under a pressure much below the guaranteed maximum, and the outlet-cocks getting out of order or breaking. Moreover, the operation of pressing, washing, and discharging was extremely slow, requiring about five hours, and the operating expense high, requiring two men on each of the three shifts at the prevailing laborer's wage of $4 per day, to say nothing of the cost of filter-cloths. It is fair to suppose that one of the high-class foreign presses of the Dehné type might have given better satisfaction. At best, however, filter-pressing is not to be compared with the system now in use, especially as regards cost of operating, and the completeness of the washing operation.

The essential points of difference between the Butters modification of the Cassel filter and the other vacuum-filtering schemes are the extreme simplicity in the design of the filter-leaf or frame, and the fact that these frames, throughout the operation, are always stationary. In the Combination plant there are 28 frames (5 by 10 ft.) set 4½ in. apart in a box 10 ft. square with a pointed bottom inclined at an angle of 50°. The slime-pulp is introduced at the point of the box, and a vacuum of 22 in. of mercury is maintained for about 20 min., during which time a cake is deposited on each side of the frame ¾ to 1 in. thick. The surplus pulp is then withdrawn to the slime reservoir and the wash introduced, consisting of a weak solution of cyanide. When the cakes are thoroughly washed, the weak solution is withdrawn into its proper vat, and water introduced, until the frames are completely immersed. The object of this final water is to assist in removing the cakes. More water is introduced into the interior of the frames under a low head. This causes the cakes to drop off clean, into the pointed bottom of the filter-box, whence they are finally removed by sluicing. The whole operation requires about 3½ hours, and about nine tons (dry) slime are treated at each charge. The plant, therefore, has a capacity of about 63 tons per day. It is operated by one man on each shift. The principal power required is for pumping the pulp and the various solutions in and out of the filter-box, and for operating the vacuum pump. In addition, the gold-bearing solution discharged from the vacuum pump is raised about 30 ft. and forced through the discarded filter-press now used for clarifying purposes. The whole consumes about 10 h.p. A 15-h.p. motor has been installed for this work, but has extra work to do not connected immediately with filtering.

The filter-plant has required no repairs since it was first operated in February of this year, and has worked in the most satisfactory manner. The cost of filtering (exclusive of power) has been reduced from 96c. per ton

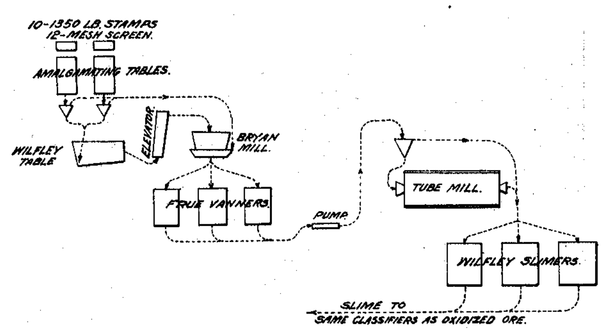

Fig. 4.—Treatment of Sulphide Ore.

of slime in January (using filter-press) to 26c. in May (using the Butters system). The power consumption appears to be about the same in the two processes.

Treatment of Sulphide Ore.—This ore is crushed to 12 mesh in two 5-stamp batteries, and run over plates to take up the small quantity of free gold present. See Fig. 4. A classifier at the end of the plate-tray removes the coarse sand and sulphide, which go to a Wilfley concentrator; the overflow of slime and fine sand passing to the outer lip of a Bryan mill. One Wilfley table, therefore, removes the coarse sulphide from the product of ten stamps crushing to 12 mesh. The tailing from the Wilfley is raised in a bucket-elevator to a Bryan mill, which crushes it to 40 mesh (No. 9 slotted screen). The tailing from the Bryan mill joins the stream of slime from the small classifier and passes to four 6-ft. Frue vanners. Here a large quantity of fine sulphide is removed. The Frue tailing is then elevated by a sand-pump to a classifier above a 4 by 12 ft. Abbé tube-mill of the trunnion type. The cone acts as a classifier as well as a de-watering device. The coarse sand passes to the tube-mill; the slime overflow joins the tailing from the tube-mill and goes to three Wilfley slime-tables of the latest pattern. The tailing from the last, consisting of slime and fine sand, is elevated to the cone-classifiers in the original mill, where it is mixed with the oxidized tailing and treated in the cyanide plant.

This plant was only operated a few days, and then shut down pending the installation of two Wilfley slime-tables, making three in all, the first having been set up experimentally. During this short run the results were very promising. Of the final product from the slime-table, 87% passed through 200 mesh, and the three stages of reduction showed a saving by concentration of over 80% of the contained gold. With the cyanide treatment of the slime tailing, a confirmation of the small mill-run is expected—namely, better than 90% recovery.

It is too soon to give the results of tube-mill work. The

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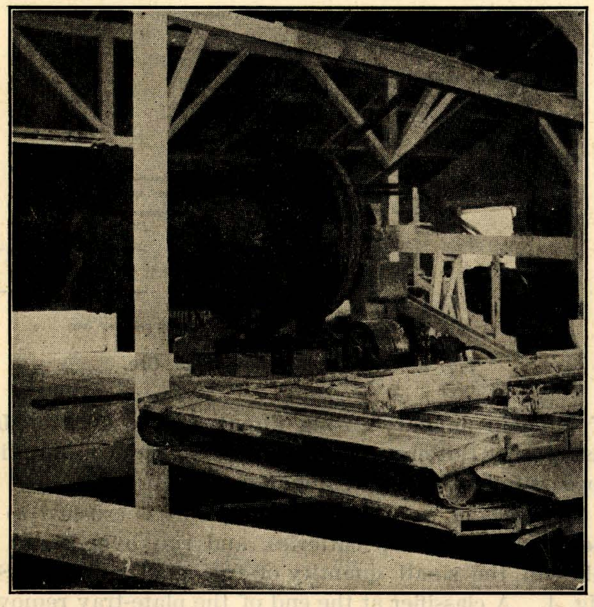
mill is lined with 2½-in. silex blocks, which will be replaced by blocks 4 in. thick.

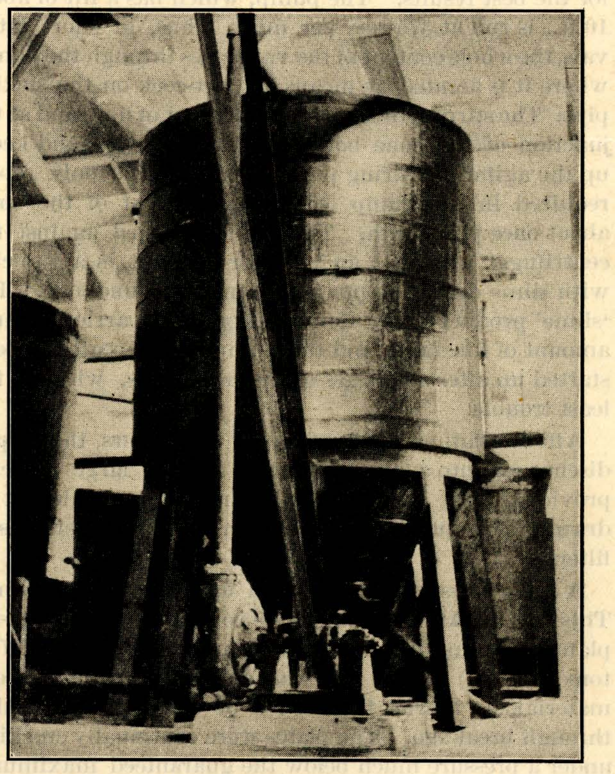
Precipitation is accomplished in the usual way by means of zinc shaving. The solutions are richer than are usually seen in cyanide mills, reaching as high as one ounce per ton in gold. Owing to the absence of silver, which undoubtedly facilitates the precipitation of gold, a very large zinc surface is required. The precipitate is refined with sulphuric acid and smelted in a pot-furnace, with gasoline, a powerful jet being maintained by means of a small upright Leyner compressor.

The following details of costs for milling and cyaniding indicate a few of the difficulties met with in an isolated district where the cost of supplies, and of labor, power, and water are unusually high. And then it must be borne in mind that the mill is of small capacity. The change from steam to electric power—the latter being furnished by a local concern operating a 90-mile transmission line—reduced the cost of power from a maximum of \$1.73 per ton (January, 1906) to \$0.76 (May, 1906). We have already noted the great reduction in cost effected by introducing a new filtering system for slime. The marked decrease in operating expense during February was due to increasing the capacity of the mill by annexing ten stamps, making twenty in all. While the total milling and cyaniding cost of \$5.828 per ton seems high, it is really not so when local conditions are considered. It is expected that further retrenchment will soon cut this down to \$5 per ton.

The recovery in the mill and cyanide plant has attained a maximum of 93% for several consecutive months, and has averaged over 90 per cent.



Tube-Mill and Wilfley Slime-Tables.



Slime-Agitation Vat.

CYANIDING COST PER TON OF ORE MILLED (1906).

Month.	Ore tons.	LABOR. Solution men.	Filter press.	Shovelers.	Tailing dam.	Butters slime filter.	Total labor.	SUPPLIES. Cyanide.	Zinc.	Acid.	Lime.	Filter cloths.	Other chemicals.	Miscellaneous.	Total supplies.	Total costs.
January	791.6	0.581	0.952	0.522			\$2.055	0.362	0.121	0.031	0.155	0.016	....		0.685	\$2.740
February	1,049.0	0.400	0.351	0.357	0.044	0.149	1.301	0.349	0.073	0.020	0.155	0.023	0.027	0.008	0.654	1.955
March	1,617.0	0.288	....	0.426	0.068	0.230	1.012	0.181	0.052	0.017	0.115		0.026	0.023	0.412	1.424
April	1,606.5	0.280	....	0.449	0.060	0.255	1.044	0.311	0.058	0.033	0.157	....		0.017	0.576	1.620
May	1,842.5	0.254	....	0.252	0.063	0.269	0.838	0.384	0.153	0.013	0.311	....	0.005	0.004	0.870	1.700

NOTE.—The average consumption of chemicals in treatment of sand and slime during 1906 has been as follows: Cyanide, 1.27 lb.; zinc, 0.6 lb.; and lime, 11 pounds.

MILLING AND CYANIDING COSTS PER TON OF ORE MILLED (1906).

Date.	Ore tons.	Mill superintendent.	Power.	Heating.	Water and pumping.	Crushing and elevating.	Stamping and amalgamating. Labor.	Stamping and amalgamating. Supplies.	Concentrating.	Cyaniding. Labor.	Cyaniding. Supplies.	Assaying and sampling. Labor.	Assaying and sampling. Supplies.	Lubricating.	Tools.	Lighting.	Maintenance and repairs. Labor.	Maintenance and repairs. Supplies.	General expense.	Total per ton. Labor.	Total per ton. Supplies.	Total costs.
January	791.6	0.303	1.734	0.795	1.263	0.303	0.592	0.241	0.597	2.055	0.685	0.378	0.003	0.097	0.090	0.009	0.442	0.679	0.002	\$4.670	\$5.598	\$10.268
February	1,049.0	0.214	1.193	0.194	0.953	0.221	0.401	0.345	0.409	1.301	0.655	0.287	0.009	0.035	0.035	0.010	0.278	0.275	0.002	3.111	3.706	6.817
March	1,617.0	0.151	1.311	......	0.618	0.294	0.335	0.227	0.310	1.012	0.412	0.268	0.005	0.035	0.012	0.047	0.849	0.998	0.136	3.385	3.536	6.921
April	1,606.5	0.125	0.875	......	0.622	0.291	0.377	0.139	0.317	1.044	0.576	0.332	0.012	0.040	0.020	0.027	0.753	1.520	0.152	3.351	3.871	7.222
May	1,842.5	0.130	0.763	......	0.271	0.271	0.327	0.399	0.283	0.858	0.870	0.312	0.004	0.026	0.036	......	0.782	0.430	0.068	3.022	2.806	5.828

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The Combination Mine.—II.

Methods of Mining.

Written for the MINING AND SCIENTIFIC PRESS
By EDGAR A. COLLINS.

Owing to the irregular shape of the orebodies, and their relatively great width, the problem of timbering the stopes merited a good deal of study. Good stulls are expensive and hard to get, anything over eight or nine feet having to be brought from the eastern slope of the Sierra, a distance of 200 miles. For this reason a stull 18 or 20 ft. long and 7 in. diam. at the small end will cost about $5. Similarly, short heavy round lagging is obtainable from the piñon pine on Montezuma mountain, about six miles distant, at a cost of 50c. apiece. Consequently, sawn lumber is used to a great extent in the mines. This is of poor quality and costs $45 to $50 per thousand, depending on the stock that the local yards are able to carry.

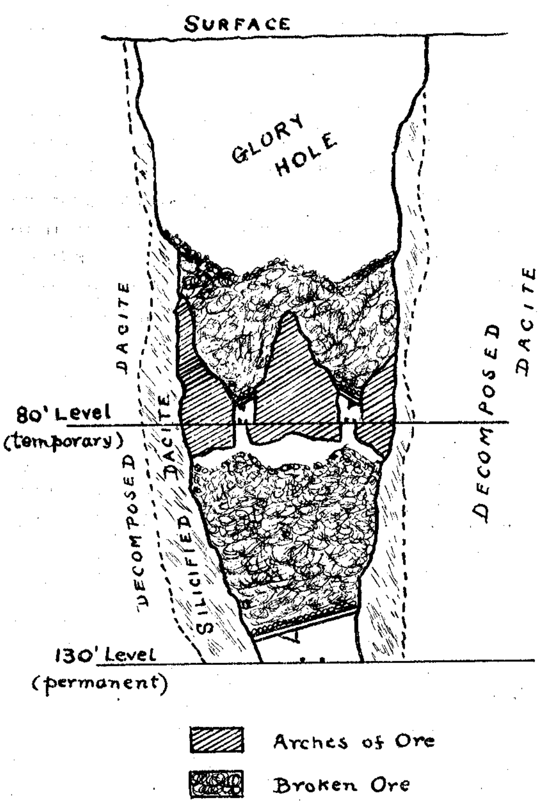

Fig. 1. Method of Stoping 'Old Shaft' Ore-Shoot.

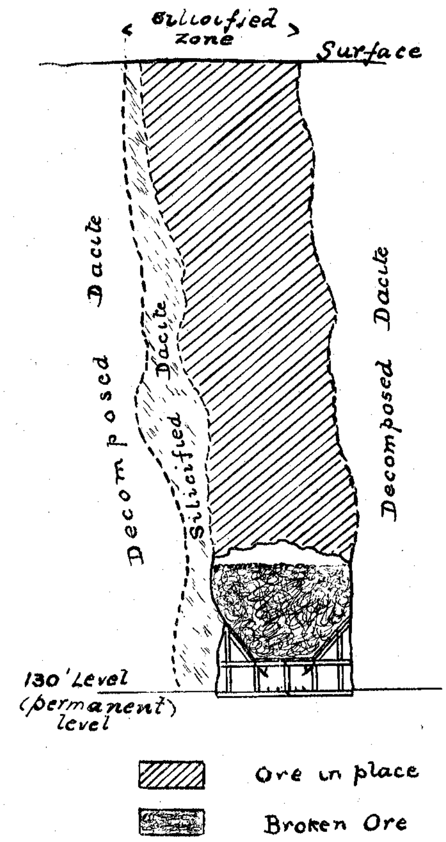

Fig. 2. Method of Stoping No. 3 East Stope.

In the upper levels of the Combination mine, where the ore is oxidized and the silicification especially pronounced, the ground is ideally easy to support. The silicified material is all wedged together and a minimum of timbers is necessary. It was therefore decided to stull and lag the cutting-out stopes over the level, and to fill with broken ore, as the stope progressed. As a further economy it was decided to timber permanently only alternate levels, and to leave arches on the intermediate levels that could be stoped out from the lower of each pair of levels. The following method was therefore established. At the first or 80-ft. level, after the drift was driven to the end of the ore-shoot, hitches were cut and short stulls were set in place, with the usual short head-boards. These were covered with lagging and a second stope commenced. At the same time the stope was widened on an incline above the timbers until the full width of the orebody was reached. The broken ore was allowed to remain in the stope, sufficient only being drawn off to allow room for the men to work. Substantial chutes were placed at every fourth set of timbers, being thus spaced with 20-ft. centres. At first it was thought that it might be necessary to carry the stope in floors and to place a line of stulls every 20 ft. or so in height, to steady the walls, and to act as a break in case difficulty was experienced in drawing the ore from the stope. It was found, however, that the ground stood so well, and so little difficulty was experienced at the chutes, that the idea of the floors was abandoned, and the block of ground was carried as one continuous stope.

As has been already remarked, the orebody had no definite walls, and the limit of the ore was simply the point at which the silicified material was too low-grade to pay. This limit varied in an extraordinary manner and it was found necessary to watch the work closely. As it was absolutely impossible to tell pay-ore from worthless vein-stuff by its appearance, recourse was had to continual panning, the gold being sufficiently free to act as a fairly reliable guide. Samples were therefore taken from every working face each morning, and panned by the mine foreman, who then made out a short report, which was turned in at the office. To further keep in touch with the ore, flat holes were drilled 6 ft. apart and 4 ft. deep, at intervals of 12 ft. in height, on both walls of the stope. The drillings were saved in two parts, those from the first two feet, and those from the second two feet, and were assayed as separate samples. If the results showed ore, the holes were blasted and the operation repeated; otherwise, the holes were left unblasted. As the stope progressed, the ore was found to extend farther and farther into the sides or walls of the stope, and eventually the stope was over 50 ft. wide at one end. By working the stope in sections, and by keeping the 'back' arched, no difficulty was encountered, nor was there a man hurt while the block of ground was being removed. Finally, a hole was made through the last 15 ft. or so to the surface, and the remainder of the arch of ground removed by underhand stoping.

On the second or permanent level (130-ft. depth) the

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whole of the pay-ore was taken out to the full width, and the level then timbered with either square sets or stulls, according to the width of the orebody. A second stope was then commenced, and continued, exactly in the same way as the first stope until within six feet of the level above (1st level). Raises were then put through the remaining arch to the level under each chute, after which tramming on the upper level was abandoned. The broken ore remaining above the upper level was run down into the lower stope until the arches over the level were reached. The arches were then stoped out as fast as they were exposed by drawing the broken ore, and at the same time any ore still remaining on the sides of the stope was removed. At the present time, March, 1907, the lower stope is full of broken ore and open quarrying is still going on around the sides. The 'glory hole' left is about 120 ft. long and will average about 30 ft. wide. After all the ore has been drawn out of both upper and lower stopes, it is proposed to fill the excavation above the 130-ft. level with waste from surface. This will be obtained partly from the present dump, and partly from future 'dead work.' It might be of interest to remark that the old incline shaft passed through the centre of both the first and second level stopes, and thus afforded excellent ventilation. A narrow stope on one of the high-grade seams of ore had also been taken for a short distance from the old incline shaft, through the middle of the block of ground. These 'old workings' caused us some little trouble when the big stope was commenced, and care was required to avoid 'caves,' but practically no timbering was necessary.

The above program of stoping was planned for the sulphide as well as the oxidized ore, down to the 230-ft. level, with the difference that in the stopes in sulphide ore, where the silicification was not so strong, and the ground therefore weaker, permanent timbers were placed at each level (50 ft. apart), and chutes were placed with centres only 15 ft. apart. It was planned to fill these stopes with waste from other parts of the mine, after the broken ore had been withdrawn. Below the 230-ft. level, the vein flattens in dip to about 38° and a different method of stoping would have to be employed. No definite plan had been decided on, but it is probable that it will be necessary to lower the ore on inclined movable tracks, laid on the foot-wall of the stopes, as is being done elsewhere, notably on the Rand in South Africa.

The Combination mine has 14 shafts and 7,100 ft. of levels. Only two of the shafts have ever been used for hoisting. The others are mainly prospecting shafts that were sunk to establish the position of the lode. Of the two working shafts, No. 1 was the original incline shaft, which constituted the main opening for the first two years of the mine's existence. It was sunk on the rich stringer that was exposed in the shallow 'tunnel' previously mentioned, and followed it from surface to the 230-ft. level. For the first 80 ft., the vein dipped slightly to the southwest. The next 50 ft. showed the streak almost vertical. From a depth of 130 ft. to 210 ft. the average dip was about 70° to the northeast. At the 210-ft. point the vein flattened to about 38° and the average dip between the 230 and 280-ft. levels is about 35°, also to the northeast. At the 280-ft. level, which is the lowest point at which the vein is exposed, the dip is about 30 to 35°. Below this level the lode has not yet been exposed. In June, 1905, a main vertical hoisting-shaft was started. The shaft, which was 9 by 5 ft. inside timbers, was divided into a hoisting-compartment of 4 ft. 8 in. by 4 ft., and a manway of 4 ft. 8 in. by 2 ft. 10 in. Between these a narrow compartment, 8 in. wide, and the full width of the shaft in length (4 ft. 8 in.), was divided into a box for the counterweight, and compartments for pipe and electric wires.

This shaft was sunk for the first 20 ft. only, the balance of the distance being raised from cross-cuts driven from the main levels of the mine. A small irregular shaped raise about 3 by 5 ft. was carried up from each level, and after connection was made, a chute was put in and the shaft trimmed to full size. In this way the shaft was sunk cheaply and expeditiously during the delay in arrival of the electric hoist, all broken rock being hoisted through the old shaft. The average cost of sinking this shaft to the 230-ft. level, including breaking, raising, mucking, timbering, and all supplies, was $22 per foot. The cost of the timber alone amounted to $9 per foot. From the 230-ft. level to the present bottom (400 ft.) the shaft was sunk by day work. Practically no water was encountered. The total cost per foot, exclusive of hoisting but including timber and all supplies and labor was $$23.12 per foot.

As a general rule development work in the lower levels was cheaper than in the oxidized zone. This was due to the less degree of silicification. Cross-cutting beyond the limits of the silicification is extremely cheap, the rock on the lowest level being much decomposed. For this reason it was not considered economical to use machine-drills in the development work. For the same reason the use of the diamond-drill for prospecting the adjoining country, as used in the mines of Tonopah, was not favored. The average cost per foot for driving, raising, sinking, and cross-cutting at the Combination, including breaking, supplies, track-laying, and timbering, where necessary, is as follows:

Average cost per foot driving and cross-cutting	$5.50 to $6.00
" " " " raising on lode	6.00 to 6.50
" " " " sinking winzes on lode	8.00
" " " " sinking prospecting shafts (practically no timber, and in soft rock down to 120 ft.)...	7.50 to 8.00
Cost of sinking main shaft, 9 by 5 ft. inside timbers, 400 ft. deep, including all labor and supplies, but no hoisting charges............	23.12 per foot
The cost of the timber alone............	6.50 per foot

There are four separate ore-shoots in the Combination mine. These are known respectively as the Old Shaft, No. 2 East, No. 4 West, and January shoots. The Old Shaft shoot has up to the present produced the bulk of the tonnage already treated. It is seen as the 'Glory Hole' on surface, and has been opened up on the 80-ft., 130-ft., and 180-ft. levels. On the 80-ft. level it is roughly 100 ft. long by 20 ft. wide, and is pretty thoroughly oxidized. On the 130-ft. level it is only about 75 ft. long by about 30 ft. wide, while the ore contains a larger percentage of sulphides. On the 180-ft. level, the ore-shoot is about 50 ft. long and 35 ft. wide where cross-cut. Here the general outline of the shoot appears to be triangular. Below this level the ore-shoot has not been found, and it may possibly be cut off, by the flatter-dipping No. 4 ore-shoot, to be described later.

The No. 2 East shoot is the longest continuous ore-shoot in the mine (about 320 ft.), and certainly one of the longest, if not the longest, in the district. For convenience it has been divided into two parts. The longer and more southerly portion, which follows the east, or hanging wall, side of the silicified zone, is known as the No. 2 East stope. This averages only three or four feet wide of pay-ore, bounded on the east side by a thin shell of almost barren silicified material, adjoining the soft bluish country rock, and on the west side by the main mass of the low-grade silicified zone. There is no defined line between ore and waste. The more northerly portion of this shoot extends into the silicified material for (as previously described) an average width of 20 ft. and a length of about 150 ft. This is known as No. 3 East stope. At the present time the whole shoot is exposed

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only on the 130-ft. level, where it is fairly well oxidized. On the 180-ft. level No. 3 East shoot has just been found and the ore is still well oxidized. It has not yet been cut on the 230-ft. level. The No. 2 East stope portion of this shoot has not yet been found on the 180-ft. level, but has been opened up on the 230-ft. level, for a distance of about 70 ft. At this point it is partially oxidized and of lower grade, but payable. Below this level the ore-shoot has not yet been found.

The No. 4 West shoot lies immediately west of the Old Shaft shoot, and parallel to it. It occurs in the same silicified mass, and is only separated by about 30 ft. of barren lode matter. It has been opened up on the 80-ft., 130-ft., and 180-ft. levels, and it is probable that the main ore-shoot, as exposed on the 230-ft. and 280-ft. levels, is a continuation of this shoot of ore, as it shows the same well defined flat dip to the northeast. On the 130-ft. level, some 300 ft. northwest of the Old Shaft shoot, on what is known as the January lode. This is the northeast extension of the horseshoe-shaped zone of silicification that enters the west side line of the Combination claim, about 350 ft. north of the main shaft. This ore-shoot occurs in the oxidized and silicified material, and is about 100 ft. long by about the width of the drift (5 ft.) The shoot in all probability extends practically to surface, a distance of about 100 ft., as the adjoining ground in the January mine has been stoped right up to the line, from this level to surface. A winze has been sunk some 47 ft. in ore, but otherwise the ore-shoot has not been developed below the 130-ft. level.

The following detailed mining costs, covering operations at the Combination mine during the year 1906, should prove interesting to those who may be contemplating the operation of mines in the district, and should

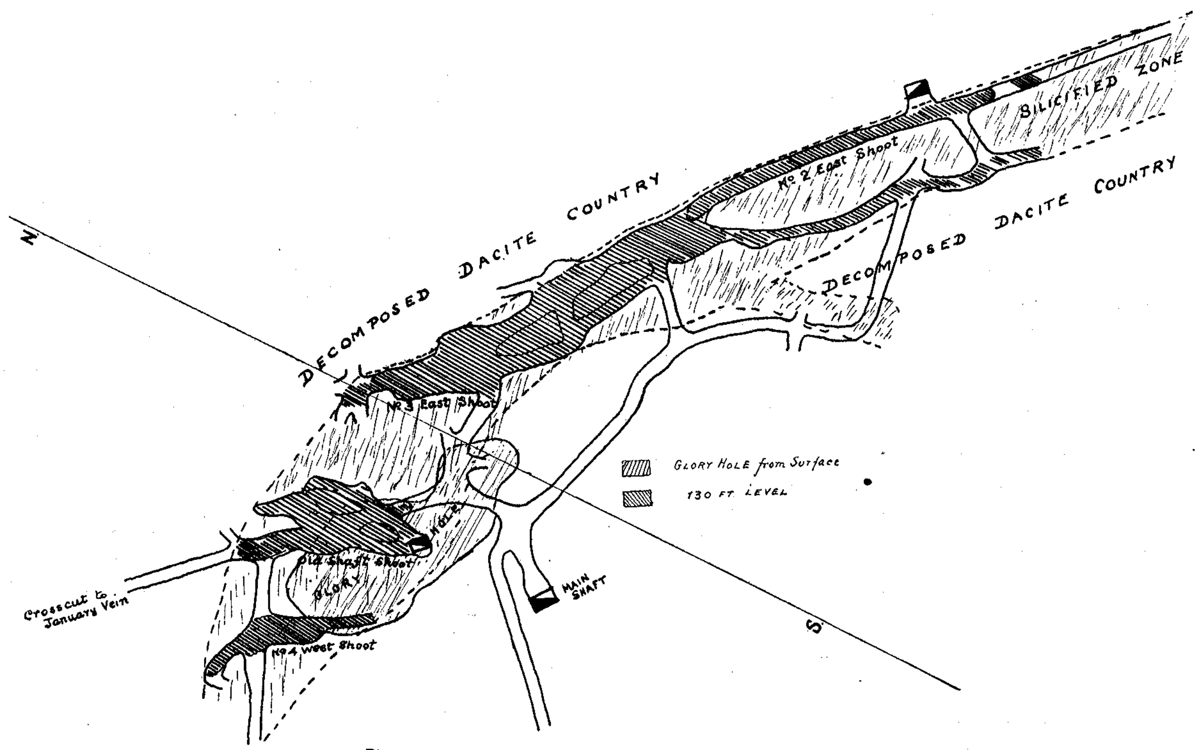

Fig. 3. Showing Ore Shoots on the 130-Ft. Level.

80-ft. level the ore-shoot is a lens of high-grade oxidized ore, about 60 ft. long, and about 15 ft. at the widest. On the 130-ft. level it is a little longer and better defined, but averages lower in grade. On the 180-ft. level, the orebody is entirely sulphide in character, and averages a higher grade, and has a length of about 100 ft. At the eastern end the lens narrows to a thin wedge, which is overlapped by a second longer ore-shoot. It is this second shoot that produced the large tonnage of high-grade sulphide ore shipped from the mine. It is a curious fact that this "all-sulphide" ore-shoot does not extend up to the 130-ft. level, although high-grade ore was stoped to within two feet of the track. On the 130-ft. level, the pay-streak is represented by a narrow clay 'gouge.' On the 230-ft. level the ore-shoot is about 225 ft. long, of which some 75 ft. was exceedingly high-grade. On the 280-ft. level the ore-shoot has been followed for a length of 100 ft. Below this level the vein has not yet been proved, but a cross-cut has been driven at the 380-ft. level to intercept the vein. At the time that I left the mine (March 1, 1907) this cross-cut had been driven a distance of 350 ft., and it was expected that the vein would be cut near the 400-ft. point. This was estimated on the supposition that the vein would continue with a dip of about 30° to the northeast.

The fourth shoot of ore, known as the January, occurs give a good idea of the comparative expense of the various items shown:

Mining Costs per Ton During Year 1906.

Foreman and shift-bosses	$0.199	Assaying	$0.162
Shaft sinking	0.305	Surveying	0.057
Driving and cross-cutting	0.940	Tracking and ditching	0.038
Raising	0.0165	Tools	0.045
Sinking winzes	0.029	Watchmen	0.118
Surface trenching	0.008	Maintenance, repairs	0.056
Stoping	0.913	Pumping	0.017
Hoisting and dumping	0 439	General	0.002
Tramming and mucking	0.365	Total	3.709

Mining.............. $2.41
Development.... 1.30

Total.............. $3.71 per ton. Of this 82% is for labor alone.

It will be noticed that in spite of the high price of labor —probably the highest paid in the world—and the heavy cost of all supplies, especially timbers, the expense of breaking ore and hoisting it to surface is not unusually high, even including development charges. The explanation of this is the fact that the ground at the Combination mine is full of soft shattered seams, which both drill and break exceptionally well, while the method of stoping employed requires comparatively little timber. Prospecting, outside the limits of the silicified lode, is cheap, because the rock is soft and decomposed. It must also be remembered that the mine is practically dry at its present depth, and the cost of pumping is therefore insignificant.

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The mill has been described in detail on several occasions, particularly by Mr. F. L. Bosqui, the consulting metallurgical engineer for the company, and it is therefore unnecessary for me to mention it further than to repeat that it is a 20-stamp mill, employing amalgamation, concentration, and cyanidation to treat the mixed oxidized and sulphide ores. Before cyaniding, the pulp is separated into sand and slime, the sand being treated by leaching, and the slime by the Butters vacuum-filter method. By mixing the ore from the various parts of the mine, the grade of ore milled is kept at about $40 or $50 per ton, the value being almost entirely in gold.

Record Sinking.

The work done in the two shafts of the Brackpan Mines Co., at Johannesburg, furnishes a new record for rapid sinking. In July the No. 2 shaft was sunk 204 ft., which is one foot better than the record for vertical shaft-sinking, made by Leslie Simson at the Simmer West in 1898. At the Brackpan No. 2 shaft, 210 ft. of timbering was placed in position during July. In the No. 1 shaft the same company has averaged 143 ft. of sinking during six months, and an average of 161 ft. during the three months ending with June. This shaft is of the seven-compartment type, and is 8 ft. wide by 39 ft. long. The Simmer West shaft has five compartments, and measures 7 ft. 6 in. by 31 feet.

Since the beginning of the year the Brackpan Mines record has been:

	Sunk. Ft.	Timbered. Ft.
No. 1 Shaft.		
January	98	83
February	131	127
March	143	158
April	148	135
May	186	196
June	150	143
July	125	135
No. 2 Shaft.		
January	77	31
February	49	57
March	100	76
April	125	135
May	152	166
June	170	158
July	204	210

The following additional particulars will be of interest:

	No. 1 Shaft.	No. 2 Shaft.
Started operations	May 12, '05.	June 1, '05.
Engines started	Sept. 16, '05.	Dec. 20, '05.
The depth then being	60 ft.	96 ft.
Depth on Aug. 6, '07	2,800 ft.	1,930 ft.

Bad ground and much water necessitated large pumps and heavy timbering in No. 2 shaft. This delayed progress from October, 1906, to April, 1907. Since then the average sinking has been 162.7 feet.

Credit for this good mining engineering is due to Charles B. Brodigan, the manager of the mine. Mr. Brodigan is an Associate of the Royal School of Mines.

COMMERCIAL asbestos includes fibrous minerals of two distinct types. The true asbestos is actinolite or tremolite and belongs to the amphibole group, and with it may be placed the other fibrous amphiboles, anthophyllite, and crocidolite. The more important asbestiform mineral, however, is the fibrous variety of serpentine known as chrysotile. Both fibrous amphibole and chrysotile possess qualities which peculiarly fit these minerals for use in the arts. The term asbestos, meaning noncombustible, thus has come to stand for mineral fiber which is more or less resistent to both heat and acids. Although the chrysotile, by reason of its chemical composition, may be affected by very high temperature and strong acids to a greater degree than the amphibole, the greater strength and flexibility of the chrysotile fiber make it the more valuable of the two.

The Prospector.

Enquiries sent to this department are answered free of charge, if submitted by subscribers who are not in arrears. The full name and post-office address of the sender must be given, otherwise no answer will be made. Those who are not subscribers must accompany their questions with a fee of $3 for each question. No assays are made.

Pearlite or Pitchstone was sent by C. B., of Reno, Nevada.

A specimen of Gneiss was sent by C. D. C., Quartzite, Arizona.

J. G. of Gem, Idaho, sends a piece of Quartzite, carrying some pyrite and galena.

The two specimens from Sisson, Cal., marked O. L. H., are: No. 1, serpentinized rock; No. 2, Rhyolite Tuff.

E. E. N., of Camacho, Mexico, sends: No. 1, Hornblende Gneiss; No. 2, Hornstone; No. 3, Quartzite; No. 4, Quartzite.

The specimen from Nogal, New Mexico, marked C. E. M., is rock containing disseminated grains and crystals of pyrite.

Specimens from the P. D. Co., Silver City, New Mexico, are: No. 1, yellow Ocher; No. 2, Diorite; No. 3, Feldspar Porphyry; No. 4, Quartz, with some Limonite and Malachite; No. 5, Feldspar Porphyry; No. 6, Epidote rock; No. 7, Quartzite; No. 8, Andesite.

The rocks from Sinaloa, Mexico, marked D. A. McD., are: No. 16, Andesite; No. 17, Quartz Porphyry; No. 18, Andesite; No. 19, Andesite; No. 20, Basalt; No. 21, altered Rhyolite or Andesite; No. 22, Andesite; No. 23, Andesite; No. 24, Rhyolite Porphyry; No. 25, Dacite, No. 26, Basalt; No. 27, Rhyolite; No. 28, Andesite; No. 29, massive Hematite.

VARNISH is a composition or an amalgamation of fossil resins, or gum, linseed oil, and spirits of turpentine, with a certain per cent of oxides worked in to serve as driers. This is varnish in general and applies to all kinds, the proportions and treatment varying according to the purposes for which the varnish is to be used. Gum, strictly speaking, is only properly applicable to those that are soluble in water, such as arabic, senegal, tragacanth, aloes, etc., all of which exude from live trees; but the gums of fossil resins used by the varnish-maker are petrified, only dissolve under great heat, and are the exudations from trees which are extinct. This resinous deposit is found buried in the earth, having laid there from a period before the creation of man. Different kinds are used for different products, the principal of which are Animi or Zanzibar, Sierre Leone, and Benguela; these are found in Africa and the East Indies. Kauri, however, is the resin most used; this comes from New Zealand, and began to be exported in quantity after the Australian gold discoveries. It is estimated since then up to 1904 $65,000,000 worth has been shipped, and in that year $52,509,085 worth was shipped to the United States and the United Kingdom. Linseed oil, the other important ingredient, needs no description. It is well known to be the expressed juice of flax seed. This is prepared by boiling, and different makers have different ideas as to the amount and manner of boiling prior to its introduction into the melted gum. Turpentine is used as a dilutant to render the treated gum and oil fluid enough to spread the varnish with a brush, and when the dilutant evaporates it has performed its mission and leaves the resultant film of gum and oil, or, in other words, it leaves all there is of what is essentially varnish.

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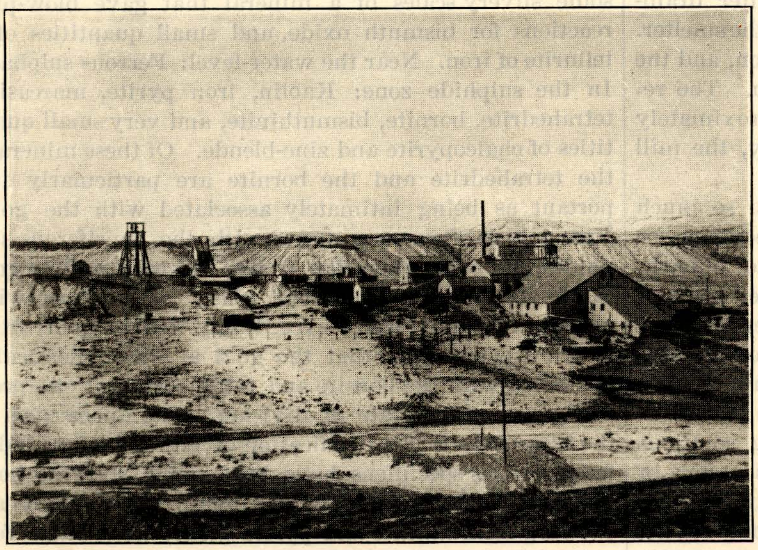
The Combination Mine.---I.

Early Development and Geologic Structure.

Written for the MINING AND SCIENTIFIC PRESS
By EDGAR A. COLLINS.

The Combination mine is situated in the Goldfield mining district, Esmeralda county, Nevada, about one-half mile in a northeasterly direction from the main street of the town of Goldfield.

The Combination No. 1 and No. 2 claims were located



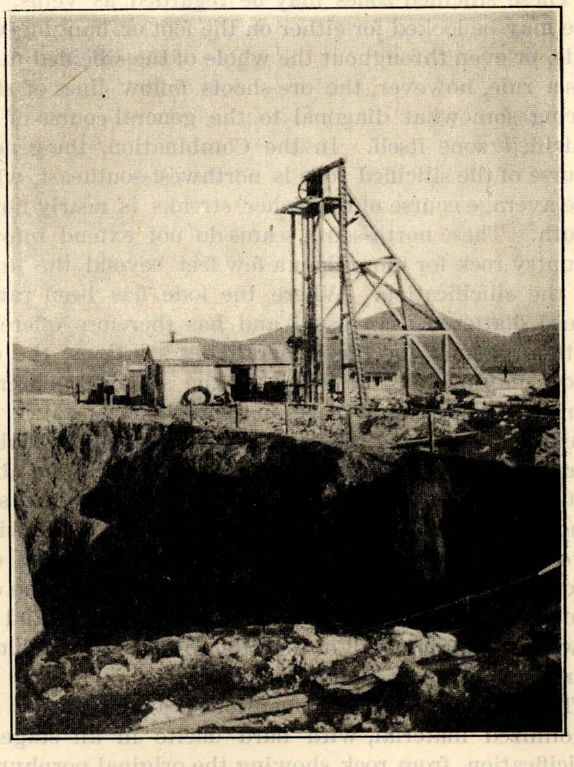
The Combination Mine and Mill.

in May, 1903, and the first discovery of mineral was made soon after. In October of the same year, the group (consisting of ten claims, aggregating 175 acres) was bonded by L. L. Patrick, acting for Arthur Winslow and J. D. Hubbard, trustees respectively for the United States & British Columbia Mining Co., and a syndicate of Chicago gentlemen. The amount of the bond was $75,000, of which $5,000 was a cash payment, and the balance spread over a period of twelve months. During the life of the bond, the purchasers had the privilege of working the property on a royalty of 25% of the net proceeds, with the understanding that the amounts paid in royalties were to apply on the purchase price of the property. As a matter of fact, the first cash payment of $5,000 was the only money actually required to finance the property, the balance of the payments being met from royalties, as they became due.

Work was commenced by Winslow and Hubbard in the latter part of October, 1903, at which time the workings consisted mainly of a shallow open-cut and drift or 'tunnel' about 90 ft. long, which exposed 30 ft. of shattered and oxidized quartz and kaolinized material. This was divided into two portions, namely, about 20 ft. in the open-cut, which, according to samples, averaged $20 per ton, and about 10 ft. near the surface of the drift, which averaged $150 per ton. The value was almost entirely in gold, which was bright and yellow, so that it made a good showing in the gold pan or horn. In addition to this cut and drift there were several other shallow cuts and pits, two of which exhibited a similar material, which sampled from $5 to $10 per ton, while others were location holes in soft decomposed and iron-stained rock.

At this time the town of Goldfield consisted of a few tents, two of which were stores, one was a so-called restaurant providing very inferior food, and three saloons. All were the result of the excitement occasioned by the bonding of the Combination and Jumbo groups of claims, and the spreading of reports of the richness of the ore. Outside of the individual prospecting which was being prosecuted at this time, the only work in progress was on these two properties, both destined to become important mines.

At the Combination, five or six men were put to work sinking a small shaft on the high-grade ore discovered in the drift, and sacking the ore broken in this manner. At the Jumbo claim on the opposite hill, John Harvey, acting for Patsy Clark of Spokane, took a bond on the property immediately after Patrick bonded the Combination, and commenced a small shaft on the hanging wall of the lode. After reaching a depth of 50 ft., he cross-cut the lode and drove a few feet in both directions. Finding the results of his sampling poor, Harvey stopped work, relinquished his bond, and by so doing just missed by a few feet one of the bonanza ore-shoots of the Goldfield district. Shipments of ore from the Combination commenced during November, 1903, and continued without interruption until the commencement of milling operations in May, 1905. The average gross value of the ore shipped during this period was $404 per ton, and it is an interesting fact that the first car of ore shipped from the mine, which averaged $160 per ton, was the lowest in grade that was ever shipped. At first the ore was hauled by teams to Candelaria, a distance of 65 miles by wagon-road, at a cost of $12.50 per ton; but with the advent of the railroad into Tonopah, the long haul was discontinued, and all ore was taken to Tonopah, and there loaded on the cars for shipment to the Selby smelter at Vallejo Junction, California. By shipping from Tonopah on the



Main Shaft and Glory Hole.

railroad, a considerable saving in time was made, but owing to the high freight-rates charged, the saving in cost of transport was practically nothing. After it was demonstrated that a satisfactory extraction could be obtained in the mill, all shipments of ore were discontinued, with the exception of a small amount of high-

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grade sulphide ore, which was, naturally, the most refractory in the mine. Excess freight-rates were charged by the railroad company on all ore assaying above $300 per ton. This excess rate amounted to 3% of all such excess. Thus on ore assaying $400 per ton, the excess freight rate was $3 per ton in addition to the regular freight charges. In consequence, the richest ore was sacked and stored pending the starting of the mill. This ore, amounting to 129 tons, was then run through a five-stamp battery, and over amalgamated plates, and the tailing then run to an empty leaching-vat, in which they were partially drained. After draining sufficiently, they were sacked and sent to the smelter. The original ore assayed 48 oz. gold per ton, and the sacked tailing assayed 18.3 oz. gold per ton. The recovery by amalgamation was therefore approximately 62%. The battery was run on day-shift only, the mill being locked and guarded at night.

This district is now so widely known, and so much has been written of its geology that it is unnecessary for me to go into the question further than to remark, briefly, that the lodes are shattered and fissured zones of silicification formed by hot solutions, under pressure from below, which, following the lines of least resistance, deposited their silica and the accompanying minerals in the zone of fracturing. It is probable that there were other subsequent flows of mineralizing solutions that to some extent concentrated the precious metals in the more fissured portions. In the main workings of the Combination mine, both country rock and lode consist of altered dacite; this is also the case in the Jumbo, Mohawk, and January mines, but in the main workings of the Florence mine, which is situated about 1,500 ft. distant on the southeast continuation of the same silicified zone or vein, the country rock in close proximity to the vein is an altered andesite.

These silicified zones may be regarded as veins, and ore may be looked for either on the foot or hanging-wall side, or even throughout the whole of the silicified mass. As a rule, however, the ore-shoots follow lines of shattering somewhat diagonal to the general course of the silicified zone itself. In the Combination, the general course of the silicified zone is northwest-southeast, while the average course of the richer streaks is nearly north-south. These north-south seams do not extend into the country rock for more than a few feet beyond the limits of the silicification. Where the lode has been rather more shattered than usual and has therefore offered a better channel for the mineralizing solutions, the ore-shoot may include the whole of the silicified material. This occurs in several places in the mine, for instance, immediately adjoining the point of discovery in the shallow 'tunnel,' the ore-shoot is from 30 to 50 ft. wide and about 120 ft. long. Again, on the second level in the split known as the East vein, the silicified zone for a width of over 20 ft. and a length of 150 ft., is one large ore-shoot. Continuing in a southeasterly direction the ore-shoot narrows to a width of four feet of pay-ore and follows the hanging wall of the lode, so that the soft bluish porphyry forms its hanging wall also.

The ore is, as already mentioned, a mixture of soft kaolinized material, with hard dacite in all stages of silicification, from rock showing the original porphyritic structure to dense flinty quartz. The best ore is found in the small stringers and veinlets, already mentioned, that traverse the main body of the lode. In the oxidized portions of the mine—above the 130-ft. level—these are largely filled with small fragments of quartz mixed with a characteristic yellow or reddish ochery material, which is mainly kaolin formed by the decomposition of the feldspar in the original rock. When the sulphide zone is reached, the rich stringers in the lode are easily followed, as the shattered clayey material is heavily impregnated with sulphides, forming a well-marked dark seam. The richest ore in the mine is usually found on the 'faces' forming the sides of these points, where the gold-bearing solutions have had the best opportunity to precipitate.

The following minerals have been observed in the Combination mine, the names being quoted in the order of their importance. In the oxidized zone: Quartz, kaolin, iron pyrite, gypsum, hydrous ferric oxide, alum, some silvery scales of a mineral that gave blow-pipe reactions for bismuth oxide, and small quantities of a tellurite of iron. Near the water-level: Ferrous sulphate. In the sulphide zone: Kaolin, iron pyrite, marcasite, tetrahedrite, bornite, bismuthinite, and very small quantities of chalcopyrite and zinc-blende. Of these minerals, the tetrahedrite and the bornite are particularly important as being intimately associated with the gold. Free gold is often seen mixed with the cupriferous sulphides, but even when this is not the case, and no gold can be seen under a powerful glass, the specimen will, almost invariably, assay exceedingly rich in gold; from which it is presumed that the gold is either chemically combined with the bornite and tetrahedrite, or else present in a very fine state of division. The appearance of the bismuthinite is no indication of either good or indifferent ore, as specimens have been found showing both free gold and tetrahedrite, and again other specimens, showing bismuthinite and pyrite alone, assay but a few dollars per ton. Similarly in the oxidized zone, no particular change in the value of the ore is noticed when the silvery specks of bismuth oxide appear. The ferrous sulphate is partly mixed with alum and was found on the 130-ft. level as plates with a bluish green color filling the joints of the soft rock forming the hanging wall of the lode. Great care was taken to preserve some of these specimens, but although placed in a tight tin vessel almost immediately, they gradually lost their color, and became dehydrated.

A section of the Combination lode shows the following peculiarities distinctly:

1. Silicification is greatest on surface and is less marked as each succeeding level is reached.

2. Gradual flattening of the vein on its dip, which reaches an angle of 30 to 35° northwest.

The first of these might well be expected in the case of deposits formed by thermal waters followed by only partial erosion. The second, however, is far more of a problem, and it is one that has yet to be studied and satisfactorily explained. The flattening of the vein commenced in the shaft just about the level at which water was first struck—210 ft. At this depth a strong seam came in from the foot-wall side of the shaft, and the ore turned with the wall. Some very rich ore was encountered in the shaft where the splice was made. The greater part of this consisted of partially rounded small boulders of flinty quartz, on the outside of which were concentric layers of the following minerals, in the order enumerated, beginning from the outside: A layer of either marcasite or iron pyrite in botryoidal form; next a layer of quartz with specks of a grayish black mineral, which was later determined as a form of tetrahedrite; and inside this a layer of rusty gold about $\frac{1}{8}$ in. thick. In all the specimens found this order of deposition was observed, and in other places in the mine at which faces of similar specimen ore were found the order was identical. It would therefore seem reasonably certain that the free gold was one of the first minerals deposited.

Very rich ore was found in this mine clear from 'grass roots' down to the bottom or 280-ft. level. Taking it

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altogether, however, the richest ore was found between the 180 and 230-ft. levels. Here the vein dipped at an angle of about 45° and consisted of a narrow 'gouge' or selvage of kaolinized material with rounded fragments of quartz heavily impregnated with sulphides. Immediately below this came several inches of the silicified dacite containing stringers and faces of free gold and rich cupriferous sulphide. In many places the rich ore consisted of a rich 'face' only—the rest of the material being low-grade. Underneath this rich seam, the value of the ore dropped immediately to low-grade ore, and sometimes to waste, there being no definite line between them.

As might be expected with such high-grade ore, stealing, or 'high-grading,' as it is generally called, flourishes to an extent unknown in the older mining districts. This is especially true in the case of the bonanza leases, which are worked under high pressure, and where anything in the shape of a delay means the loss of a considerable amount of money. The loss to the mine-owners from 'high-grading' has been estimated at several hundred thousand dollars. This is probably in excess of the true amount, but it must certainly have been very large. Many of the miners that have become expert in the business while working in the mines of Cripple Creek, obtained work in the Goldfield mines, solely for the purpose of stealing ore. These men used to wear a regular harness, to enable them to carry off 40 to 70 lb. of ore on each shift.

In the oxidized portions of the Combination mine, where the richest ore cannot be distinguished by its appearance, panning with drilling water serves to guide the 'high-grader,' and men have been known to 'horn' from 20 to 30 times in a shift. Needless to say, a great deal depends upon the honesty and fearlessness of the mine foreman.

Following the 'high-grader' came many so-called assayers, who ran assay-offices as 'blinds.' Until comparatively recently, it was impossible to do anything to stop ore-stealing. The Miner's Union would not allow change-rooms, and it was impossible to secure the conviction of a man for theft. Since the strike of last December, however, change-rooms have been built at all of the principal mines and several convictions have been secured. This will undoubtedly tend to restrict the wholesale theft of ore.

MINING IN COSTA RICA.—According to a report the following mines have been milling ore during 1906: The Abangarez Goldfields of Costa Rica; the Esperanza Mining Co. (late Boston mine) in the Abangarez district; and the Colburn Mining Co. at Pozo Azul, near Chomes. Development work is proceeding at the Montezuma mine, in the Barranca district, and the Machuca mine, in the Aguacate district. The value of the gold and silver exported was £110,645. The average rate of wages paid to day laborers in the interior has gone up, and now stands at 2s. 4.56d., and in the Limon province, on banana farms and railway work, at 4s. 4.80d. The wages of artisans of all classes have also gone up 20%. This does not, however, indicate any general advance in prosperity among the working classes, since the cost of living has also increased.

TOURMALINE IN CALIFORNIA.—The colored tourmalines of the Pala district, in San Diego county, are well known. The principal varieties are rich, deep-red rubellite, from the Pala Chief mine, and various colored tourmalines, though mainly pink rubellite, from the San Diego Co.'s property, at Mesa Grande.

Decisions Relating to Mining.

Specially Reported for the MINING AND SCIENTIFIC PRESS.

In case of unavoidable shortage of cars, a railroad company may distribute such cars as it has to the several mines on its line of road on a percentage basis, calculated on the production of the several mines. The mine owners are only entitled to their proper percentage of available cars; but such distribution cannot interfere with the right of individual owners to the exclusive use of their own cars. And the railroad company may allot an arbitrary number of cars for development to new mines which have no basis for a percentage.

United States *v.* Baltimore &c. R. Co., 154 Fed. 108. (June, '07.)

A trustee operating a mine under a lease for a term of 50 years, with the right to terminate the lease and remove the machinery if the enterprise should prove unprofitable, was held not liable to a beneficiary for surrendering the lease after prosecuting the enterprise for more than 10 years unsuccessfully.

Mexican Nat. Coal &c. Co. *v.* Frank, 154 Fed. 217. (April, '07.)

Where stockholders of a mining corporation applied to a court of equity for protection of their rights against the alleged wrongful acts of the directors, they could not recover for failing to act with reasonable diligence, or present some excuse for not having done so.

Jones *v.* Bonanza Min. &c. Co., (Utah) 91 Pac. 273. (July, '07.)

In an action for the possession of an unpatented mining claim, under a complaint alleging ownership, possession, and right of possession at a specified date and the ouster by defendant of plaintiff, the defendant may show, under a general denial, that the deed under which the plaintiff claimed title was invalid.

Holmes *v.* Salamanca Gold Min. &c. Co., (Cal.) 91 Pac. 160. (June, '07.)

Where a miner was injured by a fall of rock from the roof of a mine in which he was working, it is proper to show, in an action for damages, that the mine was inherently dangerous, and that skillful operators should have known of such condition and taken precaution against it. But it must appear that the conditions at the time of the accident were the same as those at the time to which the testimony related.

Arris *v.* Standard Plaster Co., 105 N. Y. Sup. 440. (July, '07.)

A mining corporation may issue stock to an attorney in payment for services rendered, and in the absence of fraud, it cannot be questioned by a stockholder.

Bogeler *v.* Punch, (Mo.) 103 S. W. 1001. (June, '07.)

A mining company has the right to use a stream of water for mining purposes, but it cannot pollute such stream to the injury of a lower land-owner.

Alabama &c. Coal Co. *v.* Vines (Ala.) 44 So. 377. (June, '07.)

A lease of land for the sole purpose of operating for oil, is not binding on the lessee; and on failure of the lessor to operate for oil, or pay the sum agreed upon in advance, he forfeits all rights under the lease.

Jennings-Haywood Syndicate *v.* Houssiere-Latreille Co., (La.) 44 So. 481. (June, '07.)

The owner of the legal title to coal underlying a tract of land owned by another is not required to exercise particular acts of ownership over the coal in order to retain title thereto, and, to be divested of his title by adverse possession, the possession must have been continuous and adverse for the statutory period.

Gordon *v.* Park, (Mo.) 100 S. W. 621. (March, '07.)

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GOLDFIELD, NEVADA.—I.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

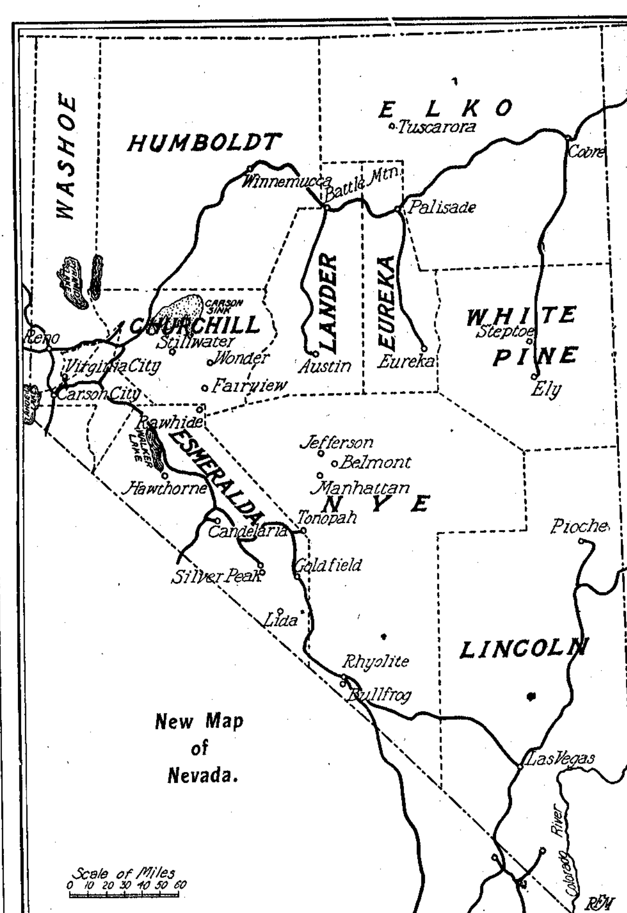
Goldfield has been twice christened. In its very infancy the camp was called Grandpa; this savored of frivolity, so the name was changed to Goldfield. It must have been a spirit of prophetic presumption that led the first discoverers to arrogate for the locality the name of The Goldfield, as if it were the one and only great gold-producing district of the world. But Fate and some fostering star have justified the baptism of the pioneers. Goldfield *is* today *the* goldfield of America.

When Harry Stimler and his partner called the place Grandpa they did not claim that it was the parent of the Mother Lode of California; it was a play on such names as Tonopah, Ivanpah, and Ibapah. These are of Indian origin; *pah* means water. Tonopah is a corruption of Tonumba-pah, which stands for 'water brush.' Iba-pah is red water and Ivan-pah is white water. Thus the Ivanpah range and the Witwatersrand are synonyms, for both mean the 'white water range,' and they sound as unlike as their sponsors, the Indian of the Nevada desert and the Dutchman of the African veldt.

Tonopah not only suggested the first name of the district, it was the parent of Goldfield. When I was in that part of Nevada in the spring to 1899, to make an examination of a copper deposit in the range east of Luning, all this part of Nevada was dead, from an industrial standpoint. Only the decadent glories of the Comstock reminded the traveler that he was in a mineral region once so famous. At Candelaria there was a little mining, and the stage road to Bodie from Hawthorne, up the canyon that led across the State line into California, recalled the excitement of a former era. The country had been abandoned by the prospector. It was one of the has-beens of history. The old trails still left faint lines across the sagebrush desert and over the bare rocky ridges and spoke of a period of strenuous search for gold and silver. Some of those abandoned paths of exploration went close to the places where rich mines have been discovered. The road from the north went close to the present site of Manhattan; in fact, there was an old camp within four miles of the one that sprang into existence in January, 1906. The old road to Montezuma, along which prospectors had traveled for fully 35 years, traversed the big flat east of Lone Mtn., and was within a few miles of both Tonopah and Goldfield. There was a trail that passed within a stone's throw of the Mizpah outcrop, which beckoned the miner for many a long day before Jim Butler thought of testing the black rock.

Like navigators that pass a treasure island in a fog and hear the muffled roar of the breakers only as a warning of danger, so these old prospectors went past the hungry-looking outcrops of silicified rock, never dreaming that they capped the wealth they were seeking with such invincible restlessness.

The first discovery in this region was made at Southern Klondike, a place half-way between Tonopah and Goldfield, which are 24 miles apart in a straight line. This first discovery was made by James Courts in 1898. The name suggests contemporaneity with the great rush to the Yukon. The outcrop of the Tonopah lode was plainly visible to those who went near it; in fact, Courts says that he and his partner used to go over the trail through the Mizpah pass and they employed the cropping of the Mizpah (or Tonopah) vein to shelter their campfire from the wind. At that time Jim Butler was at Belmont. In 1900 he made a trip from Belmont to Southern Klondike, and on his return he tried to make a short cut and so happened to tramp over the site of Tonopah. He saw the outcrop and picked up some pieces of the broken vein, taking them to Southern Klondike, where there was an assay-office run by on old man named Frank Hicks. This assayer threw the samples away, saying that he would not give a



New Map of Nevada.

dollar for 1000 tons of such stuff. Butler jokingly offered him a quarter interest for an assay. The ore was full of the black sulphide of silver (argentite) and anyone familiar with silver ores would have known that it was tremendously rich. These gold prospectors were ignorant concerning the other precious metal, so they blundered like tenderfeet.

On his way out Butler got some more of the black mineral and took it to Belmont, where he gave it to T. L. Oddie, who happened to be there in connection with the operation of a quicksilver mine. Oddie was an educated man and was then the District Attorney of Nye county; he lived at Austin, where he had been for several years as representative of the Stokes interests. Oddie sent the specimens to Walter C. Gayhart, an assayer at Austin. Butler promised Oddie a quarter interest, and Oddie promised Gayhart a half of his quarter—all for the making of an assay! Money was scarce; they were all of them 'busted.'

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Gayhart's assays showed that the selected specimens ran from $300 to $400 per ton. Oddie thereupon sent an Indian runner to Butler's ranch, which was 45 miles from Belmont, to tell him of the strike; but Butler remained inactive. He was reputed to be "the laziest white man in the world." Even his location monuments are indicative, for they are built of seven or eight stones at the most. He never did a stroke more than necessary, if as much. Meanwhile Gayhart spoke publicly of the discovery and several men left Southern Klondike with the idea of finding it; but they failed. Three months after the assay had been made, Butler went to Belmont; he arranged with Oddie and Wilse Brougher (the recorder for Nye county) to make a trip to the scene of discovery. They gathered the necessary supplies at a cost of $25, which was about all they could scrape together. This was in August, 1900. On arrival at the spot, they located seven claims and then started to work. A small shipment was taken away in their spring wagon. This lot of ore was selected stuff and weighed slightly more than a ton; it was hauled all the way to Eureka, 170 miles distant. The partners received $600 as the result of that first shipment. Thereupon, in October, they started to lease the ground they had located, dividing their claims into blocks of 100 ft. square, to be mined on a royalty of 25 per cent.

When Hicks, the assayer at Southern Klondike, heard of the find, he picked up the specimens he had thrown away and assayed them. Then he claimed a quarter interest, and the matter was finally adjusted by Butler donating him a $^1/_{32}$ interest. He had no legal, nor even moral, right to share in the discovery. Butler's action was generous—generous as the spirit of the West, and of the pioneer the world over.

On June 3, 1901, only seven months after work had begun, the property was bonded by O. A. Turner for $337,000, of which $50,000 was paid at once and distributed, according to their holdings, to the five owners, namely, Oddie, Butler, Brougher, Gayhart, and Hicks. Turner drew at sight on his friend and associate John W. Woodside, of Philadelphia, a snuff manufacturer. In this deal Turner was assisted by J. H. Anderson, who became the first president of the Tonopah Mining Company of Nevada, which was organized with a capital of $1,300,000, of which $1,000,000 was in common shares of $1 each and $300,000 was preferred stock carrying 8%, the latter being subject to recall at any time. One share of preferred entitled the subscriber to two shares of common stock. The common stock rose eventually to $22.50, so that several fortunes were made and lost before it fell to its present price of $8. The mine never needed any working capital, it was a profitable enterprise from the very start. Before the leases expired on the last day of 1901 they yielded $3,000,000 worth of ore. Up to the end of 1903 the mine yielded $5,576,000, of ore averaging $160 per ton. In the following January the late William C. Whitney, the New York traction promoter, bought Woodside's 280,000 shares for $7, and a few months earlier John Anderson sold his smaller holding to a group of Philadelphia men headed by John W. Brock and W. L. Elkins, who was a director at the time of his death. Brock retained his shares and became president, retiring a few months ago. In order to diminish operating expenses, a railroad 60 miles long was built by the Tonopah Mining Co., so as to connect the mine with the Carson & Colorado railroad, a narrow gauge line running south from Reno. Construction was begun in January and the road was completed in May, 1904. In 1905 the railroad gauge was widened to the standard, so as to correspond with a similar change on the branch of the Southern Pacific, with which it connected at Sodaville. This railroad enterprise proved highly profitable for a time and was managed so as to be more remunerative than the mine itself. It aided the development of Goldfield, to which we now return.

By his share in the Tonopah mine, Jim Butler thus became rich and the patron of other prospectors. At the end of 1902 he 'grubstaked' or outfitted Harry Stimler (a half-breed) and William Marsh, two young fellows of Tonopah about 20 years of age. They went south and found rich float on what is now Columbia mountain, which overlooks the site of Goldfield. After searching in vain for the source of the rich float, they prospected northward and found gold in the surface dirt. Thereupon, the Sandstorm was located on December 2, the name of the claim suggesting the state of the weather at the time. The Kendall was located immediately afterward. In both cases free gold was found in the débris of disintegrated rhyolite and this led to a brief excitement, but no large orebodies were uncovered and the interest of the neighboring prospectors soon waned. In the spring following a party of Tonopah men sent a couple of prospectors to the new camp, then called Grandpa, as already stated. The prospectors were A. D. Myers and Tom Murphy. They located the Combination on May 24, 1903. Here let me relate a typical tale of hard luck. R. L. Johns, a lawyer then residing at Tonopah, happened to go into a store and was told by the proprietor that a pool or syndicate was being made for the purpose of sending a couple of prospectors to Grandpa. Would he take a share? It would cost $50 at once, with $50 more to be paid later. Johns said he guessed he would, and later in the day he returned with the money, only to be told that he was too late, another man having subscribed for the share. Johns did not bother about it, until he ascertained that the other fellow had subscribed his $50 with the $50 Johns had himself lent the other fellow the day before. So the reader can imagine the feelings of Mr. Johns when the prospectors found a good lode, which became the basis for a big mine, and one-twelfth of it had been secured by the $50 lent to another man who had jumped in and subscribed for the share our legal friend had expected to take.

At the time the Combination claims were located, no ore had been found. Tom Murphy, who was one of the discoverers, says that he located the Combination because it was the vacant territory nearest the Sandstorm; all the ground south from the latter had been located. Moreover, there was a large outcrop of quartz rock on the Combination, but that did not

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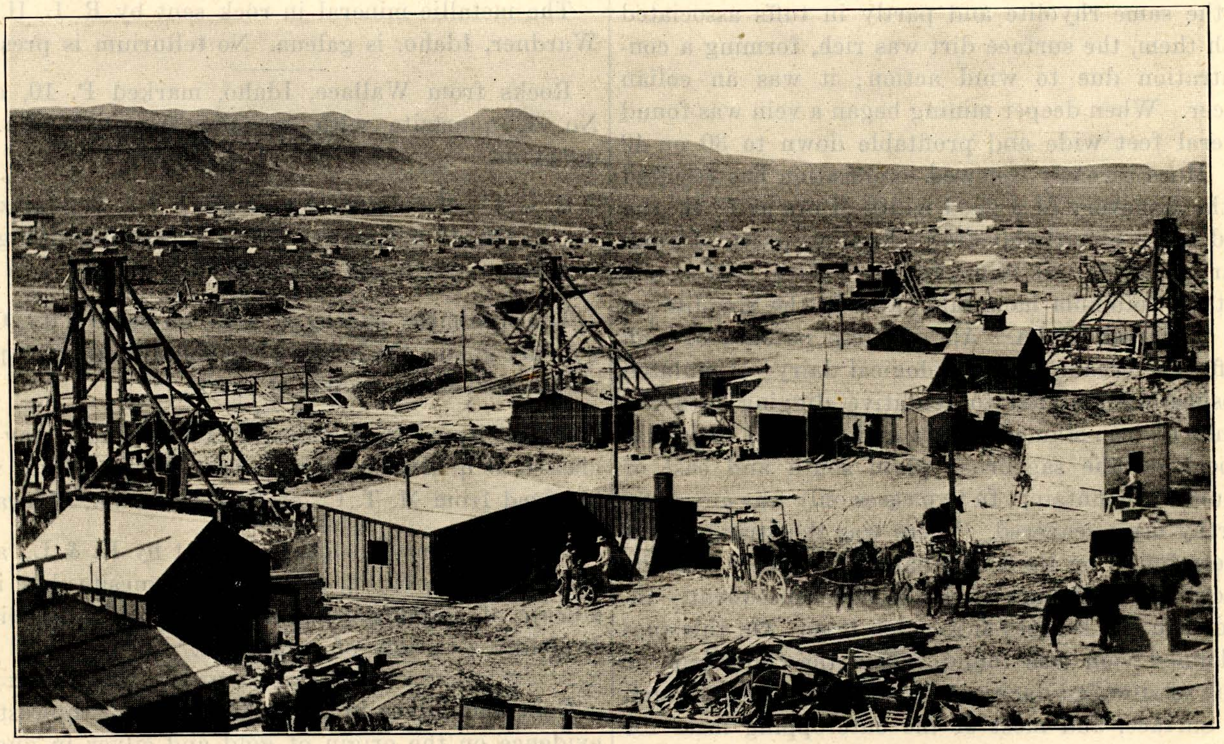
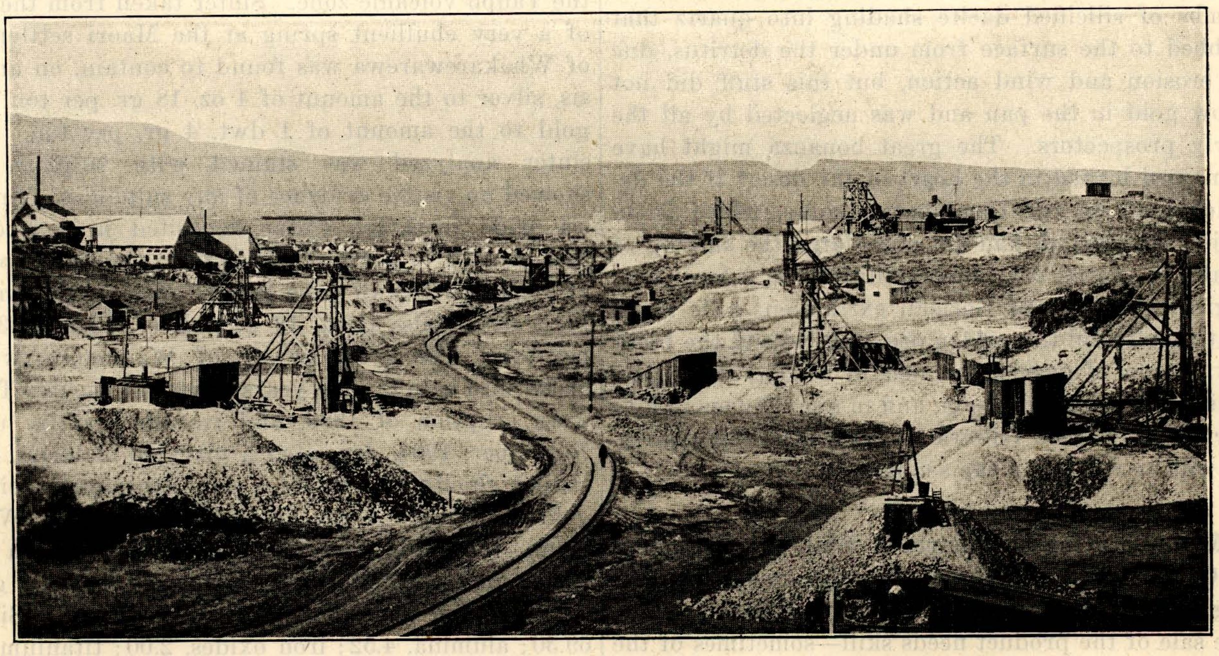
mean much, for there were many similar combs of silicified material rising above the desert all around Columbia mountain. Myers, who was Murphy's partner, made a cut near the top of the hillock because there the detritus was thin; he found a stringer carrying gold in the midst of a mass of barren quartz. This stringer panned nicely and assayed as high as

Arthur Winslow and J. D. Hubbard, who were trustees for two exploration companies. The ground was bonded for $75,000, of which $5000 was cash. This $5000 was the only money ever required, for the remaining payments were paid, as they became due, out of royalties on ore mined by successful lessees. On March 1, 1907, the Combination mine was sold



North end of Town. Frances Mohawk. Lone Mtn. Hayes-Monette Lease. Nevada-Goldfield Reduction Works. Mackenzie No. 1.



Combination Mill. Combination Fraction. Baby Florence No. 2. Railroad to Bullfrog. Mohawk. Combination Fraction. Winston Lease.

$83 per ton. Then he went down the side of the ridge and drove a short tunnel, which cut the lode at 20 ft. below the surface. He found good ore. This led to the big orebody now marked by the cavernous pit named the 'glory hole.' In the following October the Combination No. 1 and 2 claims and eight other locations were bonded by L. L. Patrick, acting for

to the Goldfield Consolidated Mines Co. for $4,000,000. In the previous 25 months the Combination had yielded $786,000 in profits to the owners, besides enriching many lessees. So ended a most profitable mining adventure; and it is pleasant to relate that among the beneficiaries was a scientific geologist and mining engineer, for Arthur Winslow was State Ge-

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ologist of Missouri before he applied his technical training to the exploitation of a bonanza.

The early discoveries made in the Goldfield district gave no indication of the marvelous wealth it was destined to produce. The first find, on the Sandstorm, was bunchy and close to the surface of the rhyolite, which occurs in thin flows. This yielded fully $500,000. On the Kendall claim, which is partly in the same rhyolite and partly in tuffs associated with them, the surface dirt was rich, forming a concentration due to wind action; it was an eolian placer. When deeper mining began a vein was found several feet wide and profitable down to 30 or 40 ft. More recently renewed exploration has resulted in the discovery of a good width of ore at 70 ft., the gold being remarkably free. Another patch of gold-bearing débris was found on the Velvet, half a mile east of Columbia mountain; also in the rhyolite on the Cimarron claim. All of these patches of rich stuff—the cream of the geological dairy—stimulated prospecting and led to tentative digging into the combs of iron-stained quartz and silicified rock that appear on the sagebrush plain south and east of Columbia mountain. In two cases only was any noteworthy ore uncovered. There was the stringer that Myers found on the Combination and there was the outcrop on that part of the Jumbo subsequently included in the Bowers & Kernick lease. On this spot a defined vein was found within the 'blow-out' or comb of silicified dacite. A stope now breaks through the surface, and most of the outcropping mass for about 20 ft. in width will assay $20 to $25. But there was nothing to indicate the presence of the wonderful orebodies of the Mohawk mine; in places there were combs of silicified dacite shading into quartz that bobbed to the surface from under the detritus, due to erosion and wind action, but this stuff did not show gold in the pan and was neglected by all the early prospectors. The great bonanza might have remained locked in the heart of the desert if the developments in the Combination mine, which is on adjoining ground, had not stimulated the sinking of shafts in search for 'impossibilities,' as a local geological authority of German extraction, calls them. But they found the golden ore, rich beyond the fondest dreams of avarice, wonderful beyond the expectation of geological vision, and in quantity enough to turn a dozen good fellows into ordinary millionaires, as will be related in the sequel.

Manufacturing as Compared With the Reduction of Ores.—In a manufacturing establishment the buying of supplies presents no special difficulty, while the sale of the product needs skill—sometimes of the highest—and even then at a sacrifice of profit. A mine reduction-plant may be considered to have no anxiety regarding its principal supply of ore, or, at least, if the supply is not forthcoming, it is helpless; on the other hand, its product is easily sold at the market quotation, at any rate, to the metals selling companies. A custom reduction-plant needs skillful buyers, who, with good money in their hands, make every endeavor to procure ore, and sometimes fail to get it in sufficient quantity or quality.

The Prospector.

Owing to the large number of specimens forwarded to this department, it has been found necessary in future to make a charge of 25 cents to subscribers for each determination, and $3 to non-subscribers.

M. F. B., of Boston, Mass., sends a specimen of serpentinized rock containing pyrite.

The metallic mineral in rock sent by R. L. H., of Wardner, Idaho, is galena. No tellurium is present.

Rocks from Wallace, Idaho, marked P. 10, are: No. 1, quartzite with specular hematite; No. 2, quartzite.

G. G. C., of Schurz, Nev., sends: No. 1, quartzite; No. 2, quartz porphyry; No. 3, limestone with gypsum veins; No. 4 and 5, quartz.

The specimens sent by E. I. F., of Baker City, Oregon, are gneiss, the larger piece containing much pyrrhotite and some veinlets of pyrite.

A specimen of porphyritic rock which may be classed as quartz-porphyry or rhyolite-porphyry was received from M. T. C., of Red Mountain, Colorado.

Minerals from Tacopa, Cal., sent by K. & L., are: No. 1, calcite; No. 2, resembles magnesite, but is a silicate of alumina, magnesia, soda, and some lime. Probably a variety of amphibole.

Gold and Silver in Thermal Springs.—Interesting evidence on the origin of gold and silver in quartz veins is given by certain hot springs in the centre of existing hydrothermal activity in New Zealand, the Taupo volcanic zone. Sinter taken from the rim of a very ebullient spring at the Maori settlement of Whakarewarewa was found to contain, on analysis, silver to the amount of 4 oz. 18 gr. per ton, and gold to the amount of 1 dwt. 4 gr. per ton. The sinter analyzed was stained with sulphur, but showed no visible evidence of any sulphides. Analysis made from the sinter deposited in a wooden trough used to conduct water from the same spring at Whakarewarewa gave the following result in the precious metals: Gold, 12 gr.; silver, 15 dwt. 3 gr. per ton. The great geyser of Waimangu, which broke into action some years after the terrible Tarawera eruption of 1886, and remained active until November, 1904, deposited a blackish material, consisting chiefly of sulphides, but containing neither gold nor silver. Some mud obtained by Dr. Wohlmann, the Government balneologist, from a hot spring in the sanatorium grounds at Rotorua gave the following somewhat remarkable analysis: Silica, 69.30; alumina, 4.52; iron oxides, 2.00; titanium oxide, 0.58; lime, 1.00; magnesia, 0.10; soda and potash, 1.30; sulphur, combined, 1.40; sulphur, free, 6.09; organic matter, 10.01; water, 3.70%. Microscopic examination of the deposit showed that it consisted mainly of quartz and amorphous silica with a little feldspar. The mud also contained 5 gr. gold and 6 dwt. 1 gr. silver per ton. It is evidently not a deposit from the spring, but is merely a silicious tufa impregnated by the thermal solutions.—J. Mackintosh Bell in *The New Zealand Mines Record.*

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GOLDFIELD, NEVADA.—II.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

The story of the mines that made Tonopah and Goldfield famous is interesting, but the most picturesque story is that of which George Wingfield is the central figure. When Tonopah was discovered by Jim Butler in 1900, Wingfield was barely 21 years old. Already he had been variously a cowboy, a gambler, a prospector. In the spring of 1901 he went to Tonopah. He came thither from Winnemucca, where he had made a little money and owned some property, but being ambitious for a larger field he left a town that was only a point to punctuate the transcontinental line of the Southern Pacific, and started for the districts that were awakening the desert to busy life. He wandered from Winnemucca to Tonopah, where he went into partnership with Tom Kendall, who owned a saloon. Shortly after he joined Jack Hennessy in starting a roulette table in a room adjacent to Kendall's saloon. This gambling establishment was named the Tonopah Club. It made money fast. In three years Wingfield and Hennessy cleared $300,000. For those unacquainted with local conditions, it may be well to add that it was honest gambling, that is, there was no trickery or device to interfere with the hazard of fortune. The players took the chances of the game, just as they do at Monte Carlo. The Tonopah Club was not as gorgeously furnished as the casino by the shore of the Mediterranean, but George Wingfield ran his place just as squarely as that absurd anachronism, the Prince of Monaco.

As soon as Wingfield began to make $75,000 to $100,000 per annum, he placed the profit from the game in mining stocks and prospects. Tonopah was flushed with newly made wealth. In 1902 and 1903 the lessees on the Mizpah claim made money hand over fist. There was activity in every kind of gambling, from roulette to grubstaking. The chances of the desert were at least as good as those of the table. Wingfield got Nixon interested in some claims called the Boston-Tonopah group, situated at the back of Oddie Mtn. This was the first business deal with Nixon. George S. Nixon was president of the First National Bank of Winnemucca; he is now Senator from Nevada. Many tales have been invented to explain the first association of the two men who later became partners in mining enterprises involving enormous sums of money, but the facts are sufficiently romantic without adornment with the tinsel of fabrication. Wingfield knew Nixon when both of them were living at Winnemucca, and Nixon was known to be a man ready to back any reasonable venture. But the Boston-Tonopah was no good; it was within the mineral belt, but not in the rich ground.

Then came the discovery of Grandpa, or Goldfield, as it was subsequently christened. Wingfield's friend, Tom Kendall, had been partner with Jim Butler in grubstaking Stimler and Marsh, the discoverers of Goldfield. Kendall himself went to see their find, thirty miles from Tonopah. Two days later he stood on Columbia Mtn., which overlooks the present town of Goldfield, and directed Stimler and Marsh to examine and locate the ground to the south, now covered by the Florence and Combination mines. They broke samples from the combs of rock that appeared above the sagebrush and tested them by panning, getting no gold; then they went to the north side of Columbia Mtn. and located the Nevada Boy, May Queen, and a lot of other claims that never amounted to anything. They also located the Booth claim, at the foot of the mountain, for William Booth of the *Bonanza*, a daily paper published at Tonopah. Thus the Combination and Florence ground was not located by the first party of prospectors. This was at the end of 1902. In May 1903 there was organized the syndicate of ten as already stated. Each member of the syndicate was assessed $100, of which $50 was paid up. They sent A. D. Myers and R. C. Hart to Goldfield. Finding some ore, they located the Combination No. 1, 2, and 3 claims, the Hazel Queen, and the Golconda group of six claims, so that ten claims were taken up at this time. Subsequently Myers and Tom Murphy located the Combination Fraction.

Up to this point Wingfield played no part in the exploration of Goldfield, but when the first excitement had passed he bought (for himself and Nixon) some of the shares of the syndicate owning the Combination group, and he also purchased an interest in the first discoveries made by Stimler and Marsh, on the Sandstorm and Kendall. He had become rich, making as much as $200,000 in a single year from the roulette table and successful speculations in local stocks at Tonopah. In the early part of 1904 he bought the various holdings of Con Crooks, who was an old prospector in possession of a number of fractional interests, from a quarter to a half, in over 30 claims, including the Sandstorm, Kendall, and Mohawk, besides groups of claims called the Laguna, Blue Bell, and Conqueror. Crooks had a partner named Harry Ramsey, who now lives in opulent retirement at Berkeley. Wingfield and Nixon bought him out also, not long after they had made their deal with Crooks. Then they purchased Jim Butler's interest in the Sandstorm and Kendall claims. Thus Wingfield and his partner began to have large and scattered holdings in the Goldfield district. At this time Wingfield offered to sell Hennessy a half of all his holdings at Goldfield for $1000, but Hennessy refused it. This is authentic. It is one of the might-have-beens of desert romance. Soon after this unconscious play with fortune, Wingfield began to make money from leases that he financed, providing capital for exploring likely-looking ground. Among these ventures was the Sweeney lease on the Florence lode. In 1905 he made $350,000 from his share in this lease and then he and Nixon, who had remained his partner throughout these transactions, bought a one-third interest in the Florence group of claims. In the autumn of 1906 they obtained control of the Red Top and Jumbo for $1,200,000. It is generally supposed that in this big deal they were financed by others, but I am able to deny such a statement. They had made money enough to swing even a deal involving over a mil-

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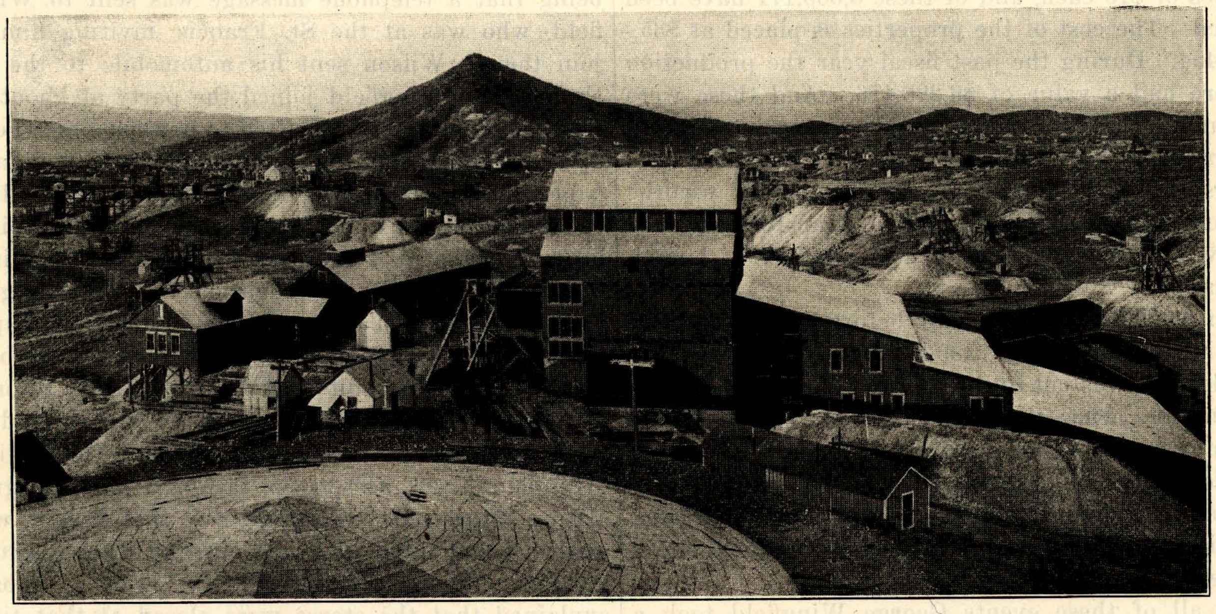

Mohawk Mine. Columbia Mtn. Florence Mill. Laguna. Winston Lease.

GOLDFIELD, LOOKING NORTH FROM THE LITTLE FLORENCE SHAFT.

THE TOWN OF GOLDFIELD WITH COLUMBIA MTN. IN BACKGROUND.

The Principal Mines Extend from Columbia Mtn. to the Right, Beyond the Town.

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lion dollars. That was the beginning of the consolidation, which was organized on November 13, 1906, under the title of the Goldfield Consolidated Mines Co. The capital is divided into 5,000,000 shares of $10 each, and of these 3,535,171 have been issued. The cost of the properties is placed at $35,085,454. During the past fiscal year the production of ore had a value of $6,296,476. And these were the mines that were hidden under a sagebrush plain diversified by combs of iron-stained rock only five years before. Talk about the creation of wealth! Never was there such a sudden transformation of the waste places of the earth into a hive of human industry.

On April 17, 1906, the day before the earthquake-fire in San Francisco, the big bonanza of the Mohawk mine was cut by the lucky lessees, and during the months when the money markets of the Pacific Coast were closed by the disaster to the City by the Golden Gate, the outpouring of golden ore from the Mohawk, Florence, and Red Top mines redressed the industrial balance and started Goldfield on a career of marvelous prosperity.

In all of these events George Wingfield took a prominent part. He received royalty as owner of claims worked by fortunate lessees and he shared directly in the profits of lessees working on other claims. When only 27 years old he was doubly a millionaire. He proved alert, quick to action, and shrewd in business. When the labor troubles began, in 1906, he showed his fearlessness in facing the agitators. Once when the union tried to boycott the only outspoken daily paper in the district, *The Goldfield Sun*, which is now *The Tribune*, he and Jack Davis took copies of the paper, went into the street, and sold them for the newsboys. The proprietor of a gambling resort is apt to get into shooting scrapes and Wingfield has often had to pull his gun, but he has never taken a human life, his cool courage proving sufficient to carry him through any awkward situation. While he has shown himself reckless, as in the incident above related, he is no fire-eater like Jack Davis, who goes about armed like an arsenal. On one occasion when the miners' strike had reached a dangerous stage, someone took a shot at Jack Davis, who rushed into the local club (the Montezuma), announcing that he had been attacked by the strikers. The electric lights in the club-rooms were turned off instantly and every member pulled his gun and stood at the windows in expectation of an attack. It proved only a false alarm, but when Davis was examined it was found that he wore three overcoats and carried a gun in each of the two pockets, making six revolvers in all. Besides these he carried a bowie knife in his belt and at his back was slung a sawed-off shot-gun. This is the burlesque of the frontiersman. In such fooleries Wingfield took no stock, although ready to assert himself whenever occasion required.

For instance, there was an episode in San Francisco last September. The *Examiner*, one of the Hearst papers, in its customary way, had exploited a story to the discredit of Wingfield. A few days later, Theodore Wores, Edgar Mizner, and J. C. Wilson happened to be at Tait's restaurant. With them was Dent Robert, managing editor of the *Examiner*. One of the party mentioned Wingfield's name and Wilson spoke of him in a kindly way, the result being that a telephone message was sent to Wingfield, who was at the St. Francis, inviting him to join them. Wilson sent his automobile to the St. Francis and Wingfield joined the party at about 11 o'clock. Soon the talk drifted to newspapers, whereupon Wingfield said that he intended to start a daily paper in San Francisco. Robert told him of the difficulties and expense incident to such an undertaking, intimating that it would cost several millions. Wingfield replied that he had the money and, turning to Robert, said: "You seem to know all about the paper business." Thereupon Wores explained that Robert was the managing editor of the *Examiner*. At once Wingfield spoke bitterly of papers in general and of the *Examiner* in particular. Robert called him a liar. Wingfield struck him in the face and pulled a gun. Mizner interfered. Wingfield saw that Robert was unarmed and told him to go out and "get a gun," and he would wait for him while he did do so. Wores explained that the stores were closed at that time of night and no revolver could be purchased until the next day. So the incident closed. Thereafter the *Examiner* was silent as to the doings of Mr. Wingfield.

Wingfield is respected by his associates and liked by those who know him. "You cannot bluff him," they say. He is "a man of his word," generous to a fault, and possessed of the mingled recklessness and honesty that mark a few of the best professional gamblers. He is the Jack Hamlin of a later day and had he lived in the golden age of Bret Harte he would have been made as famous as the virile sportsman whose memory is enshrined forever in the story of Poverty Flat.

Another chapter in the romance of the Nevada of today is contributed by the Lockhart and Parker partnership, with which is associated the development of the Tonopah Extension and Florence mines. Alexis Dupont Parker was the treasurer of our University Club at Denver when I lived there, before 1902. He was also chief accountant for the Colorado Southern Railway. Since then he has become an important mine operator, by force of luck and Tom Lockhart. Parker is a graduate of Racine (1879); he was intended for an Episcopal clergyman, but, abandoning that avocation, he decided to learn the business of railroading, from the ground up. So he went to work on the Denver & Rio Grande railroad grade. While thus employed he met T. D. Lockhart, a prospector, who had taken a laborer's job in order to make a little money preparatory to further search for mineral wealth. They became friends. Parker believed in Lockhart and undertook to grubstake him, that is, provide funds for the search after mineral wealth. He got a clerkship in the auditor's office of the D. & R. G.; he prospered; having ability he became an expert accountant and did outside work, as, for example, for the International Trust Co. of Denver. Then he left the D. & R. G. and became assistant to the chief accountant of the Colo-

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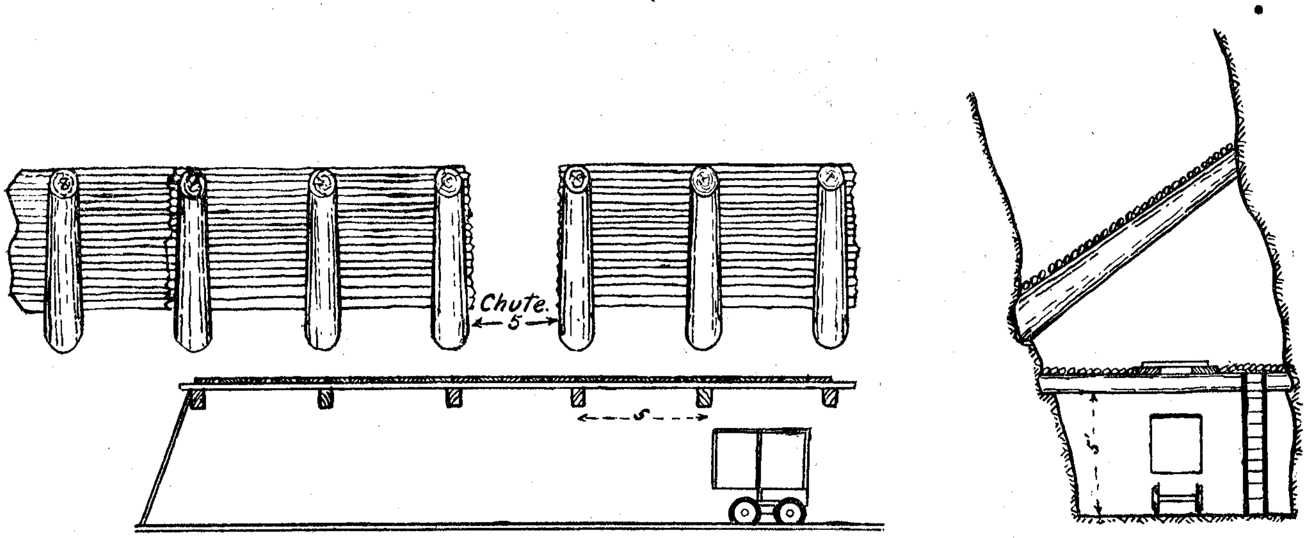
rado Southern, continuing to advance until he became vice-president and practical head of the Colorado Southern Railway. During all these years he helped Lockhart; even when a clerk on a small salary he managed to save some money to send to Lockhart, who prospected far and wide, but without any noteworthy result. In 1900 Lockhart went to Tonopah and located two claims near the Butler discovery; he was informed that he had overlapped some ground already located (the Sand Grass and Red Plume), so he moved off southward and located the Grand Trunk and C. B. & Q. claims. These were on the sagebrush plain, they showed no sign of ore of any kind, and their validity as quartz claims might easily have been questioned. Meanwhile the development on the Mizpah seemed to indicate the direction of that lode, and he began to sink a shaft through a flow of rhyolite that capped the ore-bearing formation. He started a shaft unaided; he would climb down to the bottom and fill his bucket, then climb to the surface and hoist it with his windlass. Eventually this shaft became the main opening of the Tonopah Extension mine and cut ore at 237 ft. In March 1902 Charles M. Schwab bought the Tonopah Extension for $75,000. For several years previous Parker had ceased to contribute to the expenses of Lockhart's prospecting, and legally the latter was absolved from paying his partner any portion of this stake, but he lived up to the spirit of their arrangement and handed $37,500 to Parker. They call Lockhart mean at Goldfield, and he is thrifty enough in all conscience, but his sense of honor must be far more developed than that of many more generous men. He is the type of the canny Scotchman who is as close-fisted as he is honest. After making money by the sale of the Tonopah Extension, Lockhart went to Goldfield and acquired mining property. He got hold of the Florence, paying $7000 down for a part, with an option for $25,000 on enough more of the mine to give control. Wingfield and Nixon took a lease on the Florence, and the royalty paid by them more than enabled Lockhart to complete his purchase. Soon after making this deal Lockhart overheard one of the owners chuckling over having got the best of him, but he was confident in his opinion of the value of the mine. He told Parker about it, informing him that he need not come into the deal unless he wanted to do so. Parker replied that he would share equally with him in all his ventures. He did well. The Florence has produced $3,000,000 altogether, and of this the royalty to Lockhart and Parker has amounted to fully $375,000, as owners of slightly more than one-half of the stock of the Goldfield-Florence Co. The mine is still young and is destined to add largely to the output of the district.

THE 'CHINAMAN' ORE-CHUTE.

An ore-chute with a peculiar bottom to obviate the difficulties of jamming at the doors and of sudden 'spills' or rushes of ore, which may swamp a car and cause delays until the track can again be cleared, is in use in Western Australia, under the local name of a 'Chinaman.' Its construction is clear from the accompanying cut, giving longitudinal and cross sections. Beneath the bottom of the ore-chute, where it comes to the line of stulls above the gangway or level, is a long platform or deck, supported upon sets of timbers at a convenient height above the track. This platform extends a distance of 10 ft. on either side of the chute. Along the central longitudinal line of the platform are sliding or drop-doors at intervals. The ore spills from the chute and spreads out upon the platform until the angle of rest for the pile is attained, which prevents the possibility of more ore running out. The ore is then drawn through the doors in the platform to the cars below.



The Chinaman Ore-Chute.

A concrete trestle is a somewhat novel type of structure built across the Salt river, in Illinois, by the Chicago, Burlington & Quincy railway. The structure is 477 ft. long, with six-pile bents spaced 14 ft. apart, and with an average height of 10 ft. The piles are of re-enforced concrete, 22 ft. long, and were driven by a pile-driver with a 3000-lb. hammer falling 24 ft. The piles are all vertical and are capped by deep concrete cross-girders supporting the slabs which form the floor or deck.

Re-enforced concrete line-kilns in Europe, built without fire-brick or other inside lining, have withstood for several years temperatures from 2200 to 2500° Fahrenheit.

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GOLDFIELD, NEVADA.—III.

Geological Notes.

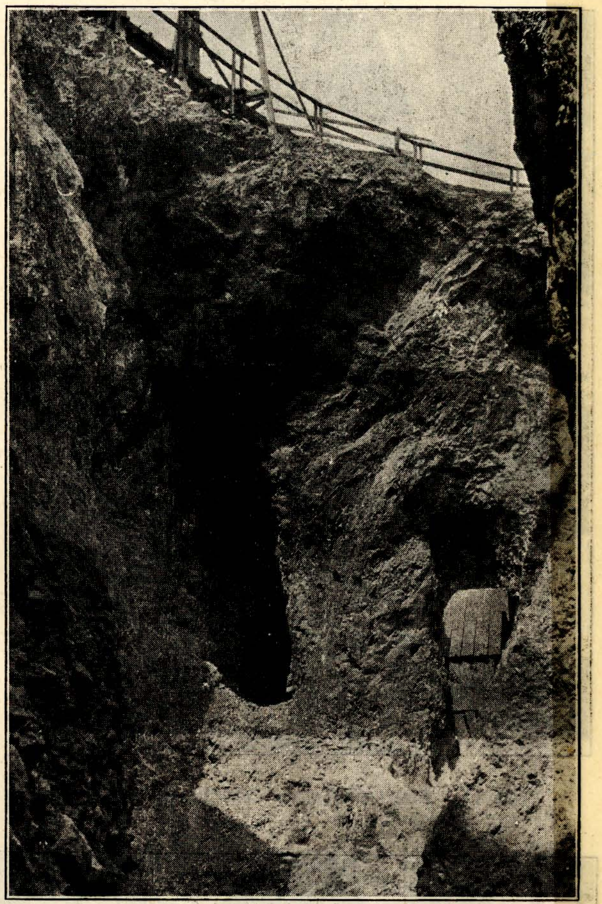
Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

Owing to the limited extent of the underground workings and the comparative newness of the mines, the geology of Goldfield is difficult to decipher. One of the first published accounts of Goldfield appears in Bulletin No. 260 of the U. S. Geological Survey, in the form of a short report by J. E. Spurr, who visited the district in November, 1904. This was long before the big discoveries. I presume that Mr. Spurr would like to see this early document destroyed, for it was written before the necessary facts were available. He emphasized the lack of persistence and want of definition exhibited by the veins of the district, and concluded that "these irregular reefs represent the horizontal sections of columns of rocks traversed by rising columns of hot waters. . . . It follows that the quartz bodies will probably, as a rule, extend deeper vertically than horizontally, and so have roughly the nature of columns or pipes." No one would recognize in these generalizations any description of the ore deposits as exposed today. Mr. Spurr has done such splendid work at Tonopah and Silver Peak, that it is surprising he should have so ill interpreted Goldfield. Undoubtedly he was the victim of that haste of publication to which the Survey is occasionally committed in the desire to please the politicians and other local powers that clamor for geological reports even before the necessary evidence is contributed by actual mining.

The next scientific investigation was made about a year later, by F. L. Ransome, who was at Goldfield at the end of 1905 and the beginning of 1906. His report appears in Bulletin No. 303 of the Survey. This work was less sketchy, although preliminary. Enough evidence was collected to warrant the comprehensive and illuminating statement that the district covers a domical uplift of Tertiary eruptives resting upon highly altered sedimentary rocks of probably Paleozoic age. This appears to be the key to the correct understanding of the local geology. In February, 1906, John B. Hastings and Charles P. Berkey contributed a valuable paper on the geology and petrography of Goldfield to the Transactions of the American Institute of Mining Engineers. This paper is often quoted by the local mine managers and it contains much useful information. The identification of the chief rock-types shows results varying slightly from those of Mr. Ransome and bearing a closer resemblance to the rocks of Tonopah. Hastings emphasizes the extreme silicification of the eruptives along systems of fracture and the evidence of wide-spread solfataric action.

The most recent description of Goldfield geology will be found in an excellent article on the Combination mine, written by Edgar A. Collins, and published in the MINING AND SCIENTIFIC PRESS of September 28 and October 5, 1907. This gives a detailed account of underground exploration by the manager of the mine that first made Goldfield famous.

When F. L. Ransome made his preliminary investigations more than two years ago, the great bonanzas had not been discovered and Goldfield gave no signs of its recent prominence as a mining district. Since then, I am informed, there has been a disposition to discourage further scientific examination by the Government geologists because of litigation as to mineral rights between different mining companies. The completion of the big merger of many of the principal mines has removed several likely points of dispute, but even now two or three important suits are pending, involving questions of extralateral right, and while I was at Goldfield I found Horace V. Winchell, W. H. Wiley, Charles J. Moore, and other mining geologists engaged in collecting evidence to serve as ammunition in the courts. The law is not always synonymous with justice, nor is



Glory Hole of the Combination Mine.

mining litigation necessarily a stimulant to the ascertainment of scientific facts. Too often these quarrels over mining territory breed a bastard sort of theory and in any event the effect of them is to silence the men best able to contribute to our knowledge of local geology. So, for the present, there is not much more to be said than that the prevailing rocks are Tertiary eruptives forming a low dome over a core of much older metamorphic sediments.

As is usual in ground that has undergone intense mineralization, the structure of the orebodies is indefinite. Obvious boundaries and simple definition are more likely characteristics of poor lodes than of rich ones. Diffuse impregnation with ore tends to obscure the relation between the vein and the rock it traverses. The prevailing rock is dacite, which has a porphyritic habit by reason of phenocrysts of feldspar. This feldspar is labradorite; in fresh dacite it is white to pale green; by alteration to alunite it becomes pink. The dacite is essentially a quartz-andesite. This is the rock in which most of the rich ore is being found. It is covered by patches of 'wash,' or débris, so that the surface exposure is incomplete, but enough evidence is available to indicate that it extends over the principal portion of the rolling plain occupied by Goldfield. Several knobs, such as Columbia and Vindicator mountains, appear to thrust themselves through the dacite, but they are really older masses that have survived ero-

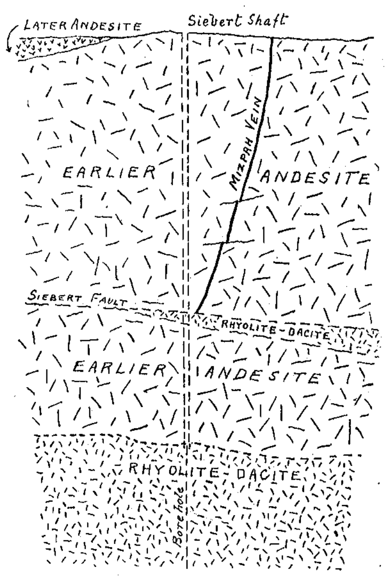


Fig. 1

sion. They represent cores of hard calcareous shale (of Cambrian age) flanked by alaskite and rhyolite. The dacite is broken by several faults, the most important being to the west of the bonanza ground

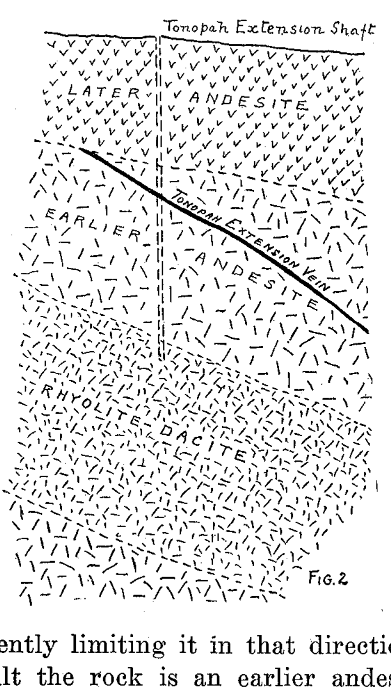


Fig. 2

and apparently limiting it in that direction. West of the fault the rock is an earlier andesite. The thickness of the dacite is not known; the deepest workings, at 600 ft. in the Mohawk, are still in this formation. On the bottom level of the Combination (380 ft.) some andesite appears under the dacite, the andesite dipping about 34° northeast. The relation between the two formations is a matter of importance, for if ascertained it would answer the question whether the dacite is an intrusion or a thick layer. At Tonopah the petrography proved the key to the mining geology in an unexpected way, as the accompanying sketches[1] illustrate. Fig. 1 shows the Mizpah vein traversing an earlier andesite and cut off by an intrusion of later rhyolite-dacite; while Fig. 2 shows the Tonopah Extension vein, which does not penetrate the later andesite and fails to reach the present surface. By reference to Fig. 1, it is seen that the Mizpah vein just missed being covered by the later andesite (shown to the extreme left of the section) and so yielded an outcrop the richness of which led to the development of the Tonopah district. It is evident that the veins were formed during a period intervening between the extrusion of the two andesites. It is likely that the deposition of ore was a sequel to the thermal activity that followed the extrusion of the older andesite.

At Goldfield the ore was found at the surface as an enriched portion of one or two ridges of quartzose rock. These ridges were combs formed by the silicification of the dacite along lines of fracture. In the rich area the chief lines of fracture run nearly north and south, and they are marked at intervals by

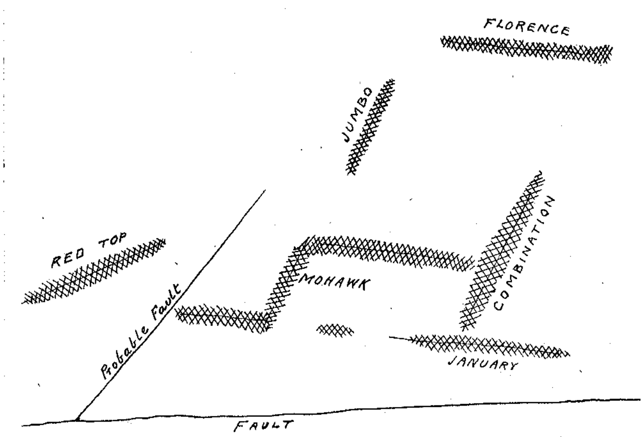


Fig. 3. Sketch Showing Relative Position of Orebodies.

these outcrops or 'blow-outs' of silicious rock. Most of them are poor, if not barren, and it was only on the Combination, January, and Jumbo that the early prospectors were able to find pay-ore. Most of the orebodies were discovered by haphazard sinking of shafts and cross-cutting into these same masses of silicified dacite at a lower level. For instance, Hayes and Monette, in their famous lease on the Mohawk, struck the first ore by cross-cutting at 150 ft. They cut it at 25 ft. from their shaft. The finest showing was just below the 150-ft. level, where they had two feet of ore assaying $15,000 per ton. This was a black heavy sulphide streak, from one inch to two feet wide; it persisted with regularity for 130 ft. on the dip. The lessees extracted a lot of 49 tons that assayed $6.50 per pound. The bottom of the zone of oxidation was between 105 and 115 ft. below the surface. In the Mohawk ground the pay-ore usually did not extend into the oxidized zone, but in the Sheets Ish and the Combination it reached to the surface.

The accompanying sketch shows the distribution of the principal orebodies as known today. It suggests that there is a system of co-ordinate fractures, which are not exactly at right angles. The north-

[1] 'Geology of the Tonopah Mining District.' J. E. Spurr, U. S. G. S.

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south members dip east from 34 to 50°; along them are found the Red Top, Mohawk, Florence, and part of the January lodes. The east-west (or more accurately, west of north) members are nearly vertical, for example, the Combination and the Jumbo. At the intersections of these lines of fracture there has been intensified mineralization, which has been supplemented by faulting, followed in turn by another period of ore deposition, leading to further enrichment. By faulting, the co-ordinated fissures have been shifted so as to accentuate the multiplicity of fracturing and assist mineralization. The lode-channel thus formed has been dislocated in many places by cross-faults but the rich streaks within the lode-channel appear only to have been diverted, that is, the later enrichment following the fault-lines. In the hard silicious mass of the lode-channel the faulting is easy to follow but in the soft outer country it is difficult to trace. Minor movements complicate the structure, especially within the larger orebodies. It is dangerous to label the individual veins and to try to trace them for any distance; they lose their identity and become part of a larger linked system. Bulges and violent changes of strike in the veins and in the main hanging wall of the lode-channel suggest an end-thrust, causing a buckling of the dacite. The best ore is found in the bulges, reaching in places from hanging to foot-wall for the full width of the lode-channel, that is, 500 to 300 ft. Thus splendid orebodies have been found. In the Mohawk the rich ore is continuous for 800 ft., and in the Combination for 500 ft., reaching out to 1000 ft. The deepest working is the winze below the 450-ft. level in the Mohawk; this is down 160 ft. on the dip, or within 50 ft. of the 600-ft. level. This winze, 8 ft. wide by 12 ft. long, has already yielded $75,000 worth of ore, and as much more is in the bins. The average has been $135 for a minimum of 6 ft. and a maximum not yet ascertained, as it exceeds the width of the winze. This is an encouraging fact, and while deeper development may at any moment break into the older andesite supposed to exist under the dacite, it is apparent that the depth already proved leaves ample room for the existence and exploitation of extensive orebodies.

The lode-channel has become, by silicification, harder than the surrounding country. If you walk into a cross-cut and see ahead some white rock fallen from the roof, you know that at that point the cross-cut has passed into the outer dacite. But while the lode-channel as a whole is hardened by silicification, the best ore is along lines softened by intense chemical action, resulting among other effects in the formation of seams and patches of the white substance called 'talc' by the miners. It is mainly alunite, a hydrous sulphate of aluminum and potassium.[2] Hugh Trenholm, superintendent of the Red Top, told me that this 'talc' is a good sign, especially when gritty. The rich ore is usually a seam of dark chalcedonic quartz, carrying needle-shaped crystals of bismuthinite (bismuth sulphide), together with gray copper, pyrite, and occasionally tellurides, such as calaverite.[3] This rich stuff is commonly flanked by mineralized and silicified rock rich enough to be classed as ore. The gold itself is not easy to see, even in the richest ore, until the specimen is wetted; then it is seen in dull yellow bands formed by minute particles of gold arranged along lines of crustification. The gold is like paint, if you please, but unlike that of Kalgoorlie, which was soft, as metal newly precipitated from solution, and required to be scratched to be detected. The burnishing developed the yellow-red glint of the noble metal. At Goldfield the gold is fine, but crystalline, and therefore quite different from the 'mustard gold' of Kalgoorlie. It appears to have been deposited with the chalcedonic quartz, which it ribbons. The gold is sufficiently free to permit of the use of the pan in ascertaining what is rich ore. Mr. Collins states that it was impossible "to tell pay-ore from worthless vein-stuff by its appearance," but panning served as a trustworthy guide. Visible gold was most abundant in the Mohawk ore after a depth of 300 ft. below the zone of oxidation had been attained, that is, over 400 ft. altogether below the surface. Some handsome specimens of free gold were extracted in the winze, at 50 to 70 ft. below the 450-ft. level, and these were identical with specimens broken in the Florence mine. The Florence is a smaller, better defined vein than the Mohawk, and I expect to see the latter become smaller and more clean-cut in depth when once below the horizon of maximum enrichment, just beneath the zone of oxidation. In the Florence a 3-ft. stope suffices; the ore thus broken in the Little Florence lease has averaged $200 to $250 per ton, down to 400 ft., where the orebody pitches into adjacent ground. It is said that 5200 tons have been shipped from this lease for a gross yield of $1,250,000. The rich streak is usually only a few inches wide; the vein is well-defined and reminded me of some I had examined in Boulder county, Colorado.[4] The Florence orebodies are not long, say 75 ft., and in some cases isolated masses of rich ore have been found in the foot-wall country, at the intersection of fractures. One of them, at the 400-ft. level, was 12 to 15 ft. diam., 70 ft. high, and roughly egg-shaped.

The mine managers speak of a hanging and a foot-wall, meaning thereby the eastern and the western limit of their exploration, for in the Red Top, I saw good stopes on a vein beyond what was once called the hanging, and a rich vein (several feet wide) beyond what was formerly regarded as the foot-wall. These 'walls' are marked by a selvage of clay, they are lines of faulting and lose themselves in other slips and gouge-seams. The most persistent of the eastern 'walls' in the Mohawk is called the hanging; it rolls both in strike and dip, so as to form large bulges, as if the lode-channel had been crumpled by pressure from the north and south. Rich ore is found within these bulges and across them, as well as along the principal 'walls'.

Every observant visitor to a new and very rich mining district wonders whether the orebodies will

[2] 'The Association of Alunite with Gold in the Goldfield District, Nevada,' by F. L. Ransome. *Economic Geology*, Vol. II, No. 7.

[3] Telluride minerals are rarely visible. In two analyses of a concentrate obtained from a mill-test on Mohawk and Combination ore, the following results were obtained: The first analysis gave 0.25% Te and Se, with 41.5% S; the other gave 0.27 Te% and Se, with 42.69% S. The concentration was in the ratio of 39 to 1, and about two-thirds of the sulphur went into the concentrate.

[4] 'The Veins of Boulder and Kalgoorlie.' By T. A. Rickard, Trans. A. I. M. E., Vol. XXXIII, p. 567.

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GOLDFIELD, NEVADA.—IV.

Rich Ore and Its Moral Effects.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

Rich ore invites theft. Wherever men find minerals so valuable that a small weight of it represents money, there the temptation to steal is inevitable, the weak guard of conventional morality is broken, and the conscience of the miner is debauched. It does not matter whether it be an old mining community like Bendigo, with its art gallery and botanic gardens, or a younger and richer district, like Kalgoorlie, or an even more recent and more extravagant camp, like Goldfield; in every case the sight of gold, or of the minerals that are known to enclose it, provokes industrial kleptomania. In Colorado and Nevada they call it 'high-grading,' that is, the removal of high-grade ore from the possession of the mine-owner to that of the mine-worker. The man that is guilty is called the 'high-grader' and the purchaser of the stolen goods is called a 'fence,' the word meaning originally a place of defence against the officers of the law. 'High-grading' is larceny.

To understand the extraordinary freedom with which rich ore was stolen at Goldfield, it is necessary to appreciate the peculiar labor conditions prevailing in this part of Nevada during the time when the bonanzas in the Mohawk, Red Top, Combination, and Florence mines were being exploited. Mining operations were conducted under leasehold; the owners of the mines were, for the most part, without experience of mining, they had no capital, and they did not know where to look for ore; so they granted leases to parties of working miners willing to work, and to adventurous operators willing to furnish the money for exploration. The areas controlled by different companies or ownerships were sub-divided into blocks, usually 300 ft. long and the width of a claim (600 ft.), and in order to stimulate speculation the lessees organized stock companies, placing the shares on the exchanges of San Francisco, Los Angeles, Salt Lake, and Goldfield. As evidence of the enterprise of some of the men that are mining at Goldfield, I may mention the lease taken by John Donnellan & Co. on the north end of the Red Top. The ground leased is on the south slope of Columbia Mtn. and near the foot of it; the area is 600 ft. square; the term of the lease is 18 months. The lessees agree to sink a two-compartment shaft to 500 ft. They expect to sink at the rate of 150 ft. per month and to complete the shaft within four months. All the money required for this work was obtained in three days by sale of stock. No better proof could be given of the venturesome spirit of the community. Of 100 leasing companies organized in 1906 and 1907, fully 20 struck rich ore and made big profits for a time. Most of them started with just enough money to pay the costs of incorporation and got the capital for their preliminary development by selling shares. Only one of the leasing companies was a close corporation. Even half a lease was incorporated. This happened to the Lewis Rogers lease, one-half interest in which was incorporated as a company of 1,000,000 shares. It only remains for the individual shareholder to incorporate himself. As soon as the incorporators had organized they went to the local or State banks and borrowed money on their stock, the bank in its turn endorsing the company as a legitimate mining scheme. The wonderful ore discovered in the Frances Mohawk, Hayes & Monette, Rogers, Little Florence, Red Top, and other leases sustained the whole fabric of fictitious finance until the day of accounting came last October, during the panic that involved the whole industrial structure of America. Then the local banks (all save one) shut their doors and the local brokers pulled down their stock-boards. The total loss among three banks was fully $1,000,000, while the surviving bank (John S. Cook) had $6,000,000 available. The rate of interest paid by the leasing companies was 1 to 2% per month; the bank officials were interested in the leases and in the stock deals that resulted. The one essential factor missing was uninterrupted prosperity and inexhaustible reserves of rich ore.

The ore was rich indeed. In the Frances Mohawk lease at a depth of 200 ft. there was 8 inches of stuff that assayed $250,000 per ton; it was free gold and calaverite or telluride of gold. The various leases on the Mohawk mine yielded a total of $50,000 per day for 106 consecutive days. One carload shipped to the Selby smelter weighed 47 tons and assayed 609 oz. of gold per ton, so that the shipment was worth $591,637, that is, more than $6 per pound. There were many places in the mines where men knew they were breaking ore worth 10 to 15 dollars per pound. It was easy to steal several pounds of ore. And they did.

At the time of my visit there was not much stealing going on—so I was told—except in the Rogers and Little Florence leases, but these happened to be the only ones in bonanza just then! It is estimated that last summer over $1,000,000 worth of ore of which no proper account was made went out from the Mohawk and Combination mines, and $500,000 to $750,000 from the Florence. Not less than $1000 per day of illicit bullion passed through the United States mint at Carson, even after the Goldfield Consolidated mines were closed down, so that most of it must have come from the two Florence leases (the Rogers and the Little Florence). Only $300,000 was recovered last year for the whole camp, despite every effort to find the stolen ore. It is most difficult to trace the 'high-grade,' for the thieves crush it at once and so destroy the chance to identify the product from any particular mine. Unless identified positively, the legal evidence is incomplete. The stolen ore is usually taken to a bogus assay-office, the proprietor of which pays about 50% of its real worth. The ore is then treated by chlorination; first it is roasted in a muffle and then leached in a barrel with a stirrer. In one case a plant was raided while in operation. The heading assayed 260 oz. gold per ton; the tailing assayed 15 oz. Larceny does not wait on accurate metallurgical method. When the owner of this particular establishment was asked what he intended to do with the tailing, he said that this residue would be sold to a man who intended to salt the oxidized

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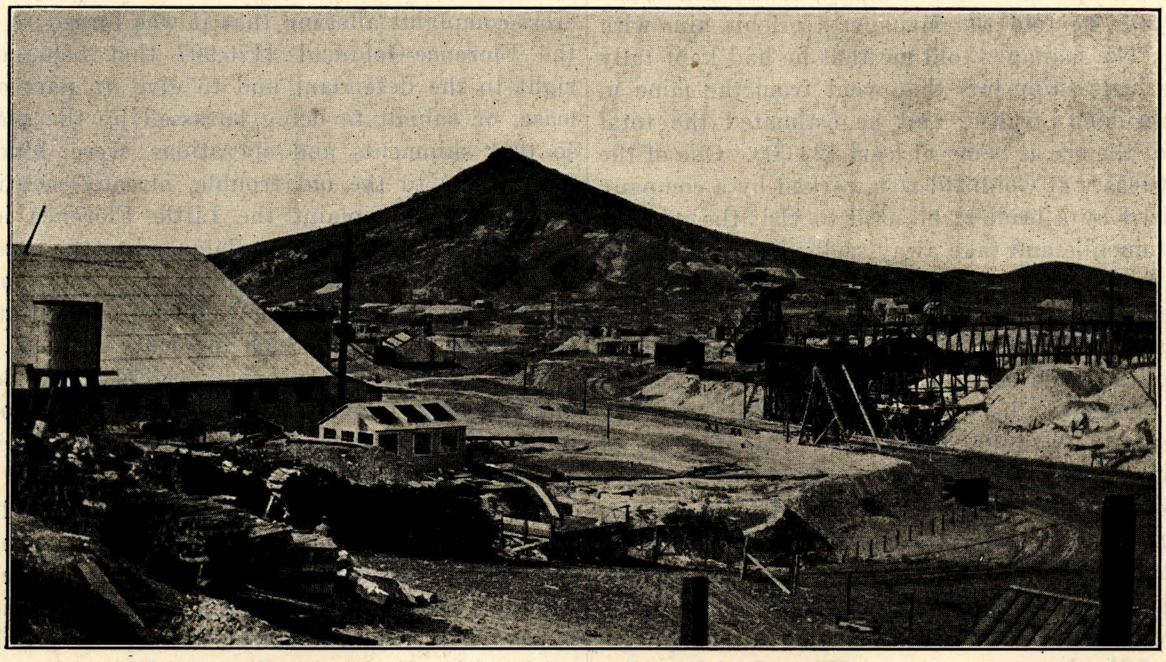
ore on a neighboring claim. Even when the circumstantial evidence seems convincing, the local justices are afraid to convict; public opinion regards 'high-grading' as rather humorous, the labor unions preach a doctrine by virtue of which a little rich ore is the miner's perquisite,* the rush to become wealthy possesses everyone, and in the excitement the moral law is left high and dry.

It was an orgie of stealing. Regular harnesses were invented for carrying ore under the working-clothes of the miners. They would step from the cage or bucket and stumble heavily laden on their way homeward without being arrested, although the officials knew why they walked so clumsily. One day, for instance, when the men were being hoisted to the surface at the Mohawk shaft, a man came up so loaded with concealed ore that on stepping out of the bucket he fell. The superintendent saw it and to strike the pail of one of his own shift-bosses with his candle-stick and detected a solid sound; he asked the man to go to the second level and perform an errand; then he opened the dinner-bucket and found it full of rich ore. Frequently the sacks that hold the wedges used by timbermen were employed as receptacles for stolen ore. The 'muckers' or shovelers would handle their shovel so as to leave the heavy fine ore on the bottom of the level and as soon as the men in the stopes shouted "fire," which they would do 10 minutes beforehand, the shovelers would have a chance to collect the rich ore while the shift-bosses and other men were on their way out to the shaft. At one time it was a rule that men should leave their dinner-buckets at the office. Some of them came on shift with their food in paper bags so as to avoid going to the office. They were warned not to do so. The next day all of them came with paper bags, and



Combination Mill. Columbia Mt. Mohawk Mine.

Goldfield, Nevada.

called out: "Some one help that son of a gun." In the Little Florence, there was a hoist underground, at the 300-ft. level, and one day the superintendent signalled from a lower level without getting any response, so he climbed up the ladder and found the engineer digging into the bonanza ore in the floor of the level. He was actually getting ready to fire a shot, and had done so previously. In order to escape notice he would time his firing so as to coincide with the blasting in the neighboring stopes, and thus the explosion would pass unheeded. In another mine, the foreman happened to go into an abandoned cross-cut and found three carloads of the richest ore lying on a platform ready to be sacked and to be removed through a disused shaft at the far end of the cross-cut. Another foreman on another occasion happened

*It may be mentioned that in olden days on the high seas the captain's graft was formally recognized and termed 'primage' because he got it first. Nowadays 'primage' is charged in addition to the freight and retained by the ship-owner. The captain does not get it; he no longer participates, he is an employee only. In nearly every industry a similar development can be traced, from open participation to surreptitious graft, and then regulation, which ended all participation in profits.

the management did not dare to dismiss them. Finally, I may relate the story of the foreman who caught a man loaded with 10 lb. of 'high-grade'; this foreman was notified that there was a watchman paid to do the watching and as he (the foreman) was not a regular watchman he had better attend to his own duties, otherwise the union would have something to say. And thus we come to the root of it all: the subservience of the lessees to the labor union. Why? Because the ore was rich, and the terms of the leases were short, the lessees could not afford to delay the extraction of ore by having trouble with the men; the men knew it, and so stole as much as they could. Everything was sacrificed to a mad rush to make money, the mines were gutted, honesty was swept aside fiercely, the law was made a dead letter.

The rights of property were not respected because the judges who preside over the local courts owed their position to political election, and this made them subservient to the majority of voters, who hap-

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pen to be mine-workers. Of the mine-owners and operators many have their homes elsewhere, in California, Utah, Colorado, or New York; they do not vote at Goldfield and they exercise scant political influence. Moreover, a large proportion of the lessees who hired the workers in the mines were ordinary miners and laborers themselves, their good fortune was not conceded as due to any superiority of character or talent, therefore the men who were employed thought it reasonable that they should participate in the luck of those by whom they were employed. It was a rough kind of socialism.

The miners stole rich ore, but so did others who might have been supposed to know better. I know of the president of a mining company (not at Goldfield, but in a neighboring district) who used to visit the mine whose operations he supervised and on the occasion of each visit he filled his dress-suit case or valise with specimen ore. Finally, at the end of the fiscal year he told the manager to debit him with $4000. The manager told me that he had kept tally on the little shipments that went from the mine in the president's valise, and he estimated the total value of the ore as being at least $22,000. One of the richest leases at Goldfield was worked by a company organized by a broker; he used to visit the mine in his automobile and take away sacks of ore; it is estimated that he removed $150,000 worth, on which he avoided the payment of the 25% royalty due to the owners of the mine, not to mention the proportion due to the shareholders in his leasing company. Every lease had its sequel of litigation caused by differences of opinion as regards the application of the law of mine and thine. Thus, the Little Florence lessees went 30 ft. over their line and took rich ore from ground not within their lease. In August, 1907, they had to pay the Florence-Goldfield Co. not less than $177,000 in liquidation of a claim for trespass, and were compelled to make a change in the management of the lease operations. When I visited that part of the mine, I found the level barred by a heavy wooden partition placed there to safeguard the rich ore remaining. While at Goldfield I found myself involved in one of these difficulties between lessor and lessee, for I made an affidavit that I found the workings of the Little Florence lease in good shape. According to the terms of the lease, the Little Florence people had to leave five feet of ground under each working level, the idea being that this arch of rock was required to conserve the safety of the workings. I noticed that levels were rather far apart; the first was at 248 ft., the second at 400 ft. from the surface. The question arose whether the ore should be left untouched under an intermediate level. The lease was about to expire (on April 26) and the lessees were gathering all the rich ore that belonged to them. They were also putting the workings in good order, for by their agreement they had to maintain the timbering and hand the mine over in ship-shape condition. I remarked to Mr. Frank Oliver, the manager, that it was a good time to visit a mine when the lease was about to expire, for all the workings appeared so accessible, clean, and safe. The lessees are under compulsion to leave the track in the principal levels, but they may remove piping and tools. Ordinarily they have three months wherein to remove the ore stacked at the surface or in the bins, and to dismantle their equipment. I was in the Little Florence on the morning of April 11. In the afternoon of the same day the Florence-Goldfield Co. asked the District Court for an injunction and cancellation of the Little Florence lease, which was to expire within a couple of weeks, on the ground of improper mining methods and the removal of an arch of rich ore under a level. This was filed on a Saturday after business hours. On a technicality the hearing was deferred to the following Monday morning. Finding that their allegations could not be sustained legally, the lawyers acting for the Florence-Goldfield notified the Little Florence people that they need not appear in court as the case would be dismissed. But before this could be done, the Little Florence company filed a cross-complaint alleging that it was forced to allow the Florence-Goldfield $176,907 that belonged of right to the defendant and to give up part of the lease, or submit to being harassed by the plaintiff so that shipments and operations were hindered. This refers to the old trouble, already mentioned. In the cross-complaint the Little Florence lessees alleged that they had been damaged to the extent of $1,500,000, and asked for this amount of damages. The shares of the Florence-Goldfield dropped at once from $4.27 to $3.95. Such incidents as these, indicating lax notions of property rights among lessees and companies, are not likely to impress the working miner with any marked respect for the possessory claims of either party.

But to understand the extraordinary conditions prevailing at Goldfield last year, it is necessary to refer to the insidious effect of having a stock exchange close to the mines and the participation of owners, lessees, and workmen in the wild speculation arising from discoveries of wonderfully rich ore. The leaders of the labor union were agitators of the worst type, their importance depended upon stirring up strife between the employers and the employees, their opportunity to make money hinged largely on the fluctuations in stocks caused by labor disturbances and the settlement of them. During the strike that was ended in April, 1907, the stock market was worked by the labor-union leaders. They would call a meeting under pretext of settling the strike, and the announcement of this fact would lead to the buying of shares by the general public; they would sell short, and then cover their shorts by open purchases made through their wives; this stimulated more buying; then the counting of votes cast at the meeting of the union would be dragged along so as to afford the time necessary to complete the stock transactions. Thus they see-sawed the market five or six times. This broke the brokers, who had been breaking the public. It was a merciless game of depredation. Never was there a more vivid example of the undermining of character in the presence of wealth; only a few sane men kept their heads in the midst of this scramble for gold, the community as a whole went crazy in the haste to get rich, and threw aside all

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the scruples of business morality and even the conventions of financial rascality.

It was amusing to hear the question discussed whether it was proper to leave 5 ft. of ore under an intermediate drift, and whether such an intermediate constituted a level. I found myself no more in sympathy with the lessee who wanted to gut the mine and was willing to precipitate a caving of the workings, than with the lessor who in his greed insisted that ore must be left untouched even though the removal of it could do no harm to the safety of the workings. From the standpoint of legitimate mining it seemed to me that the richness of the ore ought to have little to do with the removal of it, if necessary to hold the workings. The question of removal ought to be decided on its merits, as a problem in practical mining. A well known attorney told me that he had been asked whether the requirement for leaving 10 ft. of ground around the 'sides' of a shaft meant the 'ends' also. He promptly gave instructions to specify the 'ends' in all new leases. A man who risks the safety of his working shaft in order to extract a few tons of rich ore is no miner, and if he is willing to run the risk of injuring a shaft that will revert to the use of, as it already belongs to, the owner of the mine, then he is a thief, just as much as the 'high-grader'.

Thus at every turn the conclusion is reached that the exploitation of ore of unusual richness is subversive of the integrity alike of the man who digs the ore, the lessee whose property it is, and the lessor to whom the royalty is paid. All of them tried to get more than belonged to them; and when they had finished the scramble for their share, the railroad and the smelter levied a toll that was tantamount to highway robbery. There was not much to choose between them all; it was a case of catch as catch can, with the odds in favor of the one who first placed his hands on the ore—the miner himself.

State Geologists.—Representatives of twenty of the State Geological Surveys meeting at Washington on May 13, organized an Association of American State Geologists. Provision was made for an annual meeting and the appointment of various committees for the transaction of the business of the association. H. B. Kümmel of New Jersey was made president; H. F. Bain of Illinois, secretary; and J. H. Pratt of North Carolina was appointed to act with them, forming an executive committee. Messrs. W. B. Clark of Maryland, I. C. White of West Virginia, and J. H. Pratt were appointed a committee to investigate the distribution of documents by the various surveys. J. M. Clarke of New York was appointed to represent the State Geologists on the general committee on nomenclature now being organized, with Samuel Calvin of Iowa, and E. A. Smith of Alabama, as associates.

'Drift' of a diamond-drill may be caused by the use of a core-barrel until it becomes noticeably worn by abrasion. The bit is frequently renewed and is always the same size, while the pipe back of it (the core-barrel) becomes thinner by wear and is apt to sag, thus giving to the bit a slightly divergent course.

The Prospector.

This department makes a charge of 25 cents to subscribers not in arrears and $3 to non-subscribers for each determination.

From D. J. S., Seven Troughs, Nevada: Basalt with pyrite.

From W. C. B., Minneapolis, Minn., a felted aggregate of capillary crystals of black tourmaline in quartz.

From Alamos, Mexico: A quartzose metamorphic rock, perhaps a conglomerate-schist; sample too small for precise megascopic determination.

From W. B. H., Placer county, Cal.: No. 1, badly weathered volcanic andesite; No. 2, hornblende schist, little pyrite; No. 3, badly weathered volcanic, andesite-tuff; No. 4, talcose schist with pyrite; No. 5, hornblende-chlorite schist with pyrite; No. 6, fine-grained schist with magnetite.

From C. E. S., Sweetwater, Nevada: No. 1, weathered rhyolite or andesite breccia; No. 2, diabase with veins of epidote; No. 3, weathered porphyry with pyrite; No. 4, quartzite; No. 5, limonite; No. 6, epidote; No. 7, weathered volcanic rock, pyritized; No. 8, limonite and hematite in acid tuff.

From W. McF. S., Outlook, Wash.: No. 1, vesicular lava, probably andesite; No. 2, quartz and chlorite; No. 3, highly weathered acid tuff; No. 4, coarsely vesicular andesite; No. 5, weathered andesite; No. 6, chalcedony; No. 7, hematite and limonite; No. 8, aplite; No. 9, fine-grained mica schist with pyrite; No. 10, weathered tuff containing limonite, kaolin, and manganese oxide.

Received frim J. W. A., Pioche, Nevada: Complex ore containing embolite (or bromyrite), silver bromide, probably with a little chlorine, as the most important constituent occuring in small adamantine yellow-green crystals of waxy texture; also present are malachite, chrysocolla, hematite, limonite, manganese oxide, and a brilliant white crystalline substance in too small quantity for determination.

Material sent by O. H., San Francisco: No. 1, weathered andesite or andesite-tuff with pyrite; No. 2, pyritized andesitic tuff; No. 3, andesite tuff with pyrite changing to limonite; No. 4, highly weathered tuff with limonite; No. 5, weathered vesicular basalt or very basic andesite; No. 6, greatly weathered basalt or basic tuff; No. 7, fresh basalt; No. 8, fine-grained basalt with few vesicles filled with calcite and silica; No. 9, altered andesite; No. 10, andesite; No. 11, andesite; No. 12, acid volcanic brecciated and charged with silica and pyrite; No. 13, pyritized andesite veined with silica.

Steel bands for transmission of power have been successfully introduced by a firm in Charlottenburg, Germany. Such belts are made narrower and lighter than leather or rubber, and hence may be made to run much tighter and greater lengths employed. It is claimed that slipping is reduced so that an efficiency in transmission of 99% is obtained.

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GOLDFIELD, NEVADA.—V.

Metallurgical Development.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

The metallurgist waits on the miner. Until the ore is produced, the mill cannot be supplied. At Goldfield there is a good foundation for an interesting metallurgical practice because a large tonnage of rich ore is available. Already smelting, stamp-milling, cyanidation, and chlorination have been applied successfully.

No process of treatment can be used intelligently until the character of the ore is understood. The Goldfield ore posseses three dominant features: (1) high gold content, (2) the presence of tellurides as well as free gold, and (3) alunite. Owing to its richness, it has been possible to transport the product of this district to distant smelters, at Salt Lake, Denver, and San Francisco, where it forms the silicious part of a charge fed into a blast-furnace. Lead is usually the collecting medium for the precious metals, but copper is also used; although high-grade ore is rarely smelted in the copper furnaces. To the metallurgist in charge of the smelter the Goldfield ore is just so much silica needed to combine with his iron and lime in order to form a slag of suitable fusibility. The richness of the ore affects the sampling more than the actual smelting. Great care is required to obtain a true sample, and it is found necessary to pass a large proportion of a shipment through breakers and rolls before obtaining the fractional portion that is assayed. The alumina in the Goldfield ore retards the fusibility of the slag; usually there is about 7% alumina in the smelting mixture. But, on the whole, the commercial drawback to handling such rich ore is more serious than any technical difficulty; a carload of 40 tons containing $12,000 worth of gold represents a good deal of money, and it makes a great difference as to when the payment is made. The miner wants his money right away; the smelter would like to postpone payment until he gets his check from the refinery or the mint; between the date of shipment from the mine and the date when the gold and silver in the ore are available as currency, there is an interval of 60 to 90 days. Capital is tied up for three months, the miner objects to waiting, the smelter objects to loaning money even on so good a collateral as gold ore. The result is a compromise by which the miner pays his share—and sometimes more—of the interest on the suspense account.

Most mining districts begin to market their ores by shipment to the nearest smelters or custom-mills. They pay through the nose for treatment and transport, but they cannot escape this tax on their production until the reserves of ore warrant the erection of local mills. Moreover, in the early days of development it is not certain what is to be the average character of the ore and it is not possible therefore to ascertain the most suitable method of reduction. Usually a 5-stamp mill is the pioneer metallurgical unit. At Goldfield, the owners of the Combination mine were the leaders in local metallurgical research. From December 1903 to May 1905 the ore was shipped to the smelters; first it was hauled in wagons to Candelaria, a distance of 65 miles; later, on the completion of the railroad to Tonopah, it was carried thither, a distance of 30 miles. The distance was halved and time was saved, but the cost of transport remained much the same. It was decided to erect a mill at the mine. Tests were made in San Francisco and at the mine by F. L. Bosqui. It was found that crushing dry, followed by cyanidation, left some coarse gold undissolved in the tailing. Amalgamation was needed to supplement cyanidation. A small portion of the oxidized ore carried an excessive amount of acid due to decomposed sulphides. As much as 50 lb. lime per ton was added to neutralize this acid. Finally, an extraction of 90% was obtained on the oxidized ore. The sulphide ore, then unimportant, was more refractory. Pan amalgamation after roasting, concentration with oil, chlorination, and leaching, were tried in turn before it was decided to concentrate the rich gold-bearing pyrite, and cyanide the tailing from the sulphide ore.[1] When these tests had been concluded, a 10-stamp mill was built and to this plant 10 more stamps were subsequently added. The treatment of the sulphide ore is shown diagrammatically in Fig. 1, which is borrowed from Mr. Bosqui's description, already mentioned.

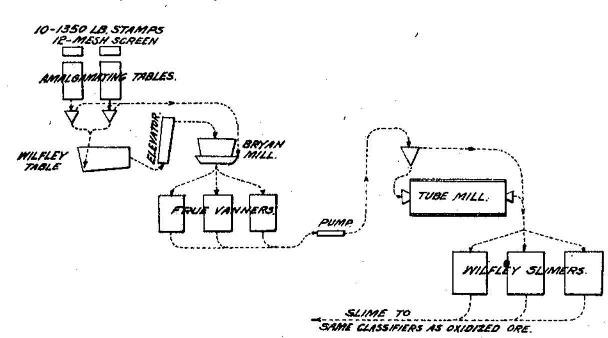

Fig. 1. Treatment of Sulphide Ore.

The Combination mill does not now impress one as a model plant, simply because it has undergone such successive additions and changes as are destructive to any unity of design, but, such as it is, this mill has solved the metallurgical problems of the district and afforded the information on which newer, larger, and prettier mills are being planned.

Before discussing the milling methods further, let us glance at the ore. It is essentially a silicified dacite enriched with gold. The altered dacite contains[2] 50% quartz, 24% kaolinite, 15% alunite, 7% pyrite, 2½% water. The alunite and kaolinite are obstacles to filtering and therefore to leaching. The alunite, being a hydrous sulphate, is destructive of cyanide and requires neutralization with lime. When the altered dacite becomes rich enough in gold to be classed as ore, it becomes more quartzose, the percentage of silica increasing from 50 to 75% or more. The ore contains pyrite (iron sulphide) in small grains, bismuthinite (bismuth sulphide) in needle-shaped steel-gray crystals, and a reddish-gray mineral resembling tetrahedrite (the gray copper of the miner).[3] The gold accompanies the gray

[1] 'Ore Treatment at the Combination Mine,' by Francis L. Bosqui. MINING AND SCIENTIFIC PRESS, October 6, 1906.
[2] 'Association of Alunite with Gold.' F. L. Ransome. *Economic Geology*. Vol. II, p. 678.

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copper and the bismuthinite, the sulphides being often arranged concentrically around fragments of brecciated dacite. The native gold is rarely coarse. It occurs usually in particles so minute and so closely packed as to resemble a streak of old-gold paint. The proportion of silver to gold is as 65 to 1, so that there is a marked difference compared to Tonopah, where the silver is predominant. The Goldfield ore also contains tellurides, not in important proportion from the mineral-collector's standpoint, but enough to interest the metallurgist. Ransome mentions the occurrence of undetermined tellurides in ore from the Jumbo Extension. Lochiel M. King states[4] that, by actual analysis, he has detected calaverite, the telluride of gold, in Mohawk ore, and that he believes the greater part of the gold in the low-grade sulphide ore to be present as a telluride. The last statement is disputed by Edgar A. Collins, the former manager of the Combination mine. However, there is no doubt that tellurides exist in the ore, whether in large or small proportion remains to be proved. They constitute part of the metallurgical problem. Neither zinc nor lead minerals are noted. A little chalcopyrite has been seen in the Florence and Sandstorm mines.

The quartz, which is the predominant accessory mineral, is usually flinty in texture. Crystalline cavities are absent, but pockets of alunite are frequent. These explain the difficulty in filtering the pulp after the ore has been crushed, and the necessity for special devices in the mill. The coarseness of some of the gold aggregates explains why amalgamation is needed to supplement cyanidation. The matrix of flinty quartz encasing the finer particles of gold necessitates fine-grinding. The fineness of much of the gold justifies the use of a solvent, such as cyanide, after amalgamation. Whether roasting and dry-crushing are better than milling raw ore is a disputed question, with the preponderance of opinion in favor of the latter practice.

The Combination mill has been described in this journal by Mr. Bosqui, so it is not necessary to repeat. The same description appears in the volume entitled 'Recent Cyanide Practice.' When I went through the mill, on April 10, I noticed a broad streak of fine gold, like a band of paint, at the head of the Deister concentrating table, which receives the re-ground product from the tube-mill. This testifies, to the advisability of using amalgamation. At the time of my visit the amalgamating plates in front of the stamp-batteries were blinded or covered with old vanner belts. (This was being done experimentally. In the new mill there will be thorough amalgamation before and after tube-milling.) From the battery the ore went to classifiers, the coarse passing to the tube-mill, while the fine was washed over supplementary amalgamating plates; both products went to the concentrators, one Deister and six vanners. The Deister table received the classified product, all finer than 200-mesh, from the cone-classifiers. From the vanners the pulp went to the cyanide annex. That band of gold on the concentrating table spoke eloquently for the usefulness of amalgamation. If you can save the gold at the very start, within or just outside the stamp-battery, why not do so? The metallurgist who pins his faith wholly on his cyanide annex is like the sportsman who does not mind missing with his right barrel, expecting to hit his bird with the left barrel, which is choke-bore and reaches farther. He wastes ammunition and is apt to miss his bird after all, for the farther it flies from him the more difficult it is to hit.

After the Combination mill came that of the Nevada Gold Reduction Co. This plant includes a sampler, with a capacity of 450 tons per diem, and a cyanide mill, treating 100 tons per diem. The surplus ore is sold to the smelting companies. The equipment of the mill includes 20 stamps, of 1250 lb. each, by which the ore is crushed to 12 mesh in a 0.50 to 1.25 lb. KCy solution. The pulp on issuing from the mortar goes over amalgamating plates and then to 8 Wilfley concentrators, classifying into three products, namely, concentrate, sand, and slime. The concentrate assays from $200 to $1000 per ton and is shipped to the smelters; the sand goes first to an Allis-Chalmers tube-mill, 5 by 22 ft., reducing it to a screen-size of less than 100 apertures per linear inch; from the tube the re-ground product passes over supplementary amalgamating plates and then to classifiers, the oversize from which is re-ground after going over a Wilfley. The leaching plant, which receives the pulp after the free gold and rich sulphides have been partly extracted by concentrators, includes eight vats of 22 by 5 ft. and three agitator-vats of 24 by 16 ft. Three vacuum-filters of the Butters type strain the gold-bearing solution, which is further clarified by passage through a 5-ton American filter-press, remodeled by E. S. Leaver, the superintendent of the mill.

Next in point of time, and a neighbor of the Nevada Gold Reduction Co.'s mill, is the Goldfield Cl Mill Co. This company has adopted the symbol of the chemical element chlorine as part of its corporate name. The plant is not yet in regular operation, although a trial run has been made. It is a most interesting departure and is due to the initiative of John E. Greenawalt, well known in Colorado as a chlorination expert.

The roasting furnace is a rabble roaster of the muffle type; it has a porous hearth and is heated with producer-gas manufactured from coal (coming from Helper, in Utah) enriched with crude oil (from Bakersfield, California). From this furnace the roasted ore goes to a cooling-cylinder, at the end of which it is moistened before delivery into a bin. From the bin it is transferred to large vats by means of a grab-bucket of 60 cu. ft. (nearly 3 tons) capacity, operated by a traveling electric crane. The chlorination vats are of wood, sealed by a wooden cover dipping into a launder filled with water. When silver ore is being treated 2% salt is added, before crushing, so as to get a good mixture. The leaching in the vats is done by a strong solution of chlorine

[3] I am informed later by Mr. C. D. Wilkinson that the above description is true of the Florence and Combination ores. In the Mohawk and Red Top the needle-shaped crystals do not appear and neither does the tetrahedrite. The occurrence of calaverite is still a question, as Mr. King made his determination by the ratio of gold and tellurium only; the higher-grade ore contains free gold, which will destroy this ratio.

[4] 'Cyanidation in Nevada.' Lochiel M. King. MINING AND SCIENTIFIC PRESS, January 25, 1908.

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in water—7 to 8 lb. free chlorine per ton of solution. Leaching requires 3 to 4 days. The seven vats are each 22 ft. diam. and 8 ft. deep. The gold solution is withdrawn by gravity and pumped into a storage-vat and thence drawn to the precipitating-boxes, resembling the zinc-box of a cyanide plant. The precipitation of the gold is effected electrolytically, the anode being graphitized carbon and the cathode lead-shaving containing 1% zinc, the presence of the zinc hastening the corrosive action. The slime of lead, gold, and silver while still moist is mixed with flux and briquetted. The briquettes are melted in a reverberatory furnace and then refined in a cupel.

The chlorine is generated electrolytically. At first salt was obtained from Hazen, in Nevada, but it was found to contain over 5% silica, from sand blown across the dry lake; the salt now used comes from Salt Lake and costs $16.25 delivered. The solution of chlorine water is made in an absorbing tower; this consists of two sheet-iron stacks lined with glazed sewer-pipe, with cement between the pipe and the outer iron. Inside is placed a filling of hollow clay balls, each of them having eight perforations. The chlorine gas is introduced at the bottom of the stacks and water is made to trickle down from the top, the large surface thus offered facilitating absorption.

The mill has a nominal capacity of 100 tons per diem. The ore is crushed to 14 mesh and the roaster is 100 ft. long. A peculiar feature of the plant is the use of piping made of hard rubber; the pumps are actually made of it and look like cast iron until one is informed of their true composition; then they look like chocolate. Of course, iron and brass would be attacked by the hydrochloric acid, hence the use of wood in the vats and rubber in the pumps and pipes. This mill was not ready to receive custom ore at the time of my visit (in April), and yet I stated, in the opening paragraph of this article, that chlorination had been applied successfully at Goldfield. A metallurgical process is applied successfully when it yields a profit to the operator. I was referring to the use of chlorination in treating 'high-grade,' that is, stolen ore. For this purpose it is convenient, and although the percentage of extraction is not anything like that to be made by Mr. Greenawalt, it is adequate for the purposes of a 'fence,' where only 50% of the value of the gold in the stolen ore is received by the thief.

Mr. Greenawalt is an enthusiastic chlorinator and deserves success. His mill should afford a market for the rich concentrate obtained as a by-product in the cyanide plants. Mr. Greenawalt keeps some gold leaf on hand and performs a simple experiment to illustrate the greater efficiency of chlorine as a solvent compared to cyanide. In one beaker he placed a 5-lb. or 0.25% cyanide solution and in the other a 7 to 8-lb. or 0.33% chlorine solution. Dropping a fragment of leaf gold into both of these solutions, it was seen that the chlorine dissolved the metal instantly, while the cyanide had not completely eaten up the gold in an hour, a few particles of gold remaining at the bottom of the beaker, where oxygen was not available. It may be proper to point out that gold leaf does not exactly simulate the condition of gold in ore; further, cyanidation may be possible without previous roasting, but chlorination is rarely successful on raw ores. In cyanidation, the rate of solution decreases rapidly as the particles of gold increase in size, and in this respect chlorine is a more energetic solvent.

On the northwestern slope of Columbia Mtn. the Goldfield Consolidated Mines Co. is building a large stamp-mill and cyanide annex, in which will be made available all the experience obtained in the Combination mill and the other plants treating ores of similar character. At the time of my visit the concrete foundations were being laid at the rate of 100 cubic yards per day, these operations being facilitated by a crushing-plant, the rock used for concrete passing first through a Gates gyratory crusher and then through a Dodge jaw-breaker; then the broken stuff was elevated to a trommel, the oversize going to bins above a concrete-mixer and the undersize to rolls, which reduced it fine enough to make sand. This sand was an angular product well suited for making concrete. The cement comes from Iola, in Kansas. Fully 5000 cu. yd. of concrete will be required. Five thousand barrels of cement have been ordered, at a cost of $4 per bbl. delivered. A barrel holds 380 lb. On my way to the mill-site I saw a clever device used in grading for a new reservoir, to be connected with the new mill. A piece of sheet iron (an old turn-sheet) ¼ in. thick, 5 ft. wide, and 12 ft. long was being used in place of a dump-cart, as shown in the photograph (Fig. 3). The broken rock was shoveled onto this piece of sheet iron, and it was easy to roll rocks upon it (avoiding any lifting) and then to drag the load to the dump, where it swung over the edge so as to discharge itself automatically. This reservoir is to be made by using the solid rock for a back wall and half of the sides, simply plastering the surface to make it water-tight. The dimensions are 65 by 150 ft., and 15 ft. high. Steel would have cost $14,000, wooden tanks $10,500, concrete construction as outlined above will cost only $8000.[5]

On arrival at the mill-site, I found a scene of great activity, for the foundations of the ore-bins were being laid, and in the temporary office I met F. L. Bosqui, metallurgical engineer, who worked out the scheme of treatment; G. B. Shipley, the representative of the Allis-Chalmers Co., which has obtained the contract to supply the machinery; and J. B. Fleming, the mechanical engineer for the Goldfield Consolidated Mines Company.

The mill is two miles north of the mines; the ore will be transported over a railroad in 25-ton hopper-bottom cars and deliver into a bin of 850 tons capacity. The mill is expected to treat 600 tons per diem. From the bins the ore passes to a Gates gyratory crusher, which delivers it by elevator to a revolving screen (4 by 14 ft.) having 1½-in. aper-

[5]In response to a later enquiry as to the progress of the work, I am informed by Mr. J. H. Mackenzie, the general manager for the Goldfield Consolidated Mines Co., that the construction of a concrete reservoir has been abandoned owing to the time required. A water-supply is needed forthwith. So eight tanks made of redwood and 30 ft. diam. with 20-ft. staves will be placed on the reservoir site and two of the same size at the mill. The total capacity will be 1,000,000 gallons.

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tures. The undersize goes to a belt-conveyor; the oversize goes to two Gates crushers and then to the belt-conveyor. This is 26 in. wide; it elevates the final product from the ore-breaker up a slope of 20° and 370 ft. long, through a Blake-Denison weighing-machine, into the sampler. From the sampler the ore is distributed by a belt running over the mill-bins, which have a capacity of 4000 tons. Suspended Challenge feeders deliver the ore into 20 mortars, each containing 5 stamps. The mortars are of the narrow Homestake pattern, but with broad bases and extra heavy, set on concrete blocks. After being crushed under the 1050-lb. stamps the ore passes over amalgamating plates made of silver-plated copper, 5 ft. wide by 16 ft. long. Thence the pulp descends to 20 double-cone classifiers, of 24-in. diam. The overflow passes directly to concentrators while the underflow goes to 6 Dorr classifiers, which thicken the pulp previous to its introduction into

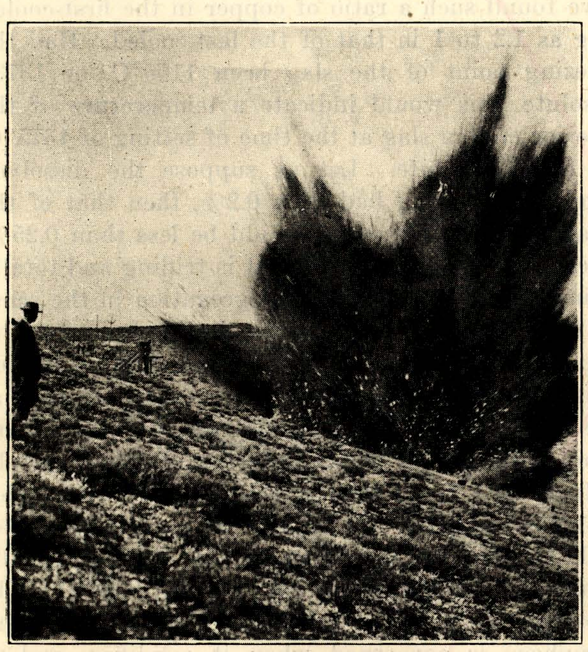

Fig. 2. Blasting for Mill Foundations.

tube-mills. There are six tube-mills, each 5 by 22 ft., of the Allis-Chalmers trunnion type, lined with 4-in. silex blocks. The product from the tubes is classified in four 48-in. double cones, the overflow from which passes to 14 secondary amalgamating plates, while the underflow is returned to the tube-mills.

The slimed pulp, of which 80% will pass a 200-mesh screen, joins the overflow from the secondary amalgamating plates and proceeds to the concentrators. Concentration will be done in one stage only, namely, after tube-milling. The concentrator used will be either the suspended Frue vanner or the Deister table, and 60 machines will be required. The stream from these vanners will join the overflow from the Dorr classifiers and pass to the settling department. This consists of 16 tanks of 30 ft. diam., 12 ft. deep, provided with 16° cone-bottoms. In these tanks, by means of serial decantation, the pulp will be reduced to a consistence of 50% moisture. Then it will be transferred, partly by gravity and partly with the aid of a large centrifugal pump, into steel agitator-vats operated on the air-lift principle and known as Pachuca tanks. There will be ten of them, 15 ft. diam. and 45 ft. deep. The water overflow from the settlers will be conveyed to a series of clarifying tanks and raised to the mill-reservoir, whence it will be again put into circulation through the mill.

From the Pachuca agitators, where the pulp will be circulated in a cyanide solution for 10 to 18 hours, the material will be transferred to large reservoirs, in which it will be kept in gentle motion by means of a stirring mechanism, and from these reservoirs it will be drawn by gravity as required into the boxes of the Butters filters. There will be two of these boxes, constructed of steel and with a capacity

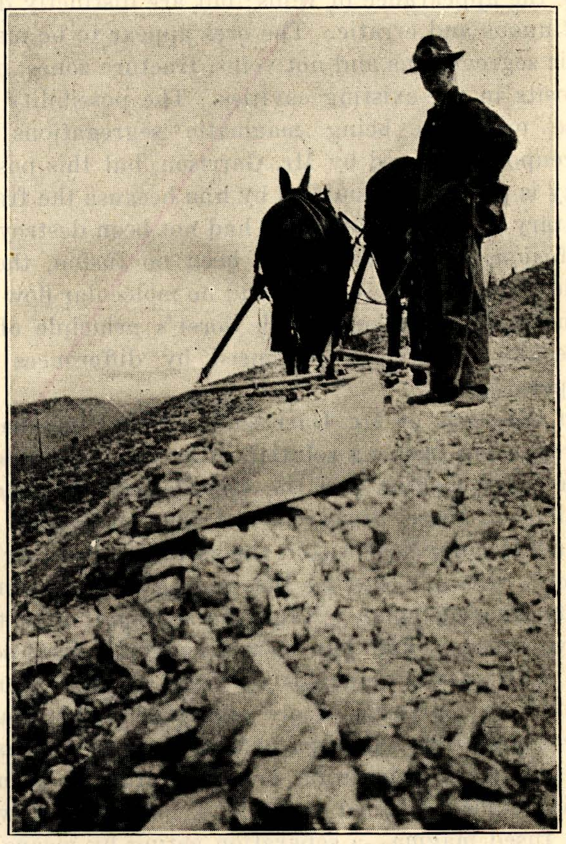

Fig. 3. Grading for Reservoir.

of 168 filter-leaves. After the deposition of the cake on the frames the surplus pulp will be drawn off by gravity into a large vat on a lower level. Similar vats will be provided for both water and solution, that is, the Butters plant will be operated by gravity throughout.

The solutions from the Butters filters will be clarified in three Perrin filter-presses, each having 50 frames 36 in. square. From these clarifying presses the solution will be drawn to the precipitation vats, where zinc dust will be added in accordance with the Merrill method. The precipitate will be gathered in four 30-frame Merrill presses, and the product melted direct (without acid treatment) in five Faber du Faur tilting furnaces.

The experience of several years in the Combination mill, and special experiments besides, justifies Mr. Bosqui, the consulting metallurgist to the Consolidated Company, in expecting an extraction of 95% of the gold, the silver content of the ore being negligible. The total cost of milling is to be under $2.50 per ton—a low figure, having regard to the high prices of water, power, and labor in this locality.

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GOLDFIELD, NEVADA.—VI.

Automobiles and Gambling.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

Mention has been made of an automobile road between Goldfield and Tonopah. The automobile is the symbol of the restless energy of these Nevada mining districts. In a bygone time men went forth to explore in sailing ships; in Mexico the mule typified transport; in the old West the patient little burro has been the mainstay of the prospector; in the North, the dog-team and the reindeer aided the mineral explorer; in Western Australia we used to ride on camels across the desert in which rich gold deposits were discovered; but in modern Nevada the mechancial genius of the age has given man a method of locomotion more swift and more flexible than any beast of burden. All the explorer needs is gasoline, instead of fodder or water. At Goldfield, gasoline costs 39 cents per gal. in cans, and 30c. in tanks. Early in the development of Tonopah, Goldfield, Rhyolite, and Manhattan the problem of rapid transit was solved by an automobile service connecting the important camps. The 28 miles between Tonopah and Goldfield has been made in 35 minutes. The usual time was 50 minutes. The regular fare used to be $6. The automobiles had three seats and carried 9 passengers apiece. In going over a bit of the old road, on the occasion of a visit to the Daisy mine, I asked Mr. C. Walter Geddes how the road was made. The automobile road between Goldfield and Tonopah was made by a stage company organized by L. L. Patrick, formerly of Leadville. First, the sagebrush was cleared by dragging a piece of T-rail placed at right angles to the line of the road. Then ruts were marked by taking an automobile over the cleared ground; along those ruts two heavy chains were dragged behind an automobile to establish a track. Another method is to take a scraper or go-devil, somewhat after the fashion of a snow-plough; this is dragged by 8 or 10 horses, so as to uproot the sagebrush and throw loose stones and other obstructions to either side. Then a double chain or pipe (6 to 8 in. diam.) is dragged over the ground; this makes parallel ruts, serving as a track of the correct gauge. The passage of the automobiles soon makes an excellent road, but after a while bumps or ridges are formed in the ruts. These riffles are of doubtful origin; some impute them to the vibration of the engine; it is probable that the traction or slip of the rear (or driving) wheels, or more probably the irregularities in tractive effort due to vibration of the sprocket-chain, causes a kick and so makes the corrugations, which gradually spoil the auto-road. When this has become a nuisance, it is customary to pull a pair of pipes along the road and smooth it. However, even with this precaution, an automobile road in the desert soon gets out of repair. Originally straight, every sway of the fast-moving machine causes a swerve from the direct line; the next machine follows its predecessor and accentuates the deviation, until the road becomes curiously serpentine, like the trails in an African forest, where the native steps aside for every fallen tree or obstructing stone, rather than clear the track and keep it straight.

The fare to Rhyolite from Goldfield (65 miles) used to be $25 for a single passenger; it is now $4.55. The railroad killed motor travel, as might be expected. Drivers of automobiles receive $8 to $10 per day, and most of them are skillful chauffeurs, both to steer and to repair. Local newspapers call them 'mahouts,' a Hindustani term borrowed from elephant service, and absurdly out of place in the desert.

To use an automobile in traveling between the mining settlements is a comparatively new departure, but the most striking use of these machines is in hastening to a new mining excitement or 'rush,' so as to be early on the ground and locate a claim near the place of discovery. Even more than this, the automobile has been used in going across country, independent of any road, in order to see a new find of golden ore. For such work a machine with a high clearance is needed. Thus a party of mining men went 140 miles north of Tonopah across the sagebrush plain. The car hit a projecting rock, hidden in the brush; the front axle was bent almost double and a hole was torn in the crank-shaft case. These up-to-date explorers made a fire with sagebrush and straightened the axle by hammering it while hot with stones. The hole in the crank-shaft case was repaired with a piece of rubber, a part of a gunny-sack, the fragment of a coal-oil can, all of which was bound with baling wire. (Good old baling wire!) They made their return serenely. If they had not been able to make these repairs, their plight would have been serious indeed.

At Goldfield the horses have become accustomed to the whirring chug-chugging motor-vehicles, which were introduced almost as soon as the horses themselves. At Oroville, in California, there is a county law forbidding motorists to travel by day, and the time is coming when special roads will be built for this purpose all over the country, for the preservation as much of the macadam roads, which the automobiles are fast destroying, as of the comfort of those who do not happen to be in the motor-car. This reminds me of a visit to the copper mines of Michigan four years ago, when automobiles were new on the Keweenaw peninsula. Mr. John R. Stanton took me from Houghton to the Atlantic mine. The scenes by the roadside were like a continuous and screaming farce, threatening at times to turn into a tragedy. We met a dozen Cornish miners on horseback with tin dinner-buckets, and as soon as their steeds saw the machine there was a separation and scattering of Cousin Jacks, dinner-pails, and frightened horses among the rocks and brush such as no African lion could have effected. On the way we met farmers' wagons, country folk going to town, single horsemen, and in nearly every case there was a threatened mixing of the scenery and the wayfarers. I presume that long before this the inhabitants of the Upper Peninsula have become accustomed to the automobile.

At Schurz, on my return from Goldfield, I saw the rush to Rawhide. This new 'excitement' is 30

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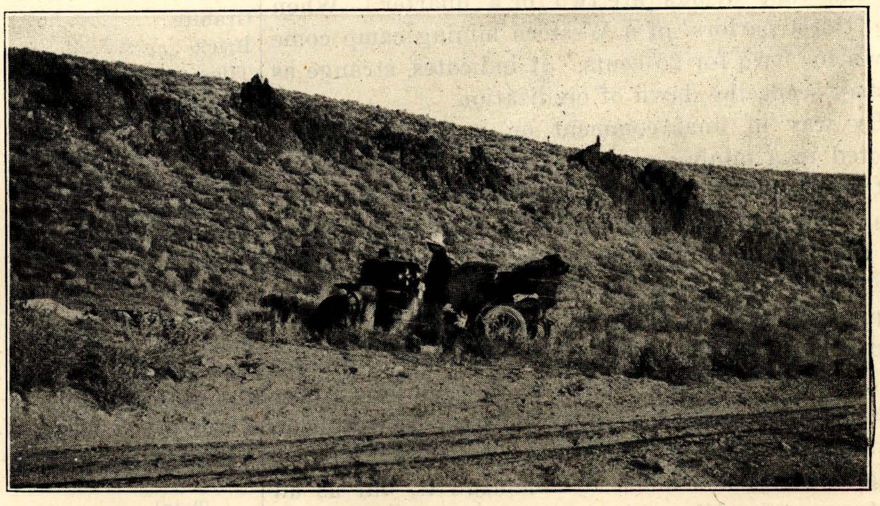
miles from Schurz, and the railroad station was crowded with automobiles awaiting passengers for the mines. The accompanying photograph shows how a Nevada rush is assisted by the automobile.

Last year ten or twelve men made a million or more apiece from the ore produced by the mines of Goldfield; of these, only five have retained their wealth. The others wasted their money, speculated wildly, got caught in the panic, and so forth. Men made money fast, but they also lost it rapidly by over-reaching.

At Schurz, on the Way to Rawhide.

On the train when I returned from Goldfield to San Francisco, there were two young brokers in our smoking compartment, and these two explained all that there was to know about mining and the business of mining. It was obvious that to them mines were made to serve as counters in a game of brokerage-finance; they themselves were the real thing; the engineers and metallurgists, even the mine-owners, were but the frill on the edge of things. The conversation drifted to gambling-saloons and it was stated how much money these establishments made, then it transpired that the proprietors of such places lost to the brokers by playing the stock-market, and finally the brokers got cleaned out by going into a lot of wild-cat schemes that exploded at the time of the panic last October. And, of course, the public lost. So everyone lost. "Where did the money go?" asked one of the smart followers of Philistine finance. It was then that I ventured to suggest to these patrons of mining, as they deemed themselves, that the money had gone into hot air. Whatever wealth had been created by the extraction of gold from the ground was still in existence, of course, but it was the difference between the intrinsic worth of the mines and the fictitious value placed upon them by flamboyant promoters and tricksters that had evaporated suddenly. It was like the settlement at the cross-roads where they passed a $1,000,000 note from one to another and thought themselves gorgeously rich, until they tried to discount that note on the outside in the purchase of a keg of beer. In New Mexico they tell a story of two Indians who invested their earnings in the purchase of a jug of whisky, intending to start a saloon and make a fortune. They were both very thirsty, and one of them had 5 cents remaining from his savings, so being thirsty, he insisted on buying whisky from his partner. His partner then felt dry, and used the same 5 cents to buy whisky from the first Indian. The exchange was continued actively until there was no more whisky in the jug. This story belongs to Charles F. Lummis.

Gambling establishments play a prominent part in the life of the mining community, and rank with the Stock Exchange as a place for the making and marring of a fortune. At the Goldfield Hotel the roulette wheel makes $11,000 profit per month nowadays, but this is small compared to the profits made by the leading gambling place, the Great Northern, during the boom days. Then on a capital of $100,000 the returns were as much as $50,000 per month. As far as I could learn, the maximum loss of an individual player was $50,000 and the maximum winning $30,000. The gambling is usually the sequel to a drunken spree, the men of sense and position who have "played the game" (roulette) will tell you that it was usually after "wining." 'Wine' means champagne. In the dry bracing air of the desert plateau, the stimulating effect of champagne is quickly felt



Broken Down in the Desert.

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and soon develops intoxication. The hotel bar and the roulette table, the ordinary saloon and its gambling annex, are eloquent witnesses to the force of juxtaposition. Besides, the players receive drinks and cigars free, that is, they are given by 'the house'. This tends to keep the gamblers in a half-fuddled condition. Their stupidity plus the chances of the table are enough to make the game profitable to the proprietor. There are 36 numbers, besides a zero and a double zero; thus the margin for the house is 2 in 38, or 5.26% in its favor. As in the other game of chance—that of the Stock Market—the average player will quit when he has made a small winning, but he will take a large loss before stopping; thus the big losses are always smaller than the big winnings. Many a man with a few dollars jingling in his pocket will drop into a gambling saloon and "take a flyer"; when his pockets are empty he will go home; if he wins, he will remain until the luck turns and he has been cleaned out. Moreover, in betting on the black or red, the even or odd, the high or low, the bets of the players are apt to balance, and thus they save the liability of the house, which gets its 5.26% steadily from each play. The license fee is $75 per month per game. The dealers are paid $8 per day of 8 hours, the regular rate being $1 per hour. The proprietor has to pay interest on his bank roll, say, $50,000. He has to insure himself against robbery, which is frequent. Even today at the Goldfield Hotel, which in equipment and comfort and prices resembles a New York hotel, there is a man with a sawed-off shot-gun sitting behind a screen on a perch placed so as to command the bank-roll of the dealer at the roulette table. This savors of the frontier, as does the 25 cents for a shoe-shine and the 10 cents for a daily newspaper. The only alleviation is the fact that 'drinks' are two for a 'quarter.' When the 'thirst parlors' of a Western mining camp come down to "two for 25 cents," it indicates, strange as it may seem, the dawn of civilization.

By way of final comment on gambling as conducted in a mining camp, I venture to say that to bet on the red or the black is more sane than to buy shares on a margin and, upon the whole, the player at the roulette table gets a better show than the speculator on the stock exchange. Certainly, the game is far less crooked than that of the race-track and in itself is less insane. The evil of the gambling in a mining camp lies mainly in its associations, for the gambling annex stands mid-way between the drinking-saloon and the brothel. As a basis for betting, roulette has much to commend it, but as an instrument in debauching manhood it is altogether hateful.

The Japanese Imperial Iron Foundry, at Wakamatsu, is unable to compete with foreign imports of steel and iron because of the higher cost of production in Japan. A sum equal to over twenty-five million dollars has already been spent on the plant. The director of the foundry states that he is awaiting the lapse of the conventional tariffs in 1911, when it will be possible to impose duties on iron and steel.

MINING IN QUEBEC.

A review of mining operations in 1907, and of future prospects in the Province of Quebec, Canada, is presented in the annual report to the Government by J. Obalski, Superintendent of Mines. Asbestos was by far the most important mineral production, both in quantity and value. No less than 12 separate companies are mining this substance, among the largest being the Asbestos Mining & Mfg. Co., of Providence, R. I., operating at Wolfestown; and the Asbestos & Asbestic Co., whose plant is at Danville. Most of the asbestos mines are open quarries. With reference to gross value, cement, granite, bricks, tiles and pottery, limestone, mica, and lime, rank next, in the order named. Two new companies have started the manufacture of cement, making three in all, the centre of the industry being near Montreal. Of the metallic minerals, copper ranks highest, the principal producers being the Eustis Mining Co. and the Nichols Chemical Co., both in the Capelton region, Sherbrooke county. Of the 29,574 tons of ore mined by all companies, 19,933 was treated at Capelton and the remainder shipped to the United States. The pig-iron industry, ranking next in importance, is practically under the control of two companies: the Canada Iron Furnace Co., Ltd., at Radnor, and John McDougall & Co., at Drummondville. The 10,047 tons produced required for reduction 22,681 tons of ore, 11,511 tons of charcoal, and 4300 tons of limestone. Chrome iron was manufactured during the year by three companies in the township of Colraine. A summary of the total production of all classes of minerals is presented in the following table:

Mineral.	Amount.	Gross Value.
Asbestos	61,985 tons	$2,455,919
Cement	....	640,000
Granite	51,873 cu. yd.	560,236
Brick	94,000 M.	525,000
Tiles and pottery	....	270,000
Limestone	97,710 cu. yd.	223,580
Mica (trimmed)	550,247 lb.	199,848
Copper ore	29,574 tons	160,455
Lime	556,000 bu.	96,000
Bog iron ore	22,681 tons	80,231
Chrome iron	6,407 tons	63,130
Calcined ochre	2,300 tons	29,430
Asbestic	29,193 tons	27,293
Mica (crude)	150 tons	24,030
Slate	4,336 squares	20,056
Raw ochre	2,700 tons	5,400
Prepared graphite	120 tons	5,000
Phosphate of lime	408 tons	3,410
Flagstone	3,000 sq. yd.	2,550
Total		$5,391,368

It will be noticed that the above list includes neither of the precious metals. Mr. Obalski estimates that between 1863 and 1878 about $2,000,000 worth of gold was taken from the valley of the river Gilbert, but that since then mining has been carried on irregularly and on a small scale. He also suggests certain localities in which he thinks it might be worth while to prospect for gold.

The strength of copper is materially reduced at temperatures exceeding 500° Fahrenheit.

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VACUUM SLIME-FILTERS AT GOLDFIELD.

Written for the MINING AND SCIENTIFIC PRESS
By ALFRED MERRITT SMITH.

The Butters vacuum filter plant at the Nevada Goldfield Reduction Co.'s mill at Goldfield, Nevada, is one of the earlier installations, and various methods of operation have been tried during the last four years, with a view to securing the most economical results. At the time the filter plant was installed it was not deemed practicable to erect the 'semi-gravity' type, whereby the stock-pulp is gravitated into and out of the filter-boxes as required from pulp-tanks placed respectively above and below the filter-boxes. The mill is situated on level ground, hence the filter-boxes were elevated quite high in order to secure sufficient dump-room for future accummulation of slime residues. About 12 ft. below the level of the filter-boxes is placed the one stock-pulp tank required, from which the pulp is pumped directly to and from the filter-boxes. The pumping is accomplished by a 6-in. Butters centrifugal pump, provided with the usual arrangement of valves for reversing the operation. The vacuum-pumps are two in number, of the Smith-Vaile 10 by 10-in. single type, and the working vacuum varies from 15 to 25 in. The filter-boxes are three in number, having 15 leaves each, 45 in all. Leaves of a special design by E. S. Leaver have been in use for over three years, the essential difference from the Butters leaves being that grooved wooden slats are used as a filling for the canvas leaf, instead of the canvas being sewed upon a cocoa matting filler.

Assuming general familiarity with the operation of vacuum slime-filters of the modern type, I will briefly describe the evolution of our filter work here to the present stage. It is known in milling circles where slime-filters of the Butters make have been adopted, that there is a continual enrichment of the wash water, or wash solution, by osmosis, which in the filtration of high-grade slime will result in material losses of gold. For example, the cakes having been formed on the leaves and the excess pulp returned to its tank, clean water is pumped or gravitated into the filter-boxes for washing, and the vacuum again applied to draw wash water through the cakes. But this clean water, coming into contact with the comparatively large area of slime cake, pregnant with gold solution, immediately absorbs and diffuses a portion of the gold and cyanide. After the required amount of this wash water has been drawn through the cakes by means of the applied vacuum, the excess of 'wash' is run back to a tank, to be used again for the same purpose, carrying with it an increment of gold and cyanide. This gold and cyanide in the wash is cumulative, increasing with each cycle of operation. The quantity of water necessary to replace that which is drawn through the leaves, and also that which is discharged with the residue in ordinary work, is not sufficient to prevent a gradual enrichment of the reserved wash.

It was our early practice here, when the wash water had increased in assay value from nothing to about $1.50 or $1.75 per ton, to discharge the whole of it into the battery solution tanks. As the original crushing is done in cyanide solution, this provided a way to save a part of the loss. A fresh supply of clean water was then taken in for filter wash. This, however, did not save the cyanide and gold remaining in the wash water which was necessary to run out the slime-residue, amounting to several cents per ton of dry slime in treating the high-grade ores of Goldfield. Double washing was next tried. The cakes were first thoroughly washed with weak barren sump-solution, the whole of the excess wash being returned to a separate tank. The boxes were re-filled with clean water, the vacuum applied for five minutes, to re-wash slightly, the cakes were dropped, and the excess of water returned to the water tank, enough water being retained to discharge the residue in the usual way. In theory this method seems almost perfect, as the loss of gold by osmosis is reduced to almost nothing, and the volume of working mill-solution is not materially increased by an additional five-minute water wash. The objections were, first, a double exposure to the air and the washing action, frequently causes much of the cake to loosen and drop off from the leaves prematurely, and second, more time and pumping is necessary to complete a cycle of operations. The first of these objections is not serious, as it cannot overcome the primary object, that is, the prevention of gold loss by osmosis, for this is obtained by saving all of the first wash solution, none of which is used to sluice out the residue.

The method now in use, which allows the filters to be worked at their full capacity, and at the same time minimizes the loss by osmosis, is as follows: The cakes being formed and the stock pulp returned, the boxes are filled with weak barren sump-solution and sufficiently washed. When the wash is completed, an excess of wash-solution is pumped back to a storage tank, enough being retained to flush out the residue. The discharged residue is run into a tailing dam, settled, and clear solution is drawn off by means of a gate or weir, to a pit, from which it is pumped back to the mill, to be used again as filter wash or as battery solution. Clean water is run into the residue pond to the amount of fifteen or twenty thousand gallons per day, as a further wash and to absorb and save a portion of the gold-bearing solution which remains in the residue. This water is returned to the mill, and is ordinarily sufficient in quantity to preserve the equilibrium of the mill solutions.

Below is a sample copy of the record kept for each filter-box charge, showing the distribution of time in a complete cycle:

Charge No. 4987. June 4, 1909.

Filter-box No. 2.	A.M.		A.M.	Hr.	Min.
Filling filter-box with stock pulp	6:52	to	7:13		21
Period vacuum applied	7:13	"	8:18	1	5
Pumping back excess pulp	8:18	"	8:35		17
Pumping on wash solution	8:37	"	9:00		23
Time washing	9:00	"	10:00	1	
Dropping cakes	10:00	"	10:07		7
Pumping back wash	10:07	"	10:22		15
Discharging residue	10:22	"	10:27		5

Total time of cycle	3 hr. 33 min.
Tons of solution from pulp	4.53
Tons of wash through cakes	3.05
Thickness of cakes	1 in.
Specific gravity of stock pulp	1.21

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GOLDFIELD, NEVADA.—VII.

Written for the MINING AND SCIENTIFIC PRESS
By T. A. RICKARD.

Goldfield has squandered its gold. The proportion of profit that has reached the owners of the mines has been pitifully small. Thus, the Florence has probably yielded \$4,000,000 worth of ore, and yet the shareholders have received only \$210,000 in dividends, that is, about 5%. There is \$350,000 in the treasury. Of course, the lessees have made fortunes, the miners have stolen fully half a million dollars, and the remainder has gone to the smelters and railroads. But a net profit of 5% on ore averaging over \$200 per ton is a miserable result from an economic standpoint. The big group of the Goldfield Consolidated produced fully \$6,250,000 gross in 1907, and out of this the shareholders have had two 10-cent dividends, making \$710,000, although the profits to the company were \$1,760,358. The lessees made about \$2,000,000 and the miners stole fully \$1,000,000—figures that made the dividends look small enough. The Consolidated group has produced \$16,500,000 to date, in 4½ years. Mention may also be made of the Tonopah mine, which has produced fully \$17,500,000, and yet the total dividends to date aggregate only \$3,450,000, about one-third of which was derived from the Tonopah-Goldfield railroad. Of course, a 100-stamp mill has been built at a cost of \$900,000, and a power plant for \$250,000, and a branch railroad, and improvements of many kinds; nevertheless it is astonishing to note how little benefit the shareholders have derived from this bonanza. If we add the production of the three mines (the Florence, Goldfield Con., and Tonopah) since their incorporation, we get a total gross yield (not including ore stolen) of fully \$38,000,000, out of which lordly sum only a driblet of \$4,370,000 has reached the shareholders. It is disgraceful, and constitutes a striking contrast with Cobalt, another boom camp where men went wild, where transportation facilities were no better than in Nevada, and where even the Government hindered rather than facilitated development. Added to these difficulties were those of a hostile climate, the long semi-arctic winters impeding the efficiency of labor far more than the burning summer heat in the dry air of the desert. The ore has yielded massive native silver, easy to steal. No effort has been made to treat the output from the mines by local reduction works until recently, so that the railroads and smelters enjoyed full opportunity to reap an excessive harvest. Yet from a total production of \$6,000,000 per annum for four years it is estimated that over 60 per cent has come into the mine-owners' pockets as profit. In 1907 the Nipissing mine yielded a gross return of \$1,245,819, from which a net profit of \$923,788 was realized, a result that is without parallel among the phenomenal gold mines of the Nevada camps. Indeed, the record of waste and loss was less startling at Guanajuato and Pachuca in the old bonanza days of the eighteenth century than at Goldfield and Tonopah in the twentieth.

Where did so much money go? As already indicated, many lessees made fortunes while operating under royalties ranging usually from 20 to 25% on the smelter returns; much ore was stolen and divided among the criminal class; a large fraction went to the railroads and smelters under cover of rates that were, in effect legalized robbery.

Until September 1907 the smelters charged:

On ore assaying:	Deduction for treatment.	Price paid for gold.
Up to \$70	\$6.00	\$19.00
Between \$70 and \$100	7.00	19.00
Between \$100 and \$300	8.00	19.50
Over \$300	9.00	19.75

An ounce of gold is worth \$20.67, so the deduction when paying \$19 per ounce on a 5-oz. ore is not less than \$8.35 per ton, that is, the real smelting charge is double that stated. The cost of smelting on a neutral basis is about \$4 per ton. As the smelters to which the Goldfield ore was sent are at Selby (near San Francisco) and Garfield (near Salt Lake), the distance it was transported ranged from 500 to 750 miles. The same rates were charged to both points. They were scaled according to the assay-value of the ore, thus:

Assay.	Per ton.
Up to \$40	\$ 7.80
\$40 to 50	10.95
50 to 60	11.55
60 to 70	12.90
70 to 80	14.25
80 to 90	15.60
90 to 100	17.15
100 to 150	18.05
150 to 200	18.85
200 to 250	19.90
250 to 300	20.90

The freight-rate is adjusted on the basis of the sampling at the smelter. Above \$300 the railroad charged \$20.90 per ton, plus 4% on all valuations above \$300. Thus on \$500 ore, the rate was 4% of 200 or \$8 plus \$20.90, making the total charge for transport not less than \$28.90, that is, an ounce and a half of gold at the smelter valuation of that metal. The 4% extra was made up from two charges of 2%, the branch road to Mina levying an extra toll of 2% on ore above \$300 valuation, while the main road from Mina to the smelter levied 2% more. This double charge ceased when the two railroad systems were consolidated, so that the extra rate is now 2% only—but that is enough.

In October the money market was disorganized, the smelters were unable to carry large supplies of ore, they became over-stocked, and levied rates that were frankly prohibitory. It is fair to add that owing to the congestion of traffic on the railroads, the smelting companies were hindered in getting cash, for the bullion was so delayed in transmission that realization on ore purchased from the mines was postponed an extra 30 days. The freight and smelter rates became:

Assay-value.	Freight and treatment.
Up to \$30	\$18.90
\$30 to 50	24.50
50 to 75	33.00
75 to 100	37.00

Besides, of course, the indirect deduction made

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by paying less than the mint price of gold. Thus on $100 ore, the total exaction became $45.35. It reminds one of the early days—50 years ago—when smelting was beginning and everything was carried across the plains in ox-wagons. In the days when it was necessary to encourage better transportation, the payment of bonuses for such a service was well enough, but now the railroads take no extraordinary risks and are entitled to no extraordinary tribute.

On December 12 the freight rates were reduced, thus:

Assay.	Per ton.
Up to $20	$ 6.00
$20 to 30	7.10
30 to 40	7.80
40 to 50	10.00
50 to 60	11.00
60 to 70	12.00
70 to 80	13.00
80 to 90	14.00
90 to 100	15.00
100 to 150	16.50
150 to 200	18.00
200 to 250	19.50
250 to 300	20.90

The addition of 2% on any excess valuation above $300 remained as heretofore. It will be noted that the improvement in rates is on the lower-grade ore. And it is to be remarked that this arbitrary charge of an extra 2% on ore worth more than $300 is not accompanied by any liability on the part of the railroad company, which refuses to hold itself liable for a valuation in excess of $100 per ton, unless compelled by means of litigation. Even if the shipper is willing to release the railroad company from any extraordinary liability on excessively rich ore, this arbitrary charge of an extra 2% is made. It is the unblushing application of the principle of charging "all that the traffic will bear."

On February 6, 1908, the smelter rates were reduced, thus:

On ore assaying:	Deduction for treatment.	Price paid for gold.
Up to $50	$ 6.00	$19.00
$50 to 60	7.00	19.00
60 to 70	8.00	19.00
70 to 100	9.00	19.00
100 to 300	10.00	19.50
Over 300	10.00+	19.50

When the gold in the ore is in excess of 5 oz. per ton, an extra charge of 50 cents per ton for each additional ounce is made, so that on 7½ oz. ore, the extra charge is $2.50, making the total direct charge for smelting $12.50, to which the deduction from the mint value of gold must be added, namely $8.77, making the total exaction not less than $21.27 per ton.

To these smelter and railroad charges must be added the cost of sampling, which is $1.50 to $2 per ton, according to tonnage. Last year it was $2 to $3 per ton. The Western Ore Purchasing Co. and the Nevada Goldfield Reduction Co. have sampling works and receive ore, both as buyers and as brokers for the smelters.

It is no wonder that metallurgical ingenuity has been busily occupied in trying to find a cheap method of treating the ore on the spot, so as to escape these impositions from the railroads and smelters. The real benefactor of Goldfield and Tonopah has been the metallurgist who has devised an effective method of applying the cyanide process to the ores of these districts. I do not say that the railroad did not help to stimulate development or that the smelter did not help in marketing the ore, but of both it is fair to say that they have "the fault of the Dutch, of giving too little and asking too much."

Now that mills have been built and others are in process of rapid erection, the smelting and railroad rates will tumble. At Millers, a junction point on the railroad 42 miles from Goldfield and 31 miles from Tonopah, there are two large mills which treat the ores of the Tonopah and Belmont mines. The Montana-Tonopah has its own mill at Tonopah. At Goldfield, the Consolidated Co. will have the Combination mill and its new 100-stamp mill, to be completed by November. Besides these are the custom plants of the Nevada Goldfield Reduction Co. and the Goldfield Cl Mill Co. Escape from the grasping hands of the railroad and smelter is assured.

With this prospect of competition, the railroad is likely to reduce its maximum rate to $12, keeping the minimum at $6, for the only ore to be sent out of the district will be the rich stuff. Smelter rates are likely to become $6 on ore under $50 and $8 on ore up to $100. It is likely that the $10 charge on ore assaying $100, and over, will be maintained, but the penalty of 2% on ore carrying more than 5 oz. gold is likely to be abolished. This is necessary if the smelting companies are to get the high-grade ore in competition with the mills. On ore above $300, the old penalty is likely to be maintained because the mills are not eager to treat this extra-rich ore. Such rules and exactions are what provoke men to wish for a benevolent despot or an interfering paternalism, to curb the frankly predatory instincts of corporate agencies. The smelting and the railroad corporations will come to an agreement as to how much each is to have. Just now the smelters are trying to get the railroad to reduce its tariff. When an adjustment has been made, it is probable that $50 ore (2½-oz. gold) will be the dividing line; ore richer than that going to the smelters, ore less than $50 going to the mills, and as the mines grow in size, milling methods improve, and competition between the mills develop, there will be a tendency to treat an increasing proportion of the output in Goldfield itself.

Some idea of the rate of production to be made from the Goldfield Consolidated mines is obtainable from the fact that the new mill will treat 600 tons per diem, in addition to the 80 tons now treated in the Combination mill. This output of 680 tons will average $30; with a recovery of 90%, this means $18,000 per day or about $5,500,000 per annum. In addition there will be 30 to 50 tons of ore daily, averaging $80 to $300 per ton, sent to the smelters. It will be economical to mix the high-grade stuff with that which will yield a small profit, for in milling as in smelting it is possible to improve results by suitable mixture of ores. Thus, by adding

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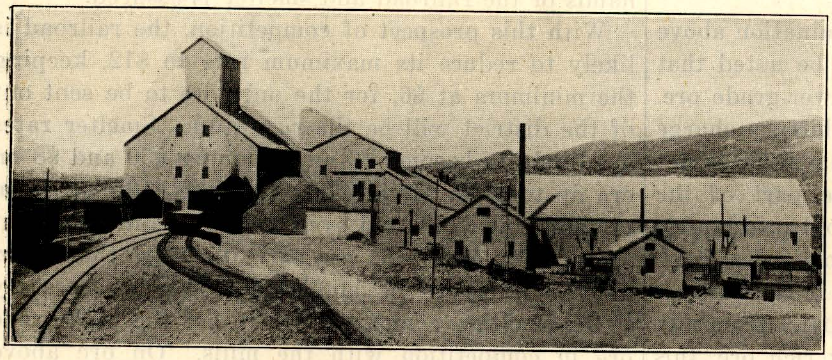
a small quantity of very rich ore to the daily feed in the Combination mill, good results are obtained. Recently the old Reilly lease-workings were tapped on the 280-ft. level of the Combination and a streak was found there 15 inches wide assaying 40 oz. gold. By distributing a ton of this stuff over 24 hours of mill-feed, the tailing was not enriched unduly and yet a good extraction was obtained by amalgamation, while the value of the concentrate was increased. In this manner the company is enabled to ship concentrates only and avoid the extortionate charges of smelters and railroads. Mining costs are from $3 to $4.50 at the Combination. In the Consolidated group the average cost of mining, development, and general expenses should not exceed $4 per ton. Transport to the mill will cost 10 cents. The average milling cost is now $6.50; in the big mill it should be not more than $2.25 per ton. Thus the total operating cost should be under $6.50. This is doing well. Kalgoorlie got down to such low costs after 10 years, Goldfield will have done it in two, by reason of the benefit derived from the experience of



Nevada–Goldfield Reduction Co's. Plant.

other gold mining districts. Let us hope that the period of squandering, of gutting mines to enrich ore-thieves, gambling saloons, bunco-brokers, railroads, and smelters is past, and that the profit will go first to the shareholders, the owners of the mines, and then, but not until then, to all the others who are the indirect beneficiaries of successful mining operations.

While the prolific production of gold has given point to the name of this Nevadan mining district, there is one feature of even greater interest to the engineer, namely, the conquering of natural obstacles. The creation of a hive of industry in a desert, by the construction of railways, the building of lines of power-transmission, and the bringing of water from a long distance—this is a fact to which every intelligent visitor must bow respectfully. It is part of the conquest of man over nature, the winning of the abomination of desolation where the sagebrush lifts an ashen face to an azure sky and barely hides the alkaline waste that was not ready for the habitation of man.

The electricity used for power and illumination at Goldfield comes from energy generated at the head of Bishop creek, in the mountains of the Sierra Nevada. The power-plant is in California, 95 miles west of Goldfield. The power is derived from the water issuing from a number of lakes, the highest of which is 9630 ft. above sea-level. There will be five power-houses eventually; at present three are built. Two of these (No. 4 and 5) generate 7500 kw., equivalent to 12,000 hp. During the present year the third plant will be completed, adding 8500 kw., or a total of 16,000 kw. One horse-power is equal to 746 watts. Both Pelton and Doble wheels are employed, the water being utilized five times over. The original transmission-line was made of aluminum wire, No. 0, B. & S., equivalent to 0.32486 inch. The second, or parallel lines, is made of copper, No. 00 wire, equivalent to 0.3648 inch. Mr. Delos A. Chappell, the president of the company (named the California & Nevada Power Co.), informed me that they were forced to take copper for their second line because of their inability to get aluminum promptly. The voltage is 55,000. The wires are strung so as to give the proper sag under varying temperature, so that the maximum stress equals one-quarter of the ultimate strength. Allowance is made for a minimum temperature of —20° F. The cost of power to the consumer ranges from $6.50 to $15 per hp.-month, as measured on the low voltage side of the consumer's static transformer by Westinghouse polyphase integrating watt-meters installed by the power company. This applies to large contracts; in small ones the measurement is by peak-load, taking 90% of the record.

The water supply of the town comes from the Goldfield Consolidated Water Co., which owns springs on Magruder Mtn., near Lida, 30 miles distant. A pipeline was built in 1907, the water being delivered on October 4, 1907. The first pipe-line was defective, but the manufacturers made good, substituting new pipe. The actual cost of water is $1 per 1000 gal. when used in the mills of the district. This may be compared with the cost of water in Western Australia, where the Government built a reservoir and pipe-line 353 miles long, at a cost of $15,700,000, and delivers water at a maximum cost of $2.40 per 1000 gallons.

A brief description of the district as it appeared from a central elevation may prove of interest to those who have not been to Goldfield and it may prompt a contrast in years to come. Standing just outside the shaft-house of the Little Florence lease on one of the many hillocks rising above the desert, the view (on April 11, 1908) was as follows: To the west on a flat plain is the town of Goldfield, with a population of about 15,000. The multitudinous small dwellings, like matchboxes of varied color, are dominated by several large structures of brick and stone, among which are conspicuous the Court House, the new hotel, the School, the News building, the Casey hotel, and the Hippodrome. The last is not the Stock Exchange, as might be supposed, but a place for prize-fights and other diversions. Overlooking the town farther to the west is the edge of a 'malapai' (Spanish *mal*, bad, and *pais*, land)

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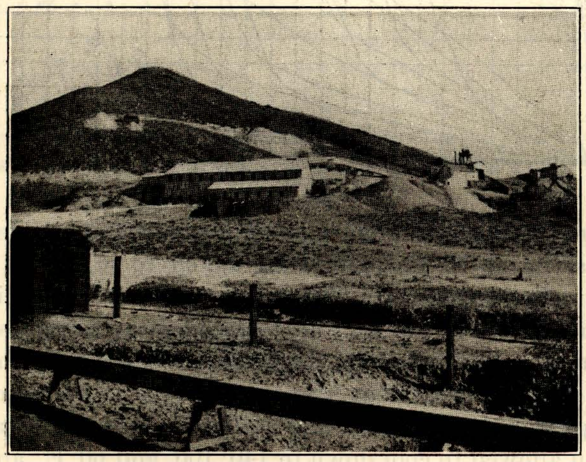
terrace, a mesa capped by basalt and underlying tuffs. In the distance is the snow-covered crest of Montezuma mountain.

Southward the malapai swings across the horizon and is broken by low hills, surmounted by remnants of lava. In the middle distance are several shafts and head-frames: the Combination Extension, White Rock, and Portland. At the foot of the hill on which the observer stands is the Florence Extension.

Turning eastward the sky is cut by a red hill, of that name. The dumps and head-frames in the intervening flat, mark leases on Consolidated ground. Due east, there is a break in the ridge forming a pass through which the Clark railroad goes to Los Angeles and also the Brock railroad to Rhyolite. The nearer of the two tracks (which connects Tonopah to Rhyolite) appears to follow the wagon-road and has a steep gradient, over 4%. Beyond the railroads are a number of broken ridges with combs of silicified rhyolite. A sprinkling of dumps and head-frames indicates the position of the C. O. D., Kansas City, Atlanta, Gold Bar, and Cimarron mines.

Swinging northward, just across the nearer line of railroad, rises a large mass of black rock, the mineral possibilities of which are now being tested in the B. Fraction. Then the view includes a large number of mines just east of the main ore-channel; among these are the Jumbo, Cleremont, and Velvet. In the immediate foreground is the new Florence mill and nearly half a mile northwest the Combination mill; beyond the last is the Western Ore Purchasing Co.'s sampler and the works of the Nevada Goldfield Reduction Co., and the Goldfield Cl Mill. All these metallurgical establishments are in the

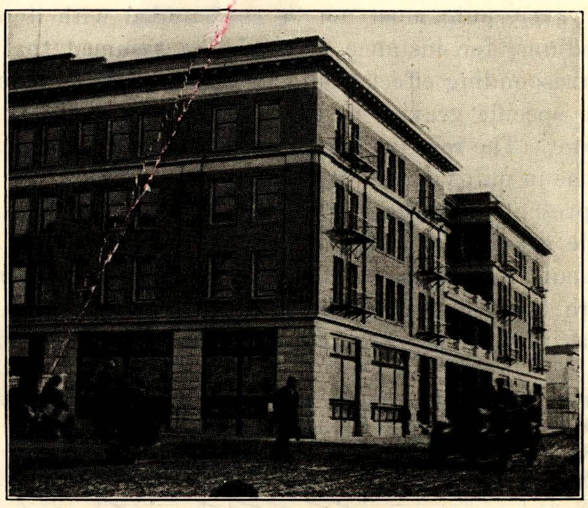


Montana-Tonopah Mill.

flat north of the town and punctuate the picture assertively.

Almost due north are the Consolidated mines, with the Mohawk shaft-house and those of the Red Top and Laguna. In the same direction farther away is the Kinkead mill and the Booth mine near the base of Columbia Mtn., a conical hill rising about 1000 ft. above the flat and dominating the view northward. On the west side of the mountain can be seen the grade of the new railroad to the Consolidated Co.'s new mill, now under construction, and below it is the old (that is oldest, for 5 years means 'old' in Nevada nowadays) town of Columbia.

Looking northwestward over the flat to the left of Columbia Mtn. is a wide desert plain of alkali and sagebrush and in the distance rises the mass of Lone Mtn. To the right of Columbia Mtn. and on the northeast horizon are the white peaks of the Toiyabe range and north of them is Jefferson peak of the Toquima range. Over the northerly edge of the base of Columbia Mtn. runs the yellow line of



The Goldfield Hotel.

the automobile road to Tonopah, the site of which is indicated by the Butler Buttes just east of a yellow hill named Gold Mtn. Between the Tonopah hills and Goldfield the desert stretches, a dreary waste, threaded by the road connecting the two mining districts, which are linked by a sequence of discovery and by the human history that has changed the waterless wilderness into a hive of industry. Overhead is a flawless sky, the vast expanse of hill and plain is bathed in brilliant sunlight and vibrant in an air as pure as the morning of Time.

Zinc-smelting in a blast-furnace has been commercially used in Sweden for some time. The blast is suppressed, and the requisite heat is obtained by the aid of an electric current instead of fuel. The furnace greatly resembles a cupola furnace for producing copper matte. It is mechanically charged at the mouth, and provided with a water-jacket. The rheophores are placed in the same position as the tuyeres of a blast-furnace. The liquid zinc is collected in a crucible, and a slag is formed, which, as in blast-furnaces, is tapped discontinuously. Such furnaces, producing 5500 tons of zinc yearly, have already been working for a few months at Leadville, Colo.; Valerdeña and Santa Barbara, Mexico. Construction and working of furnaces capable of producing 55,000 tons yearly is said to be feasible.—*Revue Scientifique.*

Colored wood is the subject of a recent patent by a Norwegian firm. Whole trunks of green trees are colored, the sap being pressed out and the dye injected in its place. It is claimed that wood treated by this process is more durable than ordinary wood, and that it will not warp.

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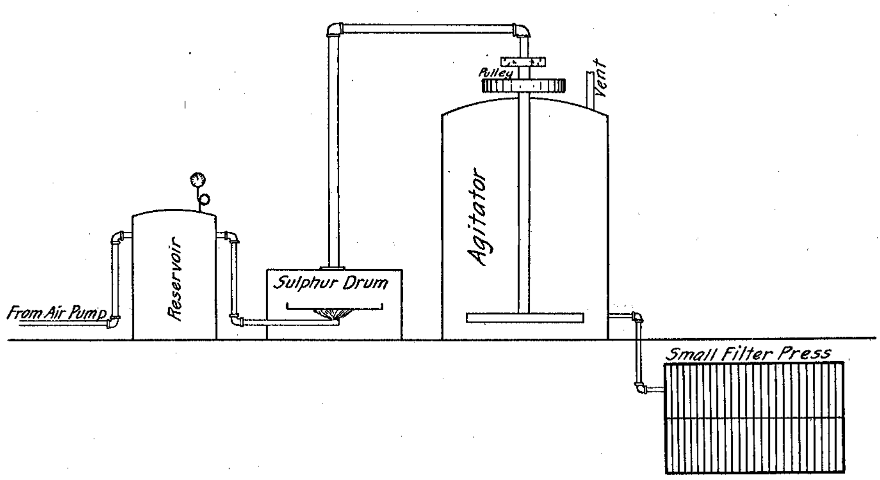
MILLING PRACTICE IN NEVADA GOLDFIELD REDUCTION WORKS.

Written for the MINING AND SCIENTIFIC PRESS
By E. S. LEAVER.

The mill of the Nevada Goldfield Reduction Works at Goldfield is a custom plant, all ore being received through the sampling works. As the ores are sampled and purchased in small lots, this allows mill-results to be followed closely, and as the ores come from various mines and leases, including surface-dumps and deep workings, the results and treatment vary considerably. The surface, or oxidized ores, readily give high extraction by simple treatment. The sulphide ores require close attention, and in depth are becoming even more complex. The deeper ores, now being marketed, require close concentration, almost absolute sliming, and not less than 10 days' cyanide contact. The treatment consists in crushing wet by stamps, amalgamating on plates, concentrating on Wilfley tables, fine-crushing of the sand in a tube-mill, re-amalgamation on plates, re-concentration on vanners, and cyaniding of the sand and slime.

The ore is crushed in weak cyanide solution (0.1%), and while this has been the practice for over a year, no particular effects are noted on the plates. The entire recovery has been better than formerly when crushing with water. The plates require more attention, but all pitting is avoided and the amalgamation has been good. By crushing in cyanide, treatment is favored, in that the ore is at once in contact with the solvent, and there is elimination of the loss of time in settling, decanting, etc. Also this practice avoids the loss in weak solution necessary in cyanide-plants where the ore is crushed in water or taken into the cyanide plant with water. In addition to the usual losses in waste-solution, the cost of water is an important item at Goldfield. Having only limited fall from the Wilfley tables to the tube-mill, advantage is taken of Wilfley tables as sizing machines, making three products, sand, slime, and concentrate. Only the sand is taken to the tube-mills, the aim being to grind it as fine as possible, and to favor passing as much of the finest sand as commercial treatment allows with the slime proper. As the ore in various mines differs, the percentage of slime and sand will vary, but approximately 20% of the original weight is treated as sand. This sand will pass the 100 mesh, and requires 10 to 20 days' double treatment by percolation, that is, the sand is collected and treated in an upper row of vats, discharged, and re-treated in a lower row of vats. Good success has attended the use of a substitute for tube-mill pebbles. The Danish pebbles were originally imported at a cost of about $60 per ton. A number of samples picked from medium low-grade ores were experimented with. The result was the finding of a close-grained compact ore which is an excellent substitute for the imported pebbles. Several tons of lumps can be selected from the ore in a short time. These lumps, varying in size, are fed into the mill, and in a short time it requires close inspection to detect them from the genuine imported pebbles. Results from these pebbles show a saving of $100 per mill per week.



The crushing or circulating solutions, after passing the concentrators, including the sand-collection vats, flow to large slime-collection vats, and the clear solution is taken off by rim-launders and decanters. This circulating solution then passes through its special zinc precipitation boxes, and is returned to the battery-tanks, to be again used under the stamps. When the slime at the collecting vats is at a proper specific gravity it is pumped to the treatment vat, where the strength of the solution is brought up if necessary. There it is agitated and aerated by circulation through centrifugal pumps, until the gold is in solution. This requires from 4 to 10 days. The slime is then separated from the gold-bearing solution by leaf-filters.

After a commercial failure with a well-known leaf-filter a number of experiments were made in the mill which led to the designing of a leaf which has given excellent results. The ordinary form or size of leaf was continued, but a filler was adopted that leaves no chance for clogging. This has been in constant use at these works for about one year, and has given perfect satisfaction. The canvas for the filter is sewed around the bottom and sides, making a bag about 5 ft. deep and 10 ft. long; the sewed edges are turned inside the bag; then, excepting about a three-inch strip along the sides and bottom, which is left for the pipe-frame work and suction, the entire filter is divided by stitching vertically in lines two inches apart, making pockets open at the top and bottom. Into these pockets are pushed grooved slats made of common lath. The grooves are about $^1/_{16}$ in. deep and about ¼ in. wide. Two grooves run lengthwise on each face, and each groove is placed so as to alternate with the groove on the opposite side, so as not to weaken the slats. Various experiments were made to determine the proper size of slat, number of grooves, etc., and the above was finally adopted.

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interfering elements is much greater in an electrolytic than in an iodide-assay. The copper can almost always be separated by precipitation on aluminum, followed by hydrogen-sulphide water in a state pure enough for an iodide assay. This is less true of the electrolytic method, where often to assure accuracy a more elaborate separation must be made. The liability to error in standardizing an iodide solution, which is very remote, is more than offset by the loss in copper attending the removal of the cathode from the electrolyte, and to the error in weighing the same. In a well-equipped laboratory, on ores where no separation is required, the time factor is much in favor of the electrolytic method.

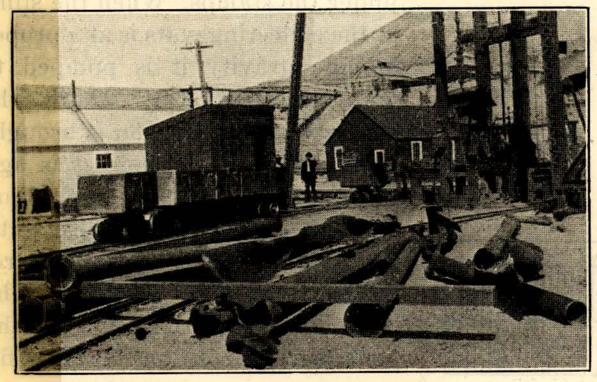
JAMES W. HOWSON.

San Francisco, July 24.

Explosion in Compressed-Air Main.

The Editor:

Sir—I enclose a photograph illustrating the effects of an explosion of gas generated in the air-compressor of the Tonopah Mining Co. of Nevada. The pipe is 6 in. diam., and the contorted piece highest in the pile was bent completely back on itself in two places. The upper side as shown in the picture was the outside of the pipe. The explosion occurred in the Mizpah shaft on April 14 of this year. It was



Effect of Compressed-Air Explosion.

confined to the pipe between the surface and the 300-ft. level, wrecking all of it. A few minor timbers were torn out; aside from this no serious damage was done. The explosion did not extend back to the receivers or the compressor, nor to the pipe below the 300-ft. level, and fortunately the cage-tender was not within those limits.

The explosion was due to gas generated in the compressor-cylinders from the oil used in them. A few weeks after this explosion there was a similar one at the compressor-plant of the Standard Oil Co. in California, and today the compressed-air line of the Tonopah Belmont Co. exploded in the shaft, and on the surface. The cylinder oil used at both the Mizpah shaft and the Belmont shaft was supplied by the Standard Oil Co., and it is to be presumed that the oil used in their own compressor was the same. This series of explosions in such close succession constitutes a serious reflection on the quality of oil furnished by the Standard Oil Company.

J. A. BURGESS.

Tonopah, Nevada, July 24.

Milling and Cyanide Practice at the San Prospero Mill, Guanajuato.

The Editor:

Sir—In J. S. Butler's description of cyanide practice at the San Prospero plant, appearing in your issue of July 25, the curve sheets and tables showing the progress of extraction during the milling operation prior to the commencement of the cyanide treatment proper, make an interesting record. The table showing the extraction of values is as follows:

	Gold Extraction, %	Silver Extraction, %
In the batteries	31.2	4.8
Between the batteries and concentrators	5.4	5.6
Between concentrators and sand plant	21.0	5.0
Total mill extraction	57.6	15.4

The extraction of the silver and gold content, 48% and 31.2%, in the crude ore during the pulping process in the battery is more remarkable when the conditions are analyzed. In a five-stamp battery of this mill, where the crushing-capacity is 3.7 tons per stamp per 24 hr., 28.5 lb. of ore must be fed to each battery per minute, and a like amount (dry weight) of pulp must issue through the battery screens in the same time. As the amount of ore undergoing the pulping process within a five-stamp mortar will approximate 85 lb., it follows that the average time required for pulping the ore in the mortar would not exceed three minutes. During this time it is known that the size of the ore-pieces found at any time in the mortar will vary from the maximum size, 1½ in., to the portion of slime resulting from the pulping operations, and it is probable that 50% of the 85 lb. of ore always in the mortar box would be large enough to remain on a ¼-in. mesh screen. What is remarkable is that ore of this disproportion of sizes when submitted to contact with 0.12% KCy solution should yield 31.2% of its gold, and 4.8% of its silver content in three minutes. If the solution of silver and gold in the pulp would proceed at the same rate all the gold in the pulp would be brought into solution in four minutes and the silver in about 60 minutes. This is a more rapid rate of solution than is generally understood to be possible in connection with the cyanidation of ore, and the fact should attract the attention of those crushing ore in cyanide solution and stimulate investigation.

It would be interesting in this connection to know the character of the ore treated, and in what form the silver occurs in it. It is probable that a large portion of the silver exists as a chloride, for silver sulphide would hardly go into solution as rapidly as stated. At any rate the statement should open up a new line of investigation and research, the results of which cannot fail to benefit the industry.

BERNARD MACDONALD.

Guanajuato, Mexico, August 1.

The crushing strength of ice at 23°F. varies between 400 and 700 lb. per square inch.

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The Cost of the Goldfield Mining Boom

By Augustus Locke

Eight years ago the gleaming desert in which Goldfield is situated was uninhabited, inaccessible, and little known. The sudden upbuilding within this desert of a large mining community, together with a mature and great mining industry, has involved a most unusual application of human energies. In general, the psychology of mining is peculiar. This, I suppose, is due to the fact that, more than in any other industry, there is room in mining for the imagination. The fortunes of mining undergo sudden and wide variations. The powers of the best developed mine are, to some degree, hidden; the results of next month's and of next year's work are, to some degree, unknown.

As a result of this uncertainty, always present, the unskillful see the industry in caricature, and an exaggerated hope of brilliant success fixes their thoughts. Such views, developed to an extreme degree, originally pervaded Goldfield. In its truest sense, the camp underwent a mining excitement. Indeed, the peculiar story of the camp is intimately woven with the story of this excitement. There were a multitude of difficulties in the way which would have turned aside or long delayed other enterprises; but here they were overcome easily and promptly by the impact of the powerful enthusiasm which accompanied the boom.

Effective as it surely was, this enthusiasm was not less illogical than it was powerful. It constituted a huge and awkward force which denied the use of reason. In its blind activities, it did much good and much harm. To estimate accurately in dollars what gains and what losses it accomplished is impossible. It is, however, possible to investigate the matter qualitatively. I propose to indicate where the gains and losses came, and to arrive at a judgment as to the debits and credits. As preliminary to the understanding of the Goldfield boom, it is necessary to realize that it consisted of two unlike parts. The first of these was the 'maelstrom' within the camp. As has been suggested above, this took the form of tremendous activity; in essence, it consisted largely of a rapid circulation of money. Money passed from hand to hand with unusual speed. It was easy to get and easy to spend. The second part was the external boom—that seductive force which caused grocers in quiet New England villages to send their good money into the maelstrom. In one way this part was similar to the first, for it encouraged easy expenditure; but in another way it was fundamentally dissimilar, for it offered no ready means of winning the expenditure back. I do not, of course, contend that the internal boom was insignificant, or that it was not primarily responsible for the external boom. I wish merely to emphasize the fact that much of the excitement within the camp was a revolution, not a progress. Much of the stir was self-neutralizing. Big gambling gains and losses, rapid changes in ownership of real estate, numerous transfers of mining stock—these, in themselves, signified little; the individual had an excellent chance to win back on Tuesday what he lost on Monday. These facts being taken account of, it is evident that the essential features of the boom may be covered in a consideration of the following: (1) The building of the town of Goldfield; (2) prospecting and developing; (3) sale of mining stocks; (4) moral effect of the boom condition.

The City of Goldfield, Nevada.

In 1907 Goldfield had a population of 20,000 and a yearly ore production of 100,000 tons. In 1910 it

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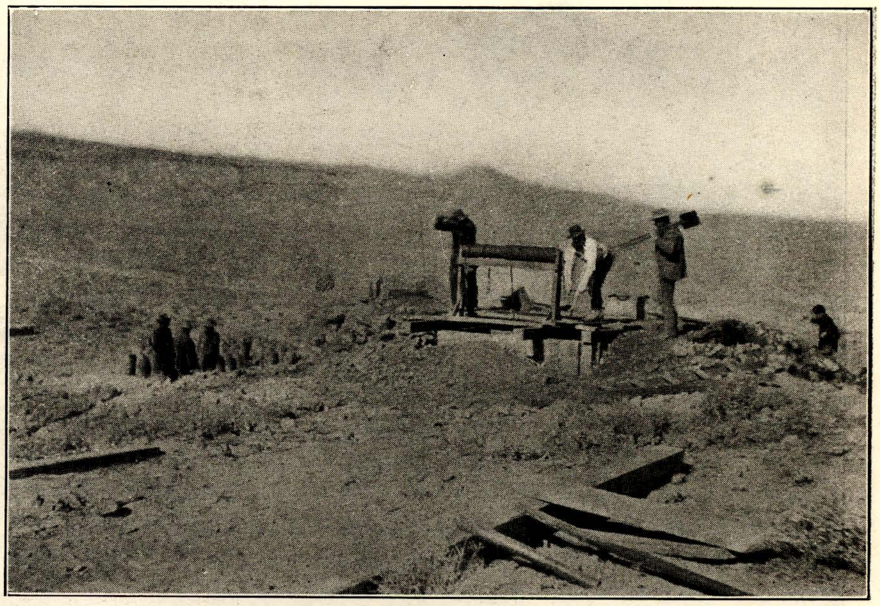
has a population of 5000 and a yearly ore production of 300,000 tons. At the earlier date the construction of the town and the development of the principal orebodies had been completed. The excess of people was therefore not required either for construction or for development. If it be assumed that the population needed varies with the tonnage produced, the actual population was something like twelve times as great as was demanded by the genuine needs of the community. In other words, eleven-twelfths of the people were on hand because of the boom. In reality the camp had from the day of its discovery been, in this meaning of the words, overpopulated. As a result, it was overbuilt. The average number of vacant buildings is about one out of three, enough in certain localities to produce a decidedly deserted appearance. There are, moreover, a considerable number of incompleted buildings. The slump in 1907 left them, with their unclad rafters, to be a permanent evidence that some persons changed their minds. The decrease in the effective selling value of real estate since 1907 is probably 75 per cent. Prices, at that time, were raised greatly above cost, and they are now much below it. The assessors' valuation of real estate in Goldfield a year or so ago was two and one half millions. It is easy from these figures to make a rough estimate of the real estate loss.



The Beginning of the Goldfield Consolidated.

Not less certain than the fact that the desert town was overbuilt, is the fact that the region about it was overprospected. A primitive race, making it a matter of religion to dig holes, could scarcely dig more pitfalls in an age than the prospectors of Goldfield dug in five years. There are holes in the tuffs, holes in the loose wash, and holes in the summits of rocky peaks. To the mind of the excitement-stricken prospector, it did not occur that an investigation of the croppings for a distance of 100 ft. down from the apex of a hill would often render as much information as to its value as the sinking of a shaft for a like distance. In 1907, something like 200 shafts produced 100,000 tons of ore. In 1910, about 20 shafts are taking out 300,000, of which amount 5 are taking out 90%. A close calculation of the cost of fruitless prospecting is not possible. An approximation of the probable maximum is, however, sufficient. Very likely the expenditure did not exceed one and one-half million dollars. The total real loss, then, through real estate depreciation and fruitless prospecting was probably between three and four million dollars. These losses were concerned with the internal boom. The losses concerned with the external boom were of a very different sort, and resulted chiefly from purchases of mining stocks. There were about 200 companies promoted in Goldfield. Almost invariably they had a capitalization of 1,000,000 shares. Each company sold, on an average, one-half of its capital stock. The average price realized is unknown; but likely was 30c. The companies which have since had great success are included in this average, but as their original selling price was low, they bring up the average little or none. The money invested, then, in the initial purchase of the mining securities of Goldfield was something like $30,000,000. These figures indicate that the external boom was several times as costly as the internal boom. So far the losses have been expressed in money. An additional and very substantial loss, the injurious moral effect of the boom condition, cannot, of course, be so expressed. The most striking effect of this sort was high-grading. Indeed, high-grading was not less a cause than an effect. Moreover, it was an effect not only of the boom condition, but of other conditions as well, chief among which was the richness of the ore. Yet the peculiar state of mind which accompanied the boom must be regarded as an essential condition.

Not over three and one-half years ago the high-grader in Goldfield was a 'perfectly good' and respected citizen. He who refused to high-grade was over nice and scrupulous, and many a man who would scorn to steal money, stole ore. Much astonishment has been expressed that such a condition should have existed. In reality its explanation is simple. The ultimate cause of the condition was the unusual excitement accompanying the hope of extreme success. This excitement was dominant. It eclipsed usual ideas and unsettled customary morailty. Goldfield got into the habit of high-grading during the days of leasing. Good miners were then not easy to get unless they were given the opportunity to high-grade. When a lease was fast expiring, it paid better to let the miner who was taking out $200 worth of ore, have $20 of it for himself, than to take out no ore at all. So high-grading was for a time actually countenanced by the owners of the ore. For the average man, the criterion of

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morality is custom. What is usual, and what the neighbors approve, is right. In Goldfield, because of the peculiar and general mental attitude, nearly everyone approved. Moreover, as Goldfield was isolated, the influence of outside opinion operated slowly. In a populous locality, the average of morality is maintained by the diffusion of ideas. But ideas diffuse very slowly over two or three hundred miles of desert. These are the reasons why high-grading became extensive, and why it was not in bad repute. There is another bad moral effect, less conspicuous, but not less important. Goldfield has seen the making of sudden fortunes; it has had the taste of extreme hope. Not one man from each thousand is destined to have great financial success. Yet 900 out of 1000 are mentally unfitted by conditions such as existed at Goldfield, for the unromantic labors which by fate they are destined always to pursue. It is difficult to get down to business and take ordinary profits, when for a year you have been dreaming of ten hundred per cent.

The effects which have now been considered are the important losses which resulted from the boom. But there were important gains as well. The mines have produced about $40,000,000. Some of the prospecting, and some of the stock investment was, therefore, very fruitful. But how much of the success was due to the boom? A useful method of arproaching this question is to consider in what way the result would have been less successful had the methods of ordinary business been applied. It is certain that the calm calculations of ordinary business would never so suddenly have established a comfortable town and a great mining industry. It would never have spent money on frenzied prospecting. Nor would it have built houses for a whimsical population which next year might migrate. But for the boom, we might still be buying water by the gallon, and extinguishing fires by primitive methods. Above all, we might still be ignorant of the existence of the Hayes-Monette orebody. In short, the boom benefited the camp by giving it the impulse for sudden development. To the owners of the productive mines, this sudden development was of enormous advantage. It is a notable fact, however, that the people who furnished the money for the boom were not those who reaped the benefit from it. The people who paid $30,000,000 for Goldfield stocks are not the owners of Consolidated. Many declare that we must have booms to develop mines. It would be more correct to say that we must have booms to develop mines at top speed. If this is an argument in favor of booms in mining, it is likewise an argument in favor of booms in other industries. It is entirely certain that, whatever the industry, the visionary attitude is very costly, and that the application of common sense is immeasurably more efficient. For the average citizen booms of any sort are excellent things to keep away from.

Minnesota furnishes from St. Louis county alone about three-fifths of the iron ore produced in the United States; the shipments during 1909 amounting to 29,282,526 tons.

A CIRCUIT TESTER FOR BLASTERS

There has always been much difficulty in detecting unexploded charges, particularly in floor shots, and until recently the blaster has had to depend on his eye to determine whether all of the holes had fired or not. This is at best an unsatisfactory method because, in a number of instances, it has been found that the wires and tamping from the hole were not blown out although the dynamite had exploded perfectly. Although the difficulty of finding a missed hole after the blast has been fired still remains, it is now possible for a blaster, with care, to determine with a great deal of certainty that each of the electric fuzes in a blast are in good condition before the shot is fired. A device known as the du Pont galvanometer or a circuit tester has been invented and placed on the market, which not only indicates whether an electrical circuit is closed or not, but also indicates, within practical limits, the amount of electric resistance in the circuit. A blaster may connect the two leading wires to the galvanometer after the holes are all connected up and the galvanometer will indicate not only whether the circuit is complete or not, but also if there is leakage through bare connections, etc., the number of electric fuses connected in series. For instance, if a blaster has 30 holes, in each of which there is an 8-ft. electric fuse, by referring to the table furnished with the galvanometer, he will see that the total amount of resistance is 33½ ohms. If the needle on the galvanometer now indicates close to that figure, he may be perfectly sure: first, that none of the electric fuses are broken; second, that there is no leakage of current through the bare ends of the connecting wires; and, third, that the leading wires are intact. If a wire of one of the electric fuses has been broken in tamping, the blaster will get no motion of the needle when he connects it to the two leading wires. By going over the bare connections over the holes, it is possible for him in a few minutes to determine which electric fuse has the open circuit. It frequently happens that the leading wires become frayed and the insulation removed in places, and this bare wire may fall over an air or steam pipe or a rail, thereby producing a short circuit through the pipe or rail. When the blaster tests his leading wires on the du Pont galvanometer under these circumstances, it will immediately show a short circuit, and the blaster, knowing that the resistance should be calculated from the number of electric fuses he has connected in series, instantly sees that something is wrong and does not attempt to fire the blast until the wires are lifted clear from the iron. In looking for a break or broken wire in a large floor blast, the instrument can be attached successively to the double lines of bore-holes at the ends so that the line containing the break is quickly found, after which the electric fuses in each bore-hole are gone over separately. It is, of course, not possible to determine positively after a blast has been fired whether the protruding wires from any bore-hole are connected to a live electric fuse or not, as the end wires are usually stripped and crumpled together by the blast when it fires.

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Operation of the Goldfield Consolidated Mill

By J. W. HUTCHINSON

CONSTRUCTION

As this article will be confined strictly to the operations at the plant, those interested in general conditions at Goldfield, Nevada, are referred to J. R. Finlay's 'Cost of Mining', T. A. Rickard's articles on Goldfield which appeared in the *Mining and Scientific Press* during 1908, and the report on the geology of the district by F. L. Ransome published by the U. S. Geological Survey. The details of construction of the 100-stamp mill and cyanide plant of this company have received publicity through Bulletin 1438 of the Allis-Chalmers Co., which is substantially correct. However, in this bulletin due credit is not given J. B. Fleming, of San Francisco, who was employed by the Goldfield Consolidated Mines Co. as mechanical engineer, and who drew up the specifications and general plans. On these specifications the Allis-Chalmers Co. bid and secured the contract for the crushing and transmission machinery and electrical apparatus. The engineers of the Allis-Chalmers Co. were not employed by the Mines company until the general plans were out and the contract had been let. This is mentioned for the sole purpose of rendering to Mr. Fleming publicly the credit which this company gives him.

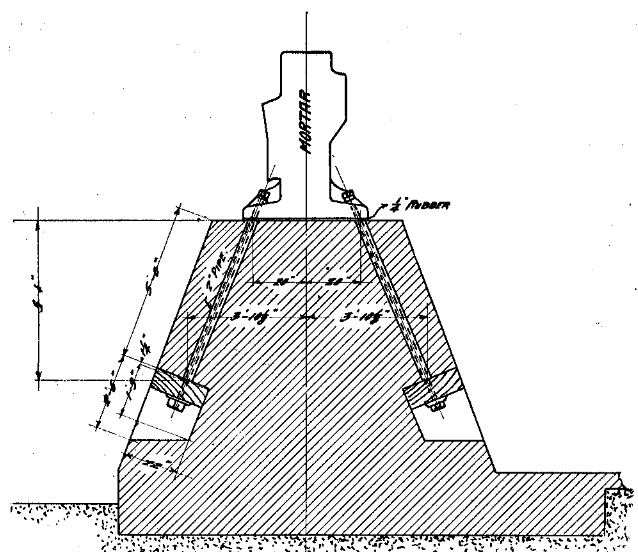
During the first year's operation, from December 26, 1908, to December 26, 1909, the stamps crushed through 12-mesh; the product, after classification, going to tube-mills. After the 20-stamp mill of this company, known as the Combination mill, was abandoned in September 1909, it was decided to increase the capacity of the 100-stamp mill from 600 to 850 tons per day.

The installation of 40 additional stamps, three tube-mills, and 25 concentrators was at first considered. This would have largely increased the floor space required, the structural steel for building, and, because of the contour of the hill, enormous concrete foundations for ore-bins and battery-block would have been needed. In addition, the increased tonnage could not have been handled for six months. The estimated cost of such construction was $175,000. As an alternate scheme, I proposed the using of six 6-ft. Chilean mills to be placed between the stamps and tube-mills. My idea was that 4-mesh screens could be used on the batteries, and a duty of 8.5 tons per stamp be obtained, followed by classification of the product, feeding oversize to the Chileans, crushing through 16-mesh, and finally, grinding this product in tube-mills after classification. This method was adopted and was in operation 90 days after the decision was reached. The total cost of the reconstruction, including 24 concentrating tables and many minor changes, was $75,000. No additional building was required. At the beginning of operations the ore was amalgamated at the batteries and below the tube-mills. With increasing depth in the mine the baseness of the ore rendered this operation unprofitable, and it was abandoned in September 1909 in favor of amalgamation of the concentrate prior to cyaniding. The floor-space occupied by the secondary amalgamating tables was used for the Chileans and concentrators and could not have been used for additional stamps and tube-mills. Doubtless a natural question here will be why the South African practice of using 4-mesh screens on the batteries and additional tube-mills was not adopted. This will be answered later.

To summarize: Through the expenditure of $75,000 the capacity of the plant was increased approximately 40%. To have accomplished the same result with stamps and tube-mills would have necessitated the expenditure of $175,000. To have added the requisite number of tube-mills to pulverize the 850 tons of 4-mesh product to 200-mesh would have increased the cost of construction over the Chilean-mill installation, and would have increased the cost of operation decidedly. Had 40 stamps and three tube-mills been put in, the cost of stamping and tube-milling would have remained the same on the increased tonnage; since the power, labor, and supplies would have increased in direct proportion. With the Chilean mills the additional labor for operating was five men. These would have been necessary in either case. For the Chilean mills 200 hp. is required. Stamps and tube-mills would have required 300 hp. The cost of supplies is approximately 2c. less per ton than for stamping and 3c. per ton for tube-milling. The total cost of pulverizing 80% of the mill-feed through 200-mesh is considerably less with three-stage reduction, as the following figures will show:

Two-Stage Reduction.	Cents.	*Three-Stage Reduction.*	Cents.
Stamping	22.1	Stamping	13.4
Tube-milling	20.6	Chilean-milling	10.0
		Tube-milling	16.6
Total	42.7	Total	40.0

I have gone into detail for the benefit of the 'doubting Thomases', many of whom have visited the plant and who have been unwilling to believe that Chilean mills could be operated at a cost even approximating 10c. per ton milled.



SECTION OF BATTERY FOUNDATION.

In order to avoid controversy, it may be mentioned here that the above costs per ton will be masked in the yearly figures by the damage to the plant by fire. For three months of the year ended October 31, 1910, the plant operated with 70 stamps and three Chilean mills. However, the above figures are representative of normal conditions. The low cost of operating Chileans is accounted for by several factors: (1) The design and construction of the mills, which were furnished by the Trent Engineering Co. of Reno; they have been most satisfactory in every respect, both as to operation and repairs. Nothing wears out or breaks except the crushing-steel. The horse-power required is 35 each. The capacity in tons pulverized is approximately 75 tons each. (2) The size of the feed to, and discharge from, the mills. Chilean mills fed with 4-mesh and discharging 16-mesh product, work most satisfactorily and produce, even at this mesh, approximately 30% of −200 slime. (3) The character of the ore, which is fairly soft.

The third proposal was to install a sufficient number of tube-mills to handle 850 tons of ¼-in.-mesh battery-product. As above stated, the cost would have been more, since the mill building would have had to be enlarged, and, from the contour of the hill, much grading and filling necessitated. In addition, experience here has not corroborated that of the 'Mines Trials Committee' at Johannesburg which found that a tube-mill is most efficient when operating on three or four-mesh feed. I do not want to be misunderstood. I believe this to be true on the Rand, where a large percentage of the tube-mill produced is leached, and the difference in the ores doubtless accounts for the difference in results. None the less, I have not been able to produce a satisfactory tonnage of −200 slime from one 5 by 22-ft. tube-mill fed with 4-mesh product and operating under con-

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ditions similar to the South African recommendations, nor have I been able to secure enough additional tonnage to compensate the increased horse-power when operating one mill at 32 instead of 27 revolutions per minute.

Conclusions based upon working tests on the Goldfield ore are: (1) 100 stamps, crushing through 4-mesh (0.18-in.) screens, yielding a tonnage of 8.5 tons per stamp, must be followed by ten 5 by 22-ft. tube-mills in order to obtain a −200 product; (2) 100 stamps, operated under above conditions and followed by six 6-ft. Chilean mills, crushing to 16-mesh will require five 5 by 22-ft. tube-mills to deliver a −200 product. This leaves the problem: Can six 6-ft. Chilean mills be operated more economically than five 5 by 22-ft. tube-mills? Operations here make me think so. The following figures may be of interest: In stamping Goldfield ore to 4-mesh, 20% of the discharge passes 200-mesh; of the remaining 80%, the Chilean mills will 'slime' 30%, or 24% of the whole. This leaves 56% to be handled by five tube-mills; all of which has passed a 16-mesh screen, and 80% of which will pass a 30-mesh. Expressed in tons, each mill is fed with 95 tons of this product. Now, by feeding 10 tube-mills direct from the battery with 4-mesh feed, 20% of which will pass a 200-mesh screen, each mill is required to handle 68 tons of 4-mesh feed. Based on figures derived from working tests, the comparative cost is as follows:

Two-Stage Reduction.		*Three-Stage Reduction.*	
100 stamps followed by 10 tube-mills fed with 4-mesh battery-feed:		100 stamps followed by 5 tube-mills fed with 16-mesh Chilean-mill product:	
	Cents.		Cents.
Stamping	13.4	Stamping	13.4
Tube-milling	30.0	Chilean-milling	10.6
		Tube-milling	16.6
Total (per ton)	43.4	Total (per ton)	40.6

From this the deductions may be made that: (1) Ore may be reduced to 4-mesh in the stamp-battery more economically than to 12-mesh; (2) for the reduction of ore particles to 16-mesh, where 'all slime' is required, stamps, followed by Chilean mills are more efficient than stamps alone; (3) ore may be reduced to −200 mesh in the tube-mills more economically when the mill is fed with 16-mesh than when fed with 4-mesh.

There is one more point to be considered. On referring to the cost of installation, it will be seen that there was a saving of $103,000 in favor of putting in the Chilean mills. Assuming that the future cost of operating these mills will be 15c., an increase of 3c. per ton of ore milled over the cost of operating 140 stamps and 9 tube-mills, and assuming a yearly tonnage of 300,000 tons, it will take ten years operation, or the milling of 3,000,000 tons of ore, to offset this original saving. It is not believed that the cost will increase to this extent. Rolls and Chilean mills or other methods of comminution were not considered, for the simple reason that the problem was to increase the capacity of a 100-stamp mill, designed to deliver an 'all-slime' product to the cyanide plant. If this article invites discussion, I shall be glad to go into details more freely at some future time.

ELEMENTS OF COST IN OPERATION

Before passing to the detail of operations it may be well to enumerate the factors governing costs and efficiency.

Water.—Water for milling was supplied during the first two years operation entirely by the local water company, coming from the Palmetto range at Lida, nearly 30 miles distant. Recently the mine-water has been conserved and neutralized, and about one-fourth of that consumed is now supplied from the mines. The local company charges at the rate of 50c. per thousand gallons; the total water consumption per ton milled is 220 gallons, or 11c. per ton. This is a lower consumption of water than any wet-crushing mill on record so far as known. This item of cost is included in supplies.

Labor.—With the exception of two or three Slavs employed in roustabout work, the entire mill crew is American. The wage-scale is $3.50 per day of eight hours for ordinary labor; $4 to $4.50 for mill-men; $5 for machinists, electricians, and carpenters. Each man makes out his own time on a distribution slip, stating hours worked, department, and whether on operation or repairs, which slip, after being checked by the foreman on shift, is delivered to the timekeeper's office, where a daily distribution of labor is made after the following form:

GOLDFIELD CONSOLIDATED MILLING & TRANSPORTATION CO.

DAILY MILL REPORT

Tons.		Assay Oz. Au.	Oz. in Heads.	Oz. in Tails.	Oz. Produced.
	Per cent time run				
	Ore received				
	Ore milled				
	Mill residues				
	Conc. plant residues				
	Conc. plant solutions				
	Mill solutions				
	Oz. amalgam				
	Totals				

CONSOLIDATED MILL

Labor Distribution

Department.	Shifts Operating.	Amount.	Shifts Repairs.	Amount.
Crushing-conveying				
Sampling				
Stamping				
Amalgamation				
Chilean-milling				
Elevating-separating				
Tube-milling				
Concentration				
Neutralizing				
Settling				
Agitation				
Filtering-discharging				
Assaying				
Precipitation				
Refining				
Steam heat				
Surface and plant				
Warehouse-office				
Watchmen				
Salaries				
Total mill labor				
Concentrate plant				
Total labor				

The entire crew, under normal conditions, including all the superintendence, men on current construction, foremen, and others, consists of approximately 90 men. The average daily pay-roll is approximately $400, and the average daily wage, including superintendence, foremen, master mechanic, electricians, etc., $4.44. Labor per ton of ore milled is $0.46; tons milled per man on shift, 9.65.

Power.—Power is supplied by the Nevada-California Power Co. at a cost of $6 per horse-power month, based on 90% of the peak load.

The average power load at present is 1500 hp., equivalent to 1.73 hp. or 32c. per ton of ore milled. A segregation of this load for the first year is shown on a chart that will be printed in the continuation of this article. The campaign of construction which has been waged has left no time to devote to bring this up to date.

Supplies.—Supplies, as usual, constitute slightly more than 60% of the total mill-costs. Naturally the distance from bases of supplies, combined with discriminating freight rates, would make this cost higher than is usual for cyanide plants in the United States. In addition to

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these factors is the exceeding baseness of the ore, which, during the second year's run has shown an increase of nearly two pounds of KCN per ton of ore milled. Had the cyanide consumption remained as low for the second year as for the first, the total cost for milling and cyaniding would have been $1.85 per ton instead of $2.12. The following table showing costs for the first three months of this fiscal year and for 1910 may be of interest:

	Nov. 1910.	Dec. 1910.	Jan. 1911.	1910 Average.
Labor	$0.46	$0.476	$0.445	$0.56
Supplies	1.24	1.285	1.263	1.25
Power	0.276	0.325	0.311	0.31
Total	$1.976	$2.086	$2.019	$2.12

In this connection it will not be amiss to note that $40 mill-heads warrant considerably higher costs than does the ore usually sent to a stamp-mill and cyanide plant, and in judging these costs the fact that the ore has averaged approximately $40 for two years, should be taken into consideration.

OPERATION

Storage and Crushing.—The plant is situated about two and one-third miles from the various working shafts, from which the ore is transported over the company's railroad to the crusher-bins in steel hopper-bottom cars of 50 tons capacity each. Originally a Blake-Dennison automatic inclined weighing machine was used. It was destroyed in the fire and replaced by a Fairbanks-Morse railroad scale, over which the cars pass en route to the crusher. The operation of weighing cars has added nothing to the working cost, and has been decidedly more satisfactory than the automatic machine in this situation, where there are such decided and sudden changes in temperature. The transportation of the ore does not come under the head of mill operations, but I shall give the cost for the benefit of those interested. These costs are based on the actual dry tons milled, which normally will average approximately 25,500 tons per month. The figures include all labor, supplies, and other items incident to transporting the ore, and mine and mill supplies, as well as the maintenance of rolling stock and roadbed.

	Nov. 1910. Cents.	Dec. 1910. Cents.	Jan. 1911. Cents.
Operation	10	6	3
Maintenance	5	5	7
Total	15	11	10

The crusher-bins, as well as the entire crusher-house and belt-way are built of wood and have a storage capacity of 800 tons. A shaker-feeder of the suspended type, driven from the main motor (150 hp.) feeds the primary crusher through rack-and-pinion ore-bin gates. The crusher is a 7½-K Gates. Set for 2½-in. product, it delivers the ore to a 48 by 14-ft. Gates type revolving trommel, with 1½-in. apertures, which delivers the undersize to a 26-in. inclined conveyor (Stephens-Adamson type) and the oversize to two No. 4K short-head crushers, set for 1½-in. product, from which the ore gravitates to above-mentioned 26-in. conveyor. Concave plates of manganese steel for the crushers last approximately seven months and crush 175,000 tons of ore. Maganese-steel screen-plates for the trommel lasted 27 months and screened 504,000 tons. The Stephens-Adamson inclined conveyor, set at an angle of 19° 54′ and 369 ft. between centres, delivers the ore to a horizontal conveyor and tripper which distributes the product to a battery storage-bin. This bin is flat bottomed and has a capacity of 4000 tons. The fire of April 8, 1910, destroyed the inclined conveyor, so it is not known what life the belt would have given. The horizontal conveyor has handled 500,000 tons, and will doubtless accomplish half that much more before it is discarded. The entire crushing and conveying system is operated for eight hours by three men, two in the crusher-house and one on the conveyors. They handle 850 to 900 tons.

The cost of crushing and conveying is as follows:

	First 3 months 1911.	Year 1910.	Year 1909.
Tons per day	850	850	600
Labor (cents per ton)	2.1	4.3	3.0
Supplies	0.1	1.3	0.5
Power	1.6	1.5	1.8
Total	3.8	7.1	5.3

The high cost for 1910 is accounted for by the fire, which destroyed the conveyor system and necessitated the building of an inclined tramway from the battery-bin to crusher-house, and its operation for three months, during which time it was necessary to tram the ore in cars over the battery-bin. The 70 stamps which were not destroyed by fire, were dropping on ore supplied by this tramway in seven days and nine hours from the beginning of the fire.

Stamping.—As stated in the beginning, all details of construction have been published in Bulletin 1438 of the Allis-Chalmers Co., and in this article no attempt will be made to give such details except on new construction. Through rack-and-pinion ore-bin gates, from battery-bin, *via* Challenge feeders, the ore is fed to ten 10-stamp batteries, five right, and five left, with 1050-lb. stamps, dropping 108 times per minute through 7 inches. The stamp-weight is distributed as follows: boss, 252; shoe, 185; tappet, 169; stem (3½-in.), 444; total, 1050 lb. The 20 mortars (10,200 lb. weight, 14 in. thick through base) are of the narrow, rapid-discharge, improved Homestake type designed for this company by J. B. Fleming and built by the Allis-Chalmers Co. on his specifications. The cam-shaft for each ten stamps (of hammered iron) is of 6½-in. diameter. During the 27 months operation, not one cam-shaft has been broken. The guides originally furnished have been discarded in favor of the 'Couture' guide, designed and patented by A. F. Couture, the master mechanic at the plant. This guide consists of one piece cast-iron (or steel) sole-plate, with tapered sockets for receiving the taper-cored wedges through which the stems operate. The distinguishing feature of the guide is the four-side taper of the sockets and wedges, and the method of dovetailing which permits of the wedges being reversed when worn. The use of these guides has materially reduced the cost of operation and increased the efficiency of the stamps.

The one-piece mortar block of concrete (mixture 1 : 3 : 3) contains 850 cu. yd. Anchor bolts for mortar and post shoes are arranged as shown in the following sketch. It may be of interest to note that the concrete withstood the fire of April 8, which destroyed the refinery, store-house, conveyor-way, and twenty stamps. The heat at the mortars was so intense that when the stamps fell, the wrought-iron anchor bolts were drawn out to a fine point. The new equipment was erected on the same foundations with no repairs. There is no sign of serious cracking in the concrete.

One man on each shift feeds the stamps, and one head batteryman on day shift, with a helper, sets all tappets, turns stems, changes shoes and dies, and does all the repair work incident to operating. As amalgamation at the battery has been abandoned, the battery-feeder is alone on his floor, and under such conditions, it seems that 100 stamps is the economic unit for operation. By this it is not meant that total costs for a 140 or 150-stamp mill will not be less, but the actual operation of the stamps will be more, for the reason that while power and supplies will increase in direct proportion to the increased tonnage, labor for 140 or 150 stamps will increase abnormally. Power is supplied for each 20 stamps by one 50-hp. Bullock motor, belted to a counter-shaft underneath the floor. A rack-and-pinion belt-tightener between the bull-wheel (built-up wood pulley) and this line-shaft controls the power for each 10 stamps.

Chrome-steel cams, bosses, and shoes, and Pennington dies have proved the most satisfactory here. In 27 months only two chrome cams have been replaced. The following comparison of wear of battery-steel may be of interest:

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Make.	Days wear.	Price per lb., cents.	Cost per ton ore, cents.
Shoes.			
Chrome (8-in.).....	58	5.43	1.88
Midvale (10-in.)....	73	6.00	2.00
Pennington (8-in)..	57	5.97	1.91
Dies.			
Chrome (7-in.).....	83	5.43	1.25
Midvale (7-in.).....	78	6.00	1.30
Pennington (7-in.)..	104	5.97	1.06
Tonopah (7-in.)....	49	4.00	1.70

The total consumption of steel (shoes and dies) per ton of ore milled in 1909 was 0.98 lb. with a stamp-duty of six tons per stamp through 12-mesh (0.048-in.) screen; in 1910, 0.604 lb.; stamp-duty, 8.46 tons, through 4-mesh (0.18-inch).

The following is a record of time lost and causes thereof for the year 1910, and for three months of the present fiscal year:

Total time lost on account of	1911. Per cent.	1910. Per cent.
Power	0.84	0.70
Water	0.08	1.43
Shoes and dies	0.28	0.20
Screens	0.01	0.08
Stems	0.17	0.11
Chilean mills	0.17	0.55
Tube-mills	0.08	0.63
Cyanide plant	0.11	0.21
Miscellaneous	1.40	0.91
Cleaning bat. tank	...	0.55
Fire loss	...	7.91
Total	3.14	13.28

A comparison of sizing tests of the 12-mesh (1909) and 4-mesh (1910) product from the stamps is given below, but is of little value owing to the increasing softness of the ore with depth.

Battery discharge through 12-mesh 0.048-in. screen. Duty 6 tons per stamp.			Battery discharge through 4-mesh 0.18-in. screen. Duty 8.46 tons per stamp.	
	Mesh.	Per cent.	Mesh.	Per cent.
Remaining on	20	15.6	10	15.0
" "	40	28.2	30	34.0
" "	60	18.1	50	10.0
" "	80	5.3	80	10.0
" "	100	5.4	100	3.0
" "	200	6.4	150	3.0
Through	200	20.0	on 200	4.0
Through			200	20.0

Originally one 24-in. cone-classifier received the 12-mesh pulp from each battery of five stamps, the spigot product from it gravitating to Dorr classifiers, the overflow to concentrators. These 20 cones were discarded in favor of two 8-ft. cones, each taking the pulp from 50 stamps. The following is a comparison of the cost of stamping through 12-mesh and 4-mesh screens:

Table 1. Stamping through 12-mesh 0.048-in. screens. Duty 6 tons.		Table 2. Normal Conditions.	Table 3. Stamping through 4-mesh 0.18-in. screens. Duty 8.46 tons.	
	Cents.	Cents.		Cents.
Labor	8.9	6.6	Labor	3.9
Supplies	6.8	7.5	Supplies	4.1
Power	6.4	8.0	Power	5.4
Total	22.1	22.1	Total	13.4

The first and third tables have been taken from the yearly mill-records, and while the various items do not show the proportionate decrease, the total reduction in cost of operating is approximately what would be expected from a 40% increase in tonnage. Several factors besides the coarser crushing have reduced the labor cost in table 1 to the figure in table 3, such as the improved guides, completion of current construction work, etc. Likewise, supplies for 12-mesh stamping in table 1 are not exact, as 200 extra shoes and 200 extra dies, furnished with the plant, did not have to be charged to cost of operation. This is stated merely to avoid confusion, as the sum of the various items in table 1 checks table 2, which is an estimate of the cost of stamping through 12-mesh under normal conditions. The discrepancy between power costs in table 1 and tables 2 and 3 is due to the recent installation of a storage battery which absorbs the mine peaks and virtually puts the mill on a meter reading.

Regrinding.—The second aisle of the mill proper contains the regrinding apparatus, which is divided into two sections, each taking the pulp from 50 stamps. Each section consists of one 7-ft. spitzkasten, three L. C. Trent Chilean mills, one bucket elevator (originally one 54-in. Frenier sand pump), one 8-ft. cone classifier, one 4-ft. spitzkasten, three Dorr classifiers, three 5 by 22-ft. tube-mills, and nine No. 3 Deister concentrators. Each section is run by one man on shift and the entire floor is kept in repair by one machinist and one helper on the day shift. This, of course, does not include the re-lining of the tube-mills. The total pulp from the 50 stamps feeding each section, gravitates through wooden launders, lined with 1-in. cast plate (grade 1¾ to 12 in.), to the 7-ft. spitzkasten, overflow from which passes to the main concentrator floor, and spigot product to three 6-ft. Chilean mills. Power for driving these mills is supplied by one 100-hp. Bullock motor belted to an overhead line-shaft. Actual meter readings show that each mill requires 35 hp. to operate when pulverizing 75 tons per day through 30-mesh, and producing 30% of −200 slime from its total feed. When the capacity of the plant was increased to 850 tons, no additional settlers or dewaterers were put in, and for this reason it is impossible to attempt close classification ahead of the Chilean mills, as the additional water required for grinding in them would prohibit economic dewatering in the cyanide plant; hence a great deal of −30-mesh product is sent to the Chilean mills, but as they have proved to be fairly efficient 'slimers,' the work of the tube-mills is materially reduced, and the apparently poor practice is in reality good. The following is a typical sizing test of the product from the Chilean mills:

	Mesh	Per cent.
Remaining on	10 mesh	Trace
" "	30 "	4.0
" "	50 "	10.0
" "	80 "	16.0
" "	100 "	9.0
" "	150 "	6.0
" "	200 "	6.0
Through	200 "	48.0
Loss		1.0
		100.0

Midvale rolled forged steel has demonstrated its superiority for use in the mills. The dies, 5¼ in. thick, last approximately 125 days; the roller shells, 4 in. thick, average 165 days. Total steel consumed per ton ore milled, 0.32 lb. Screens for a time threatened to become an item of serious expense, as it was impossible to obtain a life of more than six days from the best screens on the market. However, by inserting a strip of 4-mesh screen in the screen frame where the hardest wear fell, thus protecting the screens from unnecessary wear, their life has been doubled.

The cost per ton of operating the Chilean mills, including all repairs and upkeep, is as follows: Labor, 1.8; supplies, 4.1; power, 4.7; total, 10.6 cents.

The pulp from the Chilean mills gravitates to one 18-in. bucket elevator with 28-ft. centres, where it is joined by the tube-mill product, all of which is fed to one 8-ft. cone-classifier, in which partial dewatering is accomplished before passing to the Dorr classifiers. The overflow from this cone passes to the two-compartment 4-ft. spitzkasten, and the spigot product from the cone joins the spigot product from the first compartment en route to the Dorr classifiers. The spigot product from the second compartment feeds the nine No. 3 Deister concentrators put on this floor when the Chileans were added. The overflow from this spitzkasten, together with the overflow from the Dorr classifiers gravi-

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tates through pipes to the main concentrator floor. Each Dorr machine handles approximately 110 tons of dry ore at a dilution of 3:1, delivering a slime product, 80% of which passes a 200-mesh screen, and a feed to the tube-mills as follows:

	Per cent.
Remaining on 10 mesh	8
" " 30 "	13
" " 50 "	12
" " 80 "	23
" " 100 "	18
" " 150 "	11
" " 200 "	7
Through 200 "	7
Loss	1
Total	100

The contained moisture is 30%, which is increased to 40% as it enters the mills. The cost of elevating and classifying is as follows, in cents per ton: Labor, 1.2; supplies, 0.5 power, 0.6; total, 2.3. Revere Rubber Co.'s 'Granite' elevator belt and malleable iron bucket lasted 14 months and elevated approximately 400,000 tons dry ore.

Tube-milling.—As stated, each regrinding section contains three 5 by 22-ft. tube-mills (Gates) of the spiral-scoop feed, trunnion-discharge, type. Each battery of three mills is supplied with power from one 200-hp. Bullock motor, belted to the line-shaft. Two of the three mills are belted from this line-shaft and controlled by friction clutches. The third is geared direct from the main shaft, and controlled by cut-off clutch coupling. There is no doubt that individual drives for each mill would be more satisfactory. With the present arrangement, the repairing of one clutch hangs up 50 stamps during the operation. Silex lining has been used since the beginning of operations. This lining lasts seven months and is renewed with the following items of expense:

LABOR

Removing and replacing manhole, removing end-liners	$11.88
Removing pebbles	3.75
Removing old lining	11.25
Re-lining	63.76
Replacing pebbles	7.50
Total labor	$98.14

SUPPLIES

Cast end-liners	$92.48	
Silex, 17,710 lb. at 2.634c. per lb	466.48	
31 sacks cement at $1.10 per cwt	34.10	
Total supplies		593.06
Total cost		$691.20

The time involved is: Hours re-lining, 68; hours setting cement, 72; total hours lost, 140. While this is less time than is customary at other plants to allow the cement to set, there has never been any trouble on account of starting too soon. The silex consumed per ton of ore amounts to 6 lb.; the cost of re-lining per ton of ore milled is 2.3c.; the amount of Danish flint per ton of ore milled is 1.8 pounds.

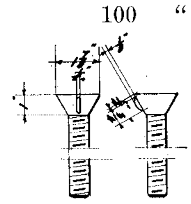
TOTAL COST OF TUBE-MILLING PER TON MILLED

Tonnage	850	850	600
Year	1911	1910	1909
	Cents.	Cents.	Cents.
Labor	1.4	2.1	2.6
Supplies	6.5	6.8	7.0
Power	8.7	9.7	11.0
Total	16.6	18.6	20.6

The design of the discharge screens on the tube-mill has been changed according to the following sketch, and the screens now last six months. The gears and pinions of manganese steel originally furnished with the mills are still in use after 27 months operation and will propably wear 12 months longer.

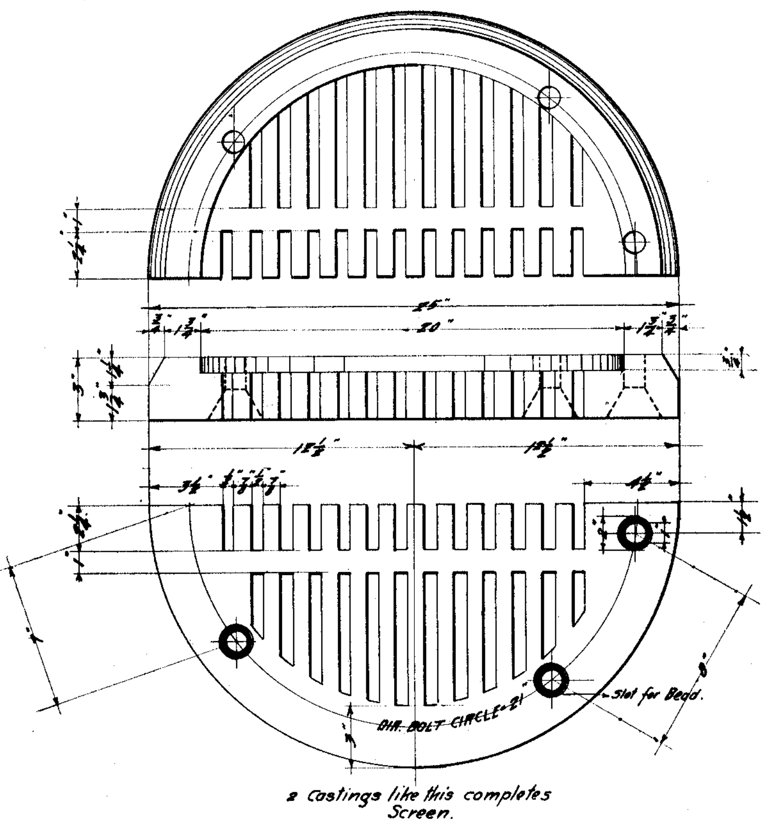
The product from the tube-mills is graded as follows:

	Per cent.
Plus 80 mesh	5.8
100 "	13.2
150 Mesh	11.6
200 "	17.8
Minus 200 "	50.6
Loss	11.0



BOLT FOR FASTENING SCREEN. SHOWING BEND ON HEAD.

Although the ore is classed as 'soft' ore, 40% of the product fed to the tube-mills is extremely hard quartz and is reduced to minus 200



CAST-IRON SCREEN FOR DISCHARGE END OF TUBE-MILLS.

product with difficulty. This was very noticeable when testing the mills with 4-mesh product from the batteries. The pulp discharged from the mills contains large quantities of coarse, rounded particles, which so accumulate after a few hours run that it becomes necessary to shut off the feed and grind the mills out.

After classification, the final product, which, as can be seen from the flow-sheet, is produced from two sources, the Dorr classifiers and spitzkasten, shows the following analysis:

	Per cent.
Plus 80 mesh	0.4
100 "	2.2
150 "	8.1
200 "	9.2
Minus 200 "	79.1
Loss	1.0
Concentration	100.0

(To be Continued)

NO CHROME IRON ORE has been produced in the Eastern States for many years, almost the entire production since 1880 coming from California. In 1909 the production of chrome ore, amounting to 598 long tons, came from California and Wyoming, the larger part being obtained from the latter State.

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Operation of the Goldfield Consolidated Mill

By J. W. Hutchinson

(*Continued from page 616.*)

CONCENTRATION

The third aisle of the mill is the main concentrator floor, containing 30 8-ft. Callow tanks, and 60 primary and 16 secondary concentrators (all No. 3 Deister slimers). The concentrators were put in after a competitive test on this ore with the suspended vanner; the Deister No. 3 machine winning by a wide margin. Each machine has the capacity, when followed by secondary concentrators in the ratio of 1:5, of handling 11 tons of dry slime, 80% of which will pass a 200-mesh screen and all of which will pass a 100-mesh, and each will concentrate therefrom 72% of the gold in the ore into 1000 lb. of concentrate. From this, 20% of the gold is recovered by amalgamation, leaving 52% to be recovered by further treatment of the concentrate. When the facts are taken into consideration, that the ore has been slimed before any of the concentrate has been removed, that 80% of the material recovered is −200-mesh, and that the ore can not be classed as a concentrating ore, the performance seems remarkable. Repeated monthly tests on a general sample of the tailing from the cyanide plant fail to show any appreciable gold recoverable by concentration. When the capacity of the plant was increased, 18 additional tables were placed on the regrinding floor, and 6 on the secondary floor. As stated, the tables on the regrinding floor take their feed from the spitzkasten, this feed assays 20% higher than the feed on the main floor. For this reason, the middling from the 18 upper tables is re-concentrated on the main floor and the tailing from the upper floor mixed with middling from the 60 primary tables.

Although 30 Callow tanks were placed on the main floor and were used for dewatering when the plant crushed 600 tons, at the present only 16 are in commission, since less water is now used for crushing. The Deister machine operates more satisfactorily on a pulp containing 3 to 3½ parts water. They are driven at 300 strokes per minute through 7/16 in. and require 0.73 hp. each to operate. One man per shift operates the 60 primary and 16 secondary tables; one man with a helper keeps all the machines in repair. This work includes all mechanical up-keep, as well as cleaning floors, machines, and pulp-thickeners. The helper brushes the mineral edge of each table every morning with a dilute HCl to remove precipitated salts. Each deck is brushed thoroughly every 60 days. No deck renewals have yet been necessary.

The concentrate from all floors, approximately 5½% by weight of the ore milled, gravitates to the concentrate-treatment plant and will be discussed later. Middling from the primary tables, amounting to 15% by weight, is re-concentrated without further regrinding on the 16 secondary tables. Reference to the flow-sheet will show the disposition made of each product. The cost for all renewals for 94 tables, including head motions, deck, riffles, etc., has averaged $150 per month for twelve months, or at the rate of $1.60 per table per month, or 0.6c. per ton milled. The labor for repairs and maintenance has been approximately $3 per table per month, or 1.2c. per ton milled, which includes everything incident to keeping the tables in first-class operating condition. The following table shows the total cost of concentrating, including the above items of repair:

Year	1911	1910	1909
Tonnage	850	850	600
	Cents.	Cents.	Cents.
Labor	3.5	4.0	4.0
Supplies	0.5	0.4	0.4
Power	1.5	1.5	1.8
Total	5.5	5.9	6.2

As pulps are being conveyed here with more variety in size of solids and degree of dilution than in the gold mills of this country or elsewhere, it is hoped the following table will prove of interest:

Product.	Dilution	Width of launder	Height of launder	Grade of launder	Dry tons handled in 24 hr.	Conveying pulp.
		In.	In.	In. per ft.	Tons.	
4-mesh from batteries	4:1	8	8	1¾	255	From 30 stamps to spitzkasten.
12-mesh from batteries	6.5:1	8	8	⅝	200	(1909) from 30 stamps to 8-ft. cone.
−4 mesh to Chileans	3.3:1	8	8	1⅛	400	From spitzkasten to three 6-ft. Chilean mills.
−30 mesh from Chileans	3.3:1	8	8	⅝	400	From three 6-ft. Chilean mills to boot of B. & B. elevator.
Tube-mill discharge, 50% −200 mesh	1.5:1	6	5	1¾	100	From one 5 by 22-ft. tube-mill to boot of B. & B. elevator.
Mixture of tube-mill and Chilean products	2.6:1	8	8	¾	700	From three 5 by 22-ft. tube-mills and three 6-ft. Chilean mills to 8-ft. cone.
Mixture of tube-mill and Chilean products	2:1	6	5	1⅛	200	Feed to one Dorr classifier.
Final product from tube-mills	3:1	5½	8	⅜	330	Feed to 30 No. 3 Deister slimers.
Product from Callow tanks	3:1	3½	3½	⅞	22	Feed to 2 No. 3 Deister slimers.
Concentrate	9:1	3	3	1 1/16	3.5	From 6 No. 3 Deister slimers to main launder.
Middling	3:1	3	3	¾	10	From 6 No. 3 Deister slimers to main launder.
Tailing	5:1	5	5	¾	52½	From 6 No. 3 Deister slimers to main launder.
Concentrate	9:1	4	10	¾	50	Main launder to concentrating plant.
Tailing	5:1	10½	10½	3/16	800	From 100 stamps.
Clear water		18	14	⅛	4000	Clear water overflow from 800 tons.

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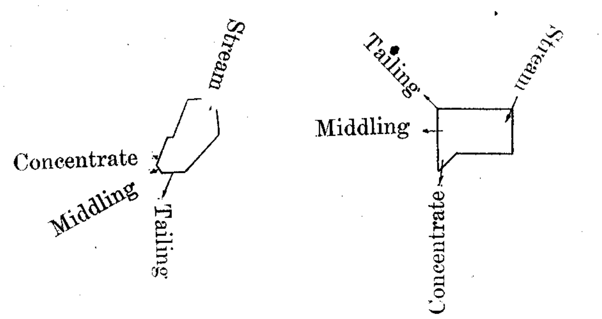
NEUTRALIZING AND DEWATERING

The lime-mixing plant is situated about 200 ft. from the mill proper and consists of storage bins (capacity 120 tons) into which the lime from the railroad cars is shoveled. Lime from these bins is slacked and dumped into two 4-ft. Wheeler pans, from which the mullers have been removed. These pans act simply as stirrers and deliver a continuous stream of milk of lime, which is laundered to the mill, with branch launders to the dewatering tanks, Pachuca agitators, and concentrate-treatment plant. The bulk of the lime consumed is added to the main launder conveying the table tailing to the dewaterers. No lime is added at the battery, and for this reason there is not the serious trouble with concentrator decks noticeable at most cyanide plants. The



FLOW OF MATERIAL OVER DEISTER TABLES.

lime for neutralizing is so regulated that the overflow water from the dewaterers titrates 0.4 to 0.5 lb. CaO per ton of water. This mill-water gravitates through two clarifying tanks to mill-water sump-tank, from which it is elevated by means of two 10 by 12 Aldrich pumps to the mill-water supply tank behind the stamps. The water leaving the batteries is a trace acid to phenol, which acidity has increased to 0.2 when the pulp reaches the concentrators. The ferrous salts generated during crushing are neutralized and oxidized en route to the dewaterers. The 16 dewatering tanks, 29 ft. 6 in. by 12 ft., with 16° false cones, are arranged in two aisles of 8 tanks with 7 ft. difference in elevation. The total capacity of the dewaterers is 96,000 cu. ft., equivalent to 120 cu. ft. per ton of ore, or 20 cu. ft. per ton of pulp. Each tank is equipped with a central well for the inflow of pulp, and peripheral launders and pipe decanters for handling the clear water. The helper in the cyanide plant regulates the lime and settlers, decants and transfers the charges. The cost of these operations is as follows:

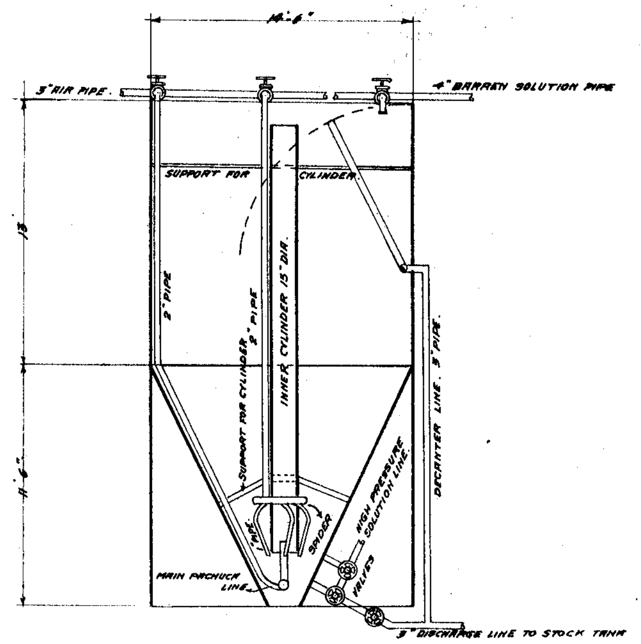
	Neutralizing, cents.	Dewatering, cents.
Labor	0.6	1.4
Supplies	4.1	4.1
Power	0.1	0.1
Total	4.8	5.6

CYANIDATION

From the dewaterers the pulp, containing 40% water, is pumped to a battery of ten 15 by 45-ft. Pachuca agitators, arranged in series, and is diluted to 1½:1 (Sp. G., 1.35) with solution from the wash-solution tank (No. 3 above) in the Butters filter system. The solution is increased to 1.2 lb. KCN per ton and the alkalinity 0.5 lb. in terms of CaO, and maintained at these strengths for 26 hours. Lead acetate, to the amount of $^1/_6$ lb., is added at the beginning of treatment and again in tanks 4 and 8 of the series. This seems unusual for gold ores, but the decomposition of the complex sulphides in the Goldfield Consolidated Mines Co.'s ore makes it necessary. All attempts to reduce the amount have resulted disastrously. Various oxidizing agents, together with bromo-cyanide, have been tested on a working scale without commercial success. The bromo-cyanide gave increased extraction, but the cost was prohibitive, except on high-grade ore. As stated, concentration tests on the tailing are made on a general monthly sample, but the loss is not in recoverable concentrate.

There is one feature which is most interesting. The assay value of the −200 product in the tailing is considerably higher, nearly 30%, in fact, than the product remaining on 200 and 150-mesh screens. This is true in both the mill and concentrate plant and is doubtless due to the presence of very brittle and insoluble alloys of gold. A picked sample of the higher-grade ore supplied to the mill gave by analysis the following percentages of elements which tend to confirm this theory: Bismuth, 0.25; antimony, 0.055; tellurium, 0.025; selenium, 0.050.

In the agitators approximately 80% of the value of the ore after concentrating is dissolved. Previous to connecting the tanks in series they were operated as 4 batteries, two of 3 and two of 2 tanks. At a gravity of 1.35, with a loss of 4 hours for transferring, the average period of agitation was 23 hours. After the publication of M. H. Kurylas' article on the results of continuous agitation at Esperanza, I wrote the management of that property asking for additional information, which was given. Using Mr. Kurylas' sketch, the Pachuca tanks were connected in series by means of 8-in. pipe connections, and experiments begun. These were not favorable to continuous agitation, and for six months thereafter the tanks were worked intermittently. The failure of the system was not explained until recently. After a new compressor with much more capacity than those previously used for agitating was put in, the tanks were again connected in series and with marked success. The only reason known here for the failure at first is that uniform agitation throughout the series is most essential. With the inadequate amount of air at



PACHUCA AGITATOR FOR CONCENTRATE TREATMENT.

that time, this could not be maintained, and the tailing, gravitating from the series to the filters, had not received uniform treatment. The following assays will give an idea of the first and second trials.

	First Trial.			*Present Operation.*		
		Oz. Au.			Oz. Au.	
	Gravity.	Pulp.	Sol.	Gravity.	Pulp.	Sol.
Heads	...	0.44	...	...	0.62	...
Agitator No. 2	1.36	0.17	0.17	1.35	0.31	0.21
" No. 4	1.36	0.16	0.17	1.34	0.26	0.256
" No. 6	1.34	0.16	0.18	1.33½	0.20	0.30
" No. 8	1.33	0.14	0.193	1.33	0.14	0.328
" No. 10	1.33	0.12	0.20	1.32	0.10	0.348
Filter Tailing	...	0.10	...	...	0.08½	...

The lighter gravity in the last tanks of the series is accounted for by the addition of solution for dissolving chemicals. The screen tests on the various tanks is as uniform as could be expected. There are so many different

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kinds of ore being fed to the mill, that it is impossible to make a definite comparison of the two methods of treatment, as the ore varies within wide limits during a week's run. The continuous system during the first run was alternated weekly with the intermittent three times, with an apparent increase in tailing loss of 30%. The present operation shows a decrease of nearly the same amount, but on a higher grade of ore. The following analysis of the solution before precipitation shows some of the difficulties encountered in the treatment:

Analysis of Cyanide Solution at G. C. M. Co. by Von Schultz & Low.

(The percentages refer to the weight of the solution. Total solids by evaporation, 0.3932%. Sp. gr. solution at 70°F., 1.002.)

	Per cent.
Silica	0.00198
Ferric oxide	0.00014
Alumina	0.00020
Manganese	Trace
Lead	Trace (faint)
Bismuth	Trace (faint)
Cadmium	Trace (doubtful)
Copper	0.03686
Arsenic	0.00010
Antimony	0.00014
Nickel	0.00010
Cobalt	0.00020
Zinc	0.00330
Selenium	None
Calcium oxide	0.02837
Magnesium oxide	0.00014
Phosphorus pentoxide	Faint trace
Sulphur trioxide	0.03282
Chlorine	0.03260
Tellurium	0.00008
Gold	0.185 (oz. Au)
Silver	0.068 (oz. Ag)

The total sulphur was determined as sulphur trioxide, and no attempt was made to determine the form of combination of the elements found.

The cost of cyaniding, which, in the method of accounting employed at the plant, is really the cost of agitating, includes all chemicals necessary for the dissolution of gold, and all labor connected with the operation of tanks, pumps, and compressors, is given below. Naturally the cost has varied considerably during the three years operation, due to the increasing cyanide consumption. The mechanical cost is practically constant. Approximately 75 cu. ft. of free air per minute is required for each tank of 85 dry tons, equivalent to 6 hp. per tank. Power per ton agitated is 2c. Maintenance and repairs for tanks, compressors, pumps, and pipe-lines is less than one cent per ton agitated. The total cost, including all the above items of chemicals and repairs, is as follows:

Year	1911	1910	1909
Tons	850	850	600
	Cents.	Cents.	Cents.
Labor	2.8	2.6	2.8
Supplies	56.6	51.0	40.3
Power	2.4	2.5	3.2
Total	61.8	56.1	46.3

It may be well to state here, in order to correct erroneous impressions given in articles on cyanidation of the low-grade surface ores at Goldfield in mills other than those of the Goldfield Consolidated Mines Co., that it is a characteristic of the sulphide ores of the district to increase in refractory elements with the increase in value. One notable shipment of high-grade ore contained nearly 2.5% tellurium with a gold content approximating 2%. With depth the baseness of the ore naturally increases. It has been demonstrated here to the satisfaction of all interested, that the lower-grade ore is more amenable to treatment by cyaniding than the higher-grade ores, and that the percentage of gold extracted does not necessarily increase with the increase in value. This is the inference made in some recent articles commenting on the extraction of one of the local mills, operating on $12 upper-level ore. It is stated that this mill makes a saving of 93% on such ore, which approximates the saving made at the Consolidated mill on $30 ore. It is interesting to note that immediately after the fire of 1910, when the higher-grade ore from the deeper levels was shipped to smelters, and the mill heads lowered to something like twice the value of those at the mill referred to, the extraction at the Goldfield Consolidated mill averaged 96%. As soon as normal operations were resumed with the mill treating all the deep-level ore, with consequently increased value in the feed, the percentage recovered dropped to 94½, which was the average for the year of 1910. Attention may be called to the performance of the old Combination mill, which for crudeness of design and lack of conveniences rivaled the plants in question. In this mill, which treated the upper-level ores, and consequently those least refractory, an average extraction of 95% was maintained. It has been the policy of the Consolidated company to include in the cost in its monthly and annual reports the residues from the concentrate treatment plant which could not be shipped at a profit, with the tailing losses at the mill. As a consequence the published report of recoveries has not been a statement of the metallurgical efficiency, but more nearly the statement of the percentage of value applicable to expenses and profits. Therefore, the greater metallurgical efficiency in the concentrate plant resulted in a lower reported extraction in the mill.

(To be Continued)

Filling Mine Workings

By Charles Enzian

The tailing from the screens preparing the smallest coal for market, or particularly for boiler fuel purposes, called the 'culm', is utilized in the anthracite district of Pennsylvania for mine filling. This culm is taken into the mines by means of bore-holes or pipe-lines in the shaft or slope, in mixtures containing about 90% water, and deposited into worked-out chambers, being confined by means of suitable dams and batteries. The water drains off and the culm remains as residue, which in the process of depositing and draining generates considerable heat due to the pyrite particles present, and this, in combination with calcareous or carbonaceous shale finely ground and held in suspension by the water, forms a binder. The aggregate becomes quite firm and will stand unsupported after some time and thus sustain considerable weight. This enables second mining to the extent of from 25 to 35% of remaining solid ground. The new opening is then filled and allowed to settle into stability, and, depending largely upon whether or not the surface is valuably improved, and assuming that a risk of subsidence would be discreet, an additional or third mining may be undertaken, leaving at least 50% of the remaining solid as permanent support and the filled area to give vertical and lateral support.

To illustrate: Assuming that the first mining was so conducted as to leave 60-ft. pillars solid, the chambers being worked 20 ft. wide; this would leave a 'remaining solid' of about 60 ft.; upon the second mining, a 15-ft. slice is taken, representing 25% of the remaining solid, or a total extraction of 44%. After filling again, another slice of 20 ft. is taken, or 44% of the remaining solid, thus giving a total extraction of 70% of original solid—an ultimate mining impracticable without the method of refilling or 'flushing' as it is termed. The flushing may be with other than vein material. A common practice at the present time is to crush all refuse material from breakers, boiler plants, and rock banks, and flush into the mines. Flushing should be done systematically and scientifically; keeping in mind the preservation of roads and air courses. This necessitates building dams at the foot or entrance of each chamber, also a drainage or overflow through the entire chamber length for the escape of accumulated water, thus relieving the dams from water pressure.—*The Black Diamond.*

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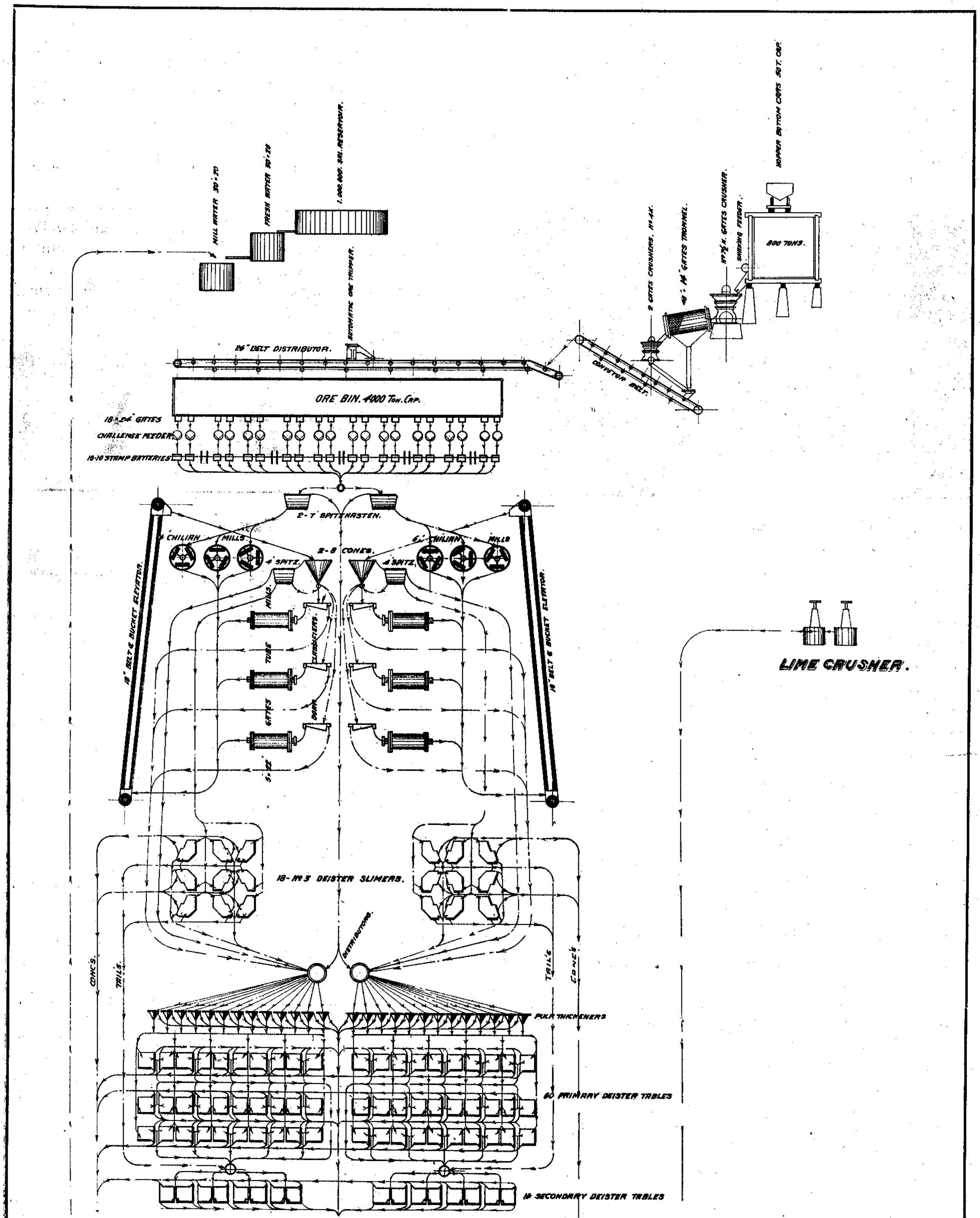
Operation of the Goldfield Consolidated Mill

By J. W. Hutchinson

(Continued from page 652.)

FILTERING

From tank No. 10 of the agitator series the pulp gravitates to two 34 by 12-ft. pulp-storage tanks fitted with mechanical stirrers. These tanks are situated about five feet above the top of the filter-boxes. On the same level are two wash-solution tanks which will be called 'No. 3 and No. 4 above' in the following description. Reference to the flow-sheet will assist in following the cycle of operations. From the pulp-storage tanks through a 16-in. pipeline the pulp gravitates to two steel filter-boxes with 6 hoppers each. The boxes contain 168 leaves each and are worked as one unit. All valves on the 16-in. filling and emptying line, are actuated hydraulically with levers from a central switchboard. After the cake has been formed the excess pulp gravitates through a 16-in. line to an excess-pulp tank from which it is elevated to the two filling tanks by means of 5-in. Morris pumps. Wash solution is run in from 'No. 4 above', which tank is filled with precipitated solution direct from the barren sump-tanks. After washing and dropping the cake, the supernatant solution is decanted to the excess-solution tank, from which it is elevated to 'No. 3 above' by means of 4-in. Krogh pumps. The solution for diluting the charges from the dewatering



Flow-Sheet of Goldfield Mill *(Continued on Opposite Page)*.

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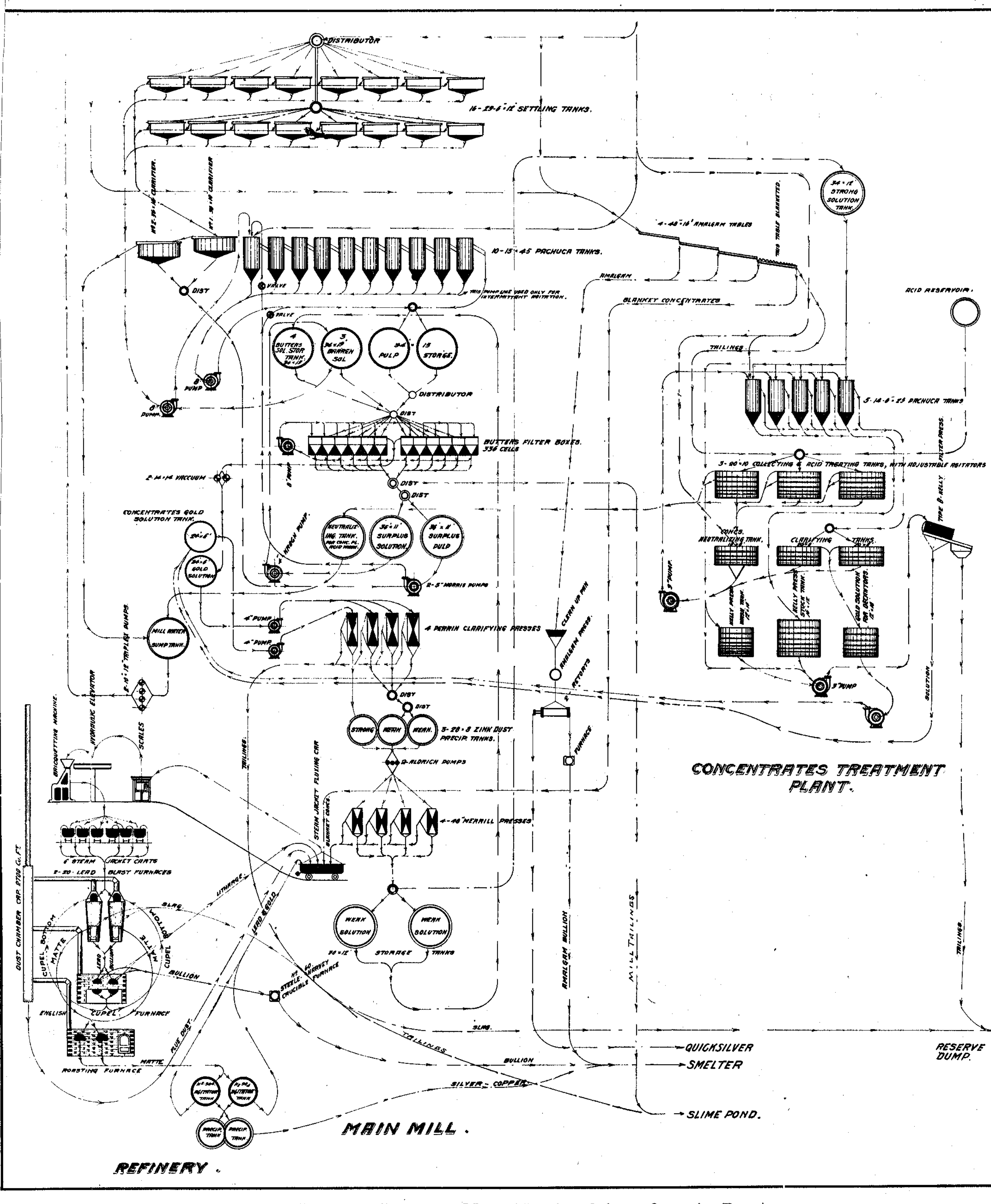
tanks is drawn from 'No. 3 above' and discharged into No. 1 Pachuca of the continuous system. In this way, by using the solution which has been in contact with the filter-cake once, for agitating the new charges, the accumulation of gold in wash solution, which is the main 'talking point' of the opponents of submerged vacuum-filtration, is avoided. The value in wash solution here is directly dependent on the efficiency of the precipitation, and while absolute recovery of dissolved metal is not claimed, the loss is unavoidable. Furthermore, it has been demonstrated beyond a doubt that additional extraction is caused by the leaching action of the wash solution through the cake, which cannot be accomplished by means of agitation unless the tailing from the agitator is treated with a freshly precipitated solution. The amount of gold dissolved on the filter may be *reduced* by longer agitation, but the point has never been reached where the wash solutions do not dissolve gold in appreciable quantities. Ordinarily, this amounts to 30c. per ton. During June of 1910, when the plant was running at 70% capacity on account of the loss of 30 stamps by fire, the pulp was agitated for six hours longer than had been customary with full tonnage. The tailing from the Pachucas assayed lower, but the filters still gave additional extraction, though to a less degree than was normally the case, indicating that all the gold dissolved by longer agitation would have been recovered on the filter. The following comparison, taken from the June records, is interesting:



FLOW-SHEET OF GOLDFIELD MILL *(Continued from Opposite Page).*

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	Oz. Au.
Tailing samples of 91 charges from Pachucas to filters (thoroughly washed with water)	0.095
Samples of 200 Butters filter discharges from same pulp	0.088
Extraction on filter	0.007

It is also believed here that the completeness of the displacement of dissolved gold in the cake, as well as the additional dissolution of gold on the filter, is not as dependent on the quantity of wash solution passed as on the length of contact. Consequently, I prefer to wash for a longer period with reduced vacuum. The time allowed for this operation ranges from 85 to 100 minutes. During this time 1⅛ tons of wash solution per ton of ore is passed through the cake, of which solution the following are typical assays, the moisture in discharged pulp being $33^1/_3\%$:

	No. 1. Oz. Au.	No. 2. Oz. Au.
Effluent solution making cake	0.192	0.20
Effluent wash at end of 10 min.	0.17	0.19
" " 20 "	0.166	0.185
" " 30 "	0.164	0.182
" " 40 "	0.158	0.168
" " 50 "	0.08	0.092
" " 60 "	0.04	0.03
" " 70 "	0.03	0.025
" " 80 "	0.025	0.022
" " 90 "	0.015	0.015
" " 100 "	0.006	0.015
Sample solution in hoppers 10 min.	0.001	0.008
" " 20 "	0.001	0.008
" " 40 "	0.001	0.007
" " 60 "	0.002	0.007
" " 80 "	0.002	0.010
" " 100 "	0.002	0.010

Assuming that the amount of solution discharged during the 10-minute intervals is constant, which is perfectly fair, the average value from the assays of the effluent wash for test No. 1 is 0.085 oz. gold, and for test No. 2, 0.092 oz. gold. Naturally this average value is not correct, since the solution is sampled intermittently during a process of gradual reduction in grade, but for the purpose in view it is sufficiently close. It will be noticed from the tables that only between the 40, 50, and 60-minute samples is there a decided drop in value. At the end of the fiftieth minute, in both cases given and in numerous tests made, 87% of the total value of the effluent wash has been picked up. The last fifty minutes and 50% of the total solution passed, are required to accumulate 13% of the total value. In both cases, based on the original assumption that the flow is constant for each 10-minute interval, only 1c. per ton of ore is removed from the pulp during the last ten minutes of washing. In test No. 1, assuming that the total moisture of $33^1/_3\%$ is of the same value as the one-hundredth minute sample, which, on the face of it, is unfair to the filter, the unwashed gold left in the pulp amounts to 0.003 oz., or 6c. per ton. In test No. 2, in which the wash solution is high, due to the higher value in the barren sumps, the unwashed value is 0.0075 oz. gold, or 15c. per ton. By referring to the subject of precipitation, it will be noticed that during the whole time of operation the barren sump solutions have averaged 0.004 oz. gold; from which fact it is safe to state that the soluble gold left in the discharge pulp has not exceeded 6c. per ton, discharged. When the filter is credited with the additional extraction obtained on it, which cannot be obtained with those filters passing large volumes of wash solution in a short period of time, the adoption of the submerged vacuum-filter has certainly been justified on this ore. The filter plant contains 33,600 sq. ft. of filter surface and averages 50 lb. of slime filtered and 120 lb. of solution recovered per square foot of filter per day. The usual Butters automatic acid-washing apparatus is used to circulate a ½% solution of HCl through the leaves to remove the carbonate of lime. Ten leaves are acid-treated each day, which makes a complete cycle in 34 days. It is worthy of special mention that the original filter-cloths are still intact after 27 months operation, and that they will undoubtedly wear 12 months longer. The tailing is discharged through twelve 12 by 12-in. flat Wheeler gate valves at the bottom of the hoppers into a tunnel, through which it gravitates to the slime pond. For the information of a certain distinguished gentleman who writes an annual review of cyanidation, and for the attention of his readers who may be misinformed, I beg to state that never, since the beginning of operation, has wash-water been added to the tailing discharged from this plant to be settled and returned to the mill for subsequent precipitation, and that the percentage of moisture in the discharged pulp does not now exceed and never has exceeded 35%. Any excess solution built up from the incoming moisture from the de-waterers is carefully precipitated, stored until the assay-value has been determined, and wasted only when the value does not warrant further expense. The following data is representative of normal operations:

Filter Cycle.	Min.		Min.
Filling boxes with pulp	10	to	10
Making cake	60	"	80
Emptying excess pulp	14	"	14
Filling with wash solution	10	"	10
Washing	85	"	100
Decanting and discharging	20	"	20

Total time per cycle.............3 hr. 15 m. to 3 hr. 50 m.
Tons per cycle...............125 to 150

COST OF FILTERING, INCLUDING UP-KEEP OF SLIME-POND

Year	1911	1910	1909
Tons	850	850	600
	Cents.	Cents.	Cents.
Labor	3.6	4.1	4.3
Supplies	1.2	1.7	1.9
Power	2.2	2.6	3.1
Total	7.0	8.4	9.1

(To be continued.)

April Copper Review

By Misha E. Appelbaum

The copper market during the greater part of April was in a stagnant condition; most sales were made at 12¼c., delivered thirty days, for both Lake and electrolytic. The domestic consumption showed a sharp shrinkage, but the export shipments were on a fairly large scale, and this with 12,000,000 lb. smaller production than the month before, resulted in an increase in the visible supply of only 3,500,000 lb.; however, the net result for the month was a decrease, since the shrinkage in the European visible supply was greater than the increase in the domestic surplus. May 11 the market is weak, and sales have already been recorded as low as 12⅛c., delivered thirty days. I can only repeat, that unless there is a sharp improvement in business, or a radical curtailment in the production takes place, lower prices will be made. It becomes more and more evident that the real curtailment will come if the price of metal should fall to 11c. A lot of smelters which do custom business with small mines, will then have a smaller production, and it is hoped, for the good of the trade, that this decline will take place as quickly as possible.

Missouri has become the State which produces the largest quantity of lead, having passed Idaho in 1907. In 1909 it produced 40.46% of the total, compared with 27.57% furnished by Idaho, 18.30% by Utah, 8.32% by Colorado, and 1.33% by Nevada, the other States contributing less than 1% each. Missouri produces over 94% of the supply of soft lead in the United States. In 1910 Missouri produced 161,659 tons out of a total of 372,227, according to the U. S. Geological Survey.

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Operation of the Goldfield Consolidated Mill

By J. W. Hutchinson

(*Continued from page 686.*)

Precipitation

The solution from the Butters filter is clarified in three 36 by 36-in. 60-frame Perrin presses, which are fed by one 4-in. Morris centrifugal pump, direct-connected to a motor. They have the capacity to clarify 2000 tons of filtered solution per day, equivalent to 1300 lb. of solution per square foot of filter-surface. To this operation must be accredited part of the success obtained here with zinc-dust precipitation. At times it has been necessary to pass part of the solution to the precipitation-tanks without clarification, and each time it has resulted in a precipitated solution of high metal content. The high-grade solution from the concentrate plant is clarified in a similar press and precipitated in the mill presses. The solution from the clarifying-presses gravitates to three 28 by 8-ft. redwood tanks, two of which are for the mill and one for the concentrate-plant solution. The usual Merrill equipment is used for feeding and emulsifying the zinc-dust, which gravitates through 1-in. rubber hose to the suction pipes of two 7 by 9-in. Aldrich pumps, which deliver the solution and zinc-dust to four 30-frame, 48-in. triangular Merrill precipitation-presses. These presses have 1680 sq. ft. of filter-surface and 140 cu. ft. of storage for precipitate, equivalent here to 1.6 tons solution per square foot of filter per day, and to ¼ cu. ft. of storage per pound of daily precipitate. Each press, when filled at a pressure not exceeding 5 lb. per square inch, will hold approximately 2500 lb. of precipitate containing 30% H_2O.

In figuring precipitating equipment for silver ores, the filter-surface is not the only consideration, as can be seen from the following instance. In evolving a process for treating a high-grade silver-gold ore, it was decided to precipitate 600 tons of solution from 120 tons of ore. The equipment, which a zinc-dust process company agreed to furnish, contained 384 sq. ft. of filter-surface, which would have been satisfactory for filtering. The storage-room in this equipment amounted to 32 cu. ft. The amount of daily precipitate at this plant is estimated at 400 lb. It can readily be seen that had this equipment been put in it would have been necessary to clean the presses every fifth day. It was finally decided to put in 800 sq. ft. of filter-surface and 70 cu. ft. of storage, and clean up three times per month. It seems that it would be advisable for the patentees of precipitating-presses to design a special press for silver ores with a greater proportion of storage room to filter-surface than the standard press in use has. It has been found necessary here not to allow the pressure to exceed 5 lb. per square inch; at a higher pressure the precipitate has a tendency to cake, which prevents the inflowing solution from filtering through the excess zinc-dust and being precipitated. This experience is in direct contradiction to the theory that precipitation is complete in the pipe-line, but it has not been possible here to corroborate this. After cleaning the presses, in order to insure satisfactory precipitation, it is necessary to use $^3/_5$ lb. of zinc dust with the first 250 tons pumped through the presses. After the filter-cloths have been coated with this excess zinc, the amount used varies from ⅛ to $^1/_5$ lb. per ton of solution. During the first 18 months operation the plant was supplied with most inferior zinc dust, which resulted in a high consumption of this chemical, as can be seen by referring to the subjoined table. In the latter part of 1910 it was decided to have this commodity shipped in metal-lined cases, similar to the cyanide package. The beneficial effect was immediate, and although the first cost is ¼c. per pound higher than the quotations for barrel packages, the consumption of zinc dust is reduced enough to offset this expense many times.

At the beginning of operations the strong and weak solutions were precipitated in separate presses. During this time the total zinc in the precipitate averaged over 30%. It was decided later to pump the strong solution, which titrates 4.5 lb. KCN and averages 1½ oz. gold per ton, through the weak-solution presses. By doing this the zinc has been reduced to 15%, and no more zinc dust is required to precipitate the 1½-oz. solution than is used on the mill-solution. This is one reason why it is believed here that the condition of the press is more important for efficient precipitation than length of pipe-line. Since the metal-lined zinc-dust package has been used, three presses are kept in constant operation, and the fourth cut-in only at the beginning of the bi-weekly clean-up. The clean-up car, which is steam-jacketed, runs on rails underneath the presses, as shown in the sketch. This change was made in order to avoid handling or transferring the precipitate, and is very satisfactory. Six hours are required for two men to clean a press and have it ready for operation. The work of cleaning is done by the refinery crew, which consists of three shifts of two men each. All filter-cloths for the clarifying and precipitating-presses are cut by the filter operators who have some little time during each cycle for this work. A double thickness of twill is used for filtering; when the outside cloth becomes worn, it is taken off, burned, and added to the precipitate.

The efficiency of this method of precipitation is shown in the following table:

	1911.	1910.	1909.
Value of Mill Solution—	Oz. Au.	Oz. Au.	Oz. Au.
Before precipitation	0.200	0.200	0.230
After precipitation	0.003	0.004	0.005
Percentage of recovery	98.5	98.0	97.8
Value of Concentrate Plant Solution—			
Before precipitation	1.56	1.22	1.40
After precipitation	0.014	0.016	0.017
Percentage of recovery	99.0	98.7	98.8

In the cost of precipitating is included the power required to elevate the solution to the press-room and the total cost of operating the clarifying-presses. The latter may seem unjust to the zinc-dust precipitation, but since it is a detail absolutely essential to the successful operation of this method, it should be charged against it. The solution from the filters, although apparently clear, contains a minute amount of flocculent slime which would cause the pressure in the precipitating-presses to rise if it were allowed to pass to them. In addition, the unclarified solution would precipitate, from a physical standpoint, on zinc-shaving, and as stated, it is thought here this cost is properly charged against the precipitation. The following table shows the cost, including cleaning-up and items mentioned above:

	1911.	1910.	1909.
Tonnage	850	850	600
	Cents.	Cents.	Cents.
Labor	1.0	0.9	1.3
Supplies	4.9	8.7	9.0
Power	1.4	1.4	1.7
Total	7.3	11.0	12.0

In spite of the fact that the heat was so intense it warped 12-in. I-beams, destroyed part of the steel structure of the press building, and burned the zinc in the presses, they passed through the refinery fire without damage, yielded 11,000 oz. of gold which they contained, without loss, and were in operation four days after the disaster.

Melting

The melting-room, as originally designed, contained a double-muffle drying-furnace and four Faber du Faur tilting furnaces for treating this precipitate. Apparently not enough experimental work was done on this part of the treatment while engaged in making the design for the

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plant. The process as outlined consisted of nitre roasting the precipitate, with subsequent melting in the tilting furnaces. The bullion from this method averaged about .250 fine in gold and silver. The analysis given will explain why. Later, acid-treating tanks were put in, which materially reduced the amount of precipitate to be melted, and by the addition of pyritic concentrate to the flux, enough copper and lead were converted to matte to raise the grade of the bullion to 425 fine, gold and silver. The following is a typical analysis of the bullion originally made at the plant:

	Per cent.
Au	34.87
Ag	5.75
Cu	40.50
Pb	12.95
Zn	3.48
Cd	1.31
Fe	0.18
Mn	Trace
As	0.10
Sb	0.15
Bi	0.07
Ni	0.16
Co	0.03
Te	0.39
S	0.10

The process was unsatisfactory from every standpoint, as the comparative tables below will show. The fire, which originated in this melting-room, through the failure of a defective bushing in the fuel-line, made it imperative to go ahead with work which had long been planned but postponed on account of other work which seemed more necessary. Much work was done with a view of finding some wet method which could be economically applied to the precipitate. The large amount of base metals made the cost of reagents prohibitive. Electrolytic parting of the base bullion would have made it necessary to carry a large stock of silver in order to secure the correct proportion of silver to gold for rapid parting. This idea was abandoned on account of the high cost. Cupellation of the briquetted precipitate, as practised at the Homestake, was impossible on account of the high percentage of copper. Laboratory work with a modification of the Tavener process, in which enough sulphur obtained from concentrate was added to matte the copper, yielded a clean lead bullion, and low-grade slag and matte. The cupellation of this base bullion left a gold bullion carrying approximately 850 parts gold and 80 parts silver.

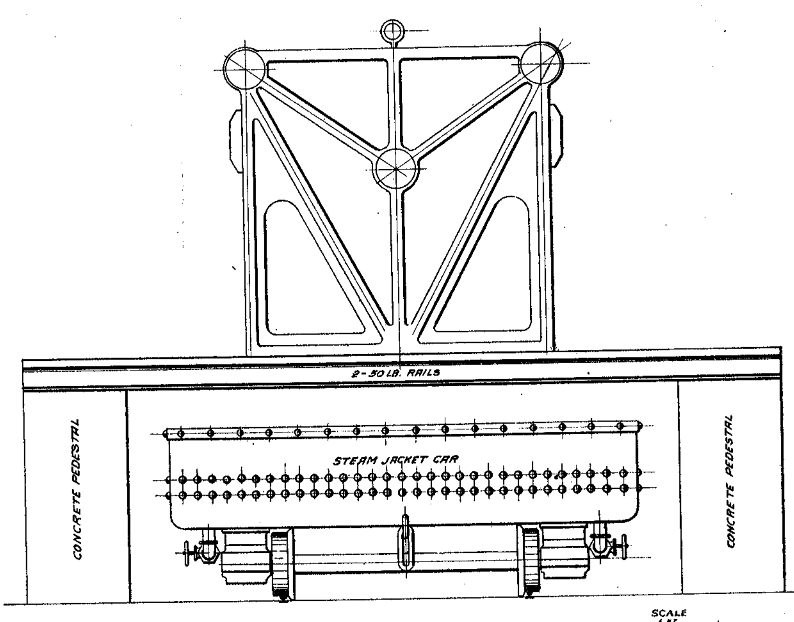
About the time it had been decided to briquette the precipitate with litharge and concentrate and smelt in a small reverberatory with suitable flux, Henry Hansen, the mill superintendent of the Pittsburg Silver Peak Co., at Blair, Nevada, told me of some experiments which he had been making with a small blast-furnace for smelting his briquetted precipitate. The refining process at that mill was at that time similar to Homestake practice. He had found the smelting of the briquetted material in the cupels a tedious operation, and had made some tests in a small blast-furnace which he was using for cleaning his cupel slags and for other clean-up work. The bullion from this furnace was cupelled, and the total time and expense of melting were reduced materially.

Smelting precipitate containing 40% copper on a lead basis did not sound very attractive as blast-furnace work at first, but after experimenting here it was decided it could be done very efficiently and economically. The precipitate is now treated in the following manner: The upper floor of the refinery is connected with the precipitating-room by a 30-in. gauge track with branches to each of the four presses. The steam-jacketed clean-up car is run underneath the press to be cleaned, the precipitate is dropped into it and hoisted to the refinery, where the total weight is determined on platform scales. After deducting the weight of the car and the moisture, fluxes are added in the following proportion:

	Parts.
Dry precipitate	100
Litharge	100 to 125
Blanket concentrate (S 35%, SiO_2 30%)	60 to 70

Flue-dust and floor sweepings in quantities to reduce the moisture to approximately 9 per cent.

The admixture of precipitate and fluxes in the car passes to a hydraulic elevator which delivers it at the feed-hopper of a two-mould Boyd press. This press was purchased from the Selby Smelting & Lead Co. and has a rated capacity of 30 tons in 24 hours. The dies have been changed to make a circular brick 4½ in. diam. and 3 in. thick, since the rectangular one was too large and unwieldy for use. Two men have no difficulty in fluxing and briquetting 2000 lb. of precipitate in 8 hours. One man feeds the press and the other receives the briquettes on rectangular trays, which are stacked in jacketed drying-carts and allowed to dry for 48 hours. Steam was used for drying during the winter when the heating plant was in operation. Having an excess of air, it was decided to heat it in one of the muffles of the drying-furnace, and allow the heated air to circulate through the trays of briquettes. This arrangement is very



MERRILL PRESS AND CLEAN-UP CAR.

satisfactory and is more economical than the use of steam jackets.

The coke-bin is situated on this floor of the refinery and the coke can be shoveled to the furnaces with no extra handling. The blast-furnaces are of the cylindrical type, 20 inches in diameter at the tuyere line with riveted steel jackets, and provided with a removable curb, mounted on wheels. The jackets are arranged to be suspended from 15-in. I-beams which pass through the concrete retaining-wall. A Connersville blower direct-connected to a 10-hp. motor supplies a blast of 3 oz. per square inch for both blast-furnaces and the double English cupelling-furnace. This blower has a capacity of 10 cu. ft. of air per revolution and is operated at present at 250 r.p.m. Arrangements are being made to reduce the speed, as the supply is in excess of the amount required. The gases from all furnaces pass to a dust-chamber which contains 2700 cu. ft. There are nine take-out hoppers inside the building through which the flue-dust is drawn after each melt. The product recovered from the flue never exceeds 400 lb., and contains less than $500 in gold from a total value in the **melt of**

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$400,000. In order to prove beyond question that gold was not escaping, an 8-in. Sturtevant exhaust fan was connected to the vertical stack and the gases filtered through muslin bags. The dust and fume collected from two separate tests of an hour's duration amounted in both cases to less than 5 lb., and assayed 1 oz. per ton.

The following analysis will give an idea of the baseness of the precipitate to be smelted:

	1911. Per cent.	1910. Per cent.	1909. Per cent.
Au	21.50	8.75	13.43
Ag	3.00	1.14	2.13
Cu	39.80	21.59	22.85
Pb	4.60	28.58	6.86
Zn	15.50	12.57	32.50
Cd	....	0.94	....
Fe	....	0.18	....
Mn	....	Trace	....
As	....	0.06	....
Sb	....	0.11	....
Bi	....	0.02	....
Ni	....	0.08	....
Co	....	Trace	....
Te	....	2.12	....
S	....	1.38	....
P	....	0.017	....
SiO_2	....	1.54	....
CO_2	....	2.74	....
Soluble alkaline salts	....	0.74	....
Moisture (105°C)	....	1.25	....
Combined water	....	3.19	....

The furnaces are 'blown-in' in the following manner: A wood fire is built in the crucible, the blast turned on, and wood thrown in from the charging floor until the crucible and lead-well are cherry red. A few charges of coke are added and the blast maintained until the whole charge is white-hot. The blast is then cut off and about 500 lb. of pig lead fed in to fill the crucible. The siphon is kept plugged with brasque in order to fill the crucible and float the ashes, charcoal, etc., which are raked out and subsequently returned to the furnace. When these have been removed the siphon is opened and blank charges of coke and slag are fed until the furnace is half full. This operation requires about 2 hours. By filling the furnace with these blank charges a bed is made for the first charge of briquettes, which prevents dusting. When the furnaces are ready, the following charge is fed:

	Lb.
Briquettes	160
Old slag	40
Borax	10
Cupel bottoms	10
Iron (oxidized)	5
Coke	25

With the exception of one run, when the slag contained 15% Zn, and became too pasty to work well, 24 hours is required to smelt 16,000 lb. of briquettes. The lead bullion contains approximately 20% gold and silver and 1% copper, which makes the lead-wells 'mushy.' In order to keep them open it is necessary to pass a hot iron rod through the siphon. The matte-fall is about one-third of the weight of the briquettes and is collected in a portable settler, from which the slag overflows into pots. The matte from the precipitate-run contains approximately 20% Pb, 50 oz. Au, and 200 oz. Ag per ton. This is stored until a sufficient quantity is on hand to make a separate run. The crucible of one furnace is then filled with brasque, and the furnace operated as a copper furnace. The separation of lead and matte is made in the portable settler. This operation reduces the lead in the matte to 10% and the gold to 3 oz. By roasting this product and leaching the copper and silver with H_2SO_4 it has been found possible to avoid shipping it, and a small plant will soon be erected for this purpose. When the slag becomes too 'zincy' for further use, it is either discarded or shipped to the smelter, according to the value.

The composition of the slag is as follows:

	Per cent.
Insoluble	24.8
FeO	26.3
CaO	5.5
ZnO	12.3
Pb	6.5
Cu	1.3

Soluble silicates and borates undetermined; gold, 1.2 oz. per ton.

Cupellation of the lead bullion is accomplished in a double English cupelling-furnace having removable tests, 3½ by 2½ ft. and 7 in. deep. The cupels are made of three parts cement and one part limestone; the latter is crushed to pass 6 mesh and the fine screened through 20 mesh and discarded. They are seasoned for six weeks before using and last for one run, cupelling approximately 4000 lb. lead bullion, after which they are broken up and fed to the blast-furnace with the next precipitate run. Thirty hours is required to cupel 8000 lb. lead bullion, during which time 25 barrels of oil is consumed. The litharge is caught in small ladles and examined for beads of bullion before it is sent to the grinding-room. Should any appear it is returned to the cupels. In finishing the cupellation it is necessary to add about 200 lb. of pig lead to each cupel in order to remove the remaining base. After the last matte and litharge have been floated off, the heat is raised and the blast increased for 20 min. to oxidize the film of base which cannot be removed mechanically. When oxidation is complete, the gold bullion is granulated by pouring into metal tubs filled with water. The material is then dried and melted with nitre and borax in a No. 60 Steele-Harvey tilting furnace. The bars thus produced, averaging 930 gold and silver, are shipped to the Selby Smelting & Lead Co. for further refining. Approximately 5% of the total lead used is lost. The matte and slag account for the greater part of this and the cupellation losses for the rest. The litharge recovered from the cupels is broken in a 4 by 6-in. Dodge crusher and pulverized to 20 mesh in a set of 7 by

CONSUMPTION OF CHEMICALS AND COST OF CONVERTING $KAuCN_2$ INTO FINE GOLD

	Per Ton Ore Milled			Per Base Oz. Bullion			Per Fine Oz. Gold		
	1909	1910	1911	1909	1910	1911	1909	1910	1911
				Fine { 354 Au / 61 Ag	378 Au / 39 Ag	850 Au / 80 Ag			
Consumption	Lb.	Lb.	Lb.				Lb.	Lb.	Lb.
KCN	1.60	2.61	3.12	0.508	0.77	1.90	1.4	2.0	2.31
Lime	8.72	8.49	8.55	2.70	2.50	5.20	7.61	6.5	6.5
Zinc dust	1.18	1.02	0.50	0.37	0.30	0.30	1.06	0.76	0.38½
Lead acetate	0.65	0.74½	0.58	0.26	0.225	0.37	0.73	0.57	0.44
Litharge			0.165			0.10			0.118
Pig lead			0.07			0.043			0.05
Lb. dry ppt. produced	1.00	0.92	0.52	0.313	0.254	0.32	0.876	0.70	0.38
Cost of	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
Precipitating	12.0	11.0	7.3	3.8	3.25	4.5	10.7	8.4	5.4
Melting	20.3	18.3	9.5	6.4	5.4	5.8	18.1	14.0	7.0
Shipping bullion	8.4	8.8	8.1	2.6	2.6	5.0	7.5	6.7	6.0
Mint charge for refining base bullion	11.8	11.03	6.7	3.7	3.25	4.1	10.6	8.4	4.9
Total cost ppt. melt. and marketing	52.5	49.13	31.6	16.5	14.50	19.4	46.9	37.5	23.3

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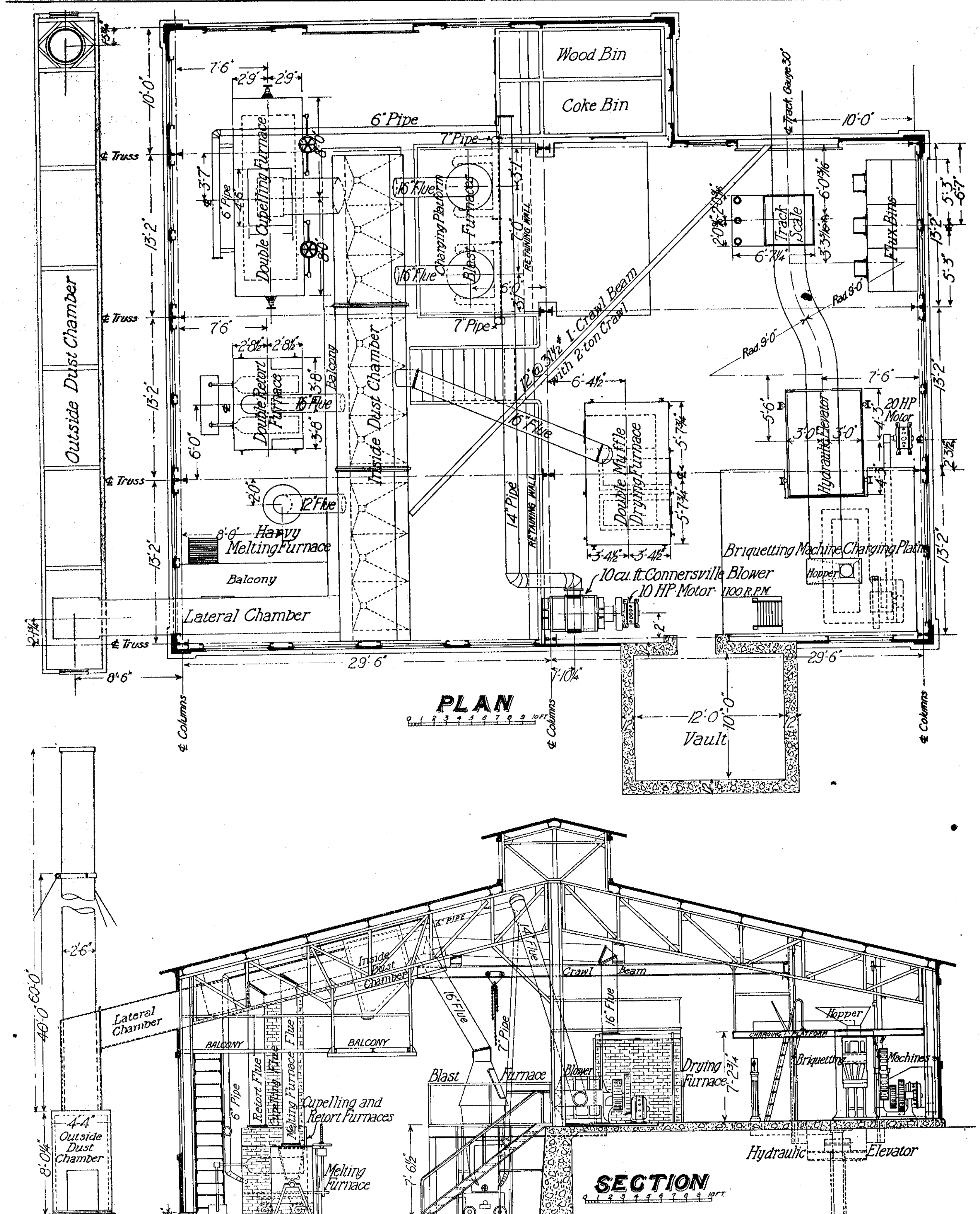

PLAN AND CROSS-SECTION OF REFINERY, GOLDFIELD CONSOLIDATED.

14-in. rolls and used for fluxing the precipitate from the next clean-up.

The cut above shows the refinery in plan and cross-section. In order to make it the more easily understood, the names of the more important features have been placed on the drawing, and it will therefore not be necessary to give any further description. The building is a steel frame covered with wire netting and portland cement plaster. Concrete was used liberally in constructing floors and foundations, and it is as nearly fireproof as such a building may be made.

It is not intended to give the impression that the process is perfect, and that the usual amount of 'grief' has not been passed through in getting started; but when the character of the product is taken into consideration, the results are very satisfactory, as can be seen from the accompanying tables:

COST OF MELTING

	Blast-Furnaces and Cupellation. 1911.	Acid Treatment and Tilting-Furnaces. 1910.	1909.
Labor	4.2	5.0	6.1
Supplies	5.2	13.2	14.0
Power	0.1	0.1	0.2
	9.5	18.3	20.3

(To be continued.)

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Operation of the Goldfield Consolidated Mill

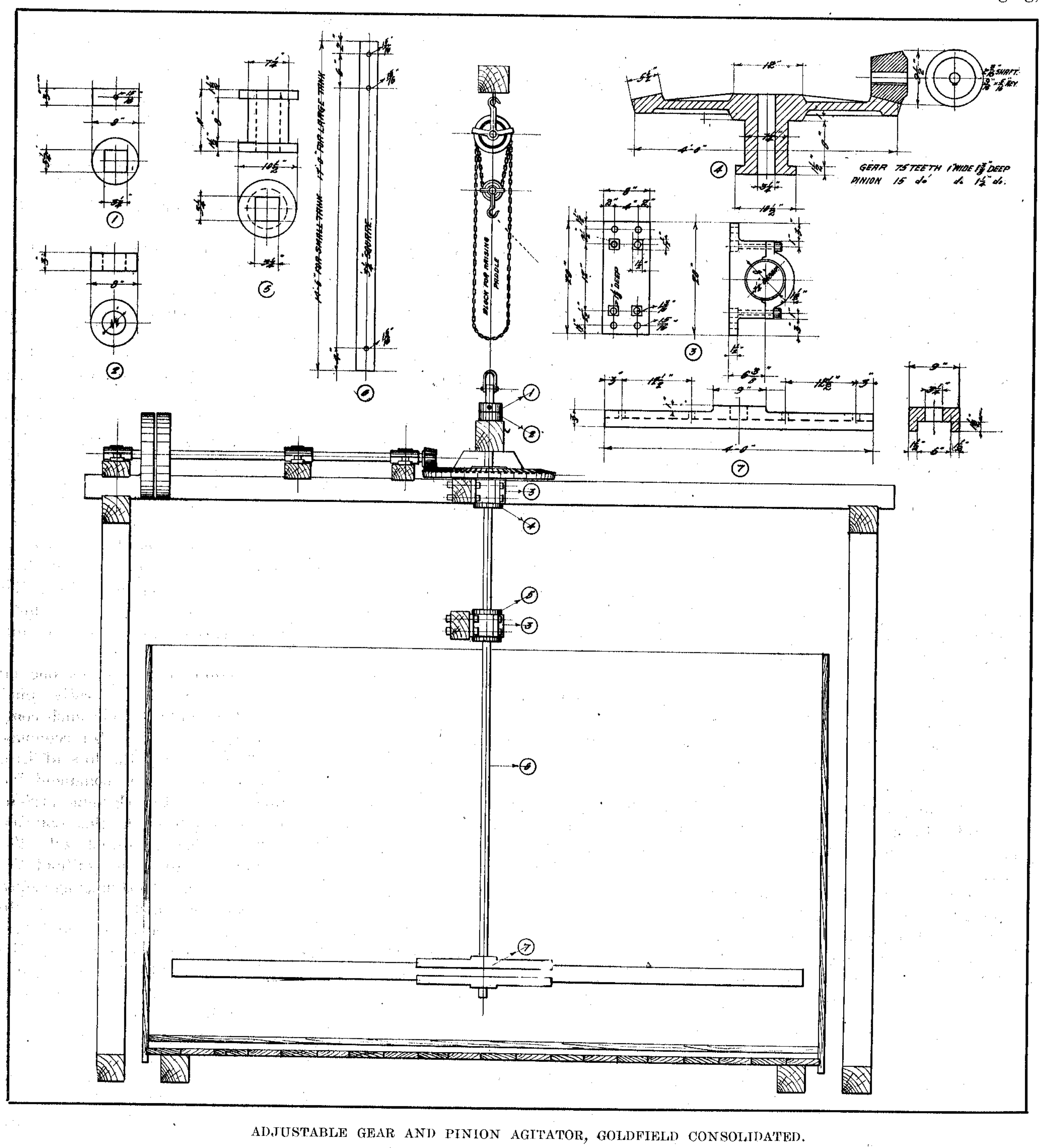
By J. W. Hutchinson

(Concluded from page 719.)

Concentrate Treatment.

Up to June 1908 very little progress had been made in evolving a process for treating the high-grade concentrate which would be produced at the mill. This product from the Combination mill of this company had been shipped to the smelters until J. H. Mackenzie assumed the management. The freight and treatment charges were so excessively high at that time that he decided to store the concentrate, pending the solution of the problem of treating it. The reports to the management by the former metallurgists of this company stated that roasting would be necessary for economical treatment, although the results of the tests made did not prove conclusively that the concentrate could be treated successfully in this way.

As can be seen from the flow-sheet on p. 686, concentration follows tube-milling. Roasting the –200-mesh concentrate, containing 30% sulphur and 20 oz. gold, to put it mildly, seems dangerous. In addition, roasting is not necessary. Much of the visible gold in the ore from this company's mines, although apparently free, is so coated that it cannot be recovered by amalgamation, and is only slowly soluble in cyanide solution. During the summer of 1908, when engaged in experimental work on the concentrate, an acid wash was applied for the purpose of removing this coating, in order to amalgamate the coarse gold if possible. It was found that this acid wash did remove the coating, and that much more gold could be recovered by amalgamating the concentrate after an acid wash had been applied. It seemed probable from this result that cyanidation of the raw concentrate would be facilitated by the same process, since it was not reasonable to suppose that only the coarser particles of gold were thus coated. Experiments were made on a laboratory scale and the results were astonishing. Approximately 90% of the gold in 20-oz. concentrate was dissolved in 8 hr. contact with a 4-lb. solution of cyanide, after a preliminary acid wash. The results were corroborated by larger tests, and were so encouraging,



ADJUSTABLE GEAR AND PINION AGITATOR, GOLDFIELD CONSOLIDATED.

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the management leased the old Kinkead mill for the purpose of treating the accumulation of concentrate from the Combination mill. During the summer of 1908 approximately 300 tons was treated and yielded 94% of the gold content, at a cost which made it seem advisable to build a plant of similar type at the 100-stamp mill. It was found during experimental work, that the solutions became inert after 8 hr. contact, and that it would be necessary to remove them and re-treat the concentrate with a solution freshly precipitated and freed of reducing agents. As originally designed and operated, the process of cyanide treatment after the acid wash consisted in agitating for 8-hr. periods in Pachuca vats, passing the pulp at the end of these periods through Dorr continuous thickeners; the clear overflow passing to the precipitating department and the thickened pulp to a second Pachuca vat, where a regenerated solution was added. This arrangement proved expensive, since it necessitated pumping the concentrate each time, and was abandoned. The process in use now consists in agitating the concentrate for 8-hr. periods, and settling and decanting in the Pachucas, and is very satisfactory. During the first year's run, much difficulty was experienced in evolving a mechanical stirrer for the pulp during the acid wash and the subsequent water washes. The sketch on the preceding page gives the details of the present arrangement, which is very successful.

The concentrate from the mill gravitates through wooden launders to four 48 in. by 16 ft. amalgamating tables. The last section of these tables is covered with carpet for removing the coarser particles of gold, which do not amalgamate. The recovery by the combined process of amalgamating and carpeting amounts to approximately 35% of the value of the concentrate, or 25% of the value of the ore. Of this amount, 10% is contained in the carpet concentrate, which is briquetted with the precipitate from the cyanide plant, and supplies the necessary sulphur for matting the copper in it. The pulp from the plates runs to three 10 by 20-ft. redwood collecting and agitating vats, fitted with the above-described adjustable stirrer. When a charge is being collected, the vertical shaft is pulled up by means of the chain-block, so that there is no difficulty in starting it up. Each vat holds one day's run, amounting at present to 50 tons of dry concentrate. After the charge has been collected, the water is decanted, leaving a pulp containing approximately 50% moisture. The agitator is started up and gradually lowered until the whole charge is in motion. To this is added 66°B. sulphuric acid in the proportion of 20 lb. per ton of concentrate, and agitation continued for 8 hours. The vat is then filled with water, after which the agitator is shut off and raised, and the charge allowed to settle. The clear solution is decanted and runs to a special tank in the mill, is neutralized there, and used in the mill for crushing purposes. Two more water washes are added in like manner, equivalent to 6 parts by weight of water per ton of dry concentrate. The last water wash is decanted as closely as possible to avoid subsequent dilution of the cyanide solution, lime added in quantities to raise the alkalinity to ½ lb. in terms of CaO, the agitator started, and when the charge is in motion, 1 lb. of lead acetate per ton of concentrate is dissolved in water and added to the pulp, which is kept in motion until time to pump to the Pachucas.

It has been found most necessary here not to let the concentrate be in contact for any length of time with a weak solution of cyanide. For this reason the amount of cyanide required to bring the solution on the charge to 4.5 lb. KCN per ton, is first dissolved in the Pachuca agitator and the charge pumped into this. Exactly what effect a dilute solution of cyanide has on a heavy sulphide ore is not known here, but that the effect is deleterious has been demonstrated beyond question by the following results. In order to avoid the small mechanical loss of cyanide due to neutralizing the concentrate with milk of lime, it was decided to slack the lime with solution from the mill. This

Rated H.P.	R.P.M.	Make	Series No.		Peak H.P.	Ave. H.P. per 24 hr. day	Ave. K.W. per 24 hr. day	Ave. K.W. hr. per 24 hr. day	Ave. K.W. hr. per 730 hr. month	K.W. hr. per ton Basis 20,000 Tons per Mo.	
				CRUSHING, SAMPLING, STAMPING AND REGRINDING.							
150	570	Bullock	42733	Crusher.	75.3	23.01	17.17	411.97	12530.81	0.6265	18.4989 K.W. hr. Per Ton for CRUSHING, SAMPLING, STAMPING AND REGRINDING.
75	680	do	40468	Elevator and Samplers.	45.6	16.40	12.23	293.62	8931.11	0.4466	
15	1130	do	32834	Distributor.	9.3	3.40	2.54	6087	1851.57	0.0926	
50	680	do	42805	Stamps.	242.5	231.44	172.65	4143.70	126037.57	6.3019	
50	680	do	42761								
50	680	do	42756								
50	680	do	42735								
50	680	do	42755								
200	570	do	50932	Tube Mills.	387.6	356.73	266.12	6386.89	194268.04	9.7134	
200	570	do	50887								
40	850	do	42783	Mill-Water Pumps.	66.0	48.40	36.11	866.55	26357.67	1.3179	
40	850	do	42823								
				CONCENTRATING.							
30	850	do	40556	Deister Slimers.	44.0	41.18	30.72	737.29	22425.82	1.1213	1.1213 K.W. hr. Per Ton for CONCENTRATING.
30	850	do	40555								
				CYANIDING.							
20	1130	do	44321	Stock Pulp Mechanical Agitators.	23.6	21.00	15.67	375.98	11436.18	0.5718	5.5631 K.W. hr. Per Ton for CYANIDING
10	850	do	44559	Excess Pulp do. do.	12.8	6.18	4.61	110.65	3365.52	0.1683	
30	850	do	40554	Pulp Pumps Excess to Stock Tanks.	35.0	10.02	7.47	179.40	5456.68	0.2728	
30	850	do	40701	Circulating Pumps Pachuca Agitators.	49.0	26.80	19.99	479.83	14594.74	0.7297	
30	850	do	44425	Circulating Pump Butters Filters.	15.6	6.90	5.15	123.54	3757.60	0.1879	
15	850	do	44131	Vacuum Pumps do. do.	15.0	9.34	6.97	167.22	5086.35	0.2543	
15	850	do	44203								
20	850	do	40576	Clarifying Presses Pump	24.5	22.01	16.42	394.07	11986.24	0.5993	
20	1130	do	44336	Circulating Pumps Wash Water.	28.5	19.43	14.49	347.88	10581.20	0.5291	
30	1130	do	44442								
15	850	do	44136	Pumps Gold Solution to Merrill Presses.	25.6	22.07	16.46	395.14	12018.87	0.6009	
15	850	do	44139								
7.5	1200	General Electric	104021	Lime Crusher.	2.2	2.00	1.49	35.81	1089.16	0.0544	
				Oil Heater.	14.8	5.00	3.73	89.52	2722.90	0.1361	
75	684	Westinghouse	504416	Air Compressors.	140.0	107.00	79.82	1915.73	58270.06	2.9135	
75	720	General Electric	108658								
15	850	Westinghouse	410418	Tramway.	12.0	0.13	0.10	2.33	70.81	0.0035	
				CONCENTRATE TREATMENT.							
50	850	Bullock	30419	Agitators.	30.0	18.20	13.58	325.85	9911.36	0.4956	2.3097 K.W. hr. Per Ton for CONCENTRATE TREATMENT.
20	1200	General Electric	174100	Kelly Press.	19.0	5.23	3.90	93.64	2848.17	0.1424	
20	1130	Bullock	44353	Pachucas.	34.5	7.83	5.84	140.19	4264.07	0.2132	
20	850	do	40605								
				REFINING.							
10	1130	Bullock	42074	Agitators, Johnson Presses, Slag Crusher.	5.5	2.80	2.09	50.13	1524.82	0.0762	0.1021 K.W. hr. Per Ton for REFINING
2	1800	General Electric	135530	Acid Wash Pump.	1.7	0.30	0.22	5.37	163.37	0.0082	
3	1800	do	126509	Fumes Collector.	1.3	0.65	0.49	11.64	353.98	0.0177	
				GENERAL.							
10	1130	Bullock	40593	Machine Shop.	12.0	1.20	0.90	21.48	653.50	0.0327	1.2940 K.W. hr. Per Ton for SHOPS & LIGHTING.
3	1800	General Electric	P.116180	Carpenter Shop.	4.0	0.30	0.22	5.37	163.37	0.0082	
				Lighting.	78.0	46.02	34.33	823.94	25061.56	1.2531	
1630.5	Total Installed Horse Power in Motors.				1454.9	1060.97	791.48	18995.61	577783.10	28.8891	

POWER ANALYSIS BY TONNAGE, GOLDFIELD CONSOLIDATED MILL.

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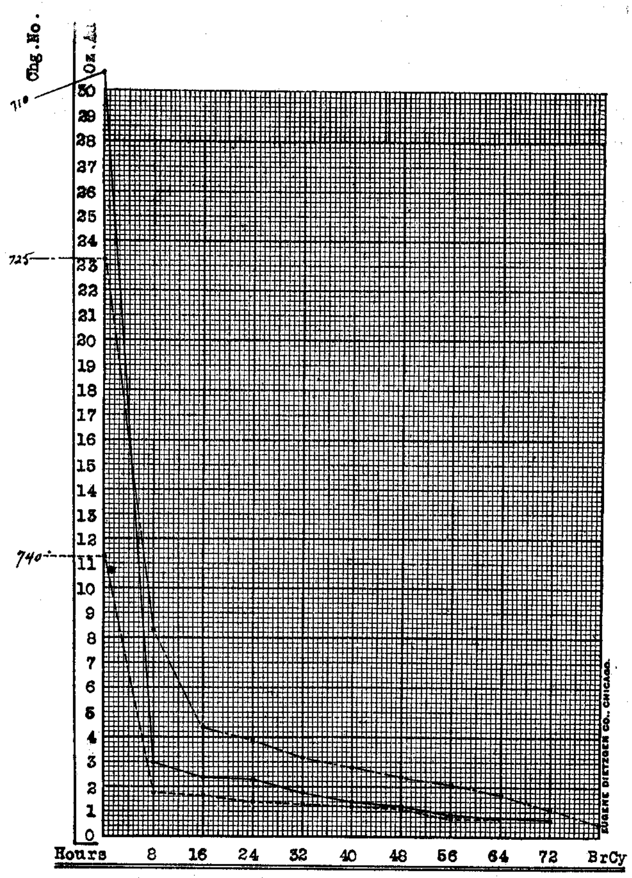
was done, and titration of the solution in the neutralizing tank showed a content of from 0.2 to 0.3 lb. KCN per ton. This practice was continued for nearly two weeks, with the result that the assay value in the tailing was doubled. Immediately on discontinuing the KCN solution in the neutralizing tank, the extraction came up to normal. This has been tried on different charges since, with the same result each time.

The same effects have been noted when treating a heavy silver sulphide ore which had been crushed in water, and then transferred to the strong solution in the agitators. When this same ore was crushed in the mill solution, and allowed to stand in the dewatering tanks in dilute cyanide solution, before being transferred to the agitators, the extraction was materially reduced and the strength of the solution used for agitating had to be increased in order to approximate the same results obtained by cyanidation after crushing in water. It is not to be understood that crushing silver sulphide in water is believed to be the most economical plan for low-grade ores, but the instance is cited to show that contact with a weak solution of cyanide is deleterious to the treatment of concentrate and ores containing large quantities of sulphides. At the end of the fourth hour, in alternate periods, peroxide of sodium is added to the charge, which increases the activity of the solution and reduces the time of treatment. One-tenth pound lead acetate per ton of dry concentrate is added to each solution at the beginning of the period.

The accompanying chart, showing the extraction graphically by periods, is representative of the various conditions



EXTRACTION CHART, GOLDFIELD CONSOLIDATED.

and grades of concentrate. For two years the extraction of gold from the pulp in the Pachuca agitators has averaged 93%, and the total extraction by amalgamating, carpeting, and cyaniding has averaged 95.23 per cent.

A Kelly filter-press (type B) containing 400 sq. ft. of filter surface is used for filtering the concentrate. It was found necessary to set this machine at a much greater inclination than is required for ordinary slime, since the heavy pulp has a tendency to pack in the bottom of the cylinder, which prevents the carriage from running out freely. The labor of one operator and two trammers is required to filter and dispose of 50 tons of concentrate in 8 hours. On this material the capacity of the Kelly press is equivalent to 750 lb. concentrate and 1200 lb. solution per square foot of filter per day of 24 hours.

The cost of cyaniding the concentrate is as follows:

Labor	$0.93
Supplies	4.44
Power	0.48
Total	$5.85

SUMMARY OF RECOVERY AND COST

	1911.	1910.	1909.
Recovery:	%	%	%
By amalgamation	17.55	15.38	10.60
By concentration	53.93	56.86	49.20
By cyanidation	22.56	22.03	32.80
Total	94.04	94.27	92.60
	1911.	1910.	1909.
Cost per ton milled:	%	%	%
Crushing-conveying	0.040	0.071	0.053
Sampling	....	0.003	0.021
Stamping	0.134	0.174	0.195
Elevating-separating	0.022	0.023	0.021
Chilean milling	0.097	0.095	....
Tube milling	0.177	0.187	0.206
Concentrating	0.057	0.059	0.062
Amalgamating	0.025	0.033	0.058
Neutralizing	0.045	0.046	0.046
Settling	0.053	0.055	0.055
Agitating	0.604	0.561	0.503
Experimental	0.102	....	....
Filtering-discharging	0.068	0.084	0.093
Assaying	0.046	0.045	0.063
Precipitating	0.074	0.110	0.120
Refining	0.098	0.183	0.203
Water service	0.098	0.112	0.110
Surface and plant	0.007	0.011	0.015
Steam heating	0.056	0.032	0.023
Watchmen	0.042	0.049	0.031
Storehouse and office	0.022	0.028	0.027
Stable	0.004	0.004	0.005
Lighting	0.021	0.018	0.019
Superintendence	0.062	0.067	0.082
General expense	0.012	0.012	0.009
Mill tools	0.002	0.003	0.005
Mechanical department	0.001	0.004	0.008
Electrical department	0.034	0.026	0.007
Return water service	0.010	....	....
Fire loss (machine shop)	....	0.026	....
Mill total	2.013	2.131	2.040
Concentrate plant total	0.381	0.312	0.276
Total, mill and conc. plant	2.394	2.433	2.316
*Mill operation	1.859	1.828	1.820
*Mill repairs	0.154	0.283	0.220
*Concentration plant operation	0.371	0.298	0.256
*Concentration plant repairs	0.010	0.014	0.020

*Included in the above but given for additional information.

On the preceding page is given a chart of the power-load analysis by tonnage which is sufficiently self-explanatory. This is the chart mentioned upon page 617, where the cost of the average power load, 1.73 hp. per ton milled, is given as 32c. per ton.

A RED AND WHITE SOLUTION for writing on blue-prints may be made by a crystal of potassium oxalate, about the size of a pea, in an ink-bottle full of water. This solution will give white lines on blue-prints; other potash solutions are yellowish. If this shows a tendency to run, owing to its too great strength, add more water and thicken slightly with mucilage. Mix this with red or any other colored ink, about half and half, and writing may be done on blue-prints in colors corresponding to the inks used.

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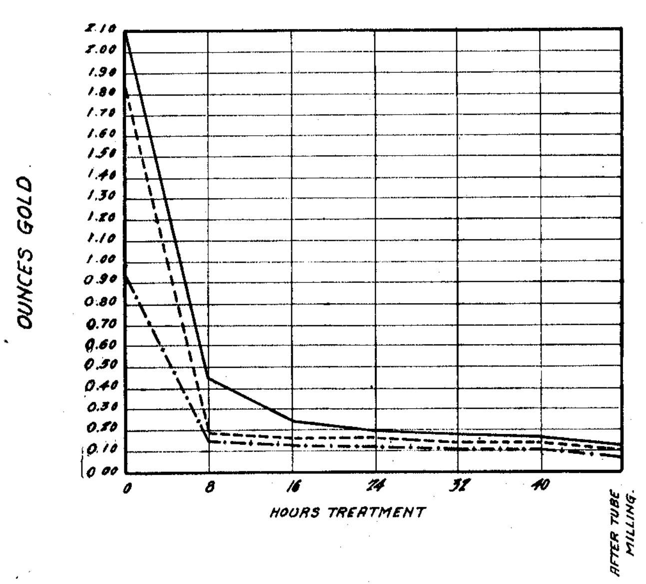
Treatment of Concentrate at the Goldfield Consolidated Mill

By J. W. Hutchinson

Since this article is intended to supplement the series which appeared in the *Mining and Scientific Press* during May and June of 1911, those interested are referred thereto for a description of the former process for treating concentrate.

It will be necessary to say here only that the acid wash, bromo-cyanide, and peroxide that were formerly applied to the raw concentrate have been discontinued. The concentrate, previous to roasting, is subjected to a preliminary treatment with alkaline cyanide for the purpose of reducing the value of the material to be roasted in order to minimize the dust loss. For instance, the loss in roasting has been determined to be approximately one-half of one per cent of the value of the material roasted. When the roasting plant was put in operation the raw concentrate entering the plant assayed 10 oz. in gold. The loss on concentrate of this grade would have been $1 per ton. The preliminary treatment given before roasting removed 85% of the value, leaving a product to be roasted valued at $30, and the loss in roasting this material was 15c. per ton. Further, it was demonstrated conclusively in the laboratory that neither a complete raw treatment nor treatment of the rich concentrate after roasting would yield so complete recovery as the combination of the processes, and that this combination treatment could be given as economically as either of the single treatments. For this reason it was decided to dispense with all the more expensive details of the raw process and to roast the tailing from this modified raw treatment.



The raw concentrate recovered from the 78 Deister concentrators (the 16 secondary tables are not now used), amounting to 6% of the weight and containing 67% of the value of the ore, is collected in flat-bottomed tanks, equipped with the adjustable square shaft agitator, which was fully described in the former articles; it is here neutralized with lime and by means of a centrifugal pump elevated to three Pachuca agitators, in which it is agitated during 8-hour periods in a two-pound solution of cyanide. Decantation at the end of the period is still practised and the charge is re-agitated with a freshly precipitated solution. Five periods of 8 hours each, followed by decantation, are sufficient to remove from 80 to 85% of the value of the concentrate. It is the intention to send to the roaster a product valued at $25 to $30, and this treatment is varied with the grade of the ore so as to accomplish this result. The pulp from the Pachucas, when dissolution is completed, is delivered to a storage tank from which it is pumped to the Kelly filter-press for filtration and drying. The latter is accomplished with air and the moisture is reduced to 12%. The consumption of cyanide during the raw treatment is 2½ lb. per ton of concentrate, and lead acetate is used in the proportion of 1 lb. per ton.

By means of a hoist and scraper, approximately 20 tons per day of the accumulation of concentrate on the dump is conveyed to the Kelly press bin and mixed with the concentrate filtered. A 14-in. conveyor, set at an angle of 17°, receives the concentrate from this bin, passing it over a Blake-Dennison automatic weighing machine en route to the bins in the roaster plant. The concentrate from this conveyor is distributed by means of a swinging bucket elevator to two bins having 45° sloping bottoms and 1620-cu. ft. capacity, from which it is fed by means of two 12-in. screw-conveyors, making three-quarters of a revolution per minute, to two slow-moving belts. These belts discharge the concentrate through the arches of the furnace between the first two rabbles. The original feeding arrangement was a 12-in. screw-conveyor, 23 ft. long from the ore-bin to the furnace. The strain caused by the packing of the concentrate made this machine impractical. The screw was cut down to 10 ft. in length and used for feeding only, and the concentrate is now conveyed from this feed-screw to the furnace by the belt above mentioned.

Roasting is accomplished in two Edwards (54 spindle) duplex furnaces, 112 ft. long by 13 ft. inside, with an effective hearth area of 1456 sq. ft. each, and each furnace is capable of roasting 40 tons per day of concentrate, of which the following is a typical analysis, the average sulphur content being 18.76 per cent:

Mesh.	Weight, per cent.	Sulphur, per cent.	Total S, per cent.
100	9.5	5.34	2.70
150	8.5	11.74	5.27
200	22.5	18.59	22.28
—200	59.0	22.37	70.30

The hearth area required per ton of concentrate roasted per day is 36.40 sq. ft., equivalent to 55 lb. of concentrate per square foot per day. The slope of the hearth is ¼ in. per foot. The space between the furnace side-walls was filled with waste from excavations to within one foot of the hearth-line. This material is decomposed surface rock full of clay. It was wetted down and tamped thoroughly and cov-

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ered with one foot of screened sand to form the hearth, in order to reduce the breakage of revolving parts. Since all details of construction can be obtained from the accompanying figures,* there is no necessity for repetition. There are two rows of 27 rabbles; 25 of these revolve at 2.25 r.p.m. and the last two on the finish at 4.5. This speed was decided on after testing the furnaces at speeds varying from 1.6 r.p.m. to the speed now used. Lower speeds did not reduce the sulphur in the discharged product, decreased the capacity, and in no way affected the dusting. In addition to being a most satisfactory furnace to operate, the Edwards has the distinct advantage of producing a minimum of dust. The fear of serious loss from this feature of roasting caused the delay in adopting the two-stage treatment. That it was groundless has been proved by operation. Only 1½% of the material roasted passes out of the furnace as dust, and only ½% is lost. When the accompanying analyses quoted and that below are considered, the performance seems remarkable:

Mineral.	Percentage.
SiO_2	51.60
Fe	19.90
S	18.93
Al_2O_3	2.00
CaO	0.20
MgO	0.10
Sb	0.08
Te and Se	0.46
Cu	0.50

One man per shift operates the entire plant, including feeding, firing, oiling, and attending cooler and elevator. The bins are filled on the day shift, and have sufficient capacity to run for 24 hours. Each furnace receives power from a 10-hp. motor, and including the feed-screw and belt, which are driven from the furnace shaft, requires 4½ hp.

At the start all the fire-boxes were used for several weeks. The first to be discontinued were the middle boxes. For several months thereafter fuel was burned in the front boxes and on the finishing hearth. As stated above, the moisture in the material fed the furnace averages 12%. Approximately 6 sq. ft. of hearth area per day ton is required to remove this moisture, and the concentrate is not thoroughly dried until the fourth rabble is reached.

An attempt was made to discontinue the fire in the front box, with the result that the moisture traveled farther down the furnace and a decided decrease in capacity was caused. It was then decided to dispense with the fires on the finishing hearth and to burn all the oil required in the front boxes. This is the present practice and has resulted in reducing the fuel consumption approximately 45%. Nine gallons of crude oil is required per ton of concentrate, equivalent to 3.3% of the weight. The sulphur begins to oxidize at the seventh rabble and is burning freely at the tenth. Doubtless because of the extremely fine state of division of the sulphide, after once becoming ignited, roasting is carried to satisfactory completion without additional fire at the finishing end. This, of course, is contrary to the usual practice when roasting previous to cyanidation, since it is customary to raise the temperature at this point, but it has been demonstrated conclusively here to be the most economical practice for this material. Were roasting ahead of chlorination being done, the scheme would not be feasible, since a dead roast would be required and the soluble sulphur would be objectionable. However, at this plant the treatment required by the roasted material prior to cyanidation, removes the sulphate and obviates the necessity of a complete roast. It is no doubt true that the fineness of the pyrite makes this practice possible and that coarser concentrate would require different treatment in the roaster. E. D. Peters in his 'Principles of Copper Smelting' makes the following observation:

> Theoretically, the smaller each particle of sulphide, the more rapid and thorough will be its oxidation. It takes an appreciable time for the oxidation process to penetrate into the centre of even a very small particle of the sulphide mineral. It begins its work upon the surface, and the more surface it is allowed to operate upon, the more oxidation there will be in a given space of time.* * * Consequently, if there were no disadvantages arising from the employment of very finely pulverized ore, it is evident that there would be no limit to the fineness at which it would be advantageous to have our ore for roasting; so that, if it were as fine as soot or rouge powder, it would roast instantaneously, * * * but in practice there are difficulties which more than offset the advantages arising from the rapid and thorough oxidation of this very fine ore. Among the more obvious of these difficulties are: the great expense of pulverizing ore to this fineness; the enormous production of flue-dust in the roasting furnaces from the fine particles carried away by the draft; the tendency of the sulphide particles to melt from the heat arising from instantaneous oxidation; the fact that such excessively fine ore lies solid in the roasting furnace, offering no interstices for the penetration of air into the deeper layers.

As can be seen from the screen analysis, the product fed the furnaces has been pulverized so that 80% passes a 150-mesh screen and this 80% contains 92½% of the total sulphur. Obviously, the problem here has been to avoid excessive loss from dusting and the melting of particles. In the matter of admitting air to the hearth, the practice here varies from the usual in that all side ports are kept closed, except six near the finish. In this way, most of the air entering the furnaces is preheated by passing over the practically roasted ore before it reaches that port of the furnace where heat is required, and thus saves fuel by not excessively reducing the temperature at this point. With a given degree of comminution the factors governing the oxidation of pyrite are, time, temperature, and oxygen. The first is inversely proportional to the last two. For instance: Oxidation at the surface of the earth is accomplished in long periods of time at low temperature and an abundance of oxygen; the same chemical result can be obtained in an assay muffle in 60 minutes with high temperature, while in practice the result is obtained in a number of hours. Since the combustion of sulphur produces dead atmosphere, and since combustion ceases entirely when the amount of SO_2 in the surrounding air reaches 12%, it follows that oxidation proceeds more rapidly in pure air than in air diluted with reducing gases. However, since a large excess of air or draft increases fuel consumption and stack losses, the economic limit of any furnace is reached

*See part II.

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when the decrease in cost of labor and power resulting from increased capacity is more than offset by the increased cost of fuel and increased stack losses. This point has been demonstrated here by repeated tests. There is no difficulty in increasing the capacity of the furnaces by burning more fuel and increasing the draft, but there is no resultant economy from such increase on this extremely fine material. Since the element of time is not only essential, but an economical factor in the commercial oxidation of pyrite, it seems that where the tonnage to be treated is sufficient to justify the increased capital outlay, greater economy can be effected by allowing greater hearth area per day-ton of concentrate roasted. The material roasted here is so out of the ordinary and requires such unusual treatment, it has been thought advisable to go quite into detail in order to explain the theory on which the practice is based.

The accompanying chart of temperatures and sulphur elimination will show the progress of the concentrate through the furnaces. All temperatures were taken with a Brown electric pyrometer, using a platinum-rhodium couple, and for the sake of uniformity the readings were taken one foot above the hearth-line. The concentrate loses 17% of its weight in roasting.

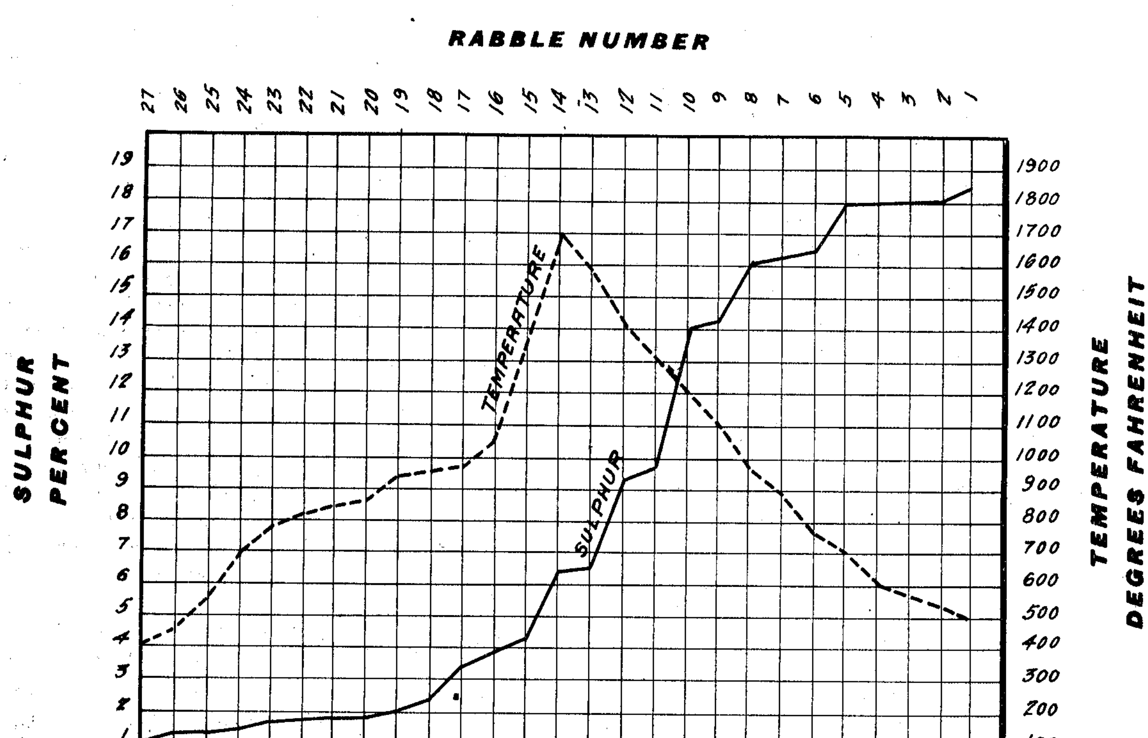

CHART SHOWING SULPHUR ELIMINATION AND TEMPERATURES AT THE RABBLES

——SULPHUR—— ----TEMPERATURE----

(To be Continued)

Silver miners of the world must consider the currency of the Orient a matter of paramount importance. No little space has been devoted to discussions of the recent tendency of India to hoard gold in the same manner as it was formerly accustomed to hoard silver. A large demand for silver has been anticipated in the awakening and reorganization of China. It is, therefore, important to silver-producing properties that it is proposed to put China on a gold-standard basis. G. Vissering, of the Bank of the Netherlands, Amsterdam, Holland, has attracted much attention by his suggestions for a reorganization of Chinese finances and the establishing of a 'gold exchange' standard similar to the system which Mr. Vissering put into force in the Dutch East Indies. China's needs for currency reform are imperative, and it is quite conceivable that in order to conform to the prevailing standard of the civilized world a gold basis will be established. But in any case, it is likely that silver will indefinitely continue in use by so conservative a people, since the Chinese distrust bank-notes unless the solvency of the banks is unimpeachable. Silver producers have during the current year taken much hope from the better outlook. A price of 63½c. ruling during the past three or four months, as compared with 54c. a year ago, has been a matter of no little encouragement; encouragement which would have resulted in considerably increased output had it not been for the unsettled conditions in Mexico during the past year and a half; conditions which preclude the enlisting of any new capital for mining enterprises of any kind in Mexico, whether gold or silver. As a result, the large low-grade silver ore-bodies, with higher prices to make developments possible, have had to see a better silver market without any progress being made in mining. In the meantime the gold output of the world shows every sign of decreasing. Only the increase on the Rand prevented a fall in output last year.

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Treatment of Concentrate at the Goldfield Consolidated Mill

By J. W. Hutchinson

*By means of iron goose-neck flues, the gases from the roasters at a temperature of 450°F. are delivered to a concrete dust-flue 264 ft. long, having a cross-section of 50 sq. ft. From this flue, 20,700 cu. ft. of gases per minute escape through a vertical steel stack, 100 ft. high and 54 in. diam., having a temperature at the base of the stack of 325°F. The velocity of the gases in the dust-flue has been determined with a Hiram anemometer to be 7½ ft. per second. The escaping gases are white, and no dust is visible to the eye. Measured quantities filtered through woolen bags indicate a stock loss of less than ½%. This figure is taken as a total loss in handling, roasting, conveying, etc., and added to the tailing loss. One per cent of the material roasted is collected in the dust-flue and has a value of 20% more than the material roasted. Apparently, all the dust is made in the feed-end of the furnace, since this dust contains 15% sulphur. Approximately 80% of the dust recovered settles in the first 150 ft. of the flue. The flue is built on the slope of a hill and at the top it is 79 ft. above the hearth. As can be seen from the drawing, a drag-chain conveys this dust down the flue, from which it is discharged through wrought-iron gates into a sump. Water is added and the resultant pulp elevated back to the Kelly press storage tank for filtration and mixing with the regular feed. The loss and inconvenience through handling this fine material dry is thus avoided.

Analysis of Roasted Product

Mesh.	Per cent.
+100	12.5
+150	15.5
+200	25.5
—200	46.0
Composition.	
SiO_2	54.60
Fe_2O_3	32.20
S (as sulphide)	0.15
S (as sulphate)	0.75
Al_2O_3	3.00
CaO	0.20
MgO	0.13
Cu	0.60
Te and Se	0.19
Sb	0.07

In line with the last rabble on each furnace there is a cast-iron discharge chute on one side of the furnace only, set at an angle of 45°, through which the roasted material is delivered to a drag-chain conveyor. This conveyor runs at a speed of 45 ft. per minute and elevates the material up an incline of 15° to the feed spout of a Baker cooler. As can be seen from the drawing, the original device for conveying the hot concentrate was a water cooled screw-conveyor. This was decidedly unsatisfactory and was discarded in favor of the drag-chain. This chain does good work, requires no repairs, and is very satisfactory. The drag-chain, sprockets, and all movable parts are housed in a brick conveyor-way built up 2 ft. above the floor line and covered with ¼-in. plate, and no dust escapes into the building.

The Baker cooler is a sheet iron cylinder, 5 ft. diam. and 22 ft. long, carried at each end on a hollow trunnion bearing, through which the ore is fed and discharged. It revolves with about 40° submergence in a water-tight concrete sump, to which the cooling water is added. Cooling is accomplished through the evaporation of water on the surface of the shell. The cooled ore is discharged dry at about 100°F. into a small concrete sump, over which there is a tight cover, where water is added to wash it to the boot of a 14-in. belt and bucket elevator with 12-ft. centres. Much difficulty was experienced in elevating the material after wetting, due to the soluble copper compounds in the roasted product. Cast-iron centrifugal pumps would not last for this reason. Phosphor bronze pumps were too soft to resist the abrasive action of the pulp. A temporary belt and bucket elevator with steel parts lasted a couple of weeks. Finally, by using brass pulleys, shafting, cups, bolts, and a canvas belt, the elevators have been made fairly satisfactory. The small elevator in the roaster building delivers the pulp to the boot of a second elevator of similar construction placed in the treatment plant, which delivers it to three 20 by 12-ft. combined collecting and agitating-tanks, fitted with the adjustable square shaft agitator. The sole plates, holding bolts for these agitators, are made of brass, and the shaft is protected from the corrosive action of the washes by means of a lead covering. The tank connections in these tanks are brass.

The roasted concentrate is delivered to one tank for 24 hours. The collected charge is then settled and decanted to a consistence of 1 to 1 and sulphuric acid added in the proportion of 20 lb. per ton of concentrate. Agitation with the sulphuric acid is continued for eight hours. Water is then added to fill the tank and the charge allowed to settle. When clear, the wash is decanted and the tank re-filled with fresh water. Four water washes are given, equivalent to eight tons of wash water per ton of concentrate. All washes are passed through two redwood tanks filled with excelsior for clarifying, and overflow from these tanks to six redwood tanks, 10 ft. diam. and 5 ft. high, arranged in series for recovering the copper. These tanks are kept filled with cyanide tins and all kinds of scrap from the mill. The average copper content of the washes is 0.4 lb. per ton, and 70% is recovered.

The thoroughly washed charge is neutralized with lime, and by means of centrifugal pumps elevated to one of four Pachuca agitators, 14 ft. diam. by 25 ft. 6 in. high. Experimental work showed that only 10% of the gold of the roasted concentrate could be recovered by amalgamation, but that no increase in final extraction could be obtained by this step, and for this reason amalgamation was not deemed necessary. The only explanation known for this peculiar

*Continued from p. 170.

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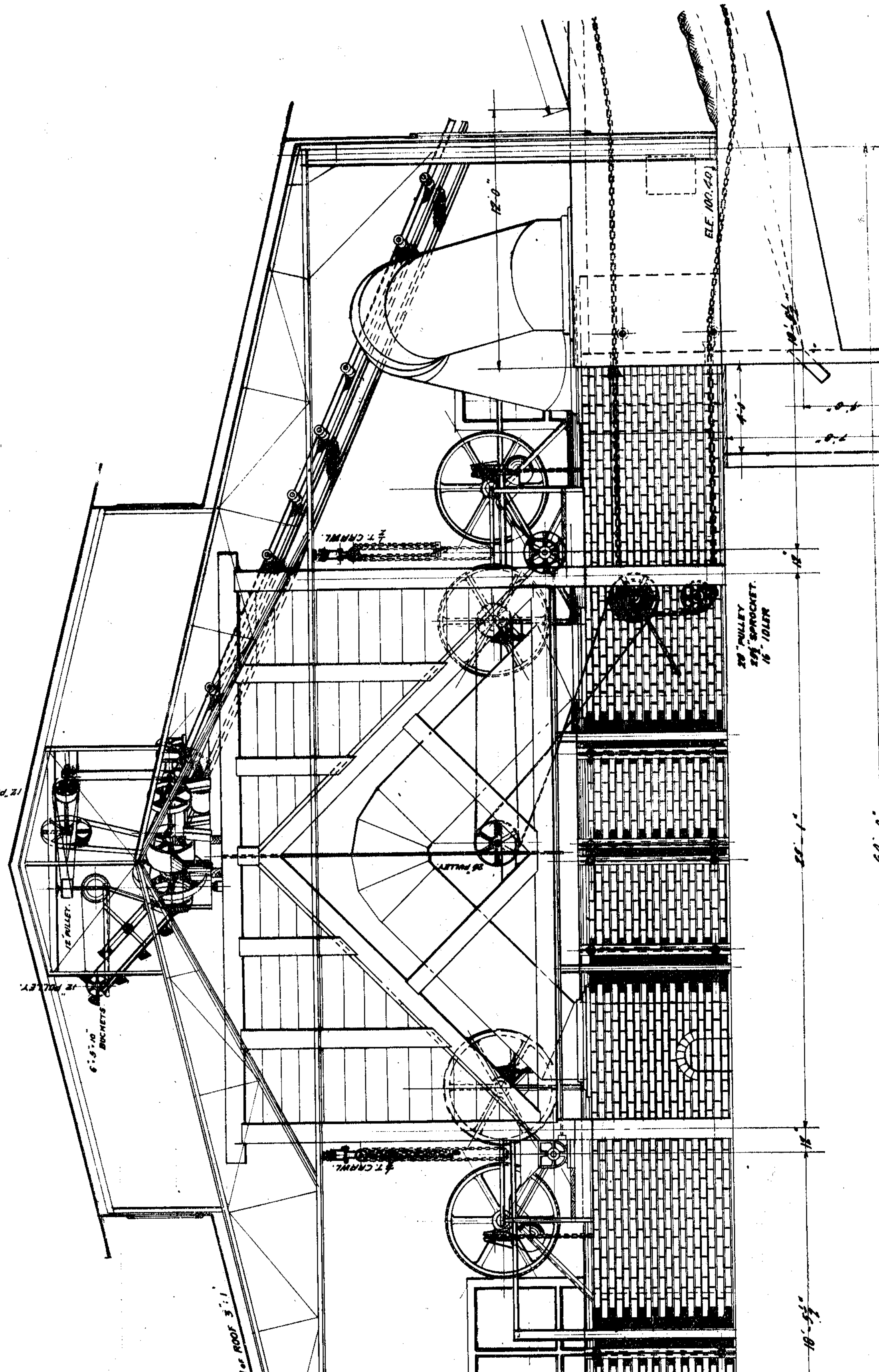

REAR ELEVATION.

CROSS-SECTION OF FURNACE SHOWING DISCHARGE END.

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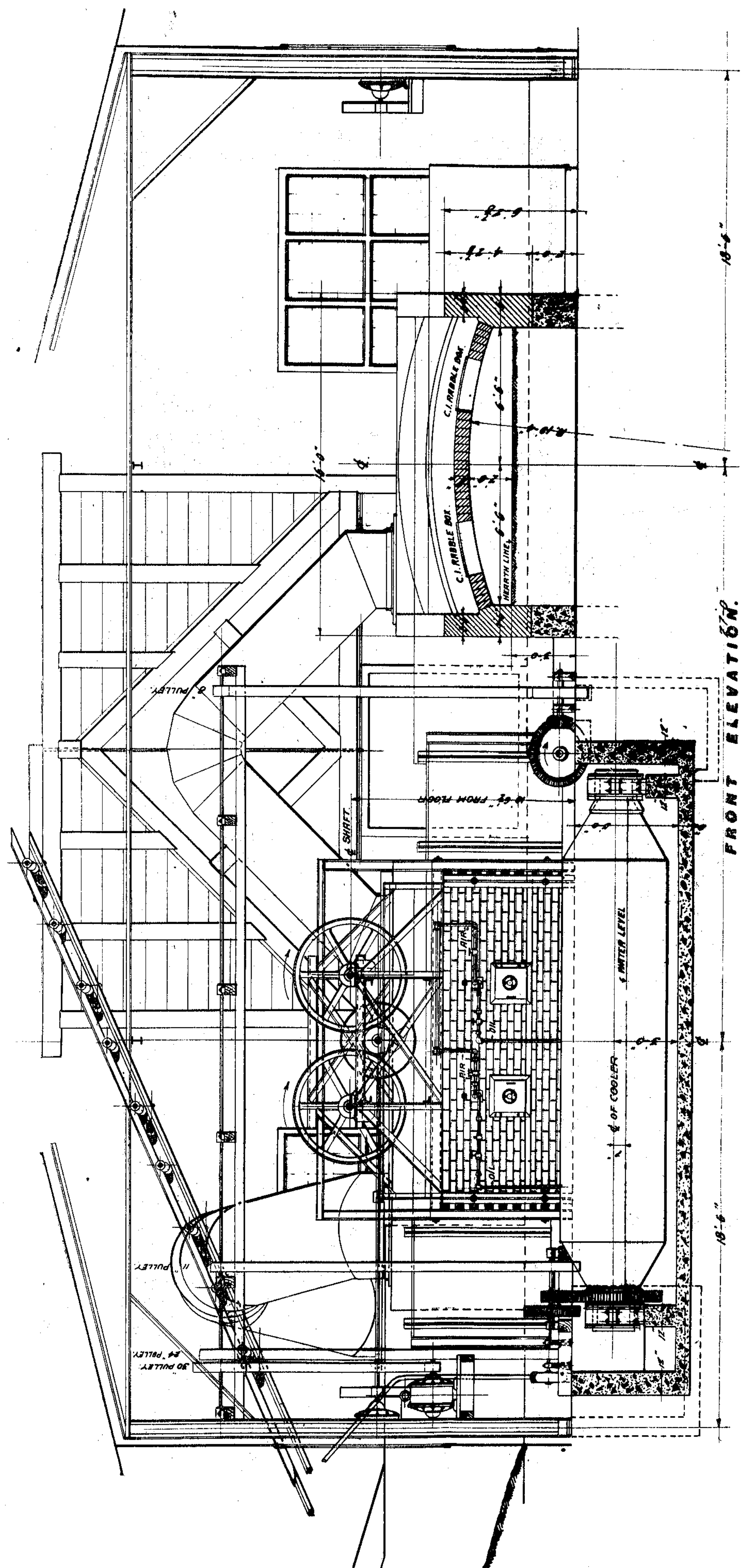

CROSS-SECTION OF FURNACE SHOWING FIRE-BOX.

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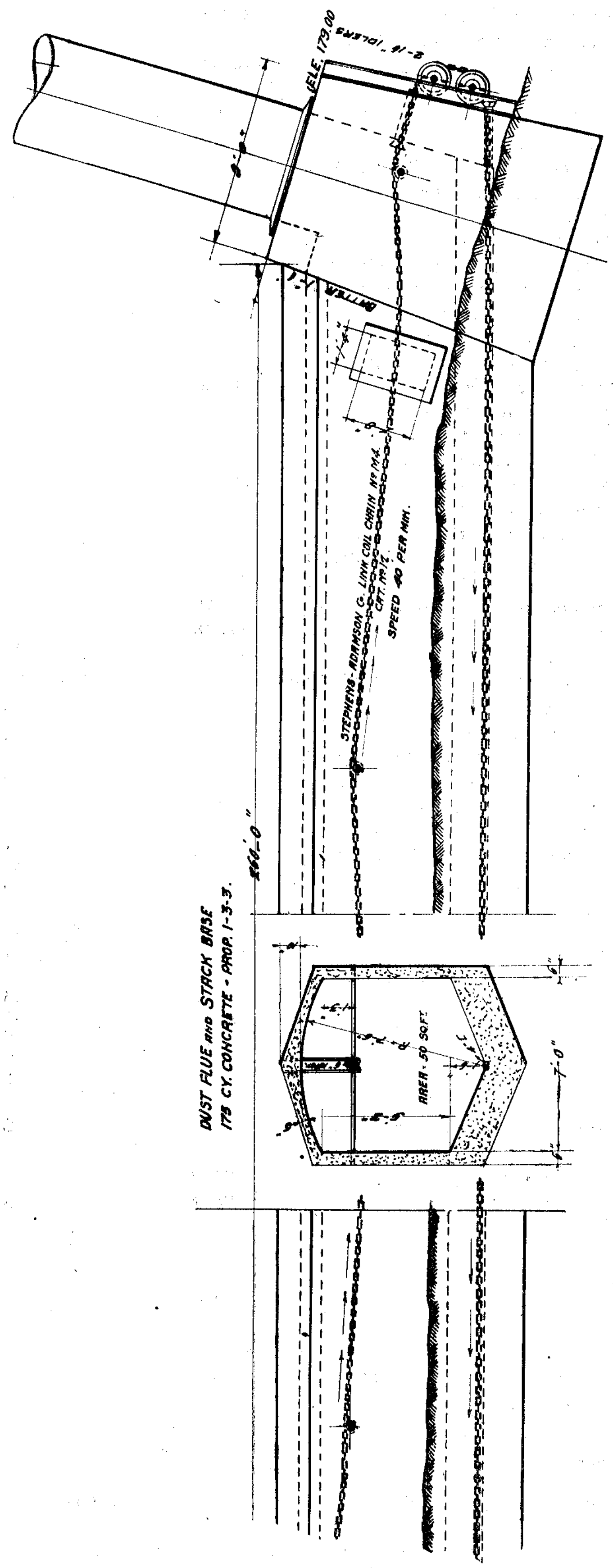

LONGITUDINAL AND CROSS-SECTION OF DISCHARGE DEVICE.

action of the gold is that the antimony, bismuth, tellurium, and selenium (though present in minute quantities only) with which the gold is so intimately associated, are not completely volatilized in the furnace, and form a film of their oxides on the surface of the gold particles. The same explanation will account for the failure of the rich roasted concentrate to yield so high a percentage of its gold to cyanidation, as can be recovered from the two-stage treatment. The following comparison is interesting:

	Au, oz.	
Assay before roasting	12.25	
Assay after roasting	15.35	
Assay after 64 hr. treatment	0.66	
Percentage recovered		95.7
Assay of raw concentrate	11.60	
Assay after 48 hr. treatment	0.96	
Percentage recovered		91.7
Assay before roasting	0.96	
Assay after roasting	1.17	
Assay after 32 hr. treatment	0.107	
Percentage recovered		92.6
Percentage of original gold recovered by raw treatment		91.7
Percentage of original gold recovered after roasting		7.6
Total percentage recovered		99.3

It is a fact proved by repeated experiments and by working tests that no more gold can be amalgamated from a 10-oz. roasted concentrate than can be amalgamated from the same material in its raw state, and that roasting the rich concentrate does not materially expedite the dissolution of the gold. It seems probable that some of the gold is partly encased in the sulphides of the volatile metals, but presents some surface to which the cyanide solution has access. After roasting, it is probable that the gold may be entirely covered with a film of the oxide of such metals, which must be removed before dissolution can proceed. Since these metals do not interact with alkaline cyanide, their removal is doubtless accomplished slowly by the alkali of the solution, and this may account for the prolonged treatment which the roasted material requires. This theory is offered merely as an explanation and the ideas of others would be appreciated.

In the above mentioned Pachuca tanks the roasted charge is agitated for eight hours in a 2-lb. solution of cyanide, containing 1.2 lb. CaO as protective alkali. At the end of eight hours, agitation is discontinued, the charge settled, decanted, and re-agitated with a freshly precipitated solution in the same manner as described above in the treatment of the raw concentrate. Five periods of agitation followed by decantation are given, and a total of three tons of solution per ton of concentrate is decanted. Consumption of chemicals amounts to 4½ lb. cyanide and 2 lb. lead acetate per ton of concentrate. After agitation is completed, the settled charge is delivered to a storage tank 18 ft. diam. by 8 ft. high, fitted with the adjustable square-shaft agitator. Placed centrally in the bottom of this tank is a 4-ft. cone with pipe connections through which the thickened pulp is fed to a 5 by 18-ft. tube-mill. The pulp issuing from the tube-mill is elevated by means of a belt and bucket elevator back to the above mentioned storage tank. This circulation grinding is continued for 16 hours, at the end of which time 95% of the material will pass a 200-mesh screen. From 80c. to $1.25 per ton is removed in this circuit. Regrinding before agitation with cyanide was the original plan, but had to be discontinued on account of the inability to settle and decant the finely pulverized ore. Since the change of solution increases extraction and since the final tailing is sent to the mill proper for filtration, it was decided to re-grind after the greater part of the gold had been removed.

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After re-grinding, the pulp is delivered by means of a centrifugal pump to the filter storage tank in the mill proper, mixed with the mill pulp, filtered, and sent to waste.

Since precipitation and refining are accomplished in the manner described in the former article, it is not necessary to give the details of these operations here. Extraction and costs are shown below:

COST PER TON

Labor		$1.02
Power		0.78
Cyanide	$1.37	
Zinc	0.06	
Lime	0.25	
Lead acetate	0.16	
Water	0.70	
Belting	0.02	
Lubrication	0.01	
Borax	0.01	
Litharge	0.02	
Pig lead	0.01	
Pebbles	0.04	
Tube-mill lining	0.00	
Filter cloth	0.03	
Assaying	0.13	
Acid	0.48	
Fuel oil	0.40	
Roaster parts	0.01	
Bromo-cyanide	0.00	
Sodium peroxide	0.00	
General stores	0.18	
Total supplies		3.88
Total costs		$5.68
First agitation		$0.73
Filtering and conveying		0.30
Roasting		0.82
Acid wash		0.77
Tube-milling		0.33
Second agitation		1.61
Assaying		0.15
Precipitation		0.10
Refining		0.08
Disposal of residue		0.03
General expense		0.02
Re-handling dump		0.04
Water		0.70
Total		$5.68*

*This may be divided into: operation, $5.32; repairs $0.36.

EXTRACTION FOR YEAR

	Au, oz.	Per cent.
Value of raw concentrate	6.58	
Value after treatment	1.23	
Recovery		81.3
Value of tailing after roasting and treating	0.097*	
Recovery		92.16
Total recovered from roasted material		17.23
Recovered from both treatments		98.53

*Based on weight before roasting.

COST OF ROASTER PLANT

Contract for:		
2 Edwards duplex 54-spindle roasting furnaces, 112 by 13 ft. inside, erected on side walls furnished by the Goldfield Con. M. & T. Co., which side walls extend to within one foot of hearth line	$33,485.17	
1 Baker cooler, 5 by 22 ft., erected on foundations furnished by the Goldfield Con. M. & T. Co., complete with driving mechanism, etc.	3,400.00	
Excavations for foundations	1,897.00	
Concrete in foundations, 218 cu. yd. at $14.25	3,106.50	
Total cost of furnaces and cooler erected		$41,888.67
Dust-flues and stack:		
Dust-flue of concrete, 264 ft. long, 6 ft. high, 7 ft. wide; walls 7 in. thick, bottom 10 in. thick with 24° slope to centre; roof 8 in. thick, arc of 10 ft. circle; concrete reinforced with ¼-in. rods, spaced 1 ft. apart both horizontally and vertically, and with a steel cable laced through these rods; total cost	$ 5,500.00	
Stack of American ingot iron, 100 ft. high by 54 ft. in diam., weight 10,620 lb., invoice $573.60, freight $519.60; total cost	1,093.20	
2 goosenecks, 60-in. diam. and 42-in. diam., weight 4480 lb., invoice, $416.00, freight $248.49; total cost	664.49	
Supplies used in erection	375.20	
Labor	311.48	
Total cost of dust-flues and stack		$ 7,944.37
Steel building:		
65 by 158 ft., containing 10,270 sq. ft., at $7398.23 erected in Goldfield	$ 7,398.23	
Foundations	1,213.87	
Covering of 'asbestos protected metal,' weight 1¾ lb. per sq. ft.	3,538.84	
Labor for laying asbestos covering	700.00	
45 windows in place at $10 each	450.00	
		$13,300.94
Ore-bins:		
2 wooden ore-bins, 1620 cu. ft. cap.:		
Supplies	$ 721.26	
Incidentals	55.93	
Labor, erection	311.70	
Total cost of ore-bins		$ 1,088.89
Transmission, elevating, and conveying machinery:		
Line shafting, etc.	$ 2,069.87	
Labor, erection	1,858.55	
Total cost		$ 3,928.42
Miscellaneous expense:		
Labor	$ 530.50	
Supplies	1,086.81	
Total miscellaneous expense		$ 1,617.31
Electrical work:		
Labor	$ 163.00	
Supplies	527.56	
Total electrical work		$ 690.56
Total roasting plant proper		$70,459.16

COST OF TREATMENT PLANT

Superstructure:		
Wooden building, 44 by 48 ft., including lumber, concrete, and covering	$ 3,109.13	
Labor (erection), excavating, moving dumps, etc.	2,447.48	
Total cost of superstructure		$ 5,556.61
Pachuca agitation:		
2 steel Pachuca agitators, 14 by 25½ ft., weight 32,860 lb.:		
Invoice	$ 1,225.00	

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Freight	384.46	
Labor (erection)	1,316.65	
Power (erection)	200.00	
Labor (foundations)	477.70	
Supplies (foundations)	283.87	
Other materials	533.61	
Total costs		$ 4,421.29
Redwood collector and agitation tanks:		
2 redwood tanks, 20 ft. diameter by 12 ft., weight 36,000 lb.:		
Invoice	$ 402.00	
Freight	265.00	
Labor (erection)	505.30	
Stirrers and other supplies	1,084.11	
Labor (foundations)	496.40	
Supplies (foundations)	197.27	
Total		$ 2,950.08
Tube-bills:		
5 by 18-ft. Gates tube-mill, weight 25,287 lb.:		
Invoice (including chain drive)	$ 1,910.00	
Freight	650.73	
Labor (erection)	297.55	
Supplies (erection)	114.13	
Labor (foundation)	298.45	
Supplies (foundation)	258.94	
Labor (lining)	111.35	
Supplies (lining)	301.64	
Total		$ 3,942.79
Miscellaneous:		
Labor	$ 530.50	
Supplies	1,086.81	
Total		$ 1,617.31
Total cost of treatment plant		$18,488.08
Total cost of roasting plant		70,459.16
Total cost of treatment and roasting plants		$88,947.24

Suspension Colloids

At a recent meeting of the Faraday Society, in London, Ruisdale Ellis read a paper entitled 'A Neutral Oil Emulsion as a Model of a Suspension Colloid.' The interface potential at the surface of the oil globules of an emulsion was measured by means of a mircoscope slide apparatus. The interface potential was found to be little affected by organic impurities in the oil, but to be altered enormously by acids, and to a lesser degree by alkalies. The maximum interface potential was found to correspond to a concentration of about 0.001 N alkali, and this was found to be the point of maximum stability of the emulsion. Surface tension measurements showed that the stability did not depend on the surface tension, but on the interface potential.

Determinations were made of the concentrations of salts with mono-, di-, and tri-valent kations required to just neutralize the charge on the oil globules. The ratio of these concentrations was found to form a geometrical series agreeing with that obtained by coagulation experiments. The coagulation of oil emulsions by colloidal ferric hydroxide was next tried, and it was found that the oil was completely precipitated within two well defined limits on either side of the iso-electric point of the oil emulsion. The coagulation appears to be due to surface precipitation effects taking place between the oil globules somewhat analogous to the condensation of water-vapor on surfaces of various curvatures.

In the subsequent discussion E. Hatschek stated that the size of the oil globules in Mr. Ellis's experiments was about 2 η, which was of course very much larger than anything in suspension colloids. It is difficult to realize how the electrical 'double-layer' would prevent collision between the globules; the theory is useful mathematically but very obscure physically. Whetham's theory with regard to the ratio of concentrations of salts with mono-, di-, and tri-valent kations required to neutralize the charge on the oil globules is untenable, as, for one thing, it is based on the assumption that the particles are stationary.

A. W. Porter controverted a number of the remarks of the previous speaker. Whetham's theory merely depends on relative motion taking place. The coming together of two globules with 'double layers' would have the effect of mutually disturbing the perfect uniformity of their electric charges, and then the particles would repel one another. F. G. Donnan, in a written communication which was read to the meeting, pointed out that no distinction could be made between suspensions and emulsions if based merely on the liquid or solid state of the disperse medium. The oil emulsions behaved as typical suspensions, forming a very striking model of a suspension colloid. Bredig's 'Lippmann Effect' theory of the stability of suspensions was not borne out by Mr. Ellis's results. Further work on emulsions is being carried out in the Muspratt laboratory.

The great interest of the subject in connection with the theory of oil flotation processes now largely in use for ore separation was remarked by H. M. Ridge. G. Senter referred to some recent work of Freundlich in which geometrical ratios in the coagulating effects of mono-, di-, and tri-valent kations were not observed. He agreed that the author's work did away with any real distinction between suspension and emulsion colloids. E. Feilmann said it had long been known that slightly alkaline media favored the formation of stable suspensions, but he gave instances where slightly acid media had the same effect. He criticized the author's suggestion that dyeing might be due to capillary coagulation of the colloidal dye.

Ore of the Braden mine, in Chile, consists of chalcopyrite, bornite, secondary chalcocite, cuprite, metallic copper, and carbonates and silicates of copper. It had proved difficult to treat by ordinary wet concentration methods, the recovery being only about 60%. Tests by the Minerals Separation flotation process resulted as follows:

	Tons treated.	Cu, %.	CuS, %.
Heads	2726.2	2.61	....
Concentrate	274.1	20.82	....
Tailing	2452.1	0.56	....
Calculated recovery		80.00	87.40
Actual recovery		80.20	87.65

Consumption of reagents was: Sulphuric acid, 3.57 lb. per ton; Texas and wood-tar oils, 2.33 lb. per ton.

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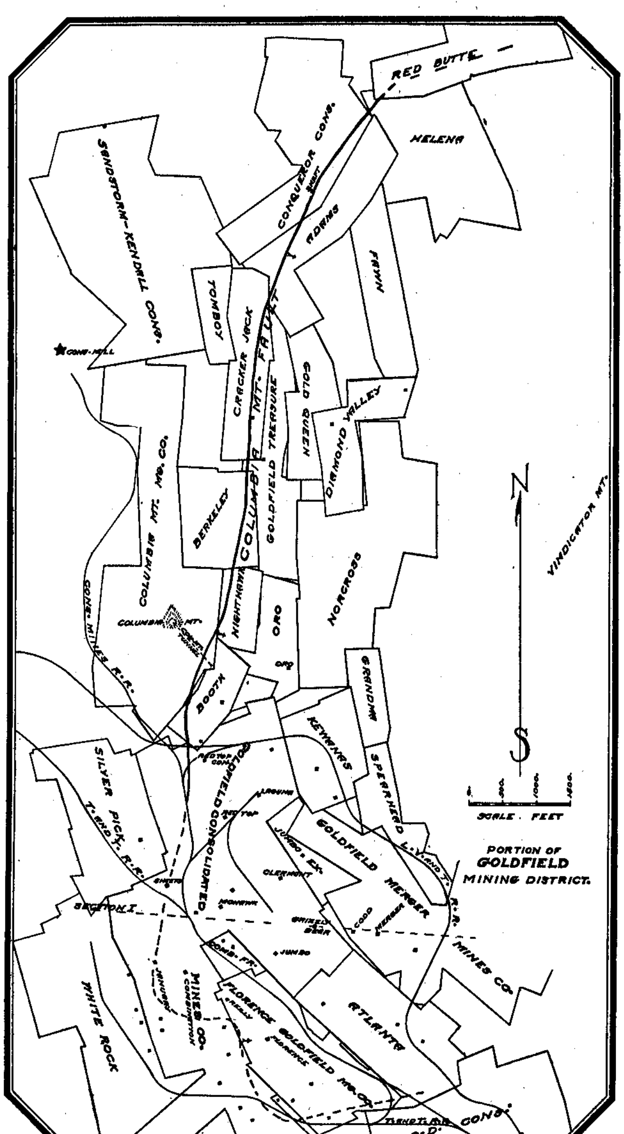
Relation of Faulting and Mineralization in Goldfield

By Corrin Barnes and E. A. Byler

That portion of the Goldfield mining district which has so far proved the most productive, includes the Goldfield Consolidated and Florence mines, and it is here intended to point out the relation existing between the Columbia Mountain fault and this mineralization.

System of Fissuring

F. L. Ransome, in his report on the Goldfield district, suggested a relation between this fault and the ore deposits of this portion of the district, but while the extent and character of the fault were well demonstrated, there was not at that time sufficient development in the underground workings to determine its relation to the ore deposits. Later developments have furnished information from which this relation may be deduced, and the general system of mineralization outlined.



GOLDFIELD DISTRICT.

This fault, as shown on the plan map, is traceable along the surface from its northern end near the Conqueror mine shaft, south a distance of some two miles to the vicinity of the Red Top, where it disappears beneath the dacite. It may be followed underground from here fairly close to its southern end, by means of the relative positions of the different formations, as disclosed in the underground workings. Its approximate position on the surface south of the Red Top is indicated on the map by a dotted line.

Between the January and the Florence it makes a sharp bend to the east, and then again to the south to its southern extremity, where it ends at an east-west fault. At its northern extremity the manner of its ending has not been disclosed, but it appears here also to end at an east-west fault which probably extends to Diamondfield. The dip of the fault is to the east, all that part north from the January having a regular dip, averaging about 28 degrees. From the Florence south the dip is much steeper to the east, the amount of which has not as yet been determined, but is probably about 60 degrees.

Mr. Ransome has concluded that the main movement of the Columbia Mountain fault was prior to the dacite intrusion, and that it occurred in the andesite and in the underlying rocks. Then came the intrusion of the dacite through the latite, to the fault and across it, and into the latite-andesite contact, lifting up the andesite which has in places since been eroded leaving the dacite at the surface. This conclusion has been disputed by some, but there seems abundance of evidence to support it, and Ransome's idea is now generally accepted.

Situation of Fault Zone

Section I, as shown in the illustration, is through the Mohawk, Grizzly Bear, and Merger Mines Co. shafts. This section furnishes data for a close location of the fault zone on its dip, and work now in progress on the Grizzly Bear and the shaft of the Merger Mines Co. will still further demonstrate it. The fault zone, shown in this section on its dip, is approximately the sloping fault-contacts of dacite with latite, and latite with shale, and at deeper workings probably the fault-contact of shale with alaskite. The downward throw on the east side of the fault has not been sufficient in amount to allow the approximately horizontal layers of latite and shale to entirely pass their corresponding layers on the west side; therefore, the fault zone in places passes through latite upon both sides, and similarly through shale. In consequence of the dacite intrusion across this fault along a portion of its length, which practically obliterated it to a depth of some three hundred feet, it follows that the relative position of the geological formations caused by the fault do not properly show above or at this intrusion, though they do below it.

After the dacite intrusion, there has been a further small movement along this fault zone, and where the intruded dacite lies across the upward extension of the zone, no pre-existing fracture is present; and the movement, while following in general the extension, has been recorded in irregular

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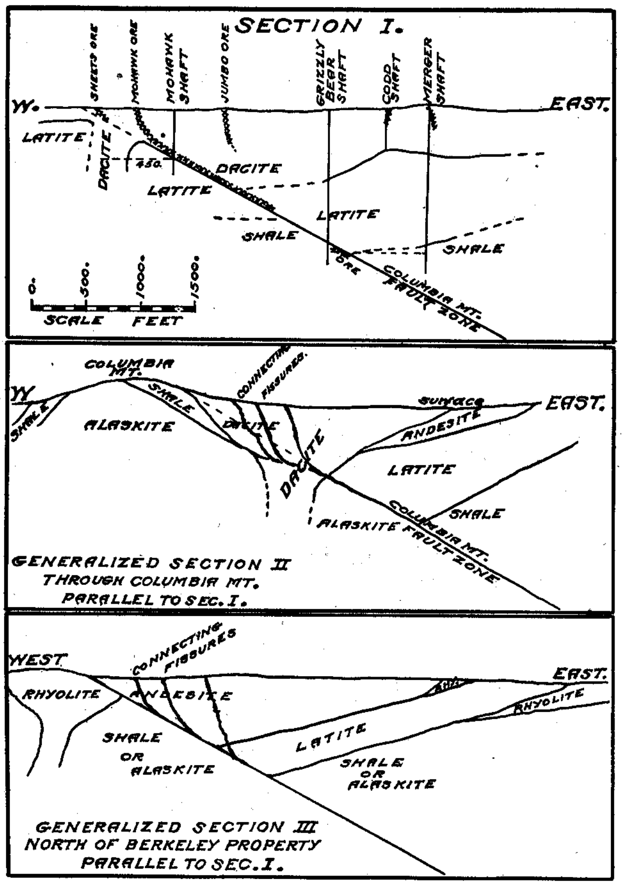
fractures extending upward to the surface, having a steeper dip and connecting below with the fault as shown in section I. The mass of eruptives and solidified dacite intrusion has settled irregularly, due to the additional movement of the fault in its original position below, and we consequently find many irregular fractures extending from this fault zone to the surface. These fractures connect below and finally with depth merge with the original fault zone; the foot-wall of which in effect forms the foot-wall of the mineralization.



SECTIONS THROUGH FAULT ZONE.

South of the January, where the fault makes a sharp bend to the east, there seems a probability of the existence of a cross-fault which would account for the difference in dip of the fault north and south of this bend. There is some evidence that this follows the east-west contact between the dacite and andesite. There is, however, no conclusive proof of the existence of this cross-fault, as there are insufficient workings penetrating the latite, in which formation would be recorded the entire movement, while in the dacite above would only be recorded the later small movement subsequent to the dacite intrusion, which may have been distributed in such a way as to make its existence not apparent.

As shown in section I, a cross-cut west from the Mohawk shaft on the 450-ft. level penetrates a body of dacite, the position of which indicates that it is a dike, and while there are not workings enough at proper depth to the north to prove it all, the evidence supports the idea that this is a dike of dacite with a north-south direction with apparently greater width north of the Laguna, which is probably the source of a portion of the dacite intrusion. If this dike is as suggested, and wider to the north of the Laguna, it will be expected that the fault in breaking its way upward through this dacite will show greater irregularity than would otherwise be expected. Generalized section II illustrates this situation and practically represents the conditions existing as far north as the end of the dacite area, in the vicinity of the Berkeley property.

Mineralization

The various orebodies of the Goldfield Consolidated and the Florence mines are directly connected with this Columbia Mountain fault zone; all of them being either in the zone itself or in the irregular fractures extending from it upward. It seems reasonable, therefore, to conclude that this fault has been the connecting channel to the deep seated source of mineralization, and the path for mineralizing solutions.

What is believed to be primary ore has already beeen found on the 1300-ft. level of the Grizzly Bear in this fault zone. The natural tendency, in a fault as flat as this, is a closing of the fracture by compression, and the consequent difficulty of the circulating waters to maintain their open channels. This accounts for the larger size of the orebodies in the upper and more nearly vertical and irregular connecting fissures than in the main channel below. It is reasonable to expect that the deposition of mineral in the channel itself as depth is increased will be of less thickness, while in the steeper intersecting veins it will not be so much restricted.

Relation of Mineralization to Faults

The mineralization appears to depend entirely upon the system of fissuring, and there appears to be no essential relation between the ore deposits and the different kinds of eruptive rock in which they exist, and references to the different kinds of rock are made here for the purpose of exhibiting the system of faulting and fissuring. There has been some cross-faulting both prior to and after the deposition of the ore, which has not been mentioned here, and has to some extent modified the uniformity of the fault and connecting fissures, but has not substantially altered the general scheme.

To the north of the Berkeley property the conditions are somewhat simplified by the absence of the dacite intrusion. This situation is represented in generalized section III. Some ore has been found in the fault zone at the Conqueror mine, which leads to the inference that there are connections along the zone to the source of the mineral solutions, through which they have penetrated this far to the north. There may be a branch from this fault or a connecting fissure extending farther to the west, which has mineralized the area including the Sandstorm and Kendall properties. This system of connected faults and branching fractures is by no means unusual and many notable similar instances are found in mining camps throughout the country. It is not intended here to convey the idea that there are not other sources of mineralization for other portions of this district, or that there may not be many more connecting mineralized fractures extending from this system, which as yet remain undiscovered and which the future will reveal.

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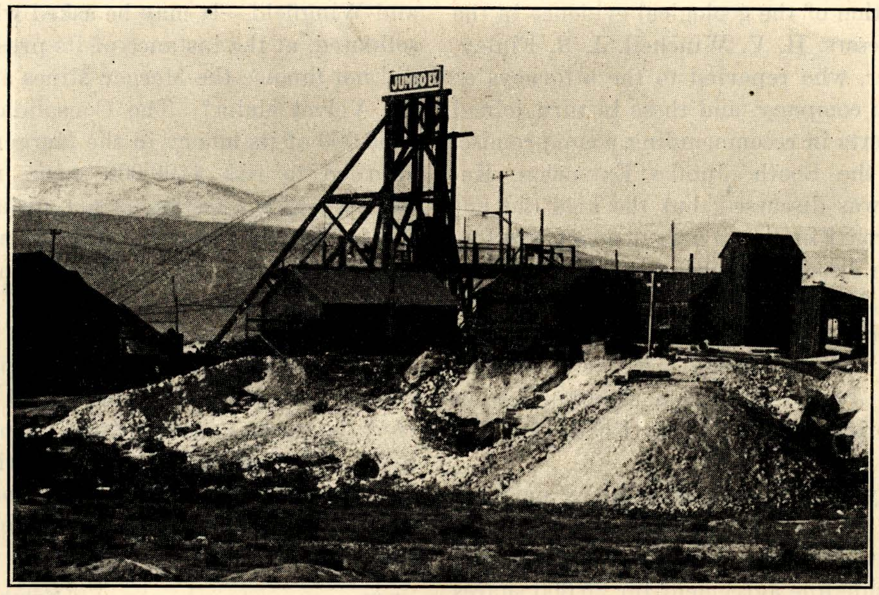
Goldfield Re-visited—I

By T. A. RICKARD

Seven years ago I made my first visit to Goldfield; it was warm then, warm with sunshine and prosperity. When I went there the other day the snow had been falling over-night, it was still falling, and the temperature was low. The air was cold and I met many people with cold feet. Goldfield has abated its exuberance of spirit and it has shrunk in size, so that it looks like a fat man who has taken the cure. The clothes of its prosperity are too ample for its severer shape. Empty houses are pitifully conspicuous. The population in 1908 was 22,000; now it is 4500. Last year it was fully 1000 less; prospects are improving; that is why it was pleasant to go to Goldfield at this time.



THE JUMBO EXTENSION MINE, GOLDFIELD.

Jumbo Extension is the main subject of talk: rich ore—and litigation. Apparently a mine must pass through the whooping cough of a lawsuit before it reaches the adult stage of development. I saw the rich ore and heard all about the disputes concerning the ownership of it. Naturally, I did not hear all that has been said on the subject, but enough to outline the story to my readers.

The Jumbo Extension property consists of two claims, the Polverde and the Velvet. Of these two the Velvet was acquired on October 5, 1914, from the Merger Mines company, owning adjoining property. The Merger Mines had failed to find any ore despite exploratory work begun in 1911 and reaching to 1772 ft., the maximum depth attained in the district. In order to assist further exploration in its Velvet claim, the Merger Mines leased the Jumbo Extension shaft in December 1912. Some low-grade ore was found, but nothing particularly encouraging. To obtain further funds, the Merger Mines then gave an option on its Velvet claim to the Jumbo Extension company, which exercised the option, as stated, in October last. This transaction might have caused no comment if immediately thereafter the Jumbo Extension company had not uncovered rich ore in the lode already cut in the Velvet claim. Shares that had been peddled for 12 to 15 cents during the major part of 1914 rose to $4 on November 30. They have not been anything like so high since, although a great deal more ore has been uncovered; but that is another story. No sooner had the discovery been heralded than the minority shareholding in the Merger Mines brought suit against the directors of that company, alleging collusion and fraud in the sale of the Velvet claim to the Jumbo Extension company. On February 21, 1915, a receiver was appointed, pending a suit to annul the deal with the Jumbo Extension company. Three days later the Reorganized Booth Mining Company, owning the Booth claims, filed an apex suit against the Jumbo Extension and the Merger Mines.

To explain this, let me refer you to the accompanying map of the territory in dispute. It will be seen that the outcrop of the rich lode traversing the Goldfield Consolidated property is shown as if it made a curve northward so as to pass through both end-lines of the Booth claim and appears to give the owner of that claim the apex rights on all ground within the end-lines projected forward in the direction of the dip of the vein. However, lodes are rarely as continuous in strike or as simple in structure as they are portrayed on maps. It is a moot question whether the vein in the Booth is an uninterrupted extension of the vein in the Red Top claim of the Goldfield Consolidated. Good evidence exists for suggesting that a fault intervenes. The vein has a flat dip, ranging from 35 to 50° east, and averaging about 40°. Geologically the Merger Mines at a vertical depth of 1750 ft. is at the same horizon as the Booth at 500 ft., although the distance between them is 3000 ft. as measured on the surface. Hence the legal ownership of ore found in the Jumbo Extension, Kewanas, and Merger Mines is threatened by the Booth apex. Before this recent entanglement supervened the Goldfield Consolidated company had entered into a mutual side-line agreement with the Jumbo Extension and the Merger companies, that is, each had waived any legal right to

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extract ore beyond their side-lines vertically projected. They agreed mutually to waive any extra-lateral rights. That agreement was renewed after the Velvet was added to the Jumbo Extension property. The Booth gave a side-line waiver to the Goldfield Consolidated as late as April 24. Now comes the humor of the position. Mr. George Wingfield is the president of the Goldfield Consolidated; he is the controlling shareholder also in the Merger Mines, and he is the owner of the Booth. He is the Pooh-Bah of the comedy. As controller of the Merger he permits the sale of the Velvet claim by the Merger to the Jumbo Extension; then, having placed the proceeds from this sale into the empty treasury of the Merger company, he and his friends are attacked by the minority shareholding in the Merger; whereupon he springs the Booth apex-suit upon them and upon the Jumbo Extension company. Both the Merger Mines and the Jumbo Extension were placed in the hands of a receiver. An examination of the geological evidence in the case was made by Messrs. H. V. Winchell, J. R. Finlay, and Frank W. Royer, who reported to the attorneys of the Jumbo Extension company, and these in turn joined with the mining experts in recommending a compromise. A consolidation of the Booth, Jumbo Extension, Kewanas, and Merger was discussed, but the idea did not find sufficient support. Finally an arrangement was formulated whereby the Jumbo Extension is to increase its capital stock from 1,250,000 shares to 1,750,000, and of this amount is to pay the Booth 300,000 shares for a side-line agreement. This proposal is to be submitted to a meeting of the Jumbo Extension shareholders on May 20, while the formal meeting of the Booth is scheduled for May 31.

Meanwhile, on May 5, the Kewanas company paid the Booth—that is, Mr. Wingfield—250,000 shares for immunity from apex demands, and the Kewanas Extension was given the same side-line agreement for 25,000 shares of its capital stock. On May 6 the receivership in the Merger case was made permanent and announcement was made before the United States Court at Seattle that suit would be brought against the majority directors for unauthorized transactions, notably the sale of the Velvet claim to the Jumbo Extension company. The assertion is made that "the policy of all the business of the Merger company has been planned and carried out by the officers and persons who dictated and directed the policy and business of the Goldfield Consolidated company and allied concerns, and not by the trustees of the Merger company." On May 11 the Court authorized the bringing of a suit by the receiver for the recovery of the Velvet claim and for the value of the ore extracted by the Jumbo Extension company. From what I could learn at Goldfield, the deal between the Merger and the Jumbo Extension was a straight transaction. The Merger needed money badly, the Velvet claim apparently had no special value, and the sale would have escaped criticism if rich ore had not been found, subsequently, in the Velvet. The only complication arises from the fact that Mr. H. O. Whittemore, a Merger director, had caused Mr. E. S. Van Dyke, a former manager of the Jumbo Extension and the person who steered the deal, to give him 10,000 shares of Jumbo Extension out of a commission of 21,000 shares received by Mr. Van Dyke from the directors of the Merger. Even this need have excited no suspicion if a rich orebody had not been discovered so soon afterward in the Velvet workings. While the original complainant was a lady shareholder, Mrs. Lucy V. S. Ames, the chief opponent of Mr. Wingfield is Mr. John Erikson, of Seattle. He is a Swede who made about $200,000 at Nome, in Alaska. When he spent his money in buying tide-lands near Seattle he was supposed to have been victimized, but the extension of the Chicago, Milwaukee & Puget Sound Railway caused the land to become exceedingly valuable as an entrance into Seattle, so Mr. Erickson made a big fortune by selling it. Part of the friction at Goldfield is imputed to an old quarrel between Messrs. Erickson and Wingfield. It may be asked why the Goldfield Consolidated, at the instance of its president, Mr. Wingfield, did not finance the Merger Mines and so save the loss of the Velvet claim? The Consolidated had already put $250,000 of its money in the Merger and could well have afforded to risk $100,000 more, rather than lose the Velvet. This question is not unfair, having regard to the further fact that the Consolidated has seen fit to go outside the district, to Aurora, in order to extend its mining activities.

If the reader will now turn to the map he will see that the Florence appears to apex the Atlanta and Merger properties. Mr. Wingfield controls the Atlanta. Here is another complication. The apex rights of the Booth will clash with the similar rights of the Florence, but the prior privilege goes to the older claim. Herewith I give the names, numbers, and dates of location of the oldest claims in this vicinity.

Name.	Number.	Date of location.
Booth	2431	May 10, 1903
Florence	2357	May 19, 1903
Red King	2361	May 19, 1903
Combination	2375	May 26, 1903
Red Top	2217	Aug. 20, 1903
Mohawk	2283	Aug. 20, 1903
Miss Jessie	2564	Aug. 29, 1903
Velvet	3843	Sept. 1, 1903
Polverde	3626	Oct. 17, 1903

These incidents should go far to emphasize the absurdity of the apex law and the possibility of its use to destroy the equity acquired by discovery and development. To lay claim to the ore in the Jumbo Extension, the Booth has to jump across half a mile of ground, that is, over the intervening property of a third company in the face of an agreement waiving apex rights with that company. The Booth then extends the shadow of its apex farther along the dip over the Merger property, to be met there by the shorter shadow of a similar right thrown forward from an outcrop that is at right angles to that traversing the Booth.

Of course, this hurts legitimate mining—and the illegitimate also, for that matter. In January last a

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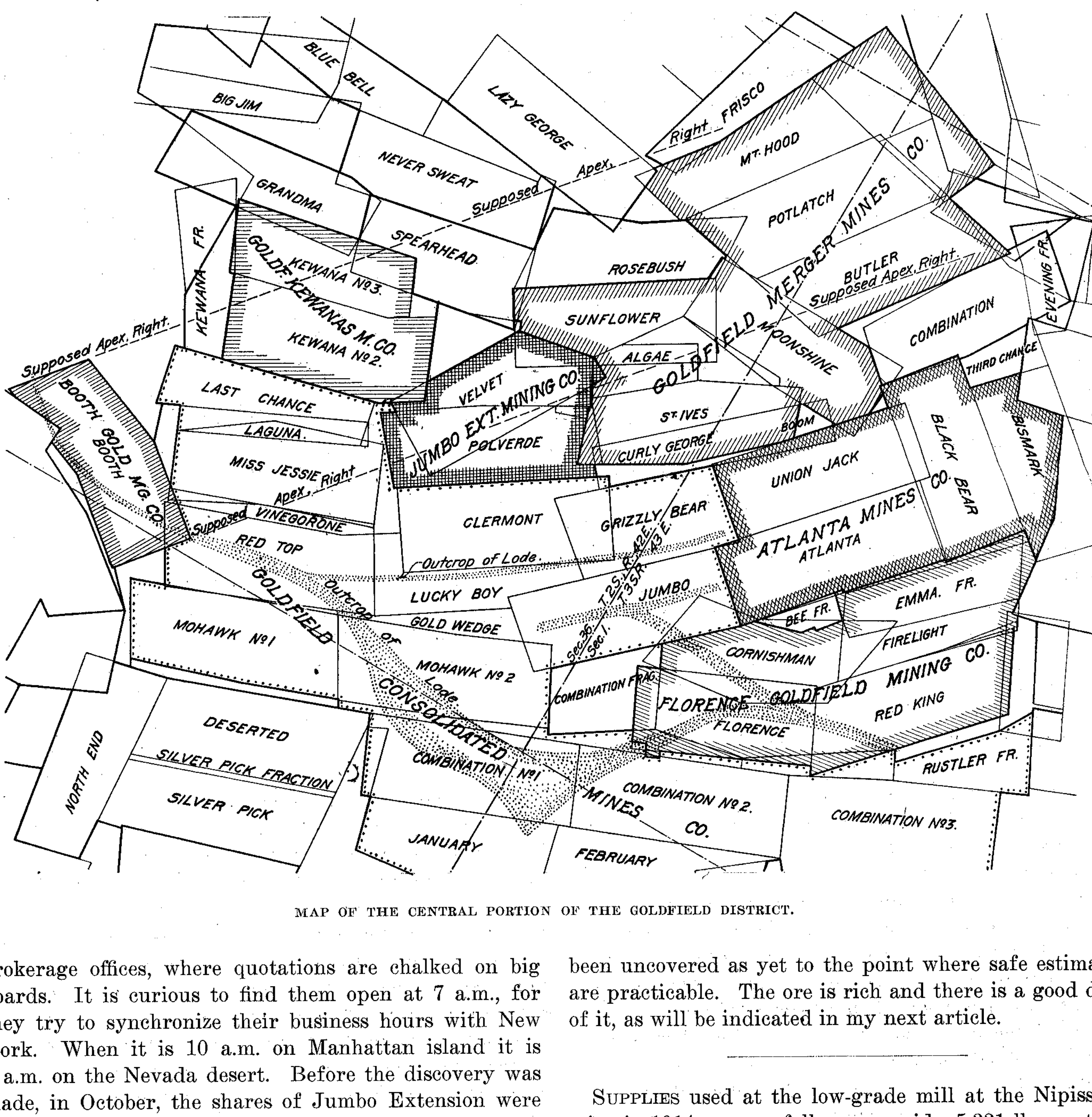
dozen companies were ready to undertake a campaign of deep mining, encouraged by the discoveries in the Jumbo Extension, when the fiasco put an end—temporarily—to such projects. Jumbo Extension shares have fluctuated widely, as the following figures show:

Nov. 30	$4.05	April 22	$1.82
Feb. 24	0.51	May 11	0.92

The company has 1,250,000 shares. Dealings are mainly at San Francisco, but also on the New York exchange. Goldfield has no exchange, but it has several and $3.15 on December 28. Simultaneously a New York curb-broker, then at Goldfield, circulated a report that the men at the mine were selling and that the mine was no good. Whereupon a small panic ensued. From $3.15 the market broke in a few days to 50 cents. Short-selling by brokers and by miners was influential in causing this debacle. Buying by New York helped to restore the quotation to a more reasonable figure. What the mine is worth is not known. Two experienced engineers have appraised the ore exposed, but the orebodies have not



MAP OF THE CENTRAL PORTION OF THE GOLDFIELD DISTRICT.

brokerage offices, where quotations are chalked on big boards. It is curious to find them open at 7 a.m., for they try to synchronize their business hours with New York. When it is 10 a.m. on Manhattan island it is 7 a.m. on the Nevada desert. Before the discovery was made, in October, the shares of Jumbo Extension were quoted at 12 to 15 cents. Many local people bought at these low prices and stood to make a nice 'clean-up,' but they used their profit to buy more stock, they 'pyramided,' forecasts being freely made that the shares would rise to $10. Some of the wiser miners gave orders to sell at $3, if the market declined. Their orders took effect automatically when the range of prices fell between $3 been uncovered as yet to the point where safe estimates are practicable. The ore is rich and there is a good deal of it, as will be indicated in my next article.

Supplies used at the low-grade mill at the Nipissing mine in 1914 were as follows: cyanide, 5.331 lb. per ton; lime, 5.749 lb.; caustic soda, 3.059 lb.; aluminum dust, 0.592 lb.; aluminum plates, 0.529 lb.; aluminum ingots, 0.172 lb.; flint pebbles, 3.826 lb.; coal for heating, 32.001 lb.; and power, 53.889 kw.-hr. The cost was $2.0457 per ton. In treating 79,125 tons the recovery was 89.64%, at a total cost of $3.989 per ton.

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Goldfield Re-visited—II

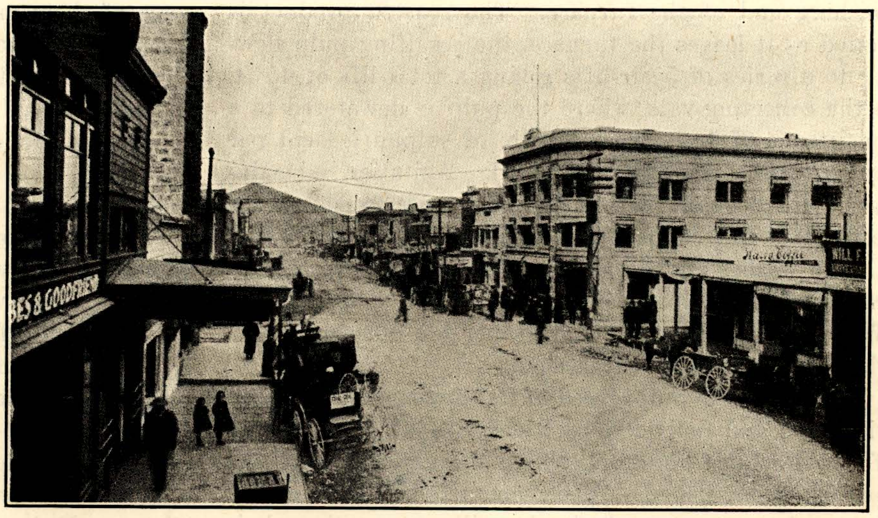
By T. A. RICKARD

On the occasion of my former visit, in April 1908, the richest ore from the mines of the district was being sent to the smelters near Salt Lake and San Francisco, while the lower-grade ore was subjected to amalgamation, cyanidation, and chlorination. Now amalgamation and chlorination have been displaced; cyanidation alone survives. At that time the Goldfield Consolidated company had just begun the erection of its big mill. F. L. Bosqui, as consulting metallurgist, expected to make an extraction of 95% by cyanidation at a total cost of "under $2.50" per ton. The mill did achieve an extraction of 95.4% at a cost of $1.60 per ton. This does great credit to J. W. Hutchinson, who was mill superintendent until he became general manager for the Goldfield Consolidated Mines Company in January last. His detailed description of the mill appeared in successive issues of the *Mining and Scientific Press* during May and June 1911.

On entering the mill the rhythmical noise of the stamps sounded to me like familiar music. The 100 stamps crush 1015 tons per day in cyanide solution through 4-mesh screens. The circulating solution contains 0.4 lb. cyanide per ton. Crushing in solution saves cyanide, but it decreases the extraction of gold in sulphide ores. The advantage, however, exceeds the disadvantage. There is also a saving of water, which, at Goldfield, costs 50 cents per 1000 gallons. The extraction made by the mill is now 92%; it has been higher. The percentage could be increased, but to do so would not be economical, for the expense would exceed the monetary gain.

The first observation to be made inside the building is the tremendous foaming of the pulp as it passes through the different machines, culminating in a vast mess of bubbles in the dewatering vats. This is due to the reaction between the lime and the sulphates in the ore. When the mill started, only 8 lb. lime per ton was required to neutralize the acidity of the ore; when the roasted concentrate was treated the addition increased to 11 lb. per ton; now it is 20 lb., this final increase being due to the treatment of caved ground, remnants, and dump, all of which contains the sulphates produced from the oxidation of the pyrite and chalcopyrite. Most of the lime is added to the ore as it passes through the first gyratory crusher, the lime being fed by a traveling belt moving 2 ft. per minute. Further rectification is made in the mill, by adding an emulsion or milk of lime.

One of the few defects of the plant is seen in the crusher-house, where there is insufficient room for screening between the bins and the first crusher, a Gates gyratory. There is also lack of grade between the trommel below the first gyratory and the two secondary crushers. Another apparent defect may be mentioned here, namely, the harnessing of three tube-mills to one motor; each might be operated separately or at least not more than two should be placed on a single drive, with ability to cut out one tube-mill when convenient. On the other hand, by the existing arrangement the high starting-torque required for tube-mill operation, necessitating 40 hp. extra, is distributed over three mills.



MAIN STREET OF GOLDFIELD.

Since the mill was constructed the only modification in the design has been the elimination of the 36 Deister tables for the treatment of middling, this change being due to the milling of lower-grade ore. At present 60 Deister tables (of the No. 3 size) are in use. On asking the manager if they were doing good work, he replied "Excellent." On the white rubber covering of the Deister tables one can see the free gold clearly, for it forms a yellow rim just above the pyrite. Another minor change is the increase in the number of leaves in the Butters vacuum-filters; 386 are in use as against 336 formerly; two boxes, containing 50 leaves, having been added.

The high price of spelter is not without interest to the superintendent, for the mill requires 425 lb. of zinc-dust per day, or about 6 tons per month.

The concentrate averages 6% by weight of the ore milled and contains 67% of its valuable contents. Formerly this product was treated with acid wash, bromo-cyanide and peroxide; later this method gave way to a preliminary treatment with alkaline cyanide, which extracted 85% of the gold, leaving a comparatively low-

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grade product to be roasted. Thus the loss in roasting was minimized. Now the raw concentrate, which has decreased in assay-value from about $200 to $30 per ton, is sent direct to the roasting-furnace.*

The daily output of concentrate is 80 tons, containing 18% sulphur. This is roasted in two Edwards duplex furnaces (112 by 13 ft.) having 54 spindles (or rabbles) arranged in two rows for each furnace. The fuel used is crude oil, 14°B., from Bakersfield, California. Nine gallons of oil is required per ton of concentrate. Contact with the heat is maintained for 10 hours. It is found necessary to fire only at the feeding end, the fineness of the material facilitating the advance of combustion so as to obviate the need for firing at the finishing end of the furnace. The roasted product is wetted as it leaves the furnace, the resulting pulp flowing to a series of 5 air-lifts giving a total lift of 45 ft. to the collecting-vats, where the pulp is dewatered to a consistence of 1:1. Then 10 lb. of sulphuric acid per ton of dry concentrate is added. Agitation is maintained for 8 hours, with subsequent water-washing, for recovering the copper and eliminating the sulphates. All the washes and decantations are passed over scrap-iron for the precipitation of the copper. The recovery of copper is at the rate of 3 lb. per ton of concentrate. This pays for the acid treatment.

Meanwhile the pulp, freed of its copper, is neutralized with lime and delivered by a centrifugal pump to one of two Pachuca agitators, where the dissolution of the gold is accomplished. When completed, the residue is delivered to storage-tanks in the mill proper; there it is filtered, and finally sent to waste with the mill-tailing. The solution goes to a separate circuit and precipitates its gold on zinc-dust. The total cost of treating the concentrate is 90 cents per ton.

Outside the mill a couple of imposing steel structures command attention; these are the dominant portions of a plant for returning the tailing to the mill for re-treatment. The tailing has been carefully impounded; it covers about 100 acres, and weighs 2,000,000 tons, averaging $1.60 per ton. To treat this residue the company has erected a Lidgerwood radial aerial cable-way with two stationary revolving towers and a traveling car. The towers are 160 ft. and 125 ft. high respectively. The lower tower moves in the arc of a circle along a standard-gauge track, having 135-lb. rail. The two tracks are 70 ft. apart, centre to centre. The rope is of 2¾-inch diameter. The capacity of the machine is 500 tons in 8 hours and the cost of it to date is $91,600. The expense of returning the tailing to the mill is estimated at 6 cents per ton.

A visit to this mill leaves a strong impression of the skill shown in its original design and an appreciation of the fact that it has accomplished its purpose most effectively. From the day the mill started (December 26, 1908) to the end of 1914, the mine has produced 1,895,338 tons of ore yielding $43,710,665. The only regret is the apparent termination of its useful activities, for the Goldfield Consolidated mine cannot continue to furnish sufficient ore for long, and no other mine in the vicinity shows any prospect of yielding a supply adequate for so large a plant.

In 1908 one of the principal technical operations in the district was 'high-grading,' a polite term for the theft of ore by the miners, who took their spoil to the 'fences' or assay-offices, where it was bought and reduced to bullion. During the summer of 1907 fully $1,000,000 worth of ore was thus stolen from the Mohawk and Combination mines. It is said that at one time 36 so-called assay-offices were running all night, some of them miniature smelters, and all brightly illuminated by electric light. That was the time of the Frances Mohawk and Hayes & Monette leases, when, in the summer of 1906, Goldfield seemed to sweat gold. In the Frances Mohawk lease at a depth of 200 ft. there was an 8-inch vein that assayed $250,000 per ton, mostly free gold and the telluride of gold (calaverite). The various leases on the Mohawk mine yielded a total of $50,000 per day for 106 consecutive days. When the principal lease was about to terminate, the lessees paid the miners $10 for 8, or even 6, hours of labor. No discipline was maintained, for a refusal to work would have prevented the lessees from gathering their golden harvest before the expiration of the lease. The miners came from underground with their jeans and jumpers stuffed with specimen ore. The worst element in the Western Federation of Miners was in control of the labor supply and the rights of property were treated humorously. Then came the big strike (not of ore, but of miners) in the fall of 1906. Troops under General Funston (recently at Vera Cruz) paraded through Goldfield and a bloody conflict was only just avoided, thanks to the respect felt for the regulars. When a committee of citizens paid a call on the General, he remarked that he had never seen so many overcoats on a hot day. All of them were armed with revolvers and sawed-off shot-guns. Every building had gun-men on its roof. On the Montezuma club, the headquarters of the mine operators, there were 15 such sentinels on the alert for trouble. Everybody expected the town to 'go up' any minute. It was like living on the edge of a volcano or the rim of a copper converter. Thus I return to my metallurgical simile.

Journalism also had its romance amid this explosive environment. Charles S. Sprague, recently identified with the Jumbo Extension mine, had formerly owned the Colorado Springs *Daily Telegraph;* at the height of the Cripple Creek boom he sold it for $65,000 and drifted to Goldfield. Arriving there he was engaged as editor of the *Tribune,* at $75 per month and one-third interest. The boom was in full swing. The only unpaid matter appearing in the paper was a 'stick'—about two inches—on the first page; all the rest of the reading matter was inserted for 60 cents per line, paid by promoters and brokers. Not every promoter was a broker, but every broker was a promoter. The 'mining game' was *in ex-*

*A description of the treatment as conducted in 1912 was written by J. W. Hutchinson for the *Mining and Scientific Press* in the issue of January 25 and February 1, 1913.

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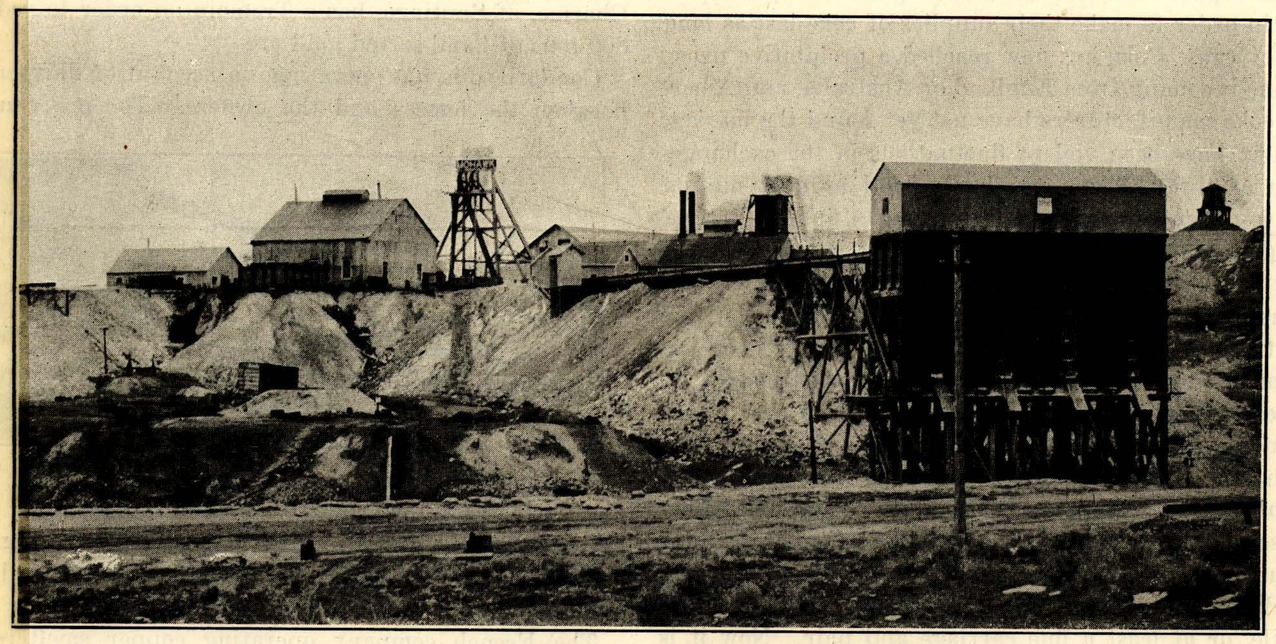
celsis. Yes, those were glorious days. When a deputation of citizens went to Carson City, to persuade the legislature that Goldfield was particularly fitted by man and nature to be the county-seat, the gentlemen from Goldfield began to throw nickels among the boys and ended by chucking $20-pieces. Having succeeded in their enterprise, they invited the legislators and their families to visit Goldfield; they chartered a train of pullmans for their guests and on their arrival in the golden city they ladled champagne from wash-tubs, because, of course, it takes time to pull corks.

Yes, those were the good old days. It seems long ago, but it was only yesterday. Among the more pleasant stories I like to recall that of Smith and Benedict, two stock-commission men from Chicago, who in 1906 had bought into a lease near the Sandstorm. They came to look at their mine, but the shaft had caved, owing to faulty methods, so they spent a night gambling with some of their acquaintances. During the evening they agreed to pay $10,000 for a half-interest in the Hayes & Monette lease. Leaving next day, they went to San Francisco and Portland, with an uneasy feeling of uncertainty as to what they had done. Not being clear about it, they telegraphed to Hayes and Monette to meet them at Reno and paid the $10,000 in cash. They were anxious to carry out their agreement as honorable men, even if they had a hazy recollection of the reasons prompting the deal. They had a code of their own. Soon after, Hayes and Monette wanted $10,000 more to develop the mine. Smith and Benedict returned to Chicago and tried to dispose of their interest, at the very moment when rich ore had been struck in the lease. They failed in their effort to sell and were forced into a fortune. Each of them made over a million dollars. Of the others, Mr. Monette is now a big banker at Los Angeles, while Mr. Hayes retired from business and placed his money in stable investments. The discoverer of Goldfield, Harry Stimler, is running a mine at Goldfield. When he located the Sandstorm and started Goldfield on its bullionic career, he was working under a grubstake from Jim Butler, the discoverer of Tonopah. It will be remembered that Tonopah was the parent of Goldfield, the one being discovered in August 1900 and the other in December 1902. After Jim Butler had made some money, his wife persuaded him to buy a ranch at Bishop, in Inyo county, California, where he now lives comfortably. Finally, T. L. Oddie, who took a prominent and intelligent part in these discoveries, became one of the owners of the Tonopah mine and was elected Governor of Nevada, for the four years from 1910 to 1914. He is now operating at Willard, near Lovelock, the new district recently named after the champion pugilist. It is a dangerous baptism: champions have their day. We hope Willard may outlive its namesake.



THE MOHAWK MINE, GOLDFIELD.

WHEN PURCHASING new equipment many firms specify something like the following: All angular moving parts or projections must be avoided so far as possible. If impossible to avoid their use, all gears, cams, eccentrics, setscrews, bolts, sprockets, chains, belts, feathers, splines, keyways, cranks, connecting rods, or other dangerous revolving or moving parts actuated by power, must be so guarded and protected as to render contact, intentional or otherwise, between them and the person or clothing of the attendant as nearly impossible as the work to be performed will permit. All guards must be so constructed and built up as to be readily removable without the use of special tools, allowing ready access to the working parts. It must be the aim of the builder to produce a machine that shall be as safe to handle and operate as human ingenuity can devise.

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Goldfield Re-visited—III

By T. A. RICKARD

GEOLOGICAL NOTES—POSTSCRIPT ON MILLING

In 1908 it was recognized that the big orebodies were in veins traversing dacite, or quartz andesite, which owes its porphyritic habit to phenocrysts of feldspar. The feldspar is labradorite, characterized in this district by alteration to alunite, forming a white clay, which is a factor both in the mining and the milling of the ore. Fortunately (thanks to the vacuum-filter) it has not hindered the milling as much as was at one time anticipated. The thickness of the dacite had not been determined in 1908. Beyond the central area it was known to be covered by andesite and on the bottom (380-ft.) level of the Combination mine, an appearance of andesite dipping 34° northeast had suggested an important change of rock in depth. At that time it was recognized that the lode-channel follows lines of silicification and alunitization. It seemed to me then that any confident expectation of continued persistence of rich ore in depth was not justified by the geologic structure of the district. The "andesite" uncovered under the dacite in the Combination workings is the rock now called 'latite.' This term was originated by F. L. Ransome; the word is derived from Latium, an Italian province; it is applied to effusive rocks intermediate between trachyte and andesite. As employed at Goldfield it represents the "earlier andesite" of the first report published by the Geological Survey. Dacite stands for the quartz-andesite group. It is named after the Roman gold-mining province of Dacia, now Transylvania. It is difficult to distinguish latite from dacite, especially where the rock is altered near lines of mineralization. Latite can be distinguished best by the streakiness of its groundmass, a distinction not available to the miner underground. The dacite is supposed to be intrusive into the latite, the upper portion of which has been eroded. In depth, the workings of the central area, largely identified with the Goldfield Consolidated property, follow the lode first through the dacite, then through the latite, and finally into alaskite and shale. Alaskite is another recent petrographic term. It was proposed by J. E. Spurr in 1900 to cover rocks consisting essentially of quartz and alkali feldspar, without regard to texture. Formerly the term 'aplite' would include such a rock. When I asked a superintendent what 'alaskite' was, he was at a loss to give a definition, so that when I suggested "a sort of bastard granite," he assented hastily. According to Spurr, it represents the last phase of magmatic differentiation. In the Jumbo Extension, at the 1017-ft. station, I saw a vein of alaskite zig-zagging through the shale. It is clearly intrusive into this underlying sedimentary formation, which ranges from a black shale to a 'short' limestone. Previously it was known from exposures on Columbia mountain, where it outcrops in a black band visible from the passing train. In 1908 I wrote of "a core of hard calcareous shale (of Cambrian age) flanked by alaskite and rhyolite." That 'rhyolite' was the 'latite' of a later nomenclature. The sequence thus observed on Columbia mountain is confirmed by the evidence obtained in the deeper mines.

At the Goldfield Consolidated office I was shown the model of the mine. This is most instructive. It indicates that of the many outcrops of quartz, only a few have proved to be the caps of orebodies. The best was the silicious comb on the Combination claim; two-thirds of its length was ore, say, 600 to 700 ft. The model shows clearly that three main lode-channels have been uncovered, namely, the Jumbo, Combination, and Mohawk. The Jumbo was ore-bearing down to 250 ft. only; it was

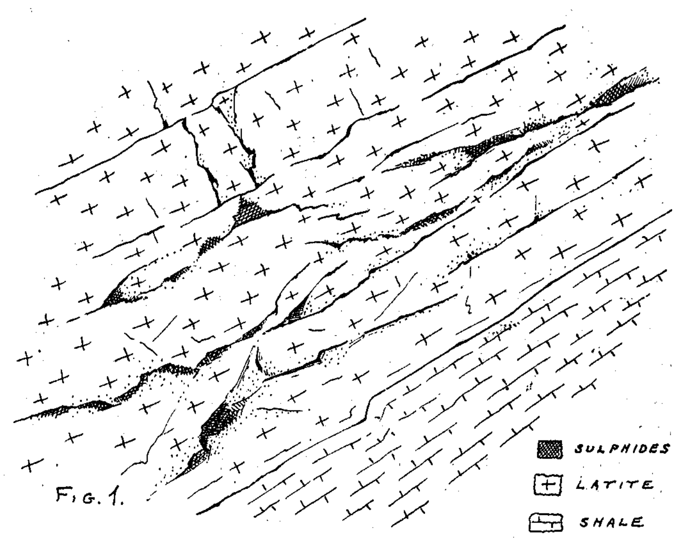

Fig. 1.

all in dacite, and did not reach the latite. The Combination ore reached down to 480 ft., that is, as far as the contact between the dacite and the latite. It was a nearly vertical vein; approaching the contact it flattened and then faded away. The Mohawk and Red Top orebodies are in a vein that dips at 40°, which is steeper than the contact between the two eruptives, but it follows their contact for 100 ft. on the dip and then penetrates the latite until the shale is reached, where the vein follows this lower contact also. The dacite is supposed to be intrusive into the latite, the upper portion of which has been eroded. The varying dips suggest a domical structure. The lower contact is the less regular, for the latite follows the irregularities of the eroded shale surface, the best ore being found where the vein is traversing the depressions in this old surface.

The deepest of the Consolidated company's shafts is the Grizzly Bear, which is down to 1400 ft. Water is being pumped out of the mine, through the Merger shaft, on adjoining property, at the rate of 220,000 gal. per day.

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Reviewing the story of the mine, it appears that ore is found in largest masses in the bends of the lode-channel where it traverses dacite. The distribution of ore is as irregular as the strike of the lode, which is no single vein, but a complex vein-system. The original boundaries of the fissuring are largely obliterated by silicification, so that 'walls' are rare, and delusive. Post-mineral faults are uncommon; only one extensive fault of this character, having a throw of 60 to 100 ft., is to be noted. The dip ranges from 30 to 45°, becoming almost horizontal in the Jumbo Extension workings where the vein reaches the shale.

The orebodies of the Combination were the most satisfactory, and they made the best showing at surface. They can be identified with the junction of the Reilly and Combination fissures, the ore extending from vein to vein. One of the rich Mohawk stopes struck 'wash' or drift 125 ft. from the surface, so that ore to that depth had probably been removed by erosion—a fortune thrown away, by Nature.

Going underground, I asked how ore was distinguished from rock. It is done by making numerous assays. The staff takes 1200 samples for assay daily. It is practically impossible to distinguish between rich ore and barren rock except in this way. Occasionally indications are noted, but they are unsafe guides. Dark streaks across the texture of the dacite and abundant alunite are mentioned as signs of richness, but the best is a line of brecciation, from one or two inches to a foot wide.

Just above the 600-ft. level of the Mohawk, I saw where the lode strikes the latite and flattens at the contact. Here a bulge of dacite into the latite terminates the orebody. Deeper the vein cuts through the latite; in this rock the orebodies, such as have been found, are rich but not so wide, although quite as long. The latite breaks cleanly but less readily, for it is finer textured than the porphyritic dacite.

On the 1000-ft. level, at a point 400 ft. west of the Grizzly Bear shaft, the visitor will see abundant green stains, indicating the presence of copper. However, much of the green discoloration seen in the mine is iron sulphate. Whether iron or copper, this sulphate is an important matter from the metallurgical standpoint. Not until the workings reached 930 ft. did the copper appear in quantity sufficient to interfere with cyanidation. The base ore is shipped to the smelters; it contains 2 oz. gold, 20 oz. silver, and 7% copper, as chalcocite and chalcopyrite. From 400 to 500 tons of such ore is being shipped from an orebody 150 ft. long and 15 ft. wide. This copper ore is characterized by the presence of famatinite, a sulph-antimonate of copper, allied to enargite, and named after an unhappy mine in the Argentine.

The lode finally strikes the shale and conforms to it, following the contact with the latite, although in places the fissuring extends into the shale itself. The rich ore is marked by needles of bismuthinite. However, plenty of bismuthinite has been found, for example, in the upper workings of the Jumbo Extension, without the accompaniment of rich ore. I saw the stoping of one of the rich veins; from 1 to 8 inches wide, averaging about 4 in., but requiring a stope 3 ft. wide, from the breaking of which the coarse waste is picked out, leaving ore assaying 2 oz. gold, 10 oz. silver, and 3% copper. A chalcocite impregnation as seen in the southeast end of the 1300-ft. level of the Grizzly Bear is shown in the accompanying sketch. Near the 1000-ft. station of the Claremont shaft, a piece of shale 10 ft. long and 3 or 4 ft. wide is seen enclosed by the latite.

These deeper workings yield patches of rich ore, but the showing is quite different from the bonanza zone nearer the surface. In the heart of the mine the orebodies were big and rich. For instance, a mass of quartz 2000 ft. long and 150 ft. wide was traversed by many veins, so that it was necessary to test it with numerous cross-cuts. One orebody yielding 60,000 tons was extracted from this mass of quartz, which reminds one of the silicious outcrops so prominent in the early days of the district before the surface had been either mined or covered with buildings.

The workings are hot, particularly the upper and older stopes, where the oxidation of the sulphides and the kaolinization of the feldspar is most pronounced. A temperature of 95 to 100°F. is reached, until free ventilation is established.

The structural geology of the shale is seen best in the Jumbo Extension, the mine that has been the subject of so much gossip, speculation, and litigation. It lies north, or on the dip, of the Goldfield Consolidated. The first ore of any consequence to the Jumbo Ex. (as it is usually known) was struck by lessees on the 790-ft. level, but it was patchy. They cut it again at 860 ft., but it was small, so they sunk the shaft to 1017 ft. and drove a level eastward, but not nearly far enough, so they failed to intersect it on the dip. Before stopping their search for the orebody, they drove a cross-cut southward to the Claremont line, thinking to cut some ore supposed to be coming down in that direction. They failed to find anything and 'quit.' The lessees having ceased to work, E. S. Van Dyke took charge for the company and extended the east drift on the 1017-ft. level, but he was unable to finish the exploration owing to a change of control, which passed to Charles S. Sprague. Under his general management, J. K. Turner, as engineer, and L. L. Dillinger, as foreman, ran the east drift 150 ft. beyond the point where the lessees had stopped it and started a raise (called Raise A) that penetrated good ore at 80 ft. above the level and 460 ft. east of the shaft. They then made other raises and subsequently extended the level itself to the present face, 950 ft. east of the shaft.

This, of course, was intelligent mining. It was based on a proper observation of the dip of the ore, which here lies at a flat angle. On the other hand, the work done by the lessees illustrates how some miners will ignore the aid of the surveyor and grope underground hopelessly, disdaining the aid of technical science.

The 1017-ft. level is in shale, for the most part, but

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for some length it is underlain by a silicious rock that may be either alaskite or quartzite, probably the former. The drift had not yet reached the ore on its dip, although the face seemed to be on the edge of the latite-shale contact at the time of my visit. In this mine the lode can be seen in the latite, on the contact, and within the shale. The last phase is shown where the old shale surface rises, as a hump, into the overlying latite. Along a given cross-section the lode becomes duplicated, the lower part following the contact while the upper, 16 to 20 ft. away, penetrates the latite. Ore has been found in shale 75 ft.

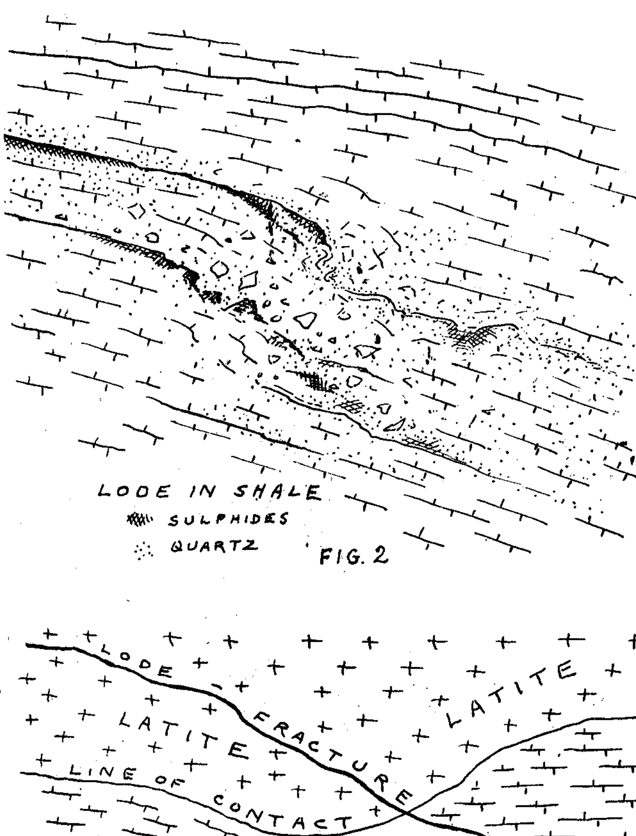

FIG. 2

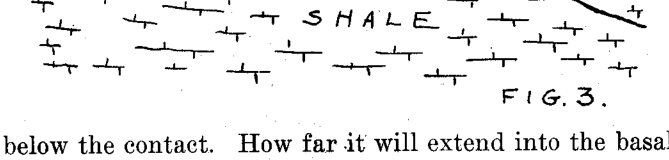

FIG. 3.

below the contact. How far it will extend into the basal formation can only be determined by further mining.

These deeper developments are exceedingly interesting. What they presage, I do not know. The evidence does not suffice to indicate the prospects of deeper mining. In most cases, in other districts, the passage of a vein-fissure from an eruptive to a sedimentary rock is not favorable. On the other hand, the lime-shale is a relatively soluble rock and as such affords a matrix suggesting the possibility of finding large replacement deposits of ore. Such orebodies as have been found are rich, although small when compared with those stoped in the dacite. When the existing complex of cross-purpose and litigation has been disentangled, we may expect to see a vigorous campaign of deepening exploration in several mines, notably the Jumbo Extension, Merger Mines, and Atlanta. The evidence to date indicates that such exploratory work is likely to be fairly productive.

POSTSCRIPT. In regard to my notes on the milling methods, given in the second article of this series, I would like to add that Mr. Bosqui, in his original scheme of treatment, did not contemplate the local reduction of the concentrate. This was evolved by Mr. Hutchinson after Mr. Bosqui's retirement and departure to South Africa. The supplementary cost therefore was not included in the estimate of $2.50 per ton. However, since 1912 the tailing-loss from the treatment of concentrate has been included in the mill-tailing. The total cost of milling, including the treatment of concentrate, was $1.88 during the year 1914. Excluding the cost of treating concentrate, the milling was done for $1.606 per ton. For the first five months of the current year, the combined cost has been $1.70, and the milling alone (excluding treatment of concentrate) $1.40 only per ton.

My reference to the prospective exhaustion of the ore reserves in the Goldfield Consolidated mines has, it appears, been taken too literally. The latest official report gives the reserve at 142,000 tons, and I have no reason to doubt the correctness of this estimate, made by such an engineer as Albert Burch, general manager up to the end of 1914. Moreover, the accumulation of tailing, now available for re-treatment, amounts to 2,000,000 tons, as stated in my second article.

As regards the tangle of conflicting interests and apex litigation, it is pleasant to record that during the past month much of the confusion has been cleared by a reasonable give-and-take on both sides of the chief controversy. This has been chronicled in our editorial columns. As far as can be learned at the time of writing, a further pacification is on foot, involving the purchase of Claude Smith's interest in the Merger Mines by Messrs. Wingfield and Erickson, who appear to have buried the hatchet.

The Motor-Truck at Tonopah

The ore from the West End mine, at Tonopah, is hauled to the old Midway mill, recently remodeled, in Pierce-Arrow motor-trucks. This method has been found economical. It is a suggestion of the 'jitney' in mining, and is of practical interest. By courtesy of the manager, John W. Chandler, we are enabled to give the details:

The distance is half a mile, or one mile for the round trip, loaded with 5 tons of ore per car going and empty returning. The truck itself, when empty, weighs 8750 lb. The road is of ordinary macadam, having an average grade of 5% and a maximum of 8½%. The truck makes 28 trips per day, requiring 15 minutes for each trip. Only 1 minute is consumed in loading. The consumption of distillate is ⅓ gal. per mile; and the consumption of oil, 1 pint per day. The driver is paid $5 and the helper $4 per shift of 8 hours. The cost of repairs is $75 per month, and the cost of hauling $450 to $500. Thus the cost of hauling is 55 cents per trip and 12½ cents per ton. Distillate costs 15 cents and oil 65 cents per gallon. The expense for tires is 6 cents per mile. Hard rubber tires are used.

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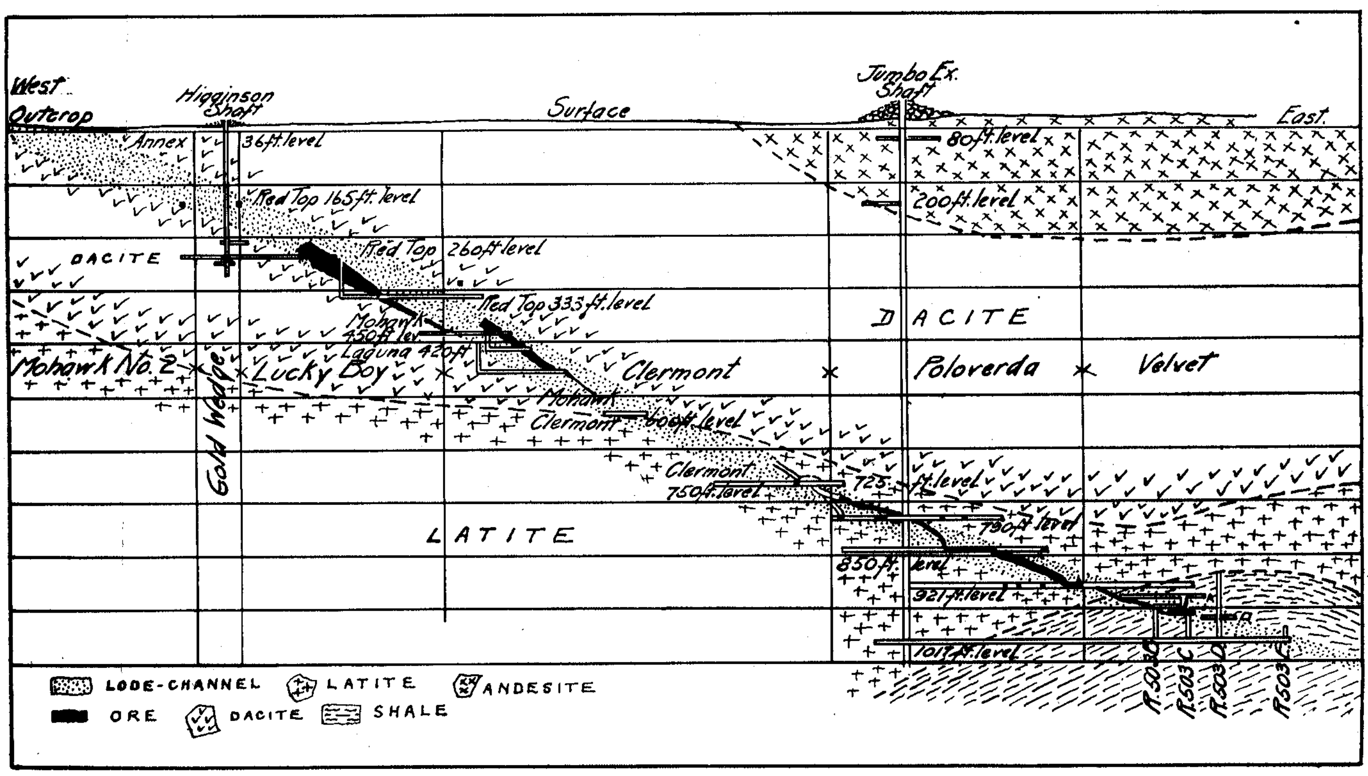
Mining in the Shale at Goldfield

By J. K. TURNER

The discovery of pay-ore in the contact-zone between the shale and latite has opened a new field for exploration in the Goldfield district, since the conditions revealed at the contact during the past few months were not heretofore known to exist. This condition, no less than the more recently ascertained fissuring of the shale, is likely to have an important bearing upon future operations in the district. Whereas until late in 1914 only one or two instances of fissuring extending into the shale, or below fissuring has taken place, although some silicification is apparent, for a negligible thickness, where the shale surface is in contact with vein-matter. Here silicification gives the shale the appearance of quartzite, and it is so classified by some authorities. There is no well-grounded supposition that the shale-latite zone will be found to contain pay-ore over the entire district, but developments up to the present time encourage the belief that valuable deposits will be found at many points in this zone where



CROSS-SECTION OF THE MOHAWK-JUMBO LODE.

the shale surface, had become known here and no fissuring of the shale had been observed, more recent development has shown fault-action displacing the shale at varying angles, together with wide fissures of ore-bearing material extending far within the shale. Apparently, however, there is no essential analogy between the fault-plane and the fissuring of vein-matter, though an exception is found in the case of what has been described as the fold-fault in the Jumbo Extension mine, where the fissure has the dip into the shale of only 20 to 30 degrees.

Shale is found throughout the Goldfield district at varying depth overlying the basic alaskite or granite. On Vindicator mountain, in the north-central area, both the shale and alaskite are found outcropping near the peak of the mountain. In the southern part of the district the surface of the shale is found at a depth of 1750 ft. The shale, a sedimentary deposit, has offered a surface impermeable to mineralizing solutions, except where even low-grade lodes or ore-channels come in contact with the shale or extend within the shale.

The overlying volcanic flows, latite, dacite, andesite, and rhyolite, this being the order of their effusion, are more extensively fractured and show a considerable degree of fault disturbance, while the shale, being more compact in structure and apparently fractured only at wide intervals, offers, with its impermeable surface and comparatively flat bedding-planes, a condition favorable to the greater concentration of the metals.

The lode itself, with a dip of about 30°, follows, for varying distances, the contact-planes between the andesite and dacite, between the dacite and latite, and between the latite and shale, finally breaking through each in turn, but continuing for a greater distance along the shale-latite contact. This bears out the theory of greater resistance to solution and therefore a higher degree of concentration of the metals, in the sedimentary than in

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the igneous formations of this and other districts.

Throughout the enriched area in the shale-latite zone, where it has been explored in the Jumbo Extension mine, the rich ore is found on the hanging-wall side. Between the rich ore and the shale is a mass of silicified latite, varying in width from 2 ft. to over 30 ft. A large part of this material is ore of commercial grade. Foot-wall seams of the rich ore have been found at a number of places in the mine, near the shale. Judging from the results of development work thus far in the contact-zone, it is apparent that the ore-channel upon coming in contact with the shale, seeks the deeper depressions or ravines in the shale's surface. The contour of this surface is most irregular and is marked by hills, ridges, and gullies, entailing a complex system of development.

In the Jumbo Extension mine the condition is presented of two distinct ore-channels, one overlying the other; the first extending along the contact-zone between the latite and shale, and the second in the form of a wide fissure, within the shale. The dip of this fissure is so nearly that of the bedding-plane of the contact that raises are driven from the workings in the fissure to develop the ore in the contact-zone, and this dip is so nearly flat that it is necessary to drive raises at frequent intervals to secure economy in stoping.

Further opportunities for interesting developments are offered at the contact of the shale and granite, the latter having been exposed at a depth of 1750 ft. in the St. Ives claim of the Merger Mines Co. Owing to its exceeding density it seems reasonable to expect an even greater concentration at the surface of the granite than upon the shale or in the other contact-zone near the surface.

AIR COOLING AND DRYING, whether the object be temperature or humidity control, has passed through the stages of free water contact, which in most important cases has been abandoned in favor of mechanical refrigeration. Numerous systems are in use where this cooling and drying has been performed by surface, city, or well water, and because the results have not been uniformly satisfactory either in obtaining the desired temperature or in reducing the humidity in the air to the proper degree for all seasons and weather conditions, they have been replaced by refrigerating equipment. All uncertainties and variations are removed by the substitution of mechanical refrigeration and yet its use is far from being as universal as it should and could be. In some classes of factory operation it has been found that the amount of labor performed by the workmen, especially in the summer months, was increased so much by mechanically-cooled breathing air that the expense of this cooling is more than offset by the greater output of the operators. For humidity control, by far the largest example of air-cooling by mechanical refrigeration is the Gayley process equipment for steel furnaces, the success of which is beyond question, and yet the application of similar equipment to other industries where humidity control is necessary or desirable is painfully slow.—*C. E. Lucke.*

Apex Litigation at Goldfield

As our readers are aware, the claim of the Booth against the Jumbo Extension has been compromised in an amicable spirit, which does credit to the two principals, Messrs. George Wingfield and Charles S. Sprague. In his geological report to the Jumbo Extension Mining Co., Mr. Horace V. Winchell summarizes the main facts in an interesting manner. We quote him as follows:

The principal veins of the productive portion of the Goldfield district occur about as shown on the above mentioned map.* Some of these veins have been large producers near the surface, and to the depth of about three hundred feet. The main vein is mineralized in lenses or shoots and has been followed downward upon its dip to the eastward for a horizontal distance of about two thousand feet and to a vertical depth of about thirteen hundred feet, thus indicating roughly an average dip of thirty-five degrees. The minor or subsidiary features, such as the old Jumbo vein, are about vertical and either die out or unite on dip with main dip. This main vein has a curved and irregular line of outcrop which is on the whole crescent shaped, and lies in the ground something like a saucer. In the case of such a vein it might be possible to start on the rim at more than one point and follow on the vein downward to the same orebody in depth. Legal rights depend upon the existence of a vein or lode and not upon the existence and position of shoots of high-grade ore within it. The vein may dip northeasterly and the ore shoots rake or pitch to the southeast within it, or the outcrop upon which the location made may have quite a different direction from that of the true strike of the vein; but the claim may still have extra-lateral rights so long as the vein can be followed generally downward between its end-line planes.

The horizontal distance from the outcrop of the vein on the Booth claim to the stopes in the Jumbo Extension ground is between 2000 and 2500 ft.; and the vertical difference in elevation is from 800 to 1200 ft., indicating (if we have the same vein in both) either a dip as low as 22 degrees or else that the vein is being followed downward on a course which is neither dip nor strike, but between the two. Observations in the ground show that the latter is the true situation. It is possible to pass downward through the Booth workings, to see the vein on the surface and on the various levels (not always well connected by raises on the vein) and to follow it along the strike with occasional interruptions due to faulting, through the workings of the Goldfield Consolidated by a devious and extended line of travel, to a connection on the vein with the Jumbo Extension mine. Absolute continuity has not been proved, and perhaps does not exist. Moreover, we travel for more than 1500 feet substantially on the strike of the vein, and pass out from the stretch of ground included within the extended Booth end lines, and yet our average course is downward; we arrive at stopes which are

*See next page.

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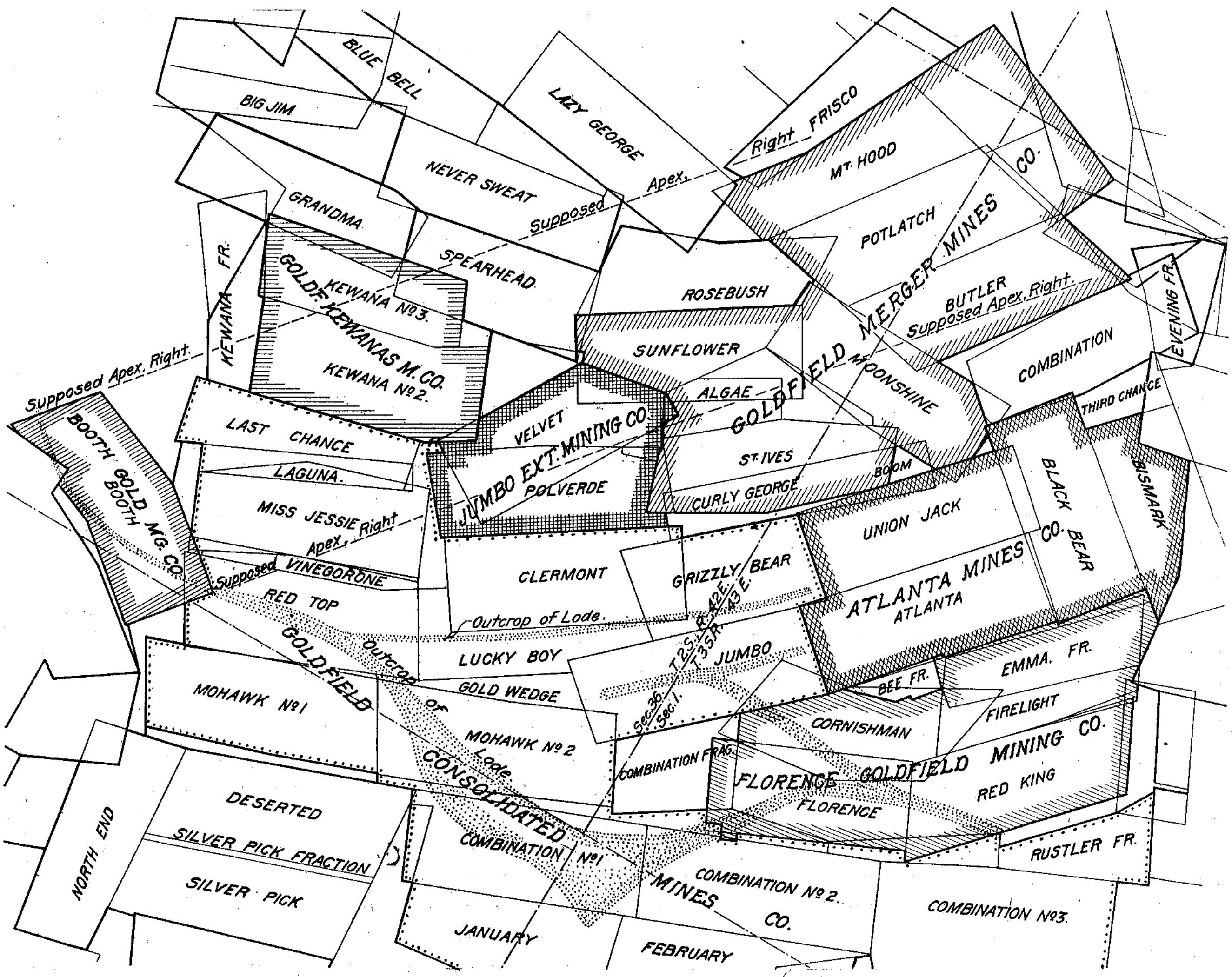
within the Booth end-lines extended; and we cannot doubt that we are on the same vein as that which outcrops on the Booth where we started. Throughout all this long belt of territory much time, money, and labor must be expended by the Booth in order to make its case positive, but it can probably be done.

Decisions in similar cases in Idaho which have repeatedly allowed extra-lateral rights more on the strike than on the dip, and have been affirmed by the higher courts, do not offer you much encouragement for contesting on that ground; and the uncertainly both as to the extent of the vertical displacement of the vein where it is seen to be faulted and as to the vein which the courts would take even if such faults were proved to be of considerable magnitude does not justify you in resting heavily on that hypothesis.

From the map it will be seen that an indefinite number of locations could have been made on the outcrop of the main vein in such a way as to include orebodies beneath the surface of the Jumbo Extension claims. As a matter of fact, such locations were made in the Mohawk No. 2 and Combination No. 1 claims, as well as in the Booth; and slight change in the direction of the east lines of the Florence, Combination Fraction, Red Top and other locations on the outcrop might have pointed them all, like the leaves of a fan, to one common centre or axis. In such cases it is held by the courts that seniority governs—the oldest location has unquestioned right of way through all the others. Here the Booth has the advantage, for it was located on May 10, 1903, at least ten days before any other on the outcrop, and nearly four months before the Velvet (located Sept. 1, 1903) and more than five months before the Poloverda (located Oct. 17, 1903).



MAP OF THE CENTRAL PORTION OF THE GOLDFIELD DISTRICT.

DAKOTA SAND, on account of its uniform character and thickness over wide areas in North America, is not considered a prospect horizon of the first class for the finding of oil, except in some limited districts, according to L. G. Huntley in Bulletin 102 of the A. I. M. E. Where structural and other conditions are favorable for concentration of oil, the relatively more lenticular bodies of sand above the Dakota, particularly in the Benton and Niobrara, are more promising for the development of production than is the Dakota sand itself, save in several regions in Canada, where the most favorable areas are in the region underlain by the sand where it begins to play out and become lenticular.

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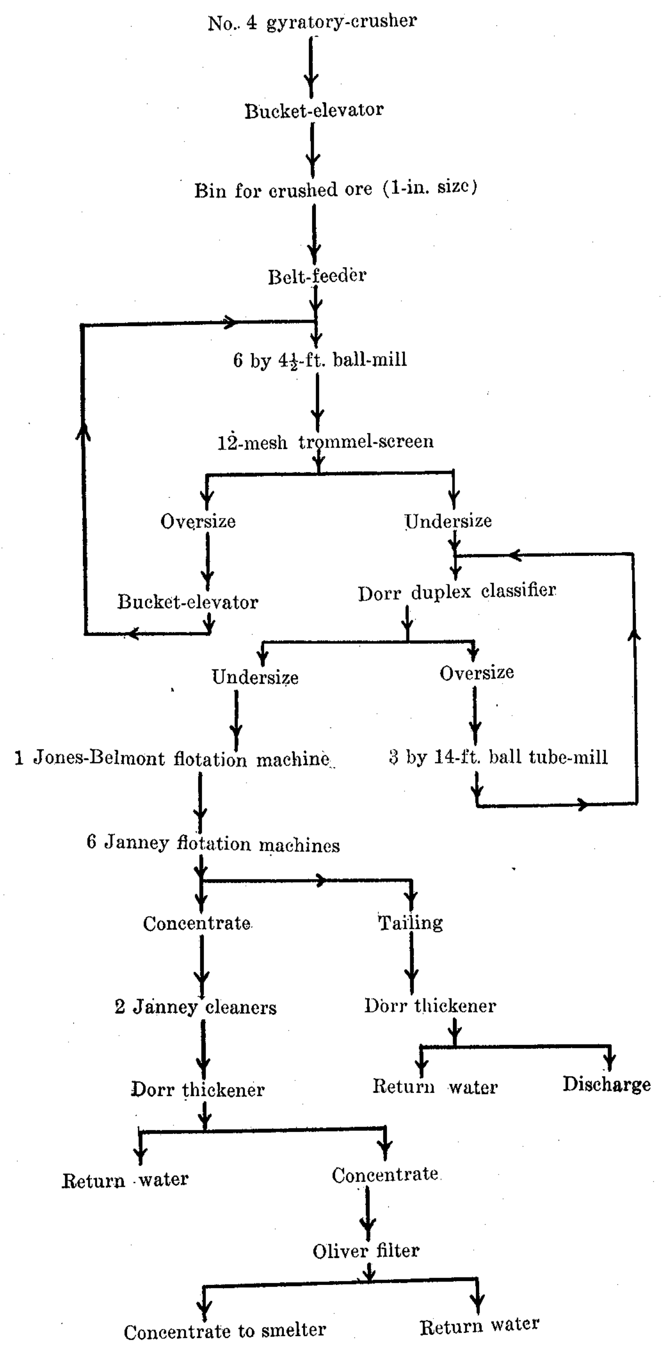
Flotation at the Florence-Goldfield Mill

By H. B. Clapp*

The Florence mine at Goldfield, Nevada, was equipped in 1909 with a 40-stamp mill and cyanide-plant, which was burned in 1912. Since then the mine has been operated for a small tonnage of shipping ore, during the extraction of which a considerable quantity of low-grade gold-copper ore was developed. Flotation has made its local treatment profitable, and a mill to treat the ore is giving satisfactory results from the following scheme of treatment:



The average extraction is 90% on a comparatively low-grade ore containing gold, silver, and copper.

*Mill manager.

The Jones-Belmont machine, which works ahead of the six Janney machines, is a new type on the market, having been designed and patented by A. H. Jones of the Tonopah Belmont Development Co. This standard-sized cell was installed in the Florence flotation plant at the request of Mr. Jones, in order that full data might be obtained under regular working conditions in comparison with the standard Janney apparatus. The Jones-Belmont has been taking the whole mill-feed of from 150 to 180 tons, and has made a higher-grade concentrate, with a lower silica-content, than the shipping concentrate taken from the two Janney machines, which clean the rougher concentrate from the six Janney machines that make a rougher concentrate. The new machine not only requires less power, but also less labor, and less oil per ton of ore to make a better extraction. The company is considering installing this type of machine in the flotation end of the mill.

Nickel-Steel

The span of the Quebec bridge that was lost during September contained 5200 tons of nickel-steel. The loss is estimated at $600,000. This material is a decided specialty when compared with ordinary carbon-steel. In fixing a price for nickel-steel the rule is to add $12.50 per ton for each 1% of nickel added. This must be open-hearth. The standard specifications of the American Society for Testing Materials for structural nickel-steel and rivet nickel-steel say that the nickel shall not be under 3.25%. Steel firms usually like to get about $12 for fabricating and $12 for erection, but the charges for this work are governed to a great extent by competition, etc. Erecting in an out-of-the-way place would command a higher figure. The fabrication and erecting costs, of course, are added to the mill price of the steel. Freight must be calculated also. Structural steel today, that is, shapes, is around 2.75c. per lb., Pittsburg. Plates are quoted at 4c., and bars at 2.75c. Prompt deliveries command premiums. The material which went into the Quebec bridge was bought at lower prices than prevail today. The Memphis bridge over the Mississippi was constructed of Mayari steel, a natural nickel-chrome steel.

THE chemical difference between sodium and potassium nitrate is in the character of the basic metal. As indicated in the names of these compounds, the metal in the one is sodium and in the other, potassium. Sodium nitrate, or Chile saltpeter, is imported into this country in large quantity from Chile. The potassium or potash nitrate has come chiefly from Germany, which controls the world's potash supply. It is practically impossible to obtain potash salts of any kind at the present time and quotations on potassium nitrate (niter) have not been published for a long time. Owing to the great demand for Chile saltpeter the price of this commodity has greatly increased and it is now bringing approximately $3 per 100 pounds.

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