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Tonopah, Nevada.

Written for the MINING AND SCIENTIFIC PRESS
By a Special Contributor.

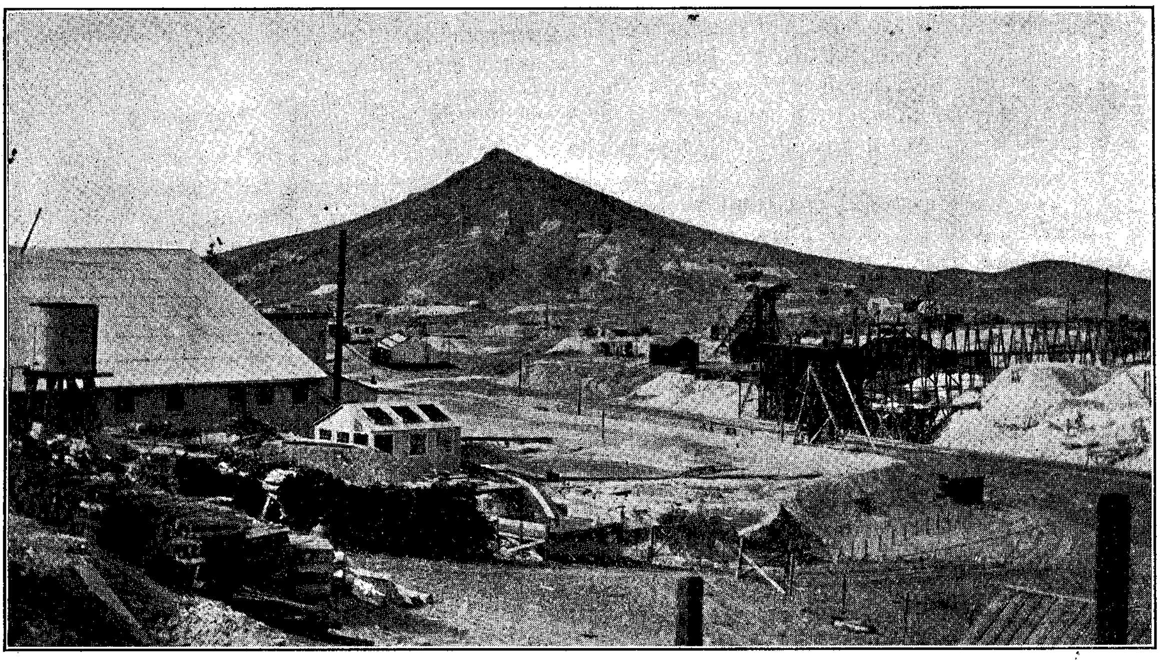
Tonopah was discovered in the spring of 1900. It is now considered the metropolis of the desert. Substantially built, with water and power transmitted many miles, this settlement is now the centre of the highest engineering talent in Nevada. Its mines are well equipped and most of the undesirable characteristics of the mining camp are comparatively suppressed. Mills will soon be treating a tonnage that will give the camp a period of remarkable prosperity.

In April of 1900, James L. Butler left Belmont to prospect in the district south of the site of Tonopah near the old camp of Southern Klondyke. On Mispah hill, he noticed the outcrops of the Burro veins, and took samples. These, like so many of the rich outcrops of Nevada, showed little to attract attention and were overlooked for a time and then thrown away by the assayer. Returning to Belmont, Butler broke off more specimens, which he handed to T. L. Oddie, on the understanding that the latter was to get an interest in the find, if he got them assayed. Oddie was then a lawyer in charge of the Stokes mine at Austin. The samples proved surprisingly rich. Butler and Oddie hastened to the discovery. It was in August that the claims were located and two tons of sorted ore were shipped while the location work was being done. This was sold for $600 and from that time the mine has paid for its own development. During 1901, the mines were given over to lessees, who probably extracted about $4,000,000. In 1902 the Tonopah Mining Co. entered the field. At this time the desert was covered by the excited population of a new mining camp. In the centre of the town, rich ore was struck in a shaft sunk in an unmineralized volcanic rock. Prospectors and promoters quickly took advantage of the find. Claims far away from real discoveries were in demand. Shafts were sunk in many places. Some were the honest investment of earnest men, many were desperately forlorn hopes, and still more were common swindles.

Rich ore was found at considerable depth by sinking through the barren andesite near the known producing area, notably at the Montana Tonopah, Desert Queen, and Tonopah Extension shafts. The geological features soon became well understood and in 1903 the gambling element found another field; the period of reckless and feverish excitement was at an end.

Tonopah has all the appearance of a mining town that depends upon the intrinsic value of its mines rather than living on the ephemeral prosperity resulting from wild speculation. The floating population, with all its unsavory attendants, has come and gone, and the mines are largely in the hands of companies that are exploiting them with the help of the best engineering talent. The town is situated in the foothills, the monotony of the desert being broken by the steep volcanic buttes. The camp is a pleasant contrast to the boisterous activity of Goldfield. Tonopah has assumed proportions commensurate with its industry, and is now settling down to work its mines with the utmost economy that the distressing conditions of the desert will allow.

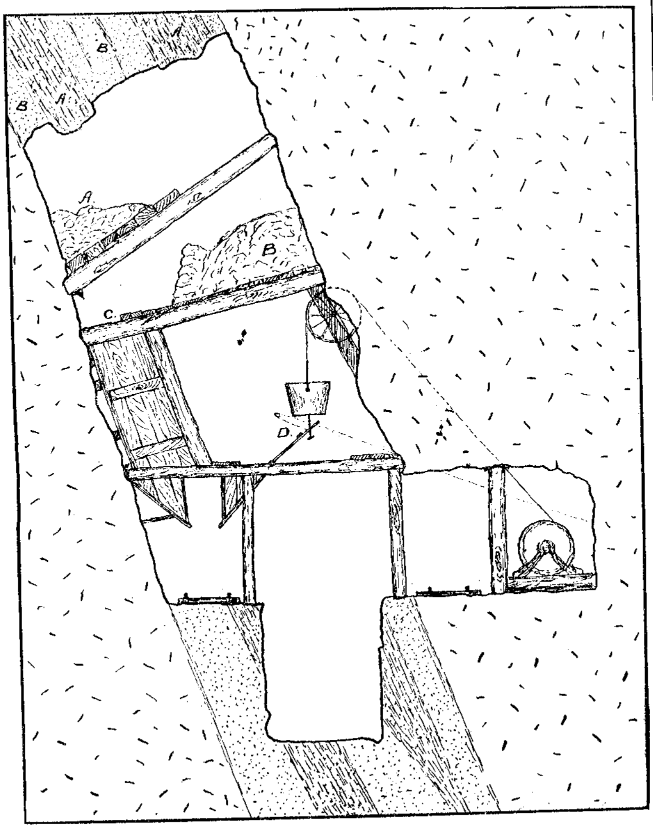
The Tonopah of Nevada and the Montana Tonopah are the principal companies operating in the camp and they are now making every effort to overcome the trying conditions, and treat their low-grade ores on a large and economic scale. At the Montana Tonopah, high-grade ore has been shipped in sufficient quantities to pay the cost of development and the construction of the mill, which is now nearing completion. No attempt has been made to increase the production beyond this point, and the wisdom of this policy is self-evident. The cost of sorting, freight, and smelting last year was $19.30 per ton, whereas it is estimated that the ore will be milled at Tonopah for $3 per ton. The shipping ore averages $50 to $60 and is largely treated by the smelters in California, where it is used for fluxing the highly basic ores of Shasta county. For this purpose it is not altogether satisfactory. The quantity of fine made by the samplers and the high percentage of alumina are detrimental to blast-furnace work, the former necessitating screening and briquetting, while the latter thickens the slag and prevents an easy separation of low-grade matte. Careful mill-tests have been made and a mill is being erected near the shaft. Water will cost $1 per 1,000 gal. and electric power $8 per h.p. per month. A plant to crush in two stages is being erected at the shaft immediately below the steel ore-bins; this will consist of one No. 5 and two No. 3 gyratory Gates crushers, which will break the ore



Montana-Tonopah Mill in Course of Construction.

to $\frac{3}{4}$-in. ring. From here the ore will be elevated by a belt-conveyor to the mill-bins. The ore will be crushed in a 40-stamp mill to 12 mesh and it is expected that 150 to 200 tons will be treated per 24 hours. This product is to be classified, the coarse material passing to eight Wilfley tables, while the slime and fine, together with the tailing from the Wilfleys, go to two Dorr classifiers, which precede the tube-mills. The latter will grind the coarse product to 150 mesh. The re-ground material then forms the overflow from the Dorr machine and the whole product will pass to two large cone-classifiers. The underflow from these cones goes to 16 Frue vanners and the overflow joins the stream of tailing from the vanners and runs to the settling-vats in the cyanide plant.

From the settlers the pulp is taken, as required, to the agitators, of which there are six of the Hendryx type.



Method of Mining.

After agitation in a cyanide solution, the pulp is drawn off to a large reservoir, where it is stored for filtering by means of the Butters vacuum filter-press. The gold and silver-bearing solution recovered from the Butters filter will be precipitated on zinc-dust in the Merrill precipitation-press, which has been so successfully used at the Homestake mine.

The mine is at present being prepared for a large output of milling ore. For this purpose chutes are being put in at 20-ft. intervals in the principal orebodies and preparations are being made to handle the ore-dumps at low cost. These will be drawn down in raises put up from the adit-level to the bottom of the dumps. The ore will be loaded in cars from these chutes and hoisted through the main shaft. A complete equipment is now installed for a constant and large supply of ore. Ample ore reserves are blocked out with chutes and bunkers for economic handling. A double-deck cage will be used with a counterbalance; a complete reserve hoisting and compressor plant, driven by a steam engine, is in place. The equipment is a double-drum double-cylinder 15 by 18-in. hoist driven by compressed air. The compressor, which is a 12-drill Ingersoll-Sergeant, is driven by a 200-h.p. motor, the power being supplied by the Nevada Power & Milling Company.

The arrangements for sorting ore in the stopes is shown in the accompanying diagram, together with the method of sinking a winze. In the vein there occur bands of shipping ore *A*, alternating with material of milling grade, *B*. These are separated underground, the shipping ore being broken down on a platform, from which it is shoveled to the chutes. The milling ore, generally below $50 per ton, is held in the stopes, as indicated. At *D* is shown an automatic dumping device, by which the hoisting engineer can empty the ore into the chute. The necessity of working out this shipping ore and leaving large quantities of milling ore in the stopes has made economic mining an impossibility. In few places has the vein been completely stoped. Generally the walls are strong and the amount of timber required will not be excessive when the milling ore is removed in a systematic way. Timber and fuel supplies are quite inadequate to meet the demand. Fire-wood is now $35 per cord, lumber $50 to $60 per 1,000 ft. and coal $37 per ton, but the price is not as serious as the scarcity of material. Several of the mines have been shut down or crippled by the fuel situation. This is one of the most important problems to be solved. The lack of locomotives and shortage of supplies prevailing in the West is only accentuated in this district.

More serious, however, are the relations between capital and labor and the prevalent tendency to make indefinite compromises. The conditions are better at Tonopah than elsewhere in Nevada, but they are serious none the less. It will require the utmost diplomacy from the Governor of the State to the Dago mucker to avoid a situation as serious to the miners as to the operators themselves.

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.

A fragment of Andesite was sent from Wallace, Idaho, by A.

The black mineral sent by A. C. W., of Randsburg, Cal., is Hornblende.

A Rhyolite Pumice stained with Iron was sent by T. R. from Los Angeles.

C. S. V., of Daggett, Cal., sends: No. 1, Dacite; No. 2, Andesite; No. 3, Andesite.

We named the rock sent by O. H., of Bodfish, several weeks ago. It is a ferruginous sillimanite Schist.

The specimens from F. G. C., Silver City, New Mexico, are: No. 1, Sandstone; No. 2, Specularite, Quartz, and Limonite.

The specimen of Clay sent by C., of Chittenden, Cal., carries considerable sand and is not a high-grade clay. It might do for brick.

The specimens sent from Casa Grande, Ariz., by W. J. F., are: No. 1, Andesite; No. 2, Andesite; No. 3, Andesite; No. 4, Pectolite and Garnet.

B. G. sends from Ydalpom, Cal.: No. 1, Andesite; No. 2, Andesite Tuff; No. 3, Andesite; No. 4, Rhyolite Tuff carrying Pyrite; No. 5, Andesite; No. 6, Andesite; No. 7, Andesite carrying stringers of Quartz.

The Desert Mill.

Written for the MINING AND SCIENTIFIC PRESS
By A. R. PARSONS.

The 100-stamp mill and power-plant of the Desert Power & Mill Co., operated by the Tonopah Mining Co. of Nevada for the purpose of milling the ore produced from its Tonopah mines, is situated at Millers, Nevada, a station on the Tonopah & Goldfield Railroad 13 miles west of Tonopah.

The entire installation, both mill and power-plant, was made by Chas. C. Moore & Co., of San Francisco, to whom a contract for the work was given. A little over one year was required to complete the mill. Mr. John H. Hopps of San Francisco acted as consulting engineer.

The power-plant contains four Babcock & Wilcox water-tube boilers of the vertical header type, provided with superheaters, set in batteries of two each. Each boiler contains 2,036 sq. ft. of heating surface. The boilers are arranged for firing with either coal or oil; the Moore oil-burning apparatus is used. Natural draft is obtained by means of a 66-in. steel stack 150 ft. high. A Green fuel-economizer utilizes the flue-gases. The minor boiler-room equipment consists of two Snow duplex boiler-feed pumps, Goubert feed-water heater, automatic relief-valves, stop and check-valves, damper-regulator, hot well, steam-traps, feed-water meter, thermometer, etc., all of which is ample and well arranged.

There are three 14 by 28 by 30 in. horizontal cross-compound side-crank McIntosh & Seymour gridiron-valve engines, each condensing, arranged for direct connection to 250 kw., 25 cycle, 2,200 volts, 150 r.p.m. alternators. There is also one 15 by 32 by 30 in. McIntosh & Seymour engine as above, directly connected to a 300-kw. alternator. All electrical equipment was furnished by the Westinghouse Electric & Manufacturing Co. The exciting current is supplied by 125-volt direct current exciters, belted from the band-wheel of the generators.

Condensation of steam from the engines takes place in Edwards condensers, equipped with power-driven air-pumps. The circulating water for condensing is pumped, by means of 8-in. double-suction Wheeler centrifugal pumps directly connected to 40-hp. motors, to a fan-driven steel water-cooling tower. Suitable switchboards with generator panels and distributing boards for mine and mill are conveniently placed in the engine-room. Step-up transformers raise the voltage from 2,200 to 22,000 for transmission over the 12-mile line to the hoists at the shafts in Tonopah. Step-down transformers lower the voltage at the mill from 2,200 to 440, all the mill-motors being 25 cycle, 440 volt Type C.

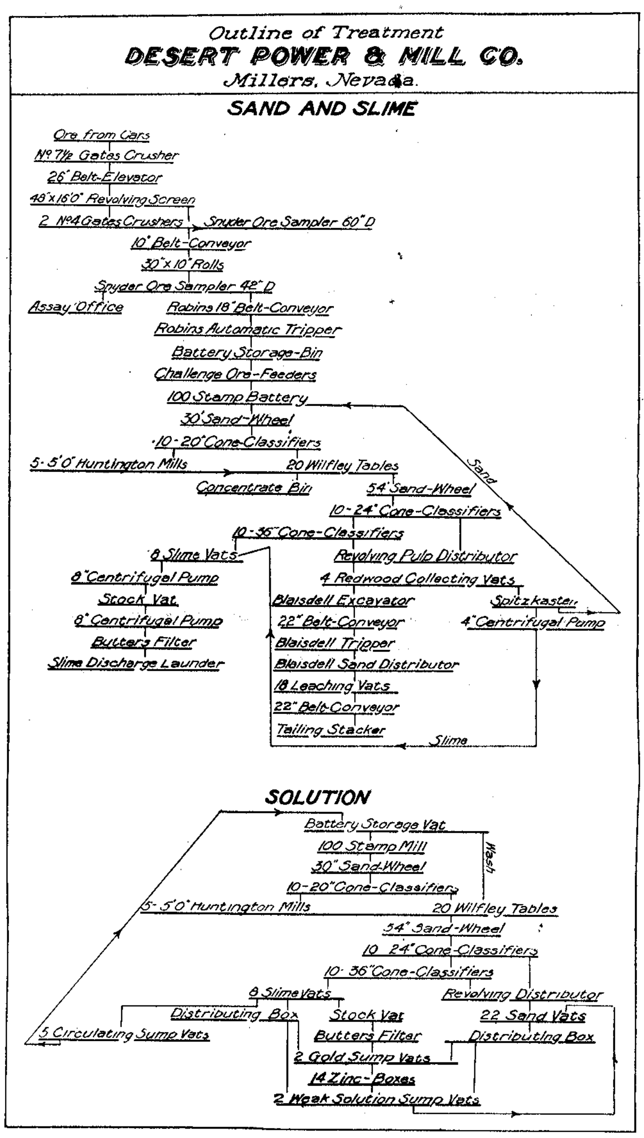
Water for the mill and power-plant is pumped by a two-stage vertical centrifugal pump from a well 60 ft. deep, situated 1,700 ft. north of the plant.

The main mill-building, 525 by 230 ft. in extreme dimensions, is erected on ground having only a 3% slope. The crude ore from the mines is taken up an incline trestle in steel hopper-bottom 50-ton railroad-cars in lots of seven cars to the crusher ore-bin.

The orebodies of Tonopah are largely replacements of andesite by quartz, forming parallel or branching veins and veinlets of quite solid quartz, separated by mineralized andesite, which frequently assays well. The ore as stoped, is therefore a mixture of quartz and 'porphyry.' Some of the early barren porphyry is sorted out before the ore is sent to the mill.

The primary ore of Tonopah consists of a gangue of quartz with some sericite and adularia, with a small percentage of the carbonates of lime, magnesia, iron, and manganese; the silver is present mostly as sulphide and sulphantimonide; gold is never visible and occurs in some form not yet determined; small amounts of pyrite and chalcopyrite occur, with traces of lead, zinc, arsenic, selenium, and other metals.

All of the ore now being treated at the Desert mill is partially oxidized. This oxidation, however, is never complete and most of the silver is still present in the form of argentite, which is probably mixed with stephanite, but in the more oxidized phases cerargyrite (horn silver), frequently containing iodine and bromine, is common. The carbonates of the primary ore are represented by oxides of iron and manganese, and gypsum. In the process of oxidation, a large proportion of the



iron, manganese, antimony, arsenic, copper, lead, zinc, and selenium originally present, has been removed. In none of the ore, however, are the base metals present in sufficient quantity to be valuable.

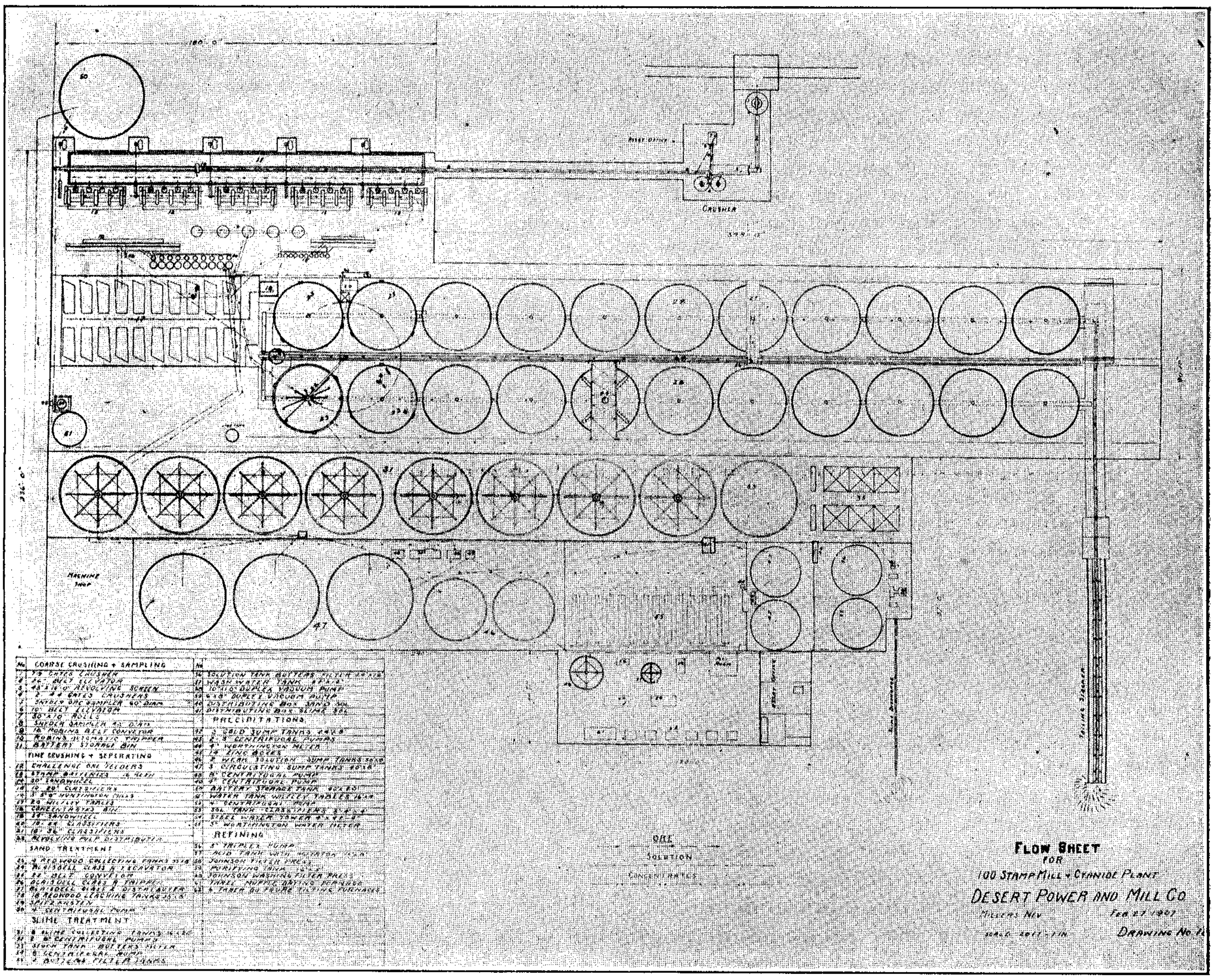
A sample of rich ore from the Valley View vein analyzed by Hillebrand of the U. S. Geological Survey gave the following results:

Ag	62.54%	38.10 as sulphides.	Au	0.62%
		24.44 as chloride, selinide, and alloy.	Fe, Mn	1.46%
			Cu, Pb, Mn	0.51%
			Se, Sb, As	0.96%

The ratio of silver to gold by weight in the ore treated at the mill is about 90 to 1.

The coarse-crushing and sampling department is in a separate building. The crushing plant has a capacity of 400 tons in 8 hours. Power for driving the machinery is supplied by a 125-hp. motor and the 18-in. Robins belt-conveyor for carrying ore from the crusher-house to battery storage-bin is driven by a 15-hp. motor.

The mine-run from the bin is fed through a finger gate to a 7½ Gates crusher, Style K. This reduces it to a maximum size of about two inches, in which condition it is elevated by means of a 26-in. steel bucket-elevator to a 48 in. by 16-ft. revolving manganese steel screen with 1¼-in. holes. The oversize from the screen passes to two 4 D Gates crushers for further reduction. The product from the smaller crushers unites with the screened product and passes through a 60-in. Snyder sampler. The reject from the sampler falls directly upon the 18-in. conveyor. The cut out from the sampler is elevated by a 10-in. steel bucket-elevator to a set of 30 by 10-in. rolls for fine grinding. The product from the rolls is passed through a 42-in. Snyder sampler, the reject from which falls upon the conveyor and the 'cut out' is taken to the assay office for further sampling and pulping. The 18-in. conveyor with troughing idlers takes the ore up a 22° incline and distributes it by an automatic tripper over the battery storage-bin, having a capacity of 1,500 tons.



Rack and pinion gates regulate the ore going to the 20 Challenge feeders, supplying the 100 stamps. The mortars of the stamp-batteries are the narrow pattern, single discharge, manufactured by the Union Iron Works. They are set on substantial concrete foundations down to hardpan, rubber sheeting ¼ in. thick being placed between the mortars and the top of the concrete. End and side liners of malleable cast-steel are used in the mortars.

Each battery of 20 stamps is driven by a 50-hp. motor, the order of dropping stamps being 1-3-5-2-4. The weight of each stamp is made up as follows: Stem, 427 lb.; tappet, 140; boss, 320; and shoe 180 pounds.

The stamps drop 104 times per minute through a height of 6½ in. and have a duty of 4.34 tons crushing through a 12-mesh wire screen, the height of discharge being three inches.

Chrome steel shoes and dies have been in use and cast iron dies containing a percentage of steel are being tried with satisfactory results, a more even wear of both shoes and dies being obtained by the combination. Both shoes and dies are worn down to a thickness of less than two inches, compensation for loss of depth being made up by cast iron false dies of varying thickness. The chrome steel dies average 1,736 stamp-hours, crushing 315 tons of ore each. The shoes average 1,011 stamp-hours, crushing 285 tons each. The steel consumption of dies and shoes is respectively 6.70 and 7.60 oz. per ton of ore crushed.

The battery-crushing takes place in a solution carrying 0.15% potassium cyanide, supplied from a 188,000 gal. storage-vat that is filled by 8-in. Butters centrifugal pumps, taking their suction from sumps at the lower end of the mill. The pulp from the battery (7 of solution to 1 of ore) flows to an inside bucket-elevator wheel, 30 ft. diam., driven by a 7½-hp. motor. The wheel elevates the pulp to ten 20-in. double cone-classifiers with hydraulic upward current of cyanide solution. Sizing tests:

Screen Mesh.	Battery product. %	Overflow from classifiers. %	Spigot discharge. %
On 20	6.82	0.21	4.40
" 30	17.58	3.71	23.27
" 40	8.85	3.42	11.37
" 50	10.32	7.04	20.46
" 60	2.83	5.10	8.30
" 80	10.62	13.85	13.39
" 100	10.82	15.90	6.67
Pass 100	31.77	50.30	11.65

The overflow from the classifiers passes directly to ten

No. 5 Wilfley concentrators. The bottom-discharge through ½-in. spigots, containing the coarse product for re-grinding, flows to five 5-ft. Huntington mills, equipped with 30-mesh screens. Three mills ordinarily re-grind the oversize product from 80 stamps. A 50-hp. motor drives the mills and a bucket-elevator for raising the product from the mills for distribution over the 10 concentrators.

The 20 Wilfley concentrators are securely anchored to a concrete floor with sufficient slope to allow drainage of any leaks or drips to the tailing-launder beneath the level of the floor, the launder also being made of concrete. The tables make 240 strokes (⅝ to ⅞ in.) per minute, the length of stroke varying with the class of material passing over each table.

Classification and dewatering of the product to the Wilfley concentrators is obtained by a series of V-boxes receiving the pulp from the launders carrying the overflow from the 10 double-cone classifiers and re-ground product from the Huntington mills. It is intended to install additional concentrators to take the middling from the present equipment, as experiments have demonstrated that an additional saving of about 10% can be made in concentrating, at the same time removing sulphides difficult to treat in the cyanide department. At present, 90 tons of ore produce 1 ton of concentrate, averaging 4.5 oz. gold and 815 oz. silver per ton. These are partially dried to about 12% moisture by draining and vacuum applied to a small vat receiving each day's output. The concentrate is sacked each day in canvas bags and shipped to the smelter in carload lots. The recovery by concentration from March to August, this year, was 15.63% of the gold and 30.32% of the silver contents of the ore milled.

A 30-hp. motor operates the 20 Wilfley concentrators, one 4-in. Butters centrifugal pump used for pumping wheel-pits, a Johnston vanner (used in experiments), and a small bucket elevator.

The tailing from the concentrators is elevated by means of a 54-ft. wheel to two sets of ten double-cone classifiers with hydraulic upward current of weak solution. The overflow from the upper set of 24-in. classifiers goes to the lower set 36-in. diam. The overflow slime and solution from the lower set of classifiers flows to the slime plant through a serious of spitzkasten for removing the very fine sand.

The sand and solution from ⅜-in. spigot-discharge of both sets of classifiers and spitzkasten flow through a wooden launder to a revolving wet distributor of the Butters & Mein type. This distributor is hung from a circular overhead trolley so that it can be swung to any one of four sand-collecting vats. These, as well as the 18 leaching-vats, are 33 ft. diam. by 8 ft. deep, provided with cocoa-matting and 10-oz. canvas filters laid on wooden strips, the filters being raised three inches above the bottom of the vat. The collecting-vats are provided with roller blind-overflow gates for a further separation of sand and slime. Some sand that overflows with the slime from these gates, is removed by a large pointed box receiving the entire overflow from collectors. The slime and solution from the pointed box are returned to the launder going to the slime-plant by means of a 4-in. Butters centrifugal pump.

For treating the sand, a collector is filled to a depth of five feet; this amount of packed sand, after transfer, fills a treatment vat. After a collector is filled, the distributor is swung to one of the other collectors, and the drain-valves opened for 24 hours; then a vacuum is applied to dry sufficiently to permit of excavating by means of a Blaisdell class A disc-excavator. This machine is mounted on a track and can be moved by electric motor and trolley to any sand-vat. A wrought-iron conical plug 22-in. diam. seated on a rubber gasket in a cast-iron flange in the vat is removed by means of a chain-block attached to the excavator. This leaves a clear opening from the top of the sand to the 20-in. Robins troughing-belt conveyor running below each row of vats, through which the excavator discharges the sand. It requires from 2 to 3 hours to excavate 250 tons of sand, the time depending upon the moisture in the material. The excavated sand is conveyed to a cross-conveyor at the east end of the sand-vats. The cross-conveyor, running up an incline, discharges to a conveyor of the same type running between the two rows of vats and above them, over a Blaisdell class A tripper. From the tripper, the sand falls upon the cross-belt of a Blaisdell class Z sand-distributor, then on to a rapidly revolving disc with speed-regulating device by means of which the sand is distributed about the vat in a fine shower, and at the same time, thoroughly aerated.

While transferring the sand from collectors to leaching-vats, lead acetate, previously dissolved in water (½ lb. per ton) is allowed to drip upon the sand on the conveyor-belt and slacked lime (4 lb. per ton) is thrown into the collector in which the excavator is working, thus thoroughly mixing the lime with the sand. After transferring, a little shoveling is done to level the sand. The first leaching solution, amounting to 30 tons, is brought up to 0.25% strength by the addition of a sufficient amount of potassium cyanide solution of known strength to the vat undergoing treatment. This pumping of strong solution is allowed to drain slowly through the partially opened drain-valves and is followed by repeated pumpings of weak solution from 0.15 to 0.20%, after which the charge is drained for transfer. This first treatment occupies five days, including time of transfer.

The second treatment averages five days and consists of repeated pumpings of strong and weak solution, that are drained and the sand transferred to another vat for the final treatment, which consists of as many pumpings of wash solution as there is time to apply, followed by two or three pumpings of water to displace all the solution. Then the vat is finally drained by vacuum for discharging from the plant. All pumpings of solution are allowed to disappear below the surface of the sand before the succeeding one is applied. Sand undergoes treatment for 12 to 15 days. Moisture in sand discharged averages 15%. Sand residues at present average 0.03 oz. gold and 3.10 oz. silver per ton.

SIZING TEST ON SAND RESIDUES.

	Screen Mesh.	Percentage.
Remaining on	20	0.15
" "	30	11.64
" "	40	13.98
" "	50	12.31
" "	60	10.48
" "	80	17.54
" "	100	12.77
Passing	100	21.05

The treated sand is discharged by the excavator upon the sand-conveying system, so arranged as to run in the direction opposite to that when transferring. The sand as it is discharged falls upon a cross-conveyor running up an incline of 25°. As the tailing-pile builds up to the stacker, an extension of 18 ft. is added at the end. Arrangements are being made for a cross-conveyor in connection with the stacker.

All leachings from the sand-vats, as well as the plant solutions, are sampled, assayed, and titrated for cyanide and alkalinity daily. Attenuated leaching solutions are sent direct to weak sumps. Centrifugal pumps, when not pumping to treatment-vats, are in service circulating solution in sumps through cones for the purpose of aerating.

All potassium cyanide used in the treatment of sand, is dissolved in a small vat from which a 2-in. pipe-line is connected to the suction of a 4-in. centrifugal pump used to pump solutions on sand. By means of a table and float arranged on the vat, the desired strength of solution can be obtained by opening the 2-in. line and allowing the requisite amount of standard solution to be drawn through the pump with the weak solution from the weak sumps.

The slime plant has eleven 3-in. redwood vats 36 ft. diam. by 20 ft. deep for collecting and agitating slime; one vat of the same dimensions used in connection with the Butters filters for stock pulp, two Butters filter-vats containing 96 filter-frames each and two tanks 24 ft. diam. and 12 ft. deep used for weak solution and water for washing slime on filters.

All of the 11 vats mentioned above are provided with rim overflow launders for receiving the clear overflow when collecting slime. The overflow goes to any of the three sump-tanks, 40 ft. diam. by 8 ft. deep, from which it is returned to the battery storage-tank by means of an 8-in. centrifugal pump. Eight of the eleven vats are provided with mechanical arm-agitators driven by a 30-hp. motor with gearing and friction-clutches over each vat. Agitators make 5 rev. per min. There are two sets of four-arm agitators quartering. The lower set, to which drags are hung for keeping the heavier fine sand in suspension, is 2 ft., and the upper set 8 ft., from the bottom. Any of the eight vats can be used for agitation although at present but four are in use at the same time, leaving four of the agitation-vats and the three regular collectors for service in receiving and collecting the slime in the slime-bearing solutions. A charge of slime is drawn from the bottom of the collecting-vats by means of an 8-in. centrifugal pump without interrupting the collecting.

Previous to receiving a charge of thick pulp from the collectors, about 150 tons of barren solution is pumped into the agitator. To this is added 1,000 lb. slacked lime and 600 lb. dissolved cyanide; the whole is agitated for one hour by the mechanical agitators and pumps. The charge of thick pulp is then pumped in. When thoroughly mixed, it has an average specific gravity of 1.144, that is, 21 parts of slime to 79 of solution. The mass is agitated for 30 hours, the mechanical agitation being assisted by compressed air, admitted through a perforated pipe running half across the bottom of the vat and by an 8-in. centrifugal pump, taking pulp from the bottom and discharging at the top of the vat. This is used when not in service for other pumping. At the end of 30 hours, the agitation is stopped and the pulp allowed to settle for six hours, when about five feet of clear solution is decanted and run to the storage-vats for precipitation. The settled slime is then pumped to a second agitation-vat into which solution equivalent to that decanted has been pumped; agitation is continued for 24 hours, when the contents are delivered to the Butters filter stock-vat.

The following table gives the average time of each operation in the Butters filter, from filling the vat to discharging the cake.

	Minutes.
Filling with slime	22
Collecting cake, 23 in. vacuum	45
Pumping back slime	20
Filling with barren solution for wash	20
Washing cake, 23 in. vacuum	30
Discharging cake	3
Settling wash	5
Running back wash to vats	30
Pumping discharged slime	10
Total	185

Pumping in connection with the Butters filters is done by an 8-in. centrifugal pump, driven by a 20-hp. motor. Vacuum for the filters is supplied by a 10 by 10 in. duplex vacuum-pump, driven by a 15-hp. motor. All valves in connection with filtering operations are operated from a platform by a system of rods and levers. The first solution coming through the vacuum-pump, upon beginning a new cycle of operations, is turbid and goes to the circulation sumps. When the solution becomes clear, usually in five minutes, it is turned to the storage-vats for precipitation.

The thick slime-cake, having been discharged, is broken up for pumping by means of 1-in. jets of water under a 70-ft. head directed downward into the hopper of the filter-vats. A duplicate pumping system is being installed to make it possible to either diminish the time of a cycle of operations or allow more time for washing. A large launder for running back wash-solution will also change the time when completed.

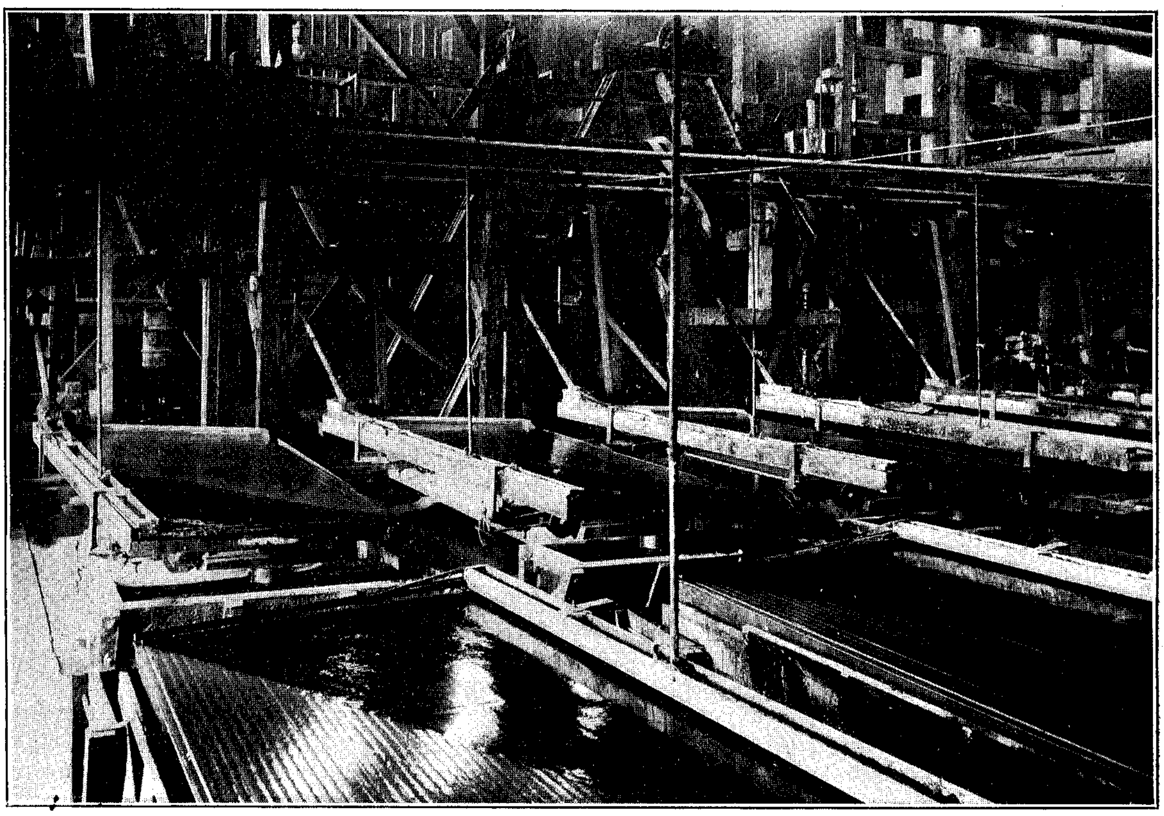
A mixture of air and water was tried for discharging cake but it was found that without careful attention and manipulation too much pressure would be used by the operator, thereby breaking the stitching in the filter-leaves. Accordingly, the cake is now discharged in the wash-solution by admitting water under a 12-ft. head. The wash-solution is allowed to settle for about five minutes before running back and then is drawn off to within about six inches of the thick slime in the hopper. By this means and the small amount of water from the jets, the slime can be easily pumped out with a moisture contents of 60%. At the present time, an average of 125 tons of dry slime is being filtered and discharged per 24 hours.

All solutions for precipitation are collected in two tanks, 24 ft. diam., 8 ft. deep. From these the solution is pumped through a 4-in. Worthington meter to the zinc-boxes by means of two 3-in. Byron Jackson centrifugal pumps, directly connected to 2-hp. motors. The precipitation room has a concrete floor sloping to a sump for drainage and collection of any drips or leaks.

There are fourteen zinc-boxes made of three-inch redwood. Each box has seven compartments, each holding 15 cu. ft. zinc shaving above the screen-trays, making the total capacity of the box 105 cu. ft., or about 1,600 lb. zinc shaving. The compartments of the boxes are arranged for an upward flow of solution.

The average amount of solution precipitated in 24 hours is 1,200 tons; the tailing from the zinc-boxes assays from a trace to 11 cents per ton, increasing in value from the time immediately after one clean-up to the time of the next. Four clean-ups are made each month; five men working over-time with two men from the melting-room make the clean-up of fourteen boxes in two days. During the first one the shaving in the two head compartments of the zinc-boxes is washed and the remainder moved up. The washing is done over the head compartment, and all precipitate is screened through a 30-mesh wire screen. The precipitate is allowed to settle and the solution is pumped by a 5 by 6 in. Knowles triplex pump through two Johnson filter-presses provided with 24 by 24 in. frames and leaves covered with 10 oz. duck. The settled precipitate in the zinc-box is bailed into tubs and dried in pans, without acid treatment, in a three-muffle drying furnace fired by coal. This precipitate, as well as that collected in the filter-presses, is thoroughly roasted, then pulverized and fluxed with 20 lb. borax, and 16 lb. bicarbonate of soda per 100 lb. precipitate. Formerly it was the custom to treat all the zinc-box product with sulphuric acid before filter-pressing; this practice required extra labor, expense, and time, and has been discontinued, as there was no appreciable increase in the fineness of the bullion produced.

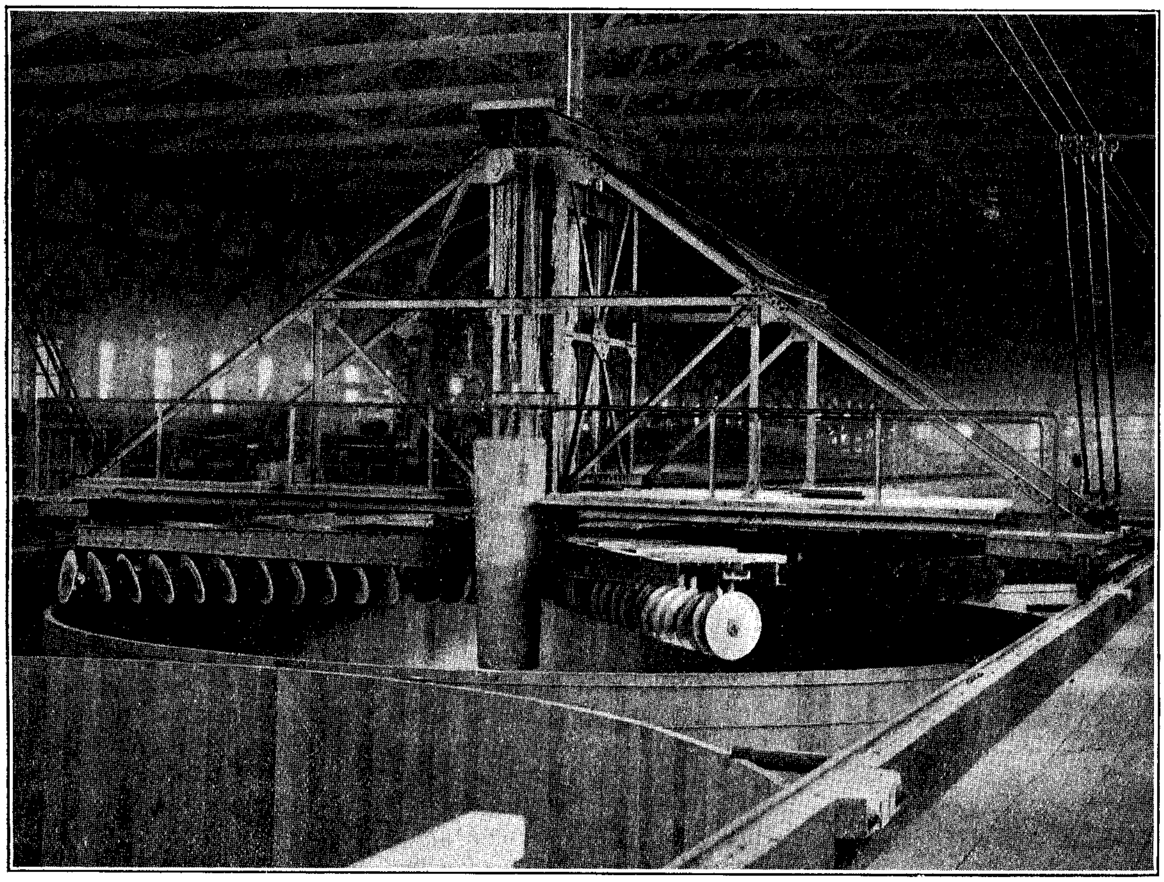
The melting is done in six Faber du Faur tilting fur-



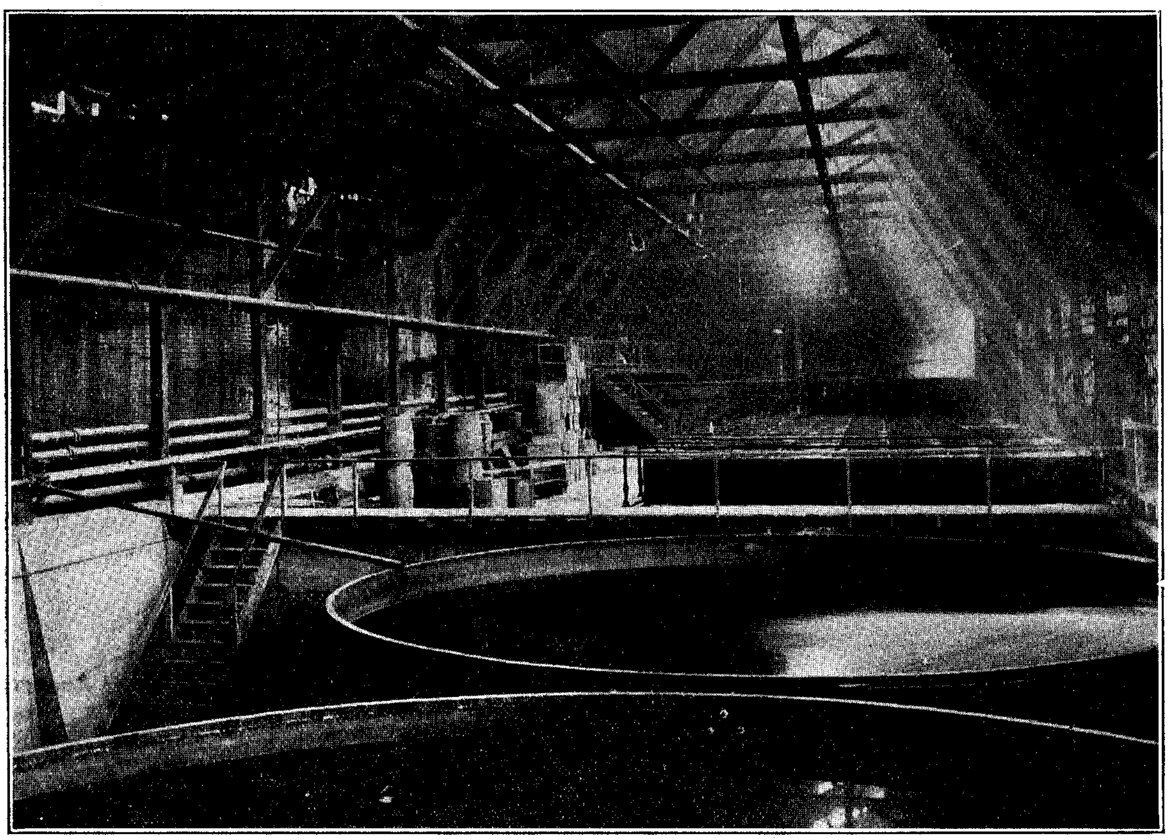
Stamps and Wilfley Tables at the Desert Mill.

The Butters Filter in the Desert Mill.

6



Blaisdell Excavator in the Desert Mill.



Sumps and Extractor Boxes.

naces, equipped with graphite retorts, with a capacity of 80 lb. fluxed precipitate. Coke is used as fuel and the first pour is made six hours after starting fires. At the present time, 2,800 lb. dried precipitate is melted into bullion in 24 hr. from the time of firing the furnaces. After a charge has melted down, sufficient precipitate is added to make a bullion bar weighing about 1,200 oz. Troy. From 70,756 lb. roasted precipitate, 47,442 lb. bullion were produced; that is, without acid treatment, 67.05% of the precipitate went into bullion. This bullion had an average fineness of gold 13.5, silver 965, total 978.5 per thousand.

Pours are made directly into bullion-molds placed upon slag-pots. The molds have a slotted overflow for slag. This procedure does away with re-melting buttons into bars, made necessary when pours are made into slag-pots. The graphite retorts in the furnaces will last for about 18 fusions, when they are discarded and new ones inserted.

Owing to the fact that the crushing takes place in cyanide solution and that the sand and slime are in contact with the solution from the time they enter the batteries, it is impossible to secure trustworthy samples of sand and slime separately and keep a record of extraction by cyanide on each product. Accordingly, the difference between the gross content of the ore and the gold and silver in the concentrate shipped, is taken as the gold and silver contents going to the cyanide plant. The extraction by cyanide is estimated from this by comparing with the total ounces of gold and silver shipped as bullion and refinery by-products, with the total contents in sand and slime residues as a check. The extraction by cyanidation from March 1 to August 1, 1907, figured in this manner, is 86.06% of the gold and 80.14% of the silver contents. Combining this with the extraction by concentration, as above, 15.63% of the gold and 30.32% of the silver, gives a total extraction by concentration and cyanidation of 88.24% of the gold and 86.20% of the silver in the ore.

The average consumption of chemicals per ton of ore, for the last five months, has been: Potassium cyanide, 3.24 lb.; lime, 7.20 lb.; lead acetate, 0.38 lb.; zinc shaving, 1.10 lb.; zinc shaving consumed per ton of solution precipitated, 0.32 pounds.

Changes and improvements in the plant now under way, such as increasing the concentration equipment by better classification, increasing the Butters filter capacity and improving the present slime agitation equipment, will increase the percentage of extraction and decrease the cost of operations.

BRAZILIAN QUARTZ CRYSTALS.—There has been considerable variation in the foreign demand for crystals of quartz of large size for optical work found in Brazil during the past few years and apparently the trade is not growing. The exports of all such crystsls from Brazil in 1904 was about $16,103. In 1905 it went up to a total of $18,132, but in 1906 it fell to a total of $10,553. Much of this fluctuation is due to the variation in the supply. There is a vast difference in the quality of the stones found and the trade varies accordingly. Most of the best stones seem to exist in a sort of circuit ranging from the central portion of Sao Paulo through Goyaz and the western portion of the State of Minas. There is a considerable supply of smaller stones and stones of inferior quality to be had in the Rio de Janeiro market at almost any time. The best stones generally are obtained by placing orders in advance. The price given as the average for exports last year was about 42c. per kilo. (2⅕ lb.) but this means little in view of the great variety of stones.

Use of the Divining Rod.

Numerous devices are used throughout this country for detecting the presence of underground water—devices ranging in complexity from the forked branch of witch-hazel, peach, or other wood, to more or less elaborate mechanical or electrical contrivances. Many of the operators of these devices, especially those that use the home-cut forked branch, are perfectly honest in the belief that the working of the rod is influenced by agencies—usually regarded as electric currents following underground streams of water—that are entirely independent of their own bodies, and many people have implicit faith in their ability to locate underground water in this way.

In experiments with a rod of this type, one of the geologists of the United States Geological Survey found that at points it turned downward independently of his will, but more complete tests showed that the downturning resulted from slight and — until watched for — unconscious changes in the inclination of his body, the effects of which were communicated through the arms and wrists to the rod. No movement of the rod from causes outside the body could be detected, and it soon became obvious that the view held by other men of science is correct — that the operation of the 'divining rod' is generally due to unconscious movements of the body or of the muscles of the hand. The experiments made show that these movements happen most frequently at places where the operator's experience has led him to believe that water may be found. The uselessness of the divining rod is indicated by the facts that the rod may be worked at will by the operator, that he fails to detect strong currents of water running in tunnels and other channels that afford no surface indications of water, and that his locations in limestone regions where water flows in well-defined channels are rarely more successful than those dependent on mere guesses. In fact, its operators are successful only in regions in which ground water occurs in a definite sheet in porous material or in more or less clayey deposits, such as the pebbly clay or till in which, although a few failures occur, wells would get water anywhere.

Ground water occurs under certain definite conditions, and as in humid regions a stream may be predicted wherever a valley is known, so one familiar with rocks and ground water conditions may predict places where ground water can be found. No appliance, either electrical or mechanical, has yet been successfully used for detecting water in places where plain common sense or mere guessing would not have shown its presence just as well. The only advantage of employing a 'water-witch,' as the operator of the divining rod is sometimes called, is that skilled services are obtained, most men so employed being keener and better observers of the occurrence and movements of ground water than the average person.

MINING LICENSES IN INDIA.—The Government of British India has decided to limit the areas granted by district officers under alluvial prospecting licenses to 1,600 acres, measuring not more than 5 miles in length except in cases of narrow belts of river gravel, on which extraordinary concessions may be granted to a length of 10 miles. The Government purposes, before the expiration of the newly granted licenses, to revise the existing general rules that govern the granting of prospecting licenses and mining leases in British India, and members of the mining community interested in the matter are invited to send to the Geological Survey department suggestions which they think should be considered before the revision is undertaken.

MILLING PLANT OF THE MONTANA-TONOPAH MINING COMPANY.

Written for the Mining and Scientific Press
By G. H. Rotherham.

The reduction of silver ores by wet milling and cyaniding, which has been studied to good purpose in Mexico, has had a comparatively limited field in this country. Doubtless the best practice is to be found at Tonopah, Nevada, where three well equipped mills are now successfully treating silver ores. The latest of these, the Montana-Tonopah Mining Co.'s plant, designed and erected under the supervision of the consulting metallurgist of the Company, F. L. Bosqui, and equipped by the Allis-Chalmers Co., possesses certain unique features which may be considered innovations in the metallurgy of silver. The ore is a gold and silver-bearing sulphide. The chief gangue-mineral is quartz, the vein-matter being a quartz replacement of the andesite. In the milling-ore the proportion of gold to silver by weight is 1 to 100. Some of the gold is undoubtedly free, the remainder being associated with the silver in the sulphide. The prevailing silver minerals are stephanite and polybasite; while beautiful specimens of ruby silver have been taken from the richer portions of the mine. Lead, copper, and zinc also occur in small quantity.

The ore, as it comes from the mine, is delivered to a 200-ton steel bin, which stands directly in front of the head-frame and adjoins the primary crushing plant. From this bin the ore is fed through a counterbalanced gate to a No. 5 K Gates gyratory breaker, which reduces it to 2-in. size. Thence it passes to a No. 5 B Gates bucket-elevator that discharges into an iron-frame revolving screen with one-inch perforations. The oversize from this screen is re-crushed in two No. 3 D Gates breakers to 1-in. size, this product then joining the undersize from the screen and passing to a 14-in. belt-conveyor, which conveys the ore to the mill-bins. This belt is 190 ft. long, and operates at an angle of 12°. It discharges onto a horizontal belt extending the length of the bins, the latter being provided with a tripper for distributing the ore. The crushing plant has a capacity of 25 tons per hour, and is operated only during one 8-hr. shift. From the bins the ore is delivered by eight suspended Challenge feeders to a battery of 40 stamps. The stamps weigh 1050 lb. and fall 100 times per minute through a 7-in. drop. The battery is the three-post, back-knee type, with wooden mortar-blocks, composed of 2-in. planks solidly spiked together and set on rubble concrete. The mortar is the Allis-Chalmers 'Homestake' narrow pattern, with extra heavy base. The battery is arranged in eight separate units, operating independently. The main counter-shaft rests low on the battery-sills, admitting of the prompt shut-down of any unit without hanging up the stamps individually. The shoes are of chrome-steel. The dies are made at a local foundry, and consist of 35% car-wheel scrap, ½% powdered manganese, and the remainder of chrome scrap. These dies give good satisfaction, whereas the chrome-steel dies were found to 'cup' badly.

The ore is crushed in cyanide solution through 20-mesh woven-wire screen, with the diameter of wire 0.016 in., and an aperture of 0.0173 in. The strength of the circulating mill solution is 0.13% cyanide of sodium. The average size of the crushed battery pulp is shown by the following screen analysis:

	Per cent.
Remaining on 30-mesh	2.53
Passing 30-mesh, on 60-mesh	25.5
Passing 60-mesh, on 80-mesh	4.8
Passing 80-mesh, on 100-mesh	6.1
Passing 100-mesh, on 200-mesh	16.0
Passing 200-mesh	45.07

From the battery the ore goes to eight 24-in. cone-classifiers, with 50° sides. The spigot-discharge feeds eight Wilfley concentrators, the slime and fine sand-overflow joining the main pulp-stream after concentration. The whole mill-stream is then elevated to two Dorr classifiers for de-watering and further classification, preparatory to tube-milling. The Dorr machines are doing satisfactory work, but, as manufactured, were too lightly constructed, and had to be extensively re-inforced. The best consistence of pulp for tube-mill purposes was found to be about 45% moisture. The excellent work done by these classifiers in separating a sandy product is shown in the following sizing tests:

PULP DISCHARGE FROM DORR CLASSIFIERS TO TUBE-MILLS.

	Per cent.
Remaining on 30-mesh	6.2
Passing 30-mesh, on 60-mesh	42.6
Passing 60-mesh, on 100-mesh	21.2
Passing 100-mesh, on 200-mesh	16.2
Passing 200-mesh	11.6
Loss	2.2

It is worthy of note that 70% of the pulp entering the mills is coarser than 100-mesh, and that the slime has been almost completely separated.

SLIME OVERFLOW FROM DORR CLASSIFIERS.

	Per cent.
Remaining on 100-mesh	2.5
Passing 100-mesh, on 200-mesh	15.5
Passing 200-mesh	82.0

The two tube-mills in use are the Allis-Chalmers, spur-gear driven, trunnion type, 5 by 22 ft., lined with 4-in. silex blocks. They make 27 r.p.m. The initial starting-load for one mill requires 60 hp., while both mills in simultaneous operation take 85. Each mill is now re-grinding 52 tons per 24 hr. This is the safe limit for this ore, considering the fineness of pulp desired for the best economical work in the cyanide plant. The size of pulp issuing from the tube-mills is as follows:

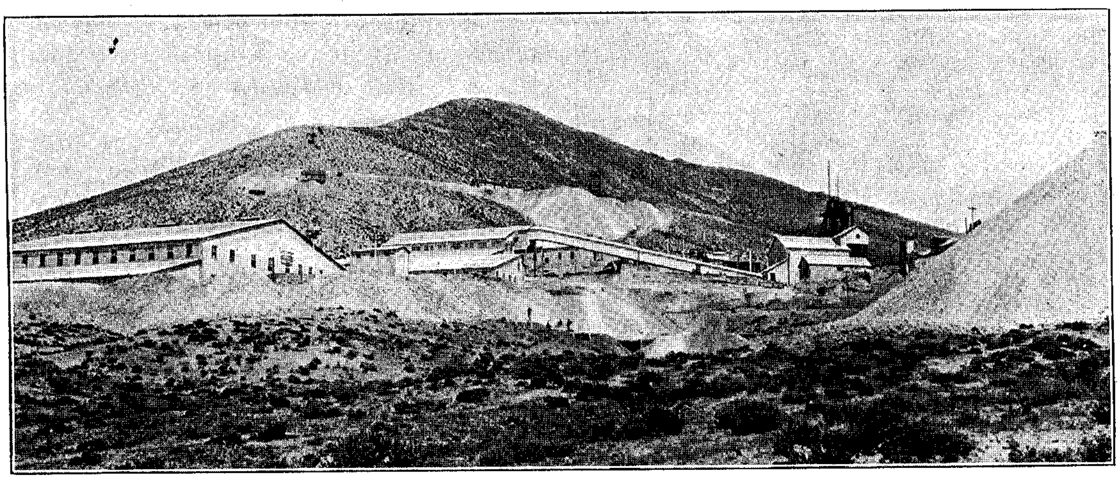
	Per cent.
Remaining on 60-mesh	1.0
Passing 60-mesh, on 80-mesh	1.5
Passing 80-mesh, on 100-mesh	4.0
Passing 100-mesh, on 200-mesh	27.5
Passing 200-mesh	66.0

The pulp from the tube-mills, after proper dilution, is re-classified in two 48-in. cones with 50° sides. The underflow from these is returned to the mills; the overflow joins the stream from the lower ends of the Dorr classifiers, and passes to three 8 by 54-in. Frenier sand-pumps, which elevate it to two thickening-cones, preparatory to secondary concentration on Frue vanners. This pulp, representing the final re-ground product, shows the following distribution of sizes:

	Per cent.
Remaining on 100-mesh	2.8
Passing 100-mesh, on 120-mesh	7.6
Passing 120-mesh, on 150-mesh	8.6
Passing 150-mesh, on 200-mesh	8.9
Passing 200-mesh	72.1

The Frue vanners, though not ideal slime-concentrating machines, do good work on this fine product. The concentrate from the Wilfleys and vanners is removed by traveling buckets to a drying-house adjoining the mill, where it is sacked for shipment to the smelter. From the lower concentrating floor the whole re-ground mill-pulp goes to the cyanide plant for treatment by agitation. All cyanide tanks are made of redwood, excepting the Hendryx agitators. These are constructed of Oregon fir, which, in my opinion, is inferior to redwood for cyaniding purposes. The mill-pulp, containing six parts of cyanide-solution to one of ore, is fed alternately to the centre of three settling-tanks 30 ft. diam. by 10 ft. deep at the centre, with a false-bottom sloping 12°. These tanks are provided with leveling rim, overflow launders, and decanting apparatus. By means of a series of decantations, the pulp is reduced to the proper consistence for agitation. Every twelve hours 75 tons of pulp, the contents of one settler, are transferred by a 6-in. Morris centrifugal pump to two of the agitators. There are six Hendryx agitators in use, 17 ft. diam. It requires 6 hp. to agitate a charge of 35 tons of ore (dry weight). The pulp in the agitator has a specific gravity of 1.28 to 1.32. A 0.2% solution of NaCy is used, and 0.3 lb. lead acetate per ton of ore. The agitation is continued for thirty-two hours. The pulp is then drawn through the same pump as mentioned above, and raised to a pulp-storage tank, 30 ft. diam. by 17 ft. deep, provided with stirring-arms making 8 r.p.m. From this tank the pulp is drawn as required into the Butters filter-boxes.

The Butters plant consists of two redwood boxes, of four compartments each. Each box contains 72 filter-leaves. Pulp and solutions are handled by an 8-in. Morris centrifugal pump, and a 12 by 10-in. Gould duplex vacuum-pump. The solution from the Butters filter is re-filtered in a 30-in. frame filter-press, whence it flows to the precipitation tanks. There are three of these, 14 by 14 ft. The zinc-dust is added as an 'emulsion' from a small feed-cone, and passes directly into the suction of the pump, which raises the solution to the Merrill precipitation-presses set up in a separate building, 100 ft. above the precipitation-tanks, and immediately above two 28 by 8-ft. solution-storage tanks. These two presses have triangular-shaped plates, 48 in. on edge. There are thirty plates in each press, with frames spaced 2 in. apart. The solutions are raised by a 5 by 6-in. Aldrich triplex pump. From the presses the barren solution runs to the storage tanks, whence it passes into the general mill-circulation. The present method is to draw from the circulating mill-supply of solution the quantity required for agitation purposes. This amount is brought to the required strength in the agitators, and, after precipitation, it passes again into the mill-circulation; so that there is really but one solution used throughout. The value of the mill-solution is kept low by precipitating about 150 tons of it daily. That portion of the mill-solution which is not used for extracting purposes overflows the rim of the settling-tanks almost clear, but is further clarified by flowing through a three-compartment, pointed, clarifying-box, each compartment being 10 ft. square at the top. Thence it flows to the mill-sump and is raised by means of an 8 by 12-in. Gould triplex pump to a 70,000-gal. concrete reservoir on the hillside above the stamps.



Montana-Tonopah Mill, Tonopah, Nevada.

Zinc-dust precipitation, as used by C. W. Merrill at the Homestake plant, South Dakota, and elsewhere, has been satisfactory in the practice at Tonopah from the first. It has the advantage of compactness, low cost for labor, and security from theft. The helper who tends the solution and water pumps gives a portion of his time only to the simple manipulation of the process. In estimating the cost of precipitation, I have not included the cost of raising the solution to the presses, inasmuch as this would have to be raised in any event for purposes of circulation; and the additional work on the pump imposed by the presses themselves may be disregarded, as the pressure at the presses never exceeds 8 lb. per square inch, and reaches this point only just before the clean-up, when the frames are loaded.

The precipitate is sacked and shipped by express to the smelter. It was found, by careful calculation, that the high cost of fuel, fluxes, and labor would not justify refining this product on the ground, although an average sample of 114 lb. refined experimentally without acid treatment yielded 966.9 oz. (Troy) bullion, of a total fineness of 946.59. The following figures refer to a period of eight months, from October 1, 1907, to June 1, 1908, during which interval 34,766 tons of ore was treated, and 95,657 tons of solution precipitated.

Zinc-dust—33,903 lb., at 8.2c. per lb.	$2,780.00
Filter cloth—1368 yd. twill, at 34.25c. per yd.	413.82
Labor—one-third time of one man, at $4 per 8-hr. shift	960.00
Total	$4,153.82

For cleaning up the two presses, twice a month, it requires the services of four men for one shift of eight hours, or $256 for eight months.

SUMMARY.

	Per ton of solution precipitated.	Per ton of ore treated.	Per oz. fine metals recovered.
Cost of precipitating.	$0.0430	$0.1190	$0.01400
Cost of cleaning up...	0.0026	0.0073	0.00086
Totals	$0.0456	$0.1263	$0.01486

The efficiency of the precipitation is shown by the following figures, giving average assays in the precipitation of 95,657 tons of solution:

	Heads, gold oz. per ton.	Tailing, gold oz. per ton.	Heads, silver oz. per ton.	Tailing, silver oz. per ton.
Strong solution	0.0352	0.00149	3.39	0.0605
Mill or weak solution.	0.0290	0.00216	2.80	0.1590*

*Excepting the month of March, 1908, when, owing to a series of breakages in a poor quality of twill-filters, the tailing from weak-solution precipitation averaged 0.58 oz. silver, this average stands 0.058 oz.

For these eight months the check between the recovery indicated by solution-assays and the actual recovery from the presses is shown in the following:

	Gold, fine oz.	Silver, fine oz.
Indicated recovery by solution-assays	2,986.31	290,715.36
Actual recovery from presses....	3,170.28	291,868.98

During the same period the average consumption of zinc-dust was as follows:

	Lb.
Consumed per ton of ore treated	0.970
Consumed per ton of solution precipitated	0.350
Consumed per ounce of metals precipitated	0.115

The total fineness of the precipitate, as taken from the presses, has ranged between 414 and 688, the average for the eight months being 517, or 51.7% precious metals. This product ranges between 12 and 20% zinc-content, and 10 to 15% silica. The presence of the latter ingredient is due to the extreme difficulty of completely clarifying the mill-solution before precipitation. To the eye these solutions appear perfectly clear, but they carry a small amount of solid matter, which is caught in the precipitation presses. A preliminary filtering system is now being installed, which will eliminate a great part of the silica, and considerably improve the grade of precipitate.

The cost of tube-milling from October 1, 1907, to June 1, 1908, works out as shown below. A 4-in. lining lasted from August 20, 1907, to April 20, 1908, or exactly eight months. During that time 31,835 ton was crushed by stamps, and 68% of this, or 21,511 tons, was re-ground.

Cost of power: 85 hp. for two mills, at $8, or $5400 in eight months.

Pebbles: 70,860 lb., at $50 per ton, or $1770 in eight months.

Silex lining: cost for two mills, using imported cement at $12.48 per bbl., $2216.92.

Labor: one-half of one man's time on each shift, at $4, $1440 in eight months.

Maintenance and repairs: on mills and Dorr classifiers, $79 per month, or $632 in eight months.

SUMMARY OF COST PER TON OF ORE STAMPED.

Power	$0.170	Maintenance and repairs	0.019
Pebbles	0.055		
Lining	0.069		
Labor	0.045		$0.358

Consumption of pebbles: 2.22 lb. per ton of ore stamped, or 3.29 lb. per ton of ore re-ground.

The following details on cost of re-lining one of the tube-mills may be of interest:

2500 silex bricks, at $0.302 per brick	$ 756.27
1 mason, 8 days, at $8 per day, lining mill	64.00
*1 mason, 10 days, at $8 per day, chipping bricks	80.00
2 helpers, 10 days, at $4	80.00
10.75 bbl. 'Heidelberg' cement, at $12.48 per bbl.	134.16
Sharpening tools	2.03
	$1,116.46

*This lining was composed of 2½-in. blocks set on edge to make a 4-in. lining, as there happened to be a large stock of the thinner bricks on hand. This made the lining unusually expensive. The mill was shut down 13 days for re-lining. The old lining wore down unevenly, being less worn at the tail-end of mill. The average thickness, when chiselled off from the shell, was about ¼ in. In places the thin cement-bedding had worn down to the shell of the mill.

The cost of filtering by the Butters process for the months of March, April, and May, 1908, was:

Labor: 1 man per shift, at $4, with occasional extras.

Maintenance and repairs: on filter leaves, filter-boxes, and pumps.

Acid: hydrochloric acid, for washing leaves.

Power: 42 hp. for operating circulating-pump, vacuum-pump, and stirrer in pulp-tank, at $8 per hp. per month.

SUMMARY.

	Labor.	Maintenance and repairs.	Acid.	Power.
March	$ 382.50	$ 6.75	$165.93	$ 336.00
April	360.00	259.06	163.43	336.00
May	364.00	142.60	160.43	336.00
Totals	$1,106.50	$408.41	$489.79	$1,008.00

Total tonnage for three months, $13,462.
Average per day filtered, 146 tons.

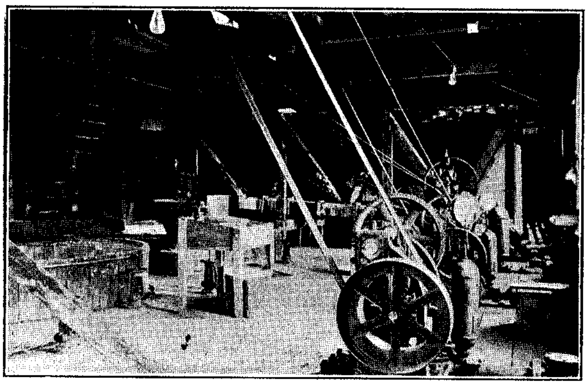
Labor per ton	$0.082
Maintenance and repairs	0.030
Acid per ton	0.036
Power per ton	0.082
Per ton of ore filtered	$0.230

CONSUMPTION OF CHEMICALS IN CYANIDE PLANT.
Pounds per ton of ore.

	Cyanide.	Zinc.	Lime.	Lead acetate.	Hydro-chloric acid.
Oct., 1907	4.98 KCy	0.91	8.64	*	*
November	2.40 KCy	0.90	7.50	*	*
December	2.09 NaCy	0.81	8.29	0.21	0.14
Jan., 1908	2.11 KCy	1.19	8.63	0.27	0.19
February	1.80 NaCy	0.76	8.32	0.31	0.33
March	2.05 NaCy	0.96	7.70	0.31	0.39
April	2.48 NaCy	1.14	8.20	0.31	0.40
May	1.98 NaCy	1.10	7.55	0.27	0.40

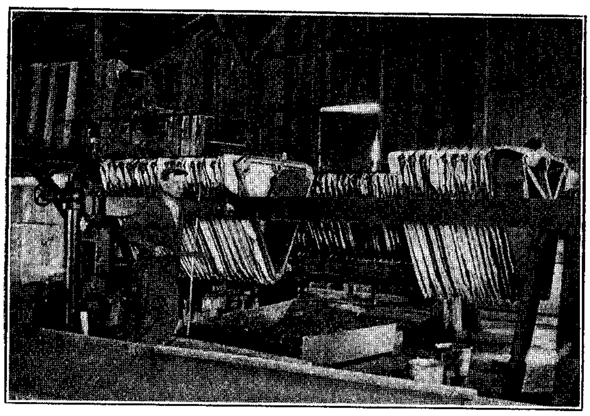
*None used.

It may be stated that although this mill has made



Pump Room, Cyanide Plant, Montana-Tonopah Mill.

the record on costs for this district, all the costs are in reality high. Water, power, and labor are expensive items, and freight rates are exorbitant. It would therefore be idle to compare these conditions with those obtaining in other countries, as in Mexico, for example. The following figures are offered merely



Zinc-dust Precipitation Presses.
Showing part of cake in place at time of clean-up.

to illustrate a few of the difficulties incident to milling operations in Nevada:

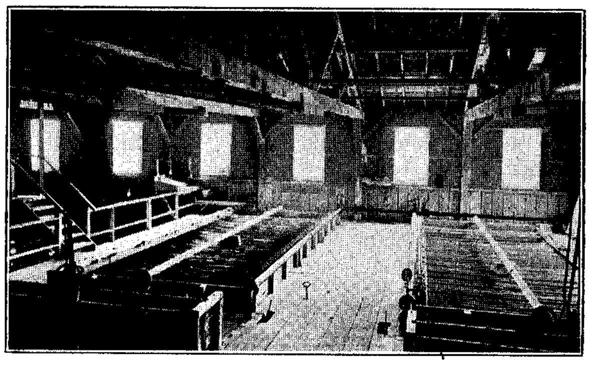
	Tonnage treated.	Cost per ton, milling and cyaniding.
October, 1907	4120	$4.25
November	4410	3.79
December	4256	3.58
January, 1908	4135	3.59
February	4383	3.59
March	4701	3.28
April	4303	4.08
May	4458	3.67

These figures do not include insurance and depreciation on plant.

In explanation of the above costs, the following items may be of interest:

Average cost of water per ton of ore treated	$0.287
Average cost of power per ton of ore treated	0.808
Average cost of labor per ton of ore treated	0.943
Total	$2.038

Crusher-men, battery-men, concentrator-men, and



Butters-Filter Plant.

solution-men receive $4.50 per shift of 8 hr.; filter-men, $4; laborers, $4; and carpenters, $6.

EXTRACTION.

(By concentration and by agitation in cyanide solution.)

	Average mill-head. Gold, oz.	Silver, oz.	Average tailing Butters filter. Gold, oz.	Silver, oz.	Extraction by bullion-yield. Gold, %	Silver, %	Total %
Oct., 1907	0.153	15.54	0.0039	2.18	97.55	85.5	88.76
November	0.135	14.97	0.0084	2.36	94.30	84.6	87.40
December	0.133	13.90	0.0080	2.14	94.30	85.4	87.70
Jan., 1908	0.163	16.34	0.0106	2.41	94.00	86.6	88.50
February	0.186	18.36	0.0105	2.01	93.50	88.0	89.40
March	0.240	23.66	0.0140	3.09	93.80	86.9	88.80
April	0.206	20.20	0.0100	1.64	95.20	89.7	91.10
May	0.219	20.44	0.0129	2.28	93.40	88.6	89.80
June	0.172	16.67	0.0110	1.80	93.40	89.7	90.60

Of this extraction approximately 40% is obtained by concentration, and the remaining 50% by cyanidation.

Coke production in the United States in 1907 amounted to 40,779,564 short tons, valued at $111,539,126, a total that passes all previous records in the history of coke-making in this country, being nearly double the output of 1900, and more than three times that of 1897. The increase over the production of 1906 was 4,378,347 short tons, or 12.02% in quantity, and $19,931,092, or 21.76% in value. The average price per ton at the ovens, $2.74, is greater by 22c. than the 1906 average, and is the highest reported in the 28 years during which statistics of coke production have been compiled by the U. S. Geological Survey, exceeding by 11c. the maximum rate previously obtained in 1873.

Alumina in silicates is subject to unexpected error in analysis, as shown by Hinrichsen. On decomposing silicious minerals with HF and H_2SO_4 and subsequently conducting the analysis on the silica-free residue, a deficiency in Al_2O_3 was noted, which was found to be due to partial volatilization of the Al as a compound with fluorine. Some fluorine also was retained in the mineral.

THE NATION'S DEPENDENCE UPON NATURE.

Written for the MINING AND SCIENTIFIC PRESS
By GEORGE OTIS SMITH.

*President Roosevelt's State papers have again and again contained stirring appeals to the lawmakers to make adequate provision for the conservation of the nation's natural resources. No other Chief Executive since Washington has given equal prominence to this sort of practical statesmanship. He has seen clearly that American natural wealth is not inexhaustible. The President's well-considered appeal should arouse a new type of patriotism no less than did Lincoln's call to arms inspire a new devotion to the nation. It was not a frenzied alarmist, but a hard-headed man of science who wrote that "future historians will date the end of barbarism from the time when generations begin to feel that they rightfully have no more than a life-estate in this sphere, with no right to squander the inheritance of their kind." The reform that is so sorely needed must gain its motive-power not from the well-intentioned enthusiasm of the few, but from the well-informed intelligence of the many. The campaign for conservation must be one of education. The United States leads the world in every important form of natural wealth, whether it be the product of the farm or of the mine. Our wheat crop forms 21% of the world's harvest, our corn crop 78, and our cotton crop 82%. So, also, in the matter of live stock, our herds and flocks comprise 17% of the world's grand total. To take the more important minerals, our output of coal is 40% of the world's production, of copper 50, of iron-ore and oil at least 60%. The increase in our per capita production and consumption of natural resources perhaps furnishes the best index of the increasing dependence of the nation upon its natural resources. At the outbreak of the Civil War the per capita consumption of coal in this country was only a little over one-half ton; in 1900 it was 3½ tons; but in 1907 the consumption had increased to 5½ tons. In 1850 our per capita consumption of lumber was 250 board feet; today it is undoubtedly close to 500 board feet. In 1870 our per capita production of iron-ore was 160 lb.; last year it exceeded half a ton.

The most valuable of all natural resources are water and soil. No more attractive field for increasing the country's available wealth lies before us today than that of determining the possibilities of our rivers and streams. Here we shall find the solution of the problems of improved inland water-transportation, successful agriculture, cheap power-development, and adequate protection from floods. The rivers of Maine now turn wheels that furnish a total of somewhat more than 210,000 hp., and with proper storage facilities the full development of her water-powers might yield 1,200,000 hp. This conservative estimate of the importance of Maine's water-power is a fair index of the value of the nation's water-resources, by far the larger part of which is yet undeveloped. Great soil-loss is occasioned by tillage methods which permit sufrace washing. The soil-waste thus started by mere negligence increases until the slopes are gullied, the bottom lands covered with waste, the streams choked, and the navigable rivers filled with bars. So universal and persistent is this drain upon our resources that it is estimated that at least one billion tons of the nation's richest soil is annually carried into the sea by our rivers. The subjects of soil and water-resources cannot be discussed without mention of the conserving influence of the forest. The records of flow of many rivers too plainly show the effect of removing the forest-cover. There is a vital relation between the forest and agriculture, manufacture, and navigation. As the great conserver of other resources the forest deserves the nation's care.

Recent surveys of the nation's coalfields indicate an area for the more accessible coals of 327,000 square miles. This is four times the area of the known coalfields of the rest of the world. In these vast coalfields the nation has available nearly 2000 billion tons of coal mineable under present conditions, or twice the tonnage estimated for the rest of the world. Apparently America's supply is so great as to render immediate alarm unnecessary, but if the present phenomenal rate of increase in consumption is maintained, the supply of easily and cheaply mined coal will be gone before the middle of the next century. The waste of our natural gas supply has been estimated at 1,000,000 cu. ft. daily, or the equivalent of 10,000,000 tons of coal per year. Nor is the record for petroleum much better. In the case of coal there is waste at every stage of mining and consumption. For each ton of merchantable coal brought to the surface at least half a ton is wasted by being left in the ground, or thrown upon the culm pile. Great also is the loss by uneconomical consumption. Less than 10% of the heat units of coal are utilized under the ordinary steam boiler, and in the locomotive the waste is said to be not less than 95%. The record of waste, however, before which every American should stand aghast is that of human life in the coal mines. So far reported, with several States unheard from, 3124 coal miners were killed last year, or an increase of 50% over 1906. It is a matter of some gratification, however, that at last the nation has made some provision for investigations that may prevent or lessen mine disasters.

The iron-ore production last year, reaching 51,000,000 tons, shows an increase of 8% over 1906 and 21% over 1905. Even should the present output continue without further increase, the known reserves will be exhausted within the present century.

The future of metal mining will be governed by the improvement of metallurgical and mining methods and by the available supplies of fuels, water, power, and mining timber, as well as by the discovery of new mining districts. In the determination of the economic value and availability of ore deposits, the fuel and water supply is a more important factor than the character of the ore itself. In a real and large sense the very life of our mining industry depends upon the conservation of all the other natural

* Abstract of Phi Beta Kappa oration of George Otis Smith, Director U. S. Geological Survey, at Colby college, Waterville, Maine.

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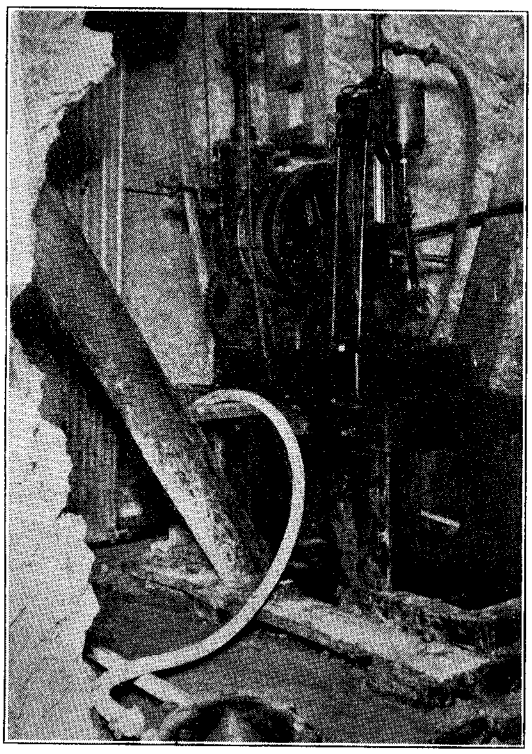
DIAMOND DRILLING AT TONOPAH.

Written for the MINING AND SCIENTIFIC PRESS
By JOHN M. FOX.

In the autumn of 1908 it was decided to put down a vertical drill-hole from the bottom, or 700-ft. level, of the Silver Top mine to gain information regarding the formation through which the Silver Top No. 1 shaft would have to pass if sunk to greater depths. A station and raise were cut close to the shaft and the drill installed. The height of the raise was sufficient to enable the runner to pull in 20-ft. lengths. A suitable sump was cut near by for return-water from the hole, thus permitting the use of the same water repeatedly, with the exception of what was lost in the hole.

The drill was a Sullivan Class 'C', provided with a hydraulic head and chuck for size 'A' rods, capable of drilling to 1500 ft. The engine, drum, and head were mounted in the usual manner, on a standard frame, which was held in place by two heavy sprags wedged to the wall of the station. Compressed air at 90 to 100 lb. pressure was used for motive power, it being tapped from a main-line through a 1-in. connection to the engine. As the level is dry, water for the drill was drawn from the water-pipe line in the shaft. Since a 700-ft. head would have provided a higher pressure for the drill than was desirable, a break in the line was made at the 400-ft. level. A large sheet-iron tank was installed at that point, and the supply-pipe from above was provided with a valve that was opened and closed automatically by a float. From the tank to the drill approximately 300 ft. gave about 123 lb. water-pressure, sufficient for the ordinary operation of the feed. In actual running, the hydraulic head was supplied entirely from this source, proving perfectly satisfactory, as it gave clean water at a constant pressure with no entrained air. The latter is objectionable, giving rise to 'jumping' in the rods, with attendant extra wear on the diamonds. As stated above, the water at the bit is used many times. As it flows from the collar of the hole, it is collected in the sump, and from there pumped by a small duplex back through the rods. The supply, of course, has to be replenished from time to time, as some water is lost.



Sullivan Drill Undergronnd.

At the start, the hole was reamed to a depth of 3 ft., and a 2-in. inside diameter pipe wedged in it. This projected high enough above the collar to allow the sludge-board to be placed, when actual drilling began. Size 'A' rods (standard), having an outside diameter of $1\frac{5}{8}$ in., were used. To these were strung a 10-ft., size 'A,' plain core-barrel, straight core-shell for 'Cossette' core-lifter, and size 'A' bit. The latter has an outside diameter of $1^{25}/_{32}$ in. With the carbons having a total offset of $^{1}/_{32}$ in., the diameter of the hole, assuming a perfect bore, would be $^{13}/_{16}$ inch.

Some trouble was experienced with the 'Cosette' core-lifter on account of the breaking of fingers, due probably to the fact that fragments of rock would wedge between the fingers and the frame on which they were mounted. As the core rose through the lifter, some irregularity on it would tend to force the fingers back against the springs which hold them out toward the centre of the core. Being unable to retreat because of the rock behind them, they would snap off. When a core-lifter is working properly, it would have little or none of the rotation of the rods, but should slip easily within the core-shell. Its relation to the core, when the latter is not broken off to the bottom, should be that of a stuffing-box to a piston-rod. Of course, such a perfect adjustment is unattainable, but it should be as closely approached as possible, otherwise the gripping edges, no matter what type of core-lifter is used, will be rapidly worn away and will lose their ability to hold the core when the time comes to pull.

The diamonds or 'carbon' in use averaged, when new, about $3\frac{1}{2}$ carats each. They were of the best quality, and at the existing market price cost $75 per carat. Six stones were set in each bit, three inside and three out, with a $^{1}/_{64}$-in. offset. The life of a new bit varied greatly in this hole; some drilled 35 to 40 ft. before becoming played out, others were retired after drilling only 2 or 3 ft. Many causes contributed to this difference in life. Vibration in the rods causes great wear and tear on both stones and metal. Short-fissured ground makes bad drilling, and variable pressure at the feed, which is hard on the bit.

Too great emphasis cannot be laid upon the advisability of buying the best carbon. Unless one is an expert by reason of long experience in judging stones, the setter should do the picking. There are few arbitrary rules by which a stone can be judged as to its worth. Its shape is almost as important as its quality, and the setter, if he be competent, is certainly the best judge of that characteristic. There are unscrupulous dealers in carbon, as there are in every line, and they have their ways of deceiving, as have the rest. Some stones are doctored to give them the proper color, porous ones are treated to give them weight, and so on. These deceptions can be avoided by dealing with responsible houses, the hard point being to pick the best stone from a number of good ones.

On account of certain conditions obtaining in the case of this particular hole, but one 8-hour shift was employed. The crew consisted of a runner and helper. As is customary in drilling, no stop was made at the lunch hour, the helper running the machine through that time. The shift went on at 7 a. m., and ceased at 3:30 p. m., unless it was deemed advisable to work through to 5, which was done many times. The runner set the bits after 3:30, and whatever repairs had to be made on the drill or pump were attended to by the helper after that time. All such overtime was credited to the crew on the scale of straight time, being figured to the nearest quarter shift.

With this outline of equipment and labor, we will consider the hole itself. Everything went smoothly, till at a depth of 87 ft., the character of the formation penetrated changed. The ground became broken, and considerable gouge kept cutting off the water. When solid formation was again reached, the hole was washed as clean as possible and portland cement mixed with water to the consistence of thick cream, was poured into the hole; 72 hours were required for the cement to become hard enough to drill. Some other soft spots were struck that gave similar trouble, and these were treated in the same way. Thus by cementing and drilling out, the hole was lowered to 225 ft., the last 15 being in firm ground.

Each morning it was found that the hole was caving worse than the preceding day, and more time was consumed in reaching bottom, the rods often having to be rotated while washing their way down. Finally at 225 ft. the rods stuck fast; hydraulic and hoist failed to start them, so an improvised jam was made by slipping a 5-ft. length of 6-in. pipe over the rods; to the pipe was fastened a rope leading over a sheave, and down. The pipe was snapped up, striking a clamp bolted securely to the rods; the latter were thus released, and when all were out, it was decided to ream to 210 ft. and case with 2-in. inside diameter pipe. This was done in the usual manner, using a reaming-bar, bit, and pilot, and following with casing instead of rods. After the last run was made, the bar, bit, and pilot were removed. A few carbon chips were set in the face of the bottom length of casing, and the latter sent home to its shoulder at 210 ft. The chips pro-

TABLE OF DIAMOND DRILL RESULTS AT TONOPAH, NEVADA.

Hole No. .... Started.............. Bottomed...........
Location............
Direction............
Elevation of collar of hole Ref. to collar of...........Shaft..............
Elevation of collar of hole Ref. to sea-level...........
Elevation of bottom of hole Ref. to collar of.........Shaft..............
Elevation of bottom of hole Ref. to sea-level...........
Length of hole..................
Diameter of hole.............. ....Rods.
Type.................

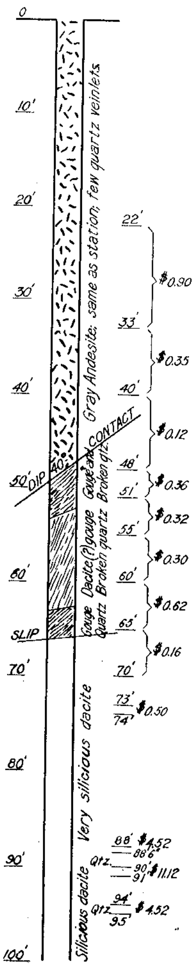
Log.						Assays.					
Date	Core Recov. Ft. In.	Drilled	Total Depth	Formation	Remarks	No.	From	To	Oz. Au	Oz. Ag	Total Value.

Month	Feet Drilled	Number of Shifts	Average Progress per Shift	Core Recovered, ft.	Core %	Labor	Power	Cement	B. S. Shop Airline, etc., Supplies Only %	Misc. Supplies	Bits Casing	Diamond Loss	Water	Total for Month	Total Cost of Hole at End of Each Mo.
September, 1908	0	0	0	...	...	102.25	...	0	0	73.93	0	0	0	176.18	176.18
Per Ft	...	...	...	...	...	...	...	0	0	...	0	0	0	...	...
October, 1908	61	7	8.7	12.0	19.5	299.47	57.44	0	30.16	33.20	0	192.20	4.80	617.27	793.45
Per Ft	...	...	...	...	...	4.91	0.94	0	0.49	0.54	0	3.15	0.08	10.12	13.00
November, 1908	114	24	4.75	36.5	32.0	352.14	130.08	7.44	102.92	3.07	35.75	159.38	18.76	809.54	1602.99
Per Ft	...	...	...	...	...	3.09	1.14	0.07	0.90	0.03	0.31	1.40	0.17	7.10	9.16
December, 1908	74	27	2.74	22.5	30.0	375.45	144.33	3 72	92.20	0.56	103.00	93.75	11.75	824.76	2427.75
Per Ft	...	...	...	...	...	5.07	1.95	0.05	1.25	0.01	1.40	1.28	0.16	11.14	9.75
January, 1909	109	25	4.36	55.0	50.0	337.14	108.15	0	31.81	62.42	46.93	187.50	8.38	782.33	3210.08
Per Ft	...	...	...	...	...	3.09	0.99	0	0.29	0.57	0.43	1.72	0.08	7.17	8.96
February, 1909	190	24	7.92	78.5	41.0	342.89	110 63	0	12.78	5.38	39.49	140 62	13.97	665.76	3875 84
Per Ft	...	...	...	...	...	1.80	0.58	0	0.06	0.03	0.21	0.74	0.07	3.50	7.07
March, 1909	129	26	4.96	47.0	36.5	372.02	160.42	5.00	24.83	22.36	...	384 37	0	969 00	4844.84
Per Ft	...	...	...	...	...	2.88	1.24	0.04	0.19	0.18	...	2.98	0	7.51	7.55

High diamond-loss in March due to breaking two carbons.
Power costs, $10 per horse-power per month.
Runners' wages, $6 per shift.
Helpers' wages, $4 per shift.
Water consumption approximates 4000-5000 gal. per shift. (All but 400 to 500 gal. lost in hole.)

vided a means of cutting away slight obstructions that might get in the way while lowering the casing for the last time. They were worthless for any other purpose, and hence were left in the hole. From here on, work progressed without serious trouble, occasional soft streaks being cemented; at 411 ft. the water was lost completely; in fact, it ran away so rapidly that the sump was siphoned dry. Sawdust and bran proved ineffectual in plugging the outlet, and until cement was used, the water continued to escape.

Experience gained in cementing sections of this hole may prove of use to others. In the first place, certain brands of cement are more adaptable to this work than others. Those reaching their initial and final set the most rapidly, are obviously the best on account of time saved. The character of the ground being drilled may cause more or less alkalinity in the water. This seems to retard the setting of some cements more than others. In making the mixture it was found that the thicker it was, the better; the limit to which the thickness may be carried, depends upon whether the cement is to be poured or not. For comparatively shallow holes, pouring answers very well; for deeper holes, placing thick cement in paper tubes about 12 to 14 in. long, and dropping into the hole, has been tried with success. Three important things to be observed to secure the best results are these: first, get the hole dry, if possible, by blowing; second, tamp the cement frequently while putting it down, which can be done by plugging the end of the rods and giving them a slight drop on the brake (of course, it takes much more time than simply pouring in the cement and allowing it to set, but the results are more certain, and in the long run, time will almost always be saved); third, if a brand of cement prove unsatisfactory, try others. One can probably be found to do the work.



Drill Record Plotted.

Vibration of the rods, which is mentioned above, can be greatly reduced, if not eliminated, by coating them with crude petroleum when lowering. This provides a lubricant that clings in spite of the scour of water and sludge.

Record of hole: there are probably as many systems of keeping the log of a diamond-drill hole as there are districts in which such work is being done. The following figures will show one such system, which is really in triplicate. The first record is composed of type specimens selected from each day's run; if the core shows no variation for that day, one piece is taken to represent the rock drilled; any change or changes shown in the hole are recorded by corresponding pieces taken from the core; each piece has painted upon it the depth it represents. The diagram of the hole is drawn to a vertical scale of 5 ft. to the inch, with the width greatly exaggerated. A plate is made for each 100 ft. drilled; on these plates are recorded depth, formation, assay values, and eventually the petrographic decription of the rock, wherever it has been deemed advisable to have slides made, will be added. The ledger record explains itself.

Assays: sludge-samples were taken from every run and assayed. Core samples were taken from time to time as a check, or wherever it was thought necessary by reason of the appearance of the core. It was realized that sludge-samples are only indicative of the presence or absence of valuable material when drilling under such conditions as the above. No reliability can be placed upon them quantitatively for many reasons: for example, a high-grade streak of narrow width would salt a 10-ft. run. If one succeeded in coring the streak and its adjacent walls, reliable information would have been gained in regard to where the valuable material came from, but without such proof, no certainty could be felt as to whether the ore were 10 ft. wide, and low in value, or one or two inches wide, and high in grade.

Costs: to one unfamiliar with conditions in Tonopah, the cost of drilling will seem strangely high. Labor and power, the two main items, probably reach a maximum there, yet in spite of this, in one month when 190 ft. were drilled, the cost per foot was only $3.50. In calculating costs for this hole, nothing was omitted that should be charged against it, and in fact, in one or two instances, it has borne unjust charges through an imperfect understanding of the conditions. The following list of charges gives an idea of how the cost was arrived at: (1) power (based on the drill's proportion of total air consumption); (2) labor; (3) diamond loss; (4) water (for the first four months only; after that, the 200-ft. level made enough water to supply both diamond and machine drills); (5) assaying; (6) supplies (includes pipe, bits, tools, cement, oil, etc.); (7) proportion of compressor repairs; (8) machine-shop work done directly on drill and equipment.

The table on page 263 will give a condensed statement in regard to progress, costs, and the like. It was kept by me for my immediate use, and has been of service in supplying quick information regarding such headings as it contains.

The Western Australian Mining Law is based on two fundamental principles: (1) that land shall be utilized for that purpose for which it is most valuable, and (2) that no man may hold mineral rights without development. Enforcement of the latter leads to the provision that none of the minerals to which the Government holds title shall be sold, but the Government may authorize the working of the mineral deposits by those conforming to the requirements of continuous development.

Water is commonly taken as weighing 62.5 lb. per cubic foot. The actual weight is 62.425 lb., at a barometric pressure of 30 in. and a temperature of 39.1° Centigrade.

to thaw the dynamite. The author, in conjunction with officials of the Aetna Powder Co., built and tested an experimental thaw-house at Aetna, Indiana. The house is about 5 by 5 ft. inside, and is divided for convenience into two compartments for storing the material to be thawed, and a separate compartment for heating the air. The space to be used for the dynamite is so arranged as not to allow room for a man to enter, requiring the attendant to remain outside the building while handling the trays, thus avoiding danger of overturning a lamp in the building. The air is heated by passing over the steam coils, and a positive circulation is secured by a stack, which has an effective height of 15 ft. The quantity of air admitted to each compartment is regulated by dampers with circular holes in the bottom of the thaw-house, and the draft is regulated by a damper in the stack. The trays containing the dynamite to be thawed are staggered on their supports, presenting a series of baffles; these prevent the air from short-circuiting, and force it to come in contact with all the trays. The house is covered with galvanized iron to protect it against creeping fires in grass or underbrush. A $^3/_{16}$-in. steel plate over the door, and gravel filling between the walls, form protection from stray bullets. Gravel is preferable to sand for filling, as the heat tends to shrink the wood on the inside, giving rise to cracks through which the sand escapes. Thermometers placed inside the house, but so arranged that they can be read from the outside, allow the attendant to keep the temperature under control. A temperature of 80°F. is considered desirable for thawing the dynamite. Good grades of gelatin dynamite may be thawed safely at higher temperatures, but with low grades of dynamite the nitro-glycerol is liable to separate if heated much above this temperature. The box containing the steam-coils is lined inside and outside with galvanized iron to protect it against fire. Where exhaust steam is used, about 70 sq. ft. of heating surface is required to raise the temperature from zero to 80°F. The capacity of the thaw-house is 540 lb. and it cost $165.

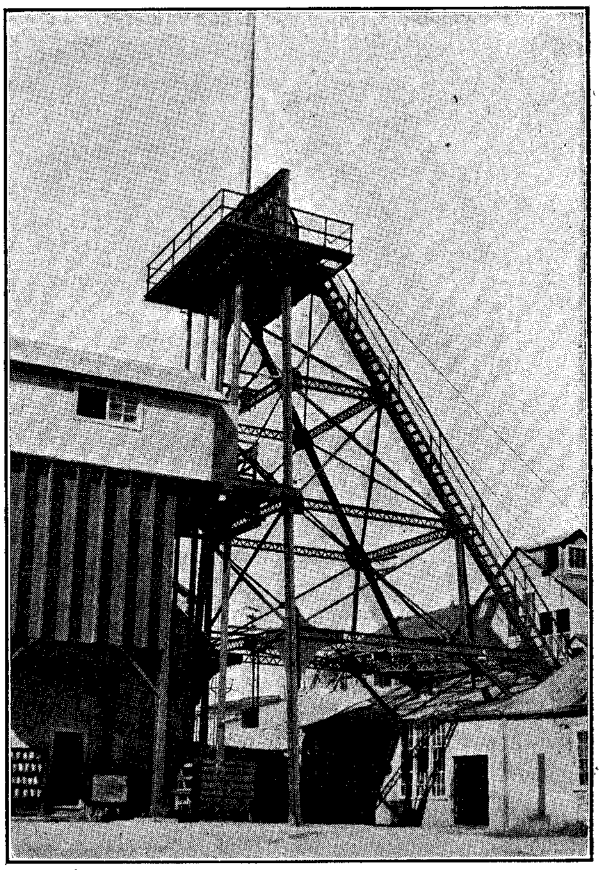
As a contribution to the literature of accidents from the use or abuse of explosives, attention may be properly called to the sixth annual report on Fourth of July injuries which appeared in the *Journal* of the American Medical Association, September 5, 1908. The total number of killed was 163, and of injured 5460. The number of cases of tetanus from wounds was 76. The statistics are carefully analyzed, and measures to prevent this foolish waste are proposed.

Oregon consumes annually about 1,200,000 bbl. portland cement. Of this quantity about 270,000 bbl. is of domestic manufacture, two-thirds coming from California. Only one cement plant, that of the Washington Cement Co., at Concrete, exists in Oregon, which began operations in 1907. It has two 100-ft. rotary kilns, crude oil being used for firing. The limestone comes from quarries at Concrete, and the clay is derived from the banks of Baker river. The cement is rather high in silica, containing 24%, in this respect resembling the cement made at the Vulcanite works in Pennsylvania. It is below 2% in magnesia.

GEOLOGICAL AND PHYSICAL CONDITIONS OF TONOPAH MINES.

By Walter P. Jenney.

*A cluster of low volcanic peaks marks the site of the mining camp of Tonopah. It has in the past been an area of repeated eruptions of lava, beginning in the early Tertiary and continuing until comparatively recent geologic time. With the dying out of the eruptive phase of volcanic activity, there followed a period of intense dynamic action, accompanied by the formation of deposits of gold and silver ore. So recent has been the cessation of volcanic outbursts in this district that the rocks, like the eruptive formations of the Comstock, still retain a portion of their original heat. Notwithstanding the comparatively late deposition of the ore, there are evidences in the mines that the formation of the deposits extended over a long period, with many disturbances and interruptions.



Montana-Tonopah Shaft.

The order of eruption is the earlier andesite, the later andesite, and the rhyolite. The earlier andesite is white or light gray in color, soft, breaking with a rough fracture; the later andesite is darker, weathering near the surface with a deep brown or purple tint, so that it is usually readily distinguished from the earlier andesite; the rhyolite breaks with a fine-grained, splintery fracture, and is commonly light greenish-gray. The general dip of the andesites is to the north, at angles usually from 15 to 45°; the rhyolite sheet, being intrusive in the older lavas, often does not conform to this dip, and in some areas is nearly horizontal. The earlier andesite is the great

*Abstract from the *Tonopah Miner*.

ore-bearing formation of the district, and for this reason is called by the miners the 'lode-porphyry'; the rhyolite in places carries workable deposits of ore; while the later andesite, known as 'cap-rock', is usually barren. Owing to causes not necessary to discuss, the veins are larger and more regular where traversing the earlier andesite, and often contract in width, or pinch out altogether on entering the rhyolite. In some instances, however, the veins pass from the lode-porphyry into the rhyolite without material change in breadth or richness.

The favorable nature of the earlier andesite as a wall-rock is shown by the production; more than 90% of the ore extracted from the mines of Tonopah has been stoped from veins in that formation. In this the ore deposits conform to the general law of selective deposition, namely, that some geological formations appear to be everywhere barren of ore; others occasionally carry small deposits, workable where the conditions are exceptionally favorable; but in each mining region certain strata are ore-bearing in a degree exceeding all other formations combined.

The earlier andesite appears at the surface in a limited area, in ground owned by the Midway and by the Tonopah Mining Co., where it extends to a depth of 500 to 600 ft.; elsewhere it is capped by the dark-colored later andesite, or by flows of rhyolite. Beneath these capping formations the earlier andesite has a much greater areal extension and stretches east and west in a broad belt, extending from the Belmont mine, at the extreme east of the district, to the Tonopah Extension and Golden Anchor properties on the west, a distance of nearly a mile. In the Tonopah and Midway workings the belt is developed over a breadth of 1000 feet.

In prospecting the Tonopah mines the problem is largely the exploration of the lode-porphyry, or earlier andesite. Owing to the fact that all the ore deposits known in the district are subsequent in formation to the intrusion of the youngest rock in the series, the rhyolite, in following the veins in depth, certain places are found where conditions have been such that the ore departs from the earlier andesite and cuts through the underlying rhyolite. In the ascent of the solution toward the surface the veins usually terminate on reaching the cap-rock; from this cause many of the productive veins are 'blind' and do not appear at the surface. Only on the Tonopah and Midway properties, where the earlier andesite-flow covers the surface, do the veins outcrop with pay-ore. Other orebodies spring from the contact of the rhyolite with the earlier andesite (which forms the foot-wall or south boundary of the lode-porphyry belt), and extend upward in the andesite. In the West End and MacNamara mines, sheet deposits of high-grade ore occur on this contact. Some of the larger veins, when followed on the dip, go down in a series of flats and pitches (like a flight of giant steps), in which the horizontal travel of the vein to the north is greater than the vertical descent on the normal dip. This tends to carry the ore-bearing fissures toward the north side of the earlier andesite belt.

There is a peculiar early-formed fault-fissure, filled with volcanic mud, which cuts through all the igneous flows and outcrops at the surface, from the Belmont mine west to beyond the Midway, known as the Mizpah dike, and while not ore-bearing, it has influenced the deposition of orebodies along its course. In the deeper levels of the mines there is evidence that the waters which formed the ore in some instances came up through channels alongside the dike.

The ore formation may be described as a broad belt of fissure veins, often closely spaced, traversing the district in a general east and west course. The extent of this mineral belt is undetermined; continuous stopes and connected levels reach from the Belmont mine, westerly through the Tonopah, Jim Butler, Montana, Midway, MacNamara, West End, and Tonopah Extension mines. With present development, ore deposits are opened along the course of the belt for nearly a mile, and are distributed over a breadth, north and south, of 3000 ft. The veins vary widely in width and often subdivide into branch veins, which not infrequently equal in size the original fissure. The great output of the camp is drawn from stopes averaging 4 to 5 ft. wide.

The ores are largely quartz, carrying silver glance (argentite), ruby silver (pyrargyrite), and the so-called 'brittle silver' (polybasite), together with chalcopyrite and other sulphides bearing gold and silver. The ratio of silver to gold in the ores is quite uniform throughout the district, being 90 to 100 oz. silver to 1 oz. gold; in the bullion produced, about one-third the value is gold. The grade of the ore as sent to the mills ranges in assay value from $12 to $50 per ton. The Tonopah ores resemble in mineral character and in the ratio of gold to silver those of the Comstock.

The Tonopah Merger Mining Co. owns the Salutation, Limerick, Limerick No. 2, Sky, Sky No. 2, and Sky No. 3, lode mining claims and fractions, and also the Golden Anchor property, embracing the Golden Anchor, Triplet, and Black Mascot patented claims. The Golden Anchor shaft is sunk to a depth of 855 ft., with levels at 400, 500, 740, and 840 ft. The shaft is equipped with machinery, air-compressor, and so forth, capable of exploring the ground to a depth of 1200 feet.

The shaft starts in the 'cap-rock', or later andesite, and continues in that formation to a depth of 550 ft., the 400 and 500-ft. levels being in later andesite. Below 550 ft. the shaft is in rhyolite to the bottom. An important development is made on the 840-ft. level, a cross-cut run north entering the lode-porphyry, or earlier andesite, at a point 290 ft. from the shaft. More important is the long cross-cut north from the Tonopah Extension shaft on the 1050-ft. which passes 530 ft. east of the Golden Anchor shaft, cutting through the intrusive rhyolite sheet, and penetrating the earlier andesite beneath the Black Mascot claim.

The workings of the Tonopah Extension mine lap the Golden Anchor territory on the south; the Tonopah Extension vein dips north, and the stopes on the 600-ft. level parallel the south boundary from which they are distant 250 ft. A 30-stamp mill, with a capacity of 135 tons per day, is now in process of construction on the Tonopah Extension.

Tonopah Extension Mill

Written for the MINING AND SCIENTIFIC PRESS
By JOHN G. KIRCHEN

The new mill of the Tonopah Extension Mining Co., at Tonopah, Nevada, has but recently been started. A brief description of the machinery and arrangement of the plant is given below.

The ore from the mine is dumped direct into a No. 4 Kennedy crusher, placed on top of an ore-bin. This bin is 15 by 15 ft. inside. The crusher is driven by a 30-hp. Westinghouse motor, 900 r. p. m., 9-in. pulley. The ore is crushed to about 1½ in. The ore-bin discharges through an 18 by 24-in. rack and pinion gate to the belt of a 16-in. troughed incline belt-conveyor, and is thus carried to the top of the battery-bins. The conveyor is driven by a 7½-hp. motor 1120 r. p. m. The distributor on the battery-bins is driven by a 5-hp. motor 1120 r. p. m. The incline distance of the conveyor is 235 ft., the total rise being 40 ft. The above crusher and conveyor were furnished by Chalmers & Williams.

The battery-bins are of wood, and are an integral part of the mill-building. They are 13 ft. wide, 15 deep, 50 long, and have a flat bottom. The inside of the bins, at the ore-bin gates, is protected from the usual wear by means of 16-lb. rails, spiked vertically on the battery-side of the ore-bin. The bins are double-floored with building-paper between, protecting the mill from dust.

The battery ore-bins feed through six 18 by 24-in. rack and pinion ore-bin gates into suspended Challenge feeders, and thence into mortars. The mortars are of the modified Homestake type with extra wide base for setting on concrete foundations. The casting is 12 in. thick under the dies, and weighs, with liners, about 9500 lb. Each stamp weighs 1050 lb., and makes 98 drops per minute. There are 30 stamps, in batteries of 10 each, with 5 stamps per mortar. Every group of 10 stamps is driven by a 30-hp. Westinghouse motor, running 690 r. p. m., placed under the ore-bins and driving the drive-shaft by means of an 11-in. endless double leather belt. The battery drive-shaft is 3 in. diam. The cam-shaft is 5⅞ in. diam., with drive-pulley of wood, 72 in. diam. by 17 face. The belt is 16-in. 5-ply rubber, with a swinging triangular-frame battery-belt tightner. The stems are 3⅜ in. diam. by 15 ft. long. The cams are keyed to the shaft by Blanton patent fasteners. The complete stamp-battery was furnished by the Union Iron Works of San Francisco. The mortar block is of concrete, reinforced with 6-in. flat iron hoisting cable, the oil and grease on the cable having first been removed by heating.

The pulp from each battery of 10 stamps flows to two Deister concentrators, where the coarse concentrate is removed. The six Deister concentrators are driven by a 5-hp. Westinghouse motor, 1120 r. p. m. The tailing flows direct to two Dorr classifiers. The classifiers are sheet-steel tanks 4 ft. 6 in. wide by 16 ft. long, with a bottom slope of 2 in. per foot. Inside are a series of rakes set transversely, which move back and forth, pushing the coarse pulp to the top, and lifting clear of it on the return motion by means of a cam and rocker arm. The slime and surplus water overflow at the lower edge of the tank. The classifiers are operated by a 5-hp. Westinghouse motor 1120 revolutions per minute.

The coarse pulp from the Dorr classifiers is fed direct to two 5 by 18-ft. trunnion-type tube-mills, with scoop feed. These mills were built by the Traylor Engineering Co., and are lined with El Oro liners, cast in blocks about 21 by 16 in., with ribs about 1½ in. thick projecting 3 in. and leaving a space between the ribs of about 3 in. These blocks are bolted to the shell. The two tube-mills are driven by a 100-hp. Westinghouse motor, 690 r. p. m. The motor belt is of double leather, 14 in. wide, connected with the main shaft from which clutch-pulleys operate the tube-mills separately. The tube-mill belts are of 6-ply rubber, 14 in. wide.

The tube-mills discharge direct to hydraulic classifying cones where, with the assistance of the solution and slime-overflow from the Dorr classifiers, the coarse particles are removed and the thin slime-pulp is delivered to three 8 by 54-in. Frenier sand-pumps. A feature here worthy of special mention is the tube-mill floor, which is made of concrete and is sloped to a central pit where all leakage finds its way, and where, together with the coarse particles from the hydraulic classifiers, it is elevated by means of a bucket-elevator, and returned to the Dorr classifiers for re-treatment. The Frenier sand-pumps elevate the slime to six 8-ft. Callow cones. The overflow from these cones goes direct to the mill solution-tank, while the thickened pulp is distributed over 12 Deister slime concentrators. These 12 machines are operated by a 10-hp. Westinghouse motor running 1120 r. p. m. The slime tailing is run to a central sump and is pumped to slime collecting tanks. The pump used is a 7 by 9-in. Aldrich triplex slime-pump. A 15-hp. Westinghouse motor running 1120 r. p. m. is used to operate the three Frenier sand-pumps and the Aldrich triplex pump.

There are four slime-collecting tanks 24 ft. diam., 16 ft. high, having a false conical bottom 25° slope toward the centre. These tanks are fitted with overflow launders and decanters. Arrangements are such that the clear solution can be discharged either to the mill-tank or the gold-tank, depending on the gold and silver the solution contains.

The thickened pulp from the slime-collecting tanks is pumped into either of the three agitating tanks by means of a 6-in. Krogh centrifugal slime-pump, operated by a 10-hp. Westinghouse motor, 1120 r. p. m. Here the solution is brought to its proper cyanide strength. These agitating tanks are 24 ft. diam. and 16 high, and represent the latest development in agitation and aeration of cyanide solution and pulp. It is known as the L. C. Trent Shoshone agitator and aerator, and works on principles that are both unique and effective. It is operated by drawing off the liquid and thin slime from the top of the tank containing the agitator and with a centrifugal pump forcing same through a central pipe to near the bottom of the tank. This central pipe has four arms or pipes radiating from it. These arms are fitted with short jet-pipes or nozzles inclined downward toward the bottom of the tank and through which all the solution and pulp drawn from the top must pass. The pressure of the solution and slime discharging through these restricted nozzles causes the arms to revolve at a speed sufficient to thoroughly stir and agitate. Air is admitted to the suction through a small valve and is introduced into the pump and forced into the solution; resulting in a complete admixture of air, solution, and slime. The solution readily becomes charged with microscopic air bubbles and the top of the tank presents a seething effervescent surface.

The feature that permits of the economic operation of this machine is the patent grit-proof step-bearing. The action of this device is such that as the tank is being filled with slime and solution, air is automatically imprisoned in an inverted chamber that surrounds the step-box, and the liquid and solid in the tank are excluded from the bearings which are maintained free from solution and slime, so that friction is no greater than were it not submerged. Numerous experiments have been made by stopping the agitator and allowing the slime to settle several hours. In each case the arms revolve within a few minutes after starting the agitator. The wear on the pump is light, owing to the fact that only fine slime and solution are drawn through it, while the coarser particles are held in suspension in the tank by the constant rising currents. Each agitator is operated by a 10-hp. Westinghouse motor, 1120 r. p. m. driving a 5-in. Price centrifugal slime-pump. The actual power consumed has not yet been determined, but it is safe to say it will be much less than the capacity of the motor. It is claimed for this agitator, that a much higher extraction is obtained, with less time for agitation and a low power-consumption. These are manufactured by the Trent Engineering Co., of Los Angeles. The pulp from the agitators is pumped to a large storage-tank with the same pump as is used to charge the agitating tanks. This storage tank is 30 ft. diam. by 20 high. The pulp is agitated and kept in suspension by an agitator similar to those above described. The important difference is that in this case the agitator is of the under-feed type, the agitating arms revolving on ball-bearings. The solution is drawn from the bottom of the storage-tank to the Blaisdell filters by gravity.

The filter-tank is constructed of steel having five hopper-bottoms. There are 100 leaves 5 by 10 ft., made of canvas-covered grooved wooden strips, similar to the Butters filter-leaf, with the exception of the cocoa matting. The bottom of the tank is so constructed that each set of 20 leaves discharges the washed pulp throught a separate hopper connected with an outlet-pipe which carries the slime to the main sluice-launder. In this department there are two more tanks placed above the filters, one being used for weak-solution, and the other for wash-water. These tanks are both 26 ft. diam. by 8 high. The vacuum for the filters is furnished by a Knowles 10 by 10-in. vertical pump. There is also an 8-in. Krogh centrifugal slime-pump, which is used to return the excess of slime from the filter-tank back to the storage-tank.

The solutions from the filters run direct to the gold-tank, which is 30 ft. diam. by 8 high, and thence to a long wooden box 4 ft. square, filled with excelsior, which serves to remove any fine slime from the solution. The overflow from this clarifying box goes direct to the zinc-boxes, of which there are eight, each having six compartments. These boxes are constructed of steel, each compartment being 3 ft. long, 31 in. wide, and 32 in. deep, with a hopper-bottom under each compartment, over which a screen with 1/4-in. mesh is placed to retain the zinc. Short-zinc will be transferred toward the head-compartment at each clean-up.

The barren solutions run to two sump-tanks 26 ft. diam. by 8 high, and is then pumped to the main mill-solution tank by a 5 by 6-in. Aldrich triplex-pump, operated by a 10-hp. Westinghouse motor, 1120 r. p. m. This motor also drives a smaller size of pump, which is used as an auxiliary. From the mill-tank, the solution is pumped by a 6 by 9-in. Aldrich triplex pump, operated by a 7½-hp. Westinghouse motor, 1120 r. p. m. to a 24 by 15-ft. mill-tank, which is situated above the batteries.

The silver and gold precipitate is sluiced from the hopper-bottom of the zinc-boxes to two tanks, 4 ft. deep by 10 diam., provided with false-bottoms of cloth over slats, which act as filters. Connection is made under the filter to the 5 by 6-in. Aldrich auxiliary pump, which serves to filter the precipitate and return the solution to the zinc-boxes.

The gold and silver precipitate is shoveled out and delivered to a steam dryer. This is a jacketed steam-box constructed of 3/8-in. steel-plate, the inside dimensions being 6 ft. wide, by 15 long, and 14 in. deep. Four inches of this depth is used as the steam-jacket. After drying, the precipitate is mixed with borax and melted in an oil-fired No. 275 Monadnock steel Harvey furnace.

The mill is run three shifts of 8 hr. each; one man is employed on each shift on the batteries and also attends the electric-driven air-compressor which supplies air to the mine; one man looks after the tube-mills, Callow cones, and Deister slimers; one solution-man to look after the agitators and solution-tanks. The filters are also attended by one man on each shift. On the day shift a roustabout, a man to attend the zinc-boxes and clean-up, is employed; these, with the mill superintendent, make a total force of 15 men.

The mill-building is a frame structure covered with corrugated galvanized iron, everything being under one roof. The site is such that the transportation of materials by gravity is not allowable in all cases, but the pumping has been reduced to a minimum, as the description shows. Electric power is supplied to the motors throughout the plant at 400 volts, by the Nevada-California Power Co. All the tanks used in the mill are of redwood, manufactured by the Pacific Tank Co. The mill is designed for a capacity of 120 tons per 24 hr. The material treated is a complex silver-gold ore carrying approximately 90 oz. silver to 1 of gold. In designing the mill I was assisted by Fred M. Field. J. P. Montague is the present mill superintendent.

Magnesium chloride and sulphate in water have a deleterious effect upon cement.

Plotting Co-ordinate Surveys

Written for the MINING AND SCIENTIFIC PRESS
By J. J. BRISTOL

(*Continued From Page 490.*)

The application of the protractor is best understood by considering a practical example of a stope-survey. In connection with this example the determination of dip and areas will be discussed. Suppose the notes of two successive surveys of the stope 17 on 1 are as follows:

STOPE 17 ON 1 WEST. UNDER 1701 A. BACKSIGHT 1701.

Survey A.

Pt.	H. A.	V. A.	S. D.	V. D.
1a	204°40′	123°00′	105.9	57.8
2a	207°40′	122°40′	100.7	54.4
3a	213°20′	121°10′	88.8	46.0
4a	218°20′	119°30′	89.4	44.0
5a	223°20′	117°40′	84.4	39.4
6a	225°40′	116°40′	80.2	36.0
7a	229°40′	115°00′	79.0	33.4
8a	231°40′	114°10′	74.9	30.7
9a	237°40′	111°10′	75.6	27.3
10a	240°30′	109°10′	73.4	24.1
11a	244°10′	107°10′	69.6	20.6
12a	247°00′	105°10′	69.5	18.2
13a	250°10′	103°30′	66.4	15.5
14a	254°20′	100°20′	64.0	11.5
15a	257°10′	98°10′	63.6	9.0

Survey B.

Pt.	H. A.	V. A.	S. D.	V. D.
1b	268°00′	90°50′	54.6	0.8
2b	264°10′	93°50′	57.6	3.8
3b	263°50′	94°10′	62.3	4.5
4b	257°10′	98°10′	63.6	9.0
5b	254°20′	100°20′	64.0	11.5
6b	250°10′	103°30′	66.4	15.5
7b	246°50′	105°30′	74.9	20.0
8b	239°10′	110°00′	74.5	25.7
9b	235°40′	111°50′	76.6	28.5
10b	231°40′	114°10′	74.9	30.7
11b	224°50′	116°10′	87.4	39.0
12b	221°50′	117°50′	94.1	44.0
13b	219°40′	119°00′	101.4	49.1
14b	218°30′	119°10′	104.8	51.2
15b	214°20′	120°30′	107.4	54.8
16b	208°40′	122°10′	110.8	59.0
17b	204°40′	123°00′	105.9	57.8

STOPE 17 ON 1 EAST. UNDER 1701 A. BACKSIGHT 1701.

Survey A.

Pt.	H. A.	V. A.	S. D.	V. D.
1a	80°50′	93°50′	29.0	2.0
2a	84°50′	97°10′	28.7	3.7
3a	99°10′	105°50′	31.1	8.5
4a	101°20′	105°30′	35.5	9.5
5a	111°50′	110°20′	37.4	13.0
6a	113°30′	110°50′	41.7	14.8
7a	120°30′	113°00′	49.1	19.2
8a	124°00′	114°10′	52.5	21.5
9a	128°10′	115°40′	53.1	23.0
10a	132°40′	117°20′	58.4	26.9
11a	138°40′	119°50′	59.1	29.4
12a	140°10′	120°00′	63.3	31.7
13a	145°00′	121°50′	68.6	36.2
14a	153°00′	123°50′	75.8	42.2
15a	156°40′	124°50′	83.1	47.5
16a	159°30′	125°20′	87.3	50.5

Survey B.

Pt.	H. A.	V. A.	S. D.	V. D.
1b	80°50′	93°50′	29.0	2.0
2b	93°10′	101°20′	30.6	6.0
3b	101°50′	104°50′	40.9	10.5
4b	106°50′	107°30′	46.0	13.9
5b	110°30′	109°09′	51.5	16.9
6b	115°00′	111°10′	52.6	19.0
7b	120°10′	113°20′	60.5	24.0
8b	127°40′	116°30′	61.6	27.5
9b	132°10′	118°10′	65.2	30.8
10b	136°40′	119°50′	72.4	36.0
11b	138°30′	120°10′	77.6	39.0
12b	144°30′	122°20′	80.4	43.0
13b	146°30′	122°50′	84.8	46.0
14b	151°50′	124°00′	85.8	48.0
15b	157°40′	125°20	85.1	49.5

To plot 17 on 1 West, Survey A, draw a line through 1701 A and 1701 (as in this instance the two stations are very near together it is best to extend the line as previously discussed in the plotting of drifts), place the protractor on the plan with the needle *I* pressed down through 1701 A, and then revolve until 204°40′ black, if line extending through backsight is used, or 204°40′ red, if the line extending from backsight through and beyond 1701 A is used; then move the scale *C* until 123° is indicated on *B*; then scale *D* until its graduated edge indicates 105.9 on the scale *C*; the needle *II* is then pressed down and a point marked on the surface of the paper. The vertical distance to be used later in the determination of the angle of dip, is read off on scale *D* and recorded opposite the station number to which it belongs. All the other points are plotted similarly, and the vertical distances recorded. The vertical distances above recorded have been read off from the protractor, and with a few exceptions will be found to agree with the calculated distances to 0.1 ft.; the exceptions to 0.2 ft. The points thus plotted for each survey are joined by right lines, inked in, thus showing the horizontal projection of the contour of the face of the stope for each survey (see Fig. 9). The freedom of the plan from pencil marks and lines is here noticeable; no erasures being necessary. As the data are not at hand and as the area stoped has no connection with the upper and lower drifts, they are not indicated on the plan of the stope shown in Fig. 9.

Before discussing the determination of dip it would, perhaps, be as well at this point to explain the use of the two sets of figures, red and black, on circle *A*. If, when plotting a stope, the line drawn through the backsight would place the protractor in a less convenient position to use, by extending the line in the opposite direction and without altering the horizontal angles as read, the red set of figures would be used and the plotting proceeded with as before. This has been done in the plotting of the stope above. Also, when making the underground survey, sometimes owing to the nature of the stope it is more convenient to read vernier II than vernier I; in this case the angle is recorded as read from vernier II and when plotting, if the line through the backsight is being used as the reference line, the red set of figures is used; if the line extending in the opposite direction, the black figures. And again it is sometimes necessary to transit the telescope to be enabled to read the vertical circle; in this case, if using the line through backsight vernier II would require the black, and vernier I the red, and if using the line extending in the opposite direction, vernier I black and vernier II red. The vertical angles would

Tonopah Geology

By J. E. Spurr

*__Conclusions in 1902.__—When I made my study of the mines of Tonopah, in the summer of 1902, I identified the highly altered and variable-appearing rock in which the principal veins were found as andesitic; and found that this rock was frequently covered by another andesite, later than the principal ore deposition, and therefore barren of ore and forming a 'cap-rock' to the ore-bearing veins. The formation which enclosed the veins I called the 'earlier andesite' (although my investigation shows the rock to be really a trachyte); the younger rock the 'later andesite.' Still younger than the 'later andesite' I found a variety of volcanic rocks, largely extrusive surface formations including tuffs, explosive breccias, and flows, but also partly intrusive. These younger rocks were chiefly rhyolitic in composition. One of the most conspicuous of these rhyolitic rocks was a rock with a glassy ground mass, usually packed full of angular inclusions of similar glassy rhyolite, so that the whole had usually the structure of a breccia. This rock occurred chiefly as surface flows in the district south of the town of Tonopah; to the north of the town, however, it was found to outcrop abundantly in the guise of an intrusive rock, younger than the 'later andesite.' To this rock the name 'Tonopah rhyolite-dacite' was given; it appeared to be plainly an autoclastic volcanic breccia. The origin seemed to be due to periods of quiescence and of partial congelations in a volcanic vent, alternating with periods of upward propulsion of the viscous lava, so that the hardened glassy exterior crusts were shattered and carried along in the upwelling still fluid portion of the same lava; and these incidents repeated a number of times produced the peculiar and characteristic structure of the finally solidified rock as we find it. Also later than the ores, and roughly contemporaneous with the Tonopah rhyolite-dacite eruptions and related intrusions (the span of the period of activity of this lava was considerable), I found a series of waterlaid tuffs formed in a lake of vast extent. Later than these tuffs, I found a number of volcanic necks, formed of distinct but closely related lavas of rhyolitic composition merging toward dacitic composition. In sum, it appeared that the productive veins had formed after the eruption of the 'earlier andesite' (which was and is still believed to be in a large measure at least a surface flow) and before the advent of the numerous other volcanic and volcanic-detrital mentioned. The whole volcanic history, including the formation of the ore deposits, was found to belong to the Tertiary, probably Miocene-Pliocene.

New Information Obtained in 1903.—Returning to Tonopah in the summer of 1903 for a brief examination of recent developments before the publication of my report, I found that a number of shafts had, after passing down through the 'earlier andesite,' penetrated, at a depth of a few hundred feet, a dense greenish glassy rock, highly altered, and essentially aphanitic, but evidently of rhyolitic nature. This rock is characterized by numerous angular light-colored or white inclusions, apparently of altered rhyolitic glass of much the same nature as the matrix, so that the whole rock appears to be an autoclastic glassy rhyolite. The most important veins seemed to be cut off by this rhyolite, whence it was concluded that the rhyolite was an intrusive sheet, younger than the 'earlier andesite' and the principal ore deposition; and this view, after recent exhaustive investigation, is still held. Since this rock was closely similar to the outcropping intrusive Tonopah rhyolite-dacite in the vicinity, it was correlated with this formation. This correlation has now been definitely abandoned, as subsequent extensive development work has proved this underground rhyolite to be of distinctly greater age than the Tonopah rhyolite-dacite (or Tonopah rhyolite, as it may be called with more simplicity and as much accuracy) and to outcrop at the surface nowhere in the surveyed and mapped district. Thus it constitutes a new formation, unexposed at the time of the original investigation. It is commonly referred to in Tonopah as the 'Upper rhyolite,' but will be here more conveniently designated as the West End rhyolite.

I observed, in the summer of 1903, that there was evidence of a second period of vein-formation, later than the extrusion of the West End rhyolite. The description of these later veins still holds. They are less definite and persistent than the veins of the first period, contain great quantities of low-grade or barren quartz, and the pay-ore, where it does occur, is spotty and usually of low grade.

The developments observed in the summer of 1903 also showed that several shafts had passed through the sheet of West End rhyolite into andesite, having apparently the general composition of the 'earlier andesite' above the sheet, and highly altered by hot-water action. No development work whatever had been done on the lower andesite body, but it was especially remarked that the alteration of this rock was entirely of the sort sometimes designated as 'propylitic,' that is, to calcite, chlorite, and pyrite, so that the rock took on a characteristic dark-green color; while the 'earlier andesite' above the West End rhyolite sheet was mainly altered to quartz, sericite, and adularia. Evidences of this propylitic alteration in this original 'earlier andesite' mass were, however, abundant in many places, so that this lower andesite was correlated with the 'earlier andesite,' although it was pointed out that this 'calcitic phase of the earlier andesite' was not associated with the ores.† At the time of this second examination in 1903 a vertical drill-hole downward from the bottom of the Mizpah shaft had penetrated a rock which I identified as rhyolite, and correlated it with the Tonopah rhyolite, and on this basis interpreted it as a barren formation, in which no pay-ore would be found. This correlation and interpretation have been confirmed by the recent exhaustive investiga-

*Abstract of report on the Geology of the Property of the Montana-Tonopah Mining Co., Tonopah, Nevada.

†Professional Paper No. 42, U. S. Geol. Surv., p. 32.

tion. In the Mizpah Extension shaft part of probably the same underground rhyolitic body was observed, and, as it still appears, correctly correlated. This deep-seated sheet became subsequently locally known as the 'lower rhyolite.'

Progress of Development Work and Modifications of Geological Views.—The extensive underground development of the succeeding years emphasized the distinction between the lower andesite body, or the 'calcitic phase of the earlier andesite,' and the upper or original 'earlier andesite' mass. The first-named rock, whose designation became usually locally abbreviated to 'calcite andesite,' was found to have a considerable lateral extent, with the general form of a sheet underlying the West End rhyolite sheet, and overlying the deeper Tonopah rhyolite (usually called locally the 'lower rhyolite') mass, which was also found to have considerable lateral extent. The green color, due to the type of alteration of the 'calcitic andesite,' was found to be quite uniform, and the scarcity of silicification or veins in this rock became increasingly apparent. Therefore there was an increasing tendency on the part of the local Tonopah geologists, who were watching the development, to question the correlation of the 'calcitic andesite' with the 'earlier andesite,' and this increasing doubt was shared by myself. These doubts took more definite form in my mind in the summer of 1908, when I returned to Tonopah for an examination of the West End and MacNamara mines. At that time I determined the fact that the West End rhyolite sheet could not be correlated with the Tonopah rhyolite, but was distinctly older. One of the strong arguments for the intrusive nature of the West End rhyolite sheet was therefore withdrawn, and a review of the whole argument became necessary, for if the rhyolite sheet were not intrusive, the main argument for the correlation of the underlying 'calcitic andesite' with the overlying original 'earlier andesite' was also withdrawn. The now more clearly exhibited (on account of new development work) uniform points of distinction between the two andesitic rocks led finally to the conclusion that the two andesitic sheets were indeed distinct and independent rock-formations. In this new light, a probable explanation appeared to be that the different formations were merely a series of regularly successive surface flows. This explanation was regarded with some favor, although it by no means explained the peculiar relations of the rocks to the mineral veins, as it still appeared that the most important veins in the 'earlier andesite' were cut off by the West End rhyolite sheet; and that the large but relatively low-grade veins of the second period, found in the West End rhyolite, did not penetrate the 'calcitic andesite' and, apparently, had not been found in the Tonopah rhyolite beneath.

In December 1909, the thesis that the different rock formations at Tonopah were a series of successive surface flows was brought out in a publication by J. A. Burgess, geologist for the Tonopah Mining Co. As a strong point in favor of this view, mention was made of the discovery, in the Mizpah mine and in the uppermost portion of the 'lower rhyolite,' of white dense banded rocks having the appearance of stratified tuffs, alternating with the coarser breccia such as is more common in this formation. Further specimens were found in cores obtained by deep drilling which could be easily interpreted, on account of their definitely banded character, as stratified. Microscopic examinations made by E. S. Larsen, of the Carnegie Institute, showed these rocks to have an essentially fragmental character, and this led to their designation as well-bedded tuffs. These considerations made the thesis that the rocks of the district were a series of successive surface formations, occurring in their normal order, with the oldest at the bottom and the youngest on top, a plausible one, which I had no difficulty in believing might prove to be correct.

Outline of Results of Recent Study.—In the early part of 1910 arrangements were made with the principal mining companies of Tonopah for a thorough geological investigation, to supplement my original report published by the Geological Survey, and to investigate the import of data subsequently exposed by development, with its bearing upon the future methods of development work in the camp. Accordingly I have spent a number of months in close detailed underground studies and mapping, and have already investigated in detail the mines of the Tonopah Mining Co., the Montana Tonopah, the Belmont, and the Midway, all adjacent and forming as a group a unit. The results of this arduous work have been finally to fix definitely and beyond doubt most of the geological relations. As is so often the case, it is the unexpected which has finally proved to be the true solution. The 'earlier andesite' still remains the oldest of the rocks, but turns out to be a true trachyte instead of an andesite, and will henceforth be called the Mizpah trachyte. In its lower portion it passes by transition into a dense banded glassy basal phase, called in this report the 'glassy trachyte.' This 'glassy trachyte' was at least several hundred feet thick, but where the exact base was, or on what older formation this flow rested, is not known. The West End rhyolite has been determined to be an intrusive sheet, mainly inserted along the zone between the 'glassy trachyte' and the Mizpah trachyte proper, although showing considerable irregularity. The 'calcitic andesite' is a distinctively intrusive sheet of considerable irregularity, younger than the West End rhyolite, and sometimes underlying this rock, directly, sometimes separated from this rock by a variable thickness of the 'glassy trachyte.' It appears to be of essentially the same age and composition as the 'later andesite,' is correlated with it, and is probably directly connected with the main later andesite mass, which appears to be essentially a surface flow. The 'lower rhyolite' is shown to be younger than the 'later andesite,' is correlated with the Tonopah rhyolite, as was done at the time of my original investigation, and is younger than the 'calcitic andesite' sheet, which it underlies and is locally known to intrude. This 'lower rhyolite' is evidently the flatly downward-pitching extension of the great mass of intrusive Tonopah rhyolite exposed on the surface half a mile or so to the north of the main producing mines. The thickness of this Tonopah rhyolite is unknown, as it has never been bottomed; in the Mizpah mine a vertical thickness of over 1900 ft. has been demonstrated by drilling. Thus is demonstrated by the well-substantiated and extraordinary condition of a series of four successive sheet-like formations of distinct characteristics, of which the oldest lies at the surface, and the youngest at the bottom, and the whole order of superposition is the reverse of the order of age. This inversion, striking as it is, is not so regular as an elementary review of the situation indicates; the impression of great regularity arises from the limited field of development underground, which has a major horizontal axis hardly more than a mile in length, and from the fact that the geological conditions in this developed area are so complex that the area appears to the conception much more important in size than it actually is. Development carried outside of this limited area would doubtless show a great irregularity of relation; and, indeed, this is already exhibited on the borders of the developed area.

The veins have finally been divided into three groups according to their age, which groups correspond essentially with those originally made. The formation of the first group followed the advent of the Mizpah trachyte and preceded the advent of the West End rhyolite. This group comprises those rich veins which have made Tonopah famous. The second group followed the intrusion of the West End rhyolite, and preceded the advent of the later andesite (including in this term the 'calcitic andesite'). It includes frequently large veins, usually low grade or barren, and locally profitable. The third group followed the intrusion of the Tonopah rhyolite and comprises rare, essentially barren veins, never profitable.

Synopsis of Final General Results.—At Tonopah the oldest rock is a trachyte flow highly altered to quartz, sericite, and adularia. The lower part of this flow is a fine flow-banded glassy trachyte. The main body of the trachyte contains the oldest and by far the most important group of mineral veins; the glassy trachyte appears practically barren. Stresses subsequent to the trachytic extrusion produced horizontal faulting near the zone of transition between the main body of trachyte and its glassy lower portion; and along here a glassy trachy-alaskitic intrusion, very full of inclusions, took place. Subsequent movement reopened this line of weakness, and a second trachy-alaskitic intrusion came in—the West End rhyolite sheet. At a subsequent epoch came an eruption of andesite (Midway andesite), largely as a surface flow, but largely also as an intrusive sheet along the old zone of weakness, but typically below the West End rhyolite sheet; at a still later epoch there was a series of rhyolitic and alaskitic surface flows and intrusive mass called the Tonopah rhyolite.

The principal veins were formed after the trachyte eruption and before the Montana breccia-West End rhyolite intrusions. They are quartz veins carrying silver and gold. A second set of veins was formed after the West End rhyolite intrusion and before the Midway andesite eruption. This second set is divided into four successive groups—**A,** large typically barren quartz veins; **B,** tungsten-bearing veins; **C,** mixed quartz and adularia veins, typically barren; **D,** small productive veins like those of the first set, following the trachyte. A third set of veins was formed after the Tonopah rhyolite intrusion. They are quartz veins containing occasional lead, zinc, and copper sulphides. All of these veins formed at shallow depths, and the different types represent various stages of temperature. The first period veins represent the normal shallow-seated type, and followed the trachyte eruption; the second period **B** veins represent an abnormally intense shortly-sustained temperature, following the trachy-alaskitic intrusion; the second period **D** veins a directly subsequent briefly-sustained stage of temperature more normal to shallow depths; the third period a relatively high but briefly-sustained temperature, following the alaskitic (Tonopah rhyolite) intrusion. No vein-formation followed the andesite eruption.

The history of faulting is long and complex; important movements have taken place at every stage of the geologic history. These movements accompanied and were due to the volcanic paroxysms; and were so intense that locally the rocks are ground almost to a powder.

Before magnetic separation can be applied, it is necessary to free, or break apart by crushing, all the constituent minerals forming the crude ore. Theoretically the point to which comminution should be carried is that at which every particle of mineral is free to be attracted or rejected according to its permeability; this is, of course, the ideal situation, but can not be carried out in practice. Nevertheless, this theoretical point should be the aim. The practical rule is to carry the crushing far enough to free the maximum number of valuable mineral particles, having due regard to the cost of crushing, and the saving effected. There is no ironclad rule to be followed for the solution of this problem; the process, as a whole, requires an intimate co-relation of the crushing and separation factors. Different ores require widely different treatment; for one ore, moderately coarse crushing will suffice; for another, excessive pulverization is required to liberate the particles of magnetite, with the consequent added cost of comminution.

New abrasives recently placed on the market are aloxite, a product for steel grinding which is successfully used in machine-shops, as it does not heat the tool or draw the temper; samite, an abrasive for cutting aluminum which does not glaze or fill when used on aluminum or other fibrous metals; and carborundum fire sand, a chemical compound of carbon and silicon which is mixed with a binding material, silicate of soda, of 52° B., which is dissolved in water before being added to the fire sand. The mixture is made plastic and is molded to the interior of a furnace for a lining. It is understood that this lining will withstand very severe conditions.

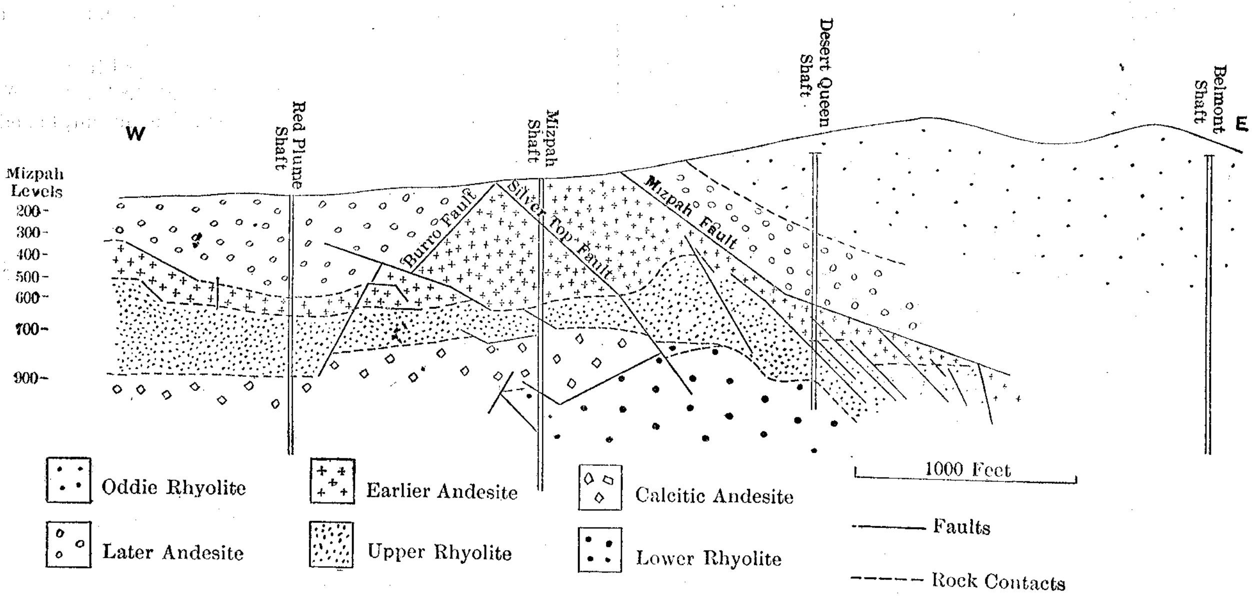
The Geology of the Tonopah Mining District

By Augustus Locke

*The important geological publications concerning the Tonopah mining district are those of Spurr and of Burgess. In these publications are presented fundamental differences of interpretation, which are the more interesting because both authorities have had ample opportunity for observation, and because both are geologists of proved ability. The general geological features of Tonopah are shown in Fig. 1, and the differences of interpretation referred to are outlined in the accompanying notes. Briefly, Burgess regards the various rocks as flows, lying in the order of their deposition. Spurr regards them in part as flows, and in part as flat-lying intrusives. The disagreement, then, concerns the rocks regarded on the one hand as intrusives, and, on the other hand, as flows. These rocks are chiefly the so-called calcitic andesite, the upper rhyolite, and the lower rhyolite. The economic importance of the question of interpretation is, of course, limited to its bearing on the probable distribution of undiscovered ore. The later andesite is generally conceded to be barren—a 'cap rock,' at whose lower contact the productive veins apex. The earlier andesite has so far yielded the bulk of the production. As has already been suggested, both Spurr and Burgess regard it as a flow, and both have essentially the same conception of its distribution. Above the bottom of the earlier andesite, therefore, the conception of the ore distribution is the same, whichever interpretation be adopted. Below the bottom of the earlier andesite, however, the matter of interpretation assumes supreme economic importance; for, while Burgess regards all the underlying rocks as older than the earlier andesite, and older than the chief ore mineralization, Spurr regards them as younger than both. Under Spurr's hypothesis, exploration in these rocks is emphatically discouraged; under Burgess', it is to a certain extent encouraged.

The important evidence appearing to favor the hypothesis that all the rocks occur in flows is as follows: (1) The locus of each rock is horizon-like. For example, the lower rhyolite is encountered at depths averaging about 1000 ft., over an area of at least a square mile. Its surface, except where it is faulted, is seldom steeper than hill slopes, and is chiefly flat or horizontal. (2) Materials closely resembling stratified volcanic tuffs occur abundantly on the upper contact of the lower rhyolite, and less abundantly on the upper contact of the upper rhyolite. (3) The contacts between the supposedly intrusive and intruded rocks are, when unfaulted, most often notably straight and regular. Nowhere have the so-called intrusives been conclusively proved to invade by means of offshoots the rocks which they have supposedly intruded. The interpretation of irregularities of contact as proof of intrusion is made difficult by the abundant faulting, and by the possibility of inter-flow erosion. (4) The andesitic cover has, over a large area, rigidly confined the rocks which underlie it. The lower rhyolite, a rock having a characteristic and unmistakable appearance, has been proved to occur on the surface only in the territory considerably north of the producing mines, and there in very small and scattered bodies which may be inclusions. (5) The productive veins in certain places pass without diminution either in size or richness from the earlier andesite down into the upper rhyolite. (6) In many places near the top and near the bottom of the upper rhyolite, there occurs an extraordinary igneous breccia, often many feet thick and crowded with foreign inclusions; the matrix is rhyolitic, and the rock looks exceedingly like a flow breccia. The upper portion of the lower rhyolites has numerous but less abundant inclusions. (7) The rhyolites, though containing abundant inclusions, and, among them some which are andesitic, have never yielded inclusions which can be positively identified as belonging to the earlier or later andesites. (8) The andesites are free from inclusions of all sorts; therefore their freedom from inclusions of rhyolite is no indication that they are older than the rhyolites.

The evidence supposedly favoring the hypothesis that some of the rocks are intrusive is as follows: (1) In the rhyolites, a banding resembling flow structure sometimes follows irregularities in the contact. (2) The rhyolites ocasionally have on their contacts with the andesites knob-like and wedge-like projections, looking like intrusive shapes. (3) In certain places the calcitic andesite is sep-



Geologic column of rocks shown in section (youngest at top).

Spurr (1910).	*Burgess.*
Oddie rhyolite (partly intrusive).	Oddie rhyolite.
Lower rhyolite.	Later andesite.
Later andesite. } probably	Earlier andesite.
Calcitic andesite (intrusive). } identical.	Upper rhyolite.
Upper rhyolite (intrusive).	Calcitic andesite.
Earlier andesite.	Lower rhyolite.

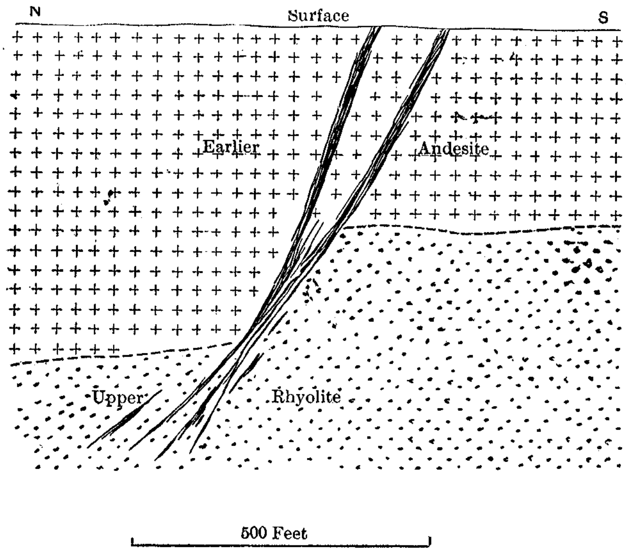
Fig. 1. EAST-WEST SECTION THROUGH MIZPAH SHAFT. (Adapted from Burgess.)

*Abstract of a paper presented at the San Francisco Meeting of the American Institute of Mining Engineers.

arated by rhyolite from the later andesite with which it is supposed by Spurr to be identical. In certain places the earlier andesite is separated by upper rhyolite from a rock called glassy trachyte, with which Spurr supposes it to be identical. (4) The profitable veins often disappear or weaken when they reach down to the lower contact of the earlier andesite.

It must be granted at the outset that the disposition of the rocks in horizons creates the presumption that they are flows. Most of the shafts penetrate similar rocks in similar succession. Thus, the lower rhyolite, so far as is known, underlies the whole district; the calcite andesite almost everywhere covers the lower rhyolite; and above these rocks come, in order, the upper rhyolite, the earlier andesite, and the later andesite. The individual sheets of rock have many irregularities in thickness; these, however, are satisfactorily attributable to inter-flow erosion and to faulting.

Again, if we conclude that the earlier andesite is the oldest rock in the district, we must conclude also that it has been floated up by the intrusive underlying rocks to a height of at least 1000 ft., and possibly to a much greater height. (The lower contact of the lower rhyolite is not known.) During the process of floating up, the andesite has retained over an area of at least a square mile, its integrity and approximate horizontality.



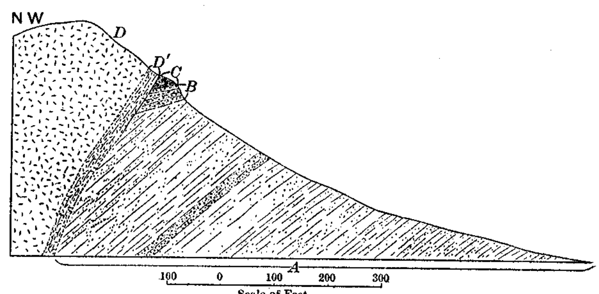
DOWNWARD EXTENSION OF TYPICAL VEIN INTO RHYOLITE.

A general view, then, of the large features of rock distribution affords strong evidence in favor of the theory of extrusion. Nevertheless, it is conceivable that pseudo-flows might result from intrusion, and this evidence is, therefore, by itself, inconclusive.

If the large facts of rock distribution fail to furnish conclusive evidence of the origin of the rocks, this evidence must be sought in the details of the rock contacts. In general, there are certain details of rock contacts whose testimony must be accepted as unimpeachable. One such detail is the existence at contacts of volcanic tuffs; genuine tuffs being proved to exist between layers of volcanic rocks, it is difficult to conceive of evidence, however abundant, which would prove that the layers are not flows. It becomes, therefore, a matter of extreme importance to determine whether or not in Tonopah the supposed tuffs are genuine. Burgess, who discovered them, believes that they are. Spurr believes that they are not. That they are tuff-like, is beyond doubt. They are somewhat soft; they possess stratification, marked by alternating bands of coarse and fine fragmental material; they cleave easily along the junctions of these bands; they lie with their structure parallel with the rock contacts. If they are not true tuffs, resulting from surface deposition, then they are conceivably attributable to one or both of two processes—flow-banding (the arrangement of inclusions along flow-lines) and movement-banding. Spurr's conception of their origin is expressed in the following: "* * * brecciated and granulated rock is often layered by the fault-movement and fault-pressure, so that it assumes all the appearance of certain varieties of surface-formed detrital tuffs."

Microscopic examination of thin sections of specimens from the Mizpah 700-ft. level yields conclusive evidence against the possibility of the production of the supposed tuffs either by flow-banding or movement-banding. (1) The tuffs are made up of sharp-cornered fragments, often crowded closely together, and are typically clastic. (2) They are distinctly layered; layers of coarse material alternate with layers of fine material, with no gradation from coarse to fine. (3) The abundant quartz phenocrysts, with one or two exceptions, when revolved in polarized light, extinguish with much suddenness. The wavy extinction, which is the invariable characteristic of strained quartzes, is strikingly absent. That a sorting out and sharp separation of coarse from fine should result from flow or movement-banding is, of course, incredible. And the significance of the unstrained quartz phenocrysts cannot be questioned. Indeed, the tuffs are so life-like and their detrital origin so obvious that their import would be ordinarily accepted as a matter of fact.

The supposedly intrusive contacts of rhyolite with other rocks at no place seen by me offer incontestable evidence of intrusion. Before such proof can be accomplished, it is necessary to prove that the irregularities were not caused by faulting, or by inter-flow erosion, or by both. Now, in localities of extensive rock alteration and abundant faulting, such proof is impossible; indeed, here the proof that the irregularities were actually caused by faulting is frequently possible. The rhyolite at certain places possesses a banding which follows to some extent irregularities of the contact, and which sometimes looks like flow-banding, and might suggest intrusion. I have failed, however, to find any such place where the evidence of intrusion was unequivocal. Usually, the banding is irregular and very discontinuous. It is quite as often oblique to the contact as parallel with it. Moreover, if contact-movement, as Spurr believes, can produce tuffs, it is very easy to conclude that it can produce apparent flow-structures.



(From *Professional Paper No. 42, U. S. Geological Survey.*)

A. Finely stratified Siebert tuffs (lake-beds) with occasional layers of rounded pumice fragments or water-worn lava.
B. Basaltic agglomerate with bombs, capped by solid basalt.
C. Basalt.
D. Brougher dacite, intrusive neck.
D'. Glassy marginal facies of dacite.

VERTICAL CROSS-SECTION OF SOUTHEAST SIDE OF SIEBERT MOUNTAIN.

The usual restriction of nearly all the probable ore deposits to the earlier andesite is one of the most interesting facts of ore occurrence with which I am familiar. In certain cases, the ore ends abruptly when it comes down to the lower contact of the andesite. In other cases, it extends down into the underlying rhyolite, ultimately, however, weakening and dying out. Occasionally, as seen in the figure, it survives for a time with a hanging wall of andesite and a foot-wall of rhyolite, ceasing shortly after it passes entirely into the rhyolite. Lastly, it passes from andesite to rhyolite without change.

To explain the superior productivity of the andesite, many hypotheses are possible: (1) The andesite is the earliest rock; the chief ore-mineralization followed it and preceded the other rocks. (Spurr's hypothesis.) (2) The

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source of the ore minerals may have been the andesite itself or the upper rhyolite. (Suggested by Burgess.) (3) The ore was deposited largely by metasomatism. The various rocks, particularly the upper rhyolite and the earlier andesite, present great contrasts in texture. Certain textural and chemical properties possessed by the andesite caused it to be more favorable to the precipitation of the ore-minerals than the other rocks. Or the andesite was more favorable to the formation of initial channels than the other rocks. (4) The path of travel of transporting agents was mainly along the andesite-rhyolite contact and upward into the andesite. (5) Ore deposition was a superficial phenomenon, effected through the decrease of heat and pressure near the surface, or through other superficial agencies. The andesite, at the time of ore deposition, was the surface rock, and, therefore, received the bulk of the ore-mineralization. (6) Post-vein faulting at the contact between upper rhyolite and lower andesite in some instances caused the disappearance of the vein at the contact. (7) Ore-mineralization occurred after the eruption of the earlier andesite and in cooling-shrinkage cracks in that rock.

Spurr, as has already been made clear, accepts the first of these hypotheses and, apparently, rejects the others. The fact that the ore frequently extends down into the rhyolite, he explains by a supposition of several periods of mineralization, of which the first and most important was earlier, and the others later, than the rhyolite. Veins contained entirely in andesite belong, then, to the first period; veins in rhyolite, or in both rhyolite and andesite, belong to later periods. That several periods of mineralization did exist is probably true. The assignment, however, of a particular vein to a particular period is often impossible. There is no mineralogical distinction whatever to be made between many veins which, according to the hypothesis, should belong to separate periods. The remaining six hypotheses cannot be easily cast aside. To prove any one of them would be difficult; to disprove would be even more difficult. Yet so long as they stand as possible explanations of the localization of the ore deposits, they must offer impassable barriers to the acceptance of the hypothesis just now considered.

Siebert mountain looks like a succession of flows, but Spurr's interpretation, shown above, regards it as partly intrusive. The fact of the matter is, however, that the supposedly intrusive rhyolite (or dacite), just above its contact with the basalt, has a very well-marked and nearly horizontal flow-structure—a fact scarcely compatible with the idea that the contact has the character shown in Spurr's section. That the rocks of the productive part of the Tonopah mining district are flows lying in the order of their deposition is proved by the occurrence of volcanic tuffs at the contacts whose interpretation has been in dispute. The other available evidence in some instances supports, and in no instance contradicts, this conclusion. There is no good reason, then, for the belief that the rocks underlying the earlier andesite are younger than the chief productive mineralization. Exploration in these rocks is accordingly relieved of the discouragement which would attend this belief.

Mineral Production of Philippines in 1910

The bulletin on Philippine Mineral Resources for 1910 has been unavoidably delayed but is now in the press. In advance of its publication, the following statistics are given:

Total metallic production, including coal........	₱505,138
Total non-metallic, chiefly structural material...	1,541,031
Total mineral production..........................	₱2,046,169
Total gold production............................	308,860
Value of iron produced...........................	20,023
Value of total coal production...................	176,255

The largest amount of gold was derived from Ambos Camarines; the iron was entirely derived from small native mines.

Electrical Iron Smelting in Sweden

By T. D. Robertson

*The success attained in smelting magnetic concentrate at Trollhättan is of great interest. The design of shaft now employed is not considered suitable for the purpose, being too narrow, but in spite of this 65% of fine concentrate caused no inconvenience in working. The inventors of the furnace are of the opinion that with a specially designed shaft, all fine concentrate could be smelted alone. Iron sands and lean magnetites, which are easily concentrated, but which are expensive to nodulize or briquette into a form suitable for blast-furnace smelting, are of frequent occurrence, and that where these are within easy reach of water-powers, there seems to be a good field open for electric smelting. Haanel, in his report on the experiments made at Sault Ste. Marie, mentions that no difficulty was experienced in smelting titaniferous ores electrically, and it is interesting that the Swedish experience bears this out, although no ores were used with more than 0.8% TiO_2. The electric smelting of ores with high sulphur content is another problem. Haanel was successful in producing low sulphur iron from sulphurous ores, but in Sweden there are practically none of these ores mined, so that this point was not confirmed on a large scale at Trollhättan; however, there can be little doubt that the electric furnace, with its reducing atmosphere and basic lining of the hearth, permitting as it does of the use of a slag rich in lime, offers the best method of producing sulphur-free pig iron from ores containing that unwelcome element.

The pig iron produced by the furnace was sent to various Swedish iron works for conversion into steel in open-hearth furnaces. The characteristic feature of electric pig iron is its freedom from oxides; and in consequence electric pig of normal silicon content (say, 1% and over) takes a longer time and more ore to convert into steel than ordinary blast-furnace gray iron. Low-carbon electric pig iron, however, is found to give surprising results, charges made up of 50% of this iron and 50% of scrap producing hot fluid steel with considerable saving of time over ordinary practice. As was to be expected, the open-hearth furnace managers looked somewhat askance at this iron at first, as they knew the disastrous effect of using low-carbon iron full of holes from the blast-furnace; however, after giving it a trial, the workmen asked for more, as they said that the furnace worked better and more rapidly with the new pig iron. Fortunately for the electric furnace in Sweden, it is more economical to make the white iron, that the steel-makers prefer, than to make high-silicon gray iron. Thus it may be maintained, on the strength of the experience gained, that for the open-hearth process, high silicon contents are detrimental rather than advantageous, while in the blast-furnace pig iron a certain quantity is necessary to neutralize the defects of a reduction process less perfect and ideal than that employed in the electric furnace. The quality of the steel produced has been thoroughly tested by making it up into various products and comparing it with steel made from Swedish charcoal blast-furnace iron. In no case was the new steel inferior, and from the most recent reports to hand, it was in several cases decidedly superior.

The Trollhättan furnace was shut down at the end of May of this year in order to make certain alterations suggested by the results of its six months' campaign. Two of these deserve mention.

(1) Since the furnace was designed, the manufacture of large carbon electrodes, of high conductivity, has made great progress. It is now possible to obtain cylindrical electrodes of 600-mm. diameter, fitted with screw joints. These have recently been installed at Trollhättan, and result in two considerable improvements, the first one being that the loss due to stump ends is done away with, as when an electrode becomes too short, a new one can be

*Excerpt from a paper presented before the American Electrochemical Society, Toronto, September 21 to 23.

Metallurgy at Tonopah

By M. W. von Bernewitz

Simplicity of operation seems to be the key-note of treatment methods at this interesting mining centre, having been brought to such a point by much intelligent experimenting. To obtain an average of 93% extraction at a cost of, say, $3 from ores carrying 30 oz. of silver and a few grains of gold per ton is good work, and not long ago would have been termed impossible, especially in such a situation as Tonopah.

Tonopah ores may be described as consisting of fine granular quartz (the silica averaging perhaps 80%), without noticeable quantities of sulphides, poor in the baser metals, and containing disseminated silver minerals, and gold. The primary metallic minerals are silver sulphides, principally polybasite, stephanite, and argentite, with occasional pyrite, chalcopyrite, galena, and blende. Silver selenide also occurs. Silver chloride, bromides, and iodides occur, mainly at least, as secondary minerals. Silver also appears in the metallic state. Gold occurs in the proportion to silver of about 1 to 100 by weight, and has been seen in the free state.

In Tonopah there are five mills—the Belmont, Extension, MacNamara, Montana, and West End—while at Millers, 12 miles north, are the Belmont and Tonopah mills, ore being shipped to these at a cost of 70c. per ton. In nearly every case gyratory crushers are used for breaking ore as it comes from the mines, the procedure being to crush first in a large crusher, up to the No. 7½ type K Gates size, pass through revolving trommels, the oversize being again reduced in No. 3 size gyratories, the final product for the stamps being about 1¼ in. Sorting is done at the Belmont and MacNamara mills; at the former on a pan conveyor from which 15% is rejected; and at the latter on a 30-in. rubber belt, from which 6% is sorted out. From the crushing department, the ore is taken to mill bins by 20-in. belt conveyors, or bucket elevators, and distributed by the usual automatic devices.

Tonopah millmen have not been troubled with the heavy-stamp mania, although the ore is fairly hard, and the 320 stamps at work vary between 1100 lb. at the Montana to 1400 at the MacNamara, the latter having probably the toughest ore in the district. Several different methods of driving stamps by motors are to be noticed, and will be separately described later. Foundations are usually of concrete, and give satisfaction. The new Belmont mill has sheet-lead under its mortars, but this is being replaced with rubber. There are no high-speed stamps, the average being perhaps 103 drops per minute with 7-in. drop. Both square-mesh and ton-cap screens are used, sizes varying from 6 to 20 mesh, and the stamp-duty is from 4 to 8 tons per day. The Challenge feeder is now a simple contrivance compared with the original, and may be described as a double monkey-wrench grab, which turns the gear and feed plate. There is no amalgamation at Tonopah, nor is it necessary on this class of ore. Crushing is done in weak and warm (from 50 to 80°F.) cyanide solutions, so the ore is in contact with solution from the stamps to filtration. This is necessary as well as the heating, which, although somewhat expensive, quickens the solution and accelerates the dissolving action. Solutions are usually heated to about 95°, and in one case 120°, by live steam introduced in the agitators.

The practice of using hot solutions is briefly as follows: At the new Belmont mill the temperature at the stamps is from 60 to 70°F., and at the Pachuca agitators exhaust steam from the mill air-compressor is fed in, increasing it from 90 to 100°. In the *Mining and Scientific Press* of January 27, 1912, A. H. Jones, metallurgist at this plant, gave some valuable data on this subject. On an ore carrying 0.05 oz. of gold and 18.2 oz. of silver per ton, 60 hours' agitation with both 60 and 90° solutions, the tailing averaged 0.0175 and 3.45, and 0.0125 and 1.90 oz. respectively. Tests on 48 and 69 hours at similar temperatures gave as marked results. Besides the effect on extraction, the hot solutions flowing through the mill kept the whole place at a good working temperature. At the Montana-Tonopah, ore is crushed in 50 to 60° solution, which is increased to 110° at the Hendryx agitators by live steam. It is found also that the heat aids settling. There is a marked decrease in extraction without hot solutions.

The MacNamara mill recently had experience with cold solutions, owing to an enforced shut-down for two days. The Trent agitators are usually kept at from 115 to 120° by live steam and it was found that it took several days to heat everything again, in the meantime the time of agitation had to be increased and extraction fell off considerably. Heat is necessary in the summer, but less steam is used. The cost is about 30c. per ton treated. The Extension ore is crushed in 80° solution, and live steam is added to the Trent agitators as soon as possible, making the temperature up to 120°. It was found that this was better than 90° and extraction has improved 1.5 to 2% during the past few months, it being 94.5% at present. About 2100 tons of solution is circulating in the mill, and it takes 7 days to heat this if it should get cold, meanwhile extraction falls off.

Cost of heating is 18c. per ton. It has been found at the Belmont mill, at Millers, that in passing through a tube-mill the temperature of the solutions increases, presumably by the grinding action of the pebbles, mill liners, and ore particles. A test taken while I was there showed feed temperature at 65°, and discharge 70°. In the *Mining and Scientific Press* of February 24, 1912, Noel Cunningham, at Millers, contributed the results of some experiments, proving that laboratory work had shown greatly improved results from hot solution. At another plant treating Tonopah ore, crushing is done in 76 to 80° solution, increased to 95° in the agitators by live steam in coils. Recent tests showed a saving of 24 to 32c. at a cost of 11c. per ton. The same temperature is kept up in summer and winter. At the Mexican mill, Virginia City, solution is heated to about 96°, at a cost of 12c. per ton, results being improved by this system, the average extraction being 92 per cent.

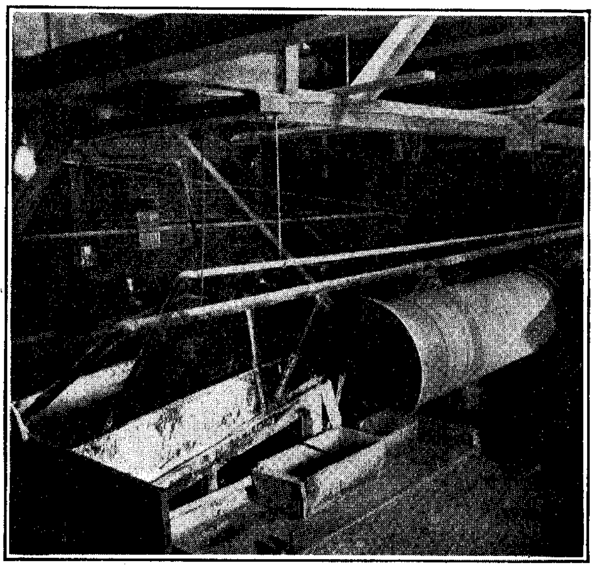
It is to be hoped that millmen in Cobalt, Mexico, and Waihi will give their experience with heating solutions. When I was with the Waihi company in 1898, heating was tried, but I kept no data. It may be interesting to mention that, at Kalgoorlie, treating an ore containing practically no silver, some argument was raised as to the benefit of hot solutions in treatment. The heating there is not intentional, as at Tonopah, but comes about through the hot roasted ore being mixed with solution, bringing it up to nearly 200°. The discussion resolved into whether by cooling prior to mixing there would be less consumption of cyanide, and less trouble with sulphates being deposited in launders, pipes, and pans, or whether there were benefits derived from the resulting hot pulp. The Associated, Associated Northern, and Kalgurli, more particularly, found that there was no appreciable decomposition, and up to 50% of the gold was dissolved at that point; while the Great Boulder, Perseverance, and South Kalgurli preferred to cool the ore before mixing.

Tonopah ores carry as much as 3% of pyrite, but concentration is not always employed, it being done only at the Belmont, Montana, Tonopah, and West End. It would seem that if the grade of the ore and percentage of mineral is not too high, tables are not necessary, and this varies from time to time in the various plants. At any rate, a very close saving is not attempted. The Extension company dispensed with their Deister tables, selling them to the West End. The Belmont, Montana, and Tonopah use Wilfley tables. Concentrate is collected, steam dried in large trays, sacked, and shipped to smelters. Freight and treatment cost nearly $70 per ton. It seems a pity that such a high cost is necessary, and that the product could not be handled locally by some central plant, which would treat the combined output at about $10 per ton.

All-sliming is the standard method, with the exception of the Tonopah mill at Millers, where three products are made; concentrate, sand, and slime. At this plant reduction is by stamps, and Chilean and Huntington mills; while at Tonopah the procedure is as follows. The pulp from the stamps is fed into Dorr duplex classifiers making 12 strokes per minute, from which slime overflows and coarse material is fed into tube-mills by means of a special feed. Discharge from these is elevated to the Dorr classifiers, and a further classification takes place, further grinding in the tube-mill, and so on. The only product which escapes to the slime plant is from the Dorr machines. This is termed the closed-circuit system, and is a good one. At the West End, the Dorr classifier discharge is:

	%
100 mesh	99
150 "	93
200 "	89

There are 16 tube-mills in the district, varying from 5 by



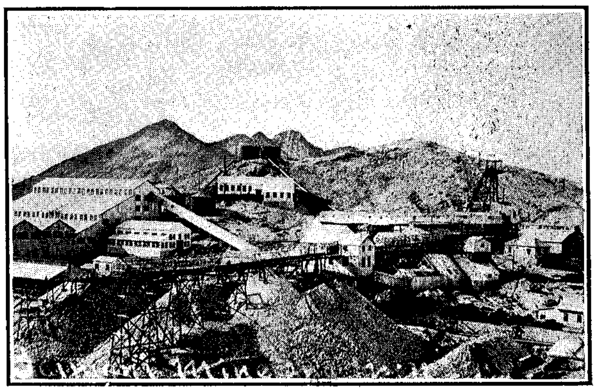
DORR CLASSIFIER, TUBE-MILL, CLOSED CIRCUIT.

18 ft. to 5 by 22 ft., revolving at about 26 r.p.m. Usually the locally-made smooth liners give good results. These are ½ in. thicker at the feed than discharge end, and last over nine months. At the Extension mill, there are two classifiers and two tube-mills in the closed-circuit, arranged thus in series: coarse pulp from No. 1 classifier is fed into No. 1 tube-mill which is fitted with ribbed liners, and is discharged to a bucket elevator, which in turn lifts the partly-ground material to No. 2 classifier which feeds No. 2 tube-mill fitted with smooth liners, the product being returned to the No. 2 classifier. The consumption of pebbles is 4.5 lb. per ton milled. It is claimed that ribbed liners are better on coarse pulp than smooth ones, and the latter can do the final grinding better. The ribs pick up pebbles and toss them about more in the mill, this giving extra impact for coarse material which is not necessary for the fine. At the Goldfield Consolidated mill this is also the practice. Pebbles remain in a better rounded shape with ribbed than when used with smooth liners, which tend to produce flat surfaces. When discussing the action of various crushing machines in the *Mining and Scientific Press* of December 10, 1910, I mentioned the corrugated liners used at Kalgoorlie affording a long life and extra grinding results. Some of the Tonopah mills find that it is not necessary to get an extremely fine product, the pulp being 73 to 89% through 200 mesh.

Various types of thickeners or dewaterers are in use, the practice being to allow the clear solution to overflow and decant off as much as possible for battery storage. When it gets too high in gold content it is decanted to the tank for precipitation. As at many other mining centres there is quite a difference of opinion regarding the efficiency of agitators, the Trent being used at the MacNamara, Montana, and West End; the Hendryx at the Montana; Pachuca tanks at the new Belmont mill; and ordinary mechanical agitators and air-lifts at the Belmont and Tonopah at Millers, these being in series at the Belmont plant. Centrifugal pumps and air at about 20 lb. pressure are used for the Trent system; and better results are obtained if pulp is drawn off near the top of a full vat and pumped through the arms as usual. Agitation proceeds for upwards of 48 hours. At the new Belmont mill, slime is first agitated in six Pachuca tanks, and from these it is elevated to Dorr thickeners by an air-lift, prior to going to another set of six Pachucas, making a total of 48 hours' agitation, the idea being to get rid of as much valuable solution as possible before sending slime to the filter-plant. Cyanide and lead acetate are added to the agitators, the former being from 2 to 5 lb. solution, while regular addition of the acetate is found necessary at all mills. Lime is usually slacked, and added to the tube-mill feed. Consumption of chemicals at the Extension is as follows:

	lb.
Lead acetate, per ton	0.9
Cyanide, per ton	2.5
Lime, per ton	3.5

Agitated slime is drawn off to stock-tanks, which serve the purpose of storage from agitators and excess from



NEW MILL, BELMONT COMPANY.

filter-plants. The latter have little of special note about them, they being of the ordinary stationary leaf type which have been described so often in technical papers.

Zinc-dust precipitation is used at the new Belmont and Montana mills, and zinc shavings at the Belmont, Extension, MacNamara, Tonopah, and West End. Methods of dealing with precipitate vary somewhat. At the new Belmont precipitate is dried, mixed with 5% of borax and smelted in double-compartment, oil-fired Rockwell furnaces, lined with carborundum, kaolin, and water glass. At the Extension it is dried, fluxed, and smelted in oil-fired Steele-Harvey tilting furnaces, which contain a No. 250 graphite crucible, while at the Tonopah mill the fine zinc shaving precipitate is incompletely dried, mixed with crude borax which swells up through the mass, and then is smelted in six coke-fired tilting furnaces. Crucibles last from 90 to 130 hours, and are turned once. Tonopah bullion will average 950 fine in silver, and a trifle over 10 in gold, and is sampled by being bored at opposite corners of top and bottom of bars. The bullion is shipped by freight like any other merchandise.

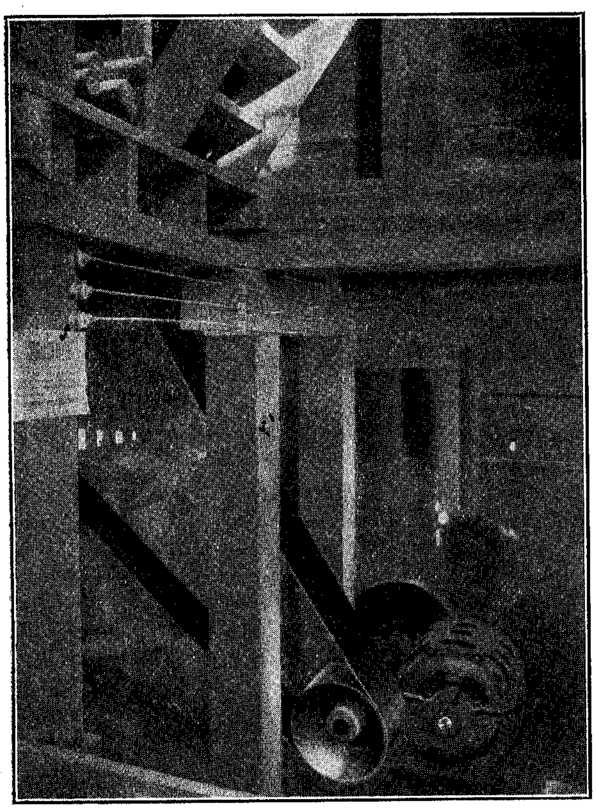
The mills at Tonopah are fortunate in having such a good supply of electric power as is available, and mechanical details are thus much simplified. I do not think there is a single engine in any mill there, also every machine having a motor drive. Chain drives are fast becoming popular, and apparently there is no dissatisfaction with them, once their peculiarities are understood. Such items as motor drives, and elevating pulp will be discussed later. In conclusion, it seems to me that generally the treatment of silver ores at Tonopah has been simplified to a fine point with good results, and is creditable to all concerned. There is not a dismantled mill, nor a badly designed one, nor any built for a mine without ore, to detract from the good work of the district.

21

The MacNamara Mill, Tonopah

By M. W. von Bernewitz

This modern little plant has been in operation for a year and is satisfactorily treating a hard ore carrying 20 oz. silver and a few grains of gold per ton. An electric hoist, with flat ropes, at the main shaft hauls self-dumping skips of 2000-lb. capacity to the surface, where the ore is emptied into a storage-bin. The ore is fed upon a 36-in. rubber sorting-belt; about 6% is discarded as waste, the remainder then passing through a No. 3 Kennedy gyratory crusher, and is taken to the mill-bin by bucket-elevator. This bin has a flat bottom, is of 200 tons' capacity, and has the ordinary rack-and-pinion gate on the ore-chutes to the feeders.



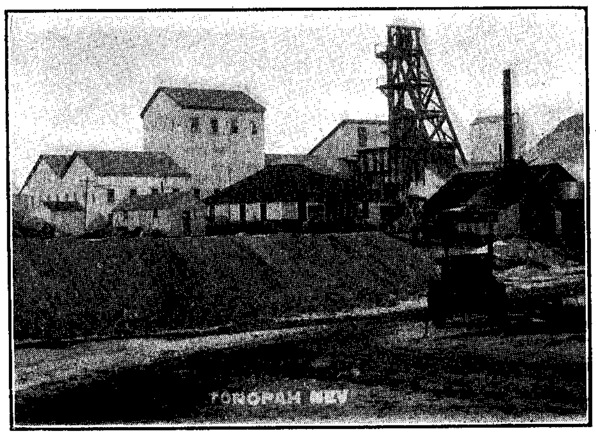
RACK-GEARED MOTOR DRIVING STAMPS.

The mill includes ten 1400-lb. stamps, dropping 8 in., 98 times per minute, and crushing 7 tons per stamp per day, through a No. 12 ton-cap slotted screen. As will be seen from the accompanying illustration, each 5-stamp battery is driven by a 20-hp. Westinghouse back-geared motor. This may be described as a type 'M.S.', 440-volt, 60-cycle, 3-phase motor, running at 555 r.p.m., the pinion on the armature shaft engaging with spur wheel on the short pulley-shaft, which runs in bearings on the back of the motor, there being a 14-in. belt driving the bull-wheel on the 6½-in. cam-shaft. These motors have given complete satisfaction, and they were recently opened after about eight months' work, inspection showing that the tool marks were hardly worn off the gear-wheels, which work in grease. No doubt this style of drive reduces friction greatly, as there is no counter-shaft, except the little pulley-shaft, and no tightening gear for the belt. The friction in this case amounts to about 90%. Careful tests have shown that five stamps absorb 17.75 hp., including 1.61 hp. for friction, or 3.55 hp. per stamp. As a check on this reading, the following old rule can be used:

$$\frac{1400 \times 98 \times 8}{33{,}000 \times 12} = 2.77 + 20\% \text{ for friction} = 3.32 \text{ hp.}$$

It may be added that at the new Belmont 60-stamp mill there is a 60-hp. motor for 20 stamps, which weigh 1250 lb. each. This drives a short jack-shaft, which in turn drives two batteries of 10 stamps. This is similar to the drive at the Goldfield Consolidated, only there the motors are 50 hp. each, and at the Desert mill of the Tonopah Mining Co. the same is true. The West End mill has four units of five stamps each, actuated from a common motor-driven counter-shaft by friction clutch-pulleys, which arrangement gives no trouble. Besides the two motors driving the MacNamara stamps, there is a back-geared motor driving a Dorr classifier and agitator in the thickening-tank. This method of transmitting power is well worth the attention of all engineers.

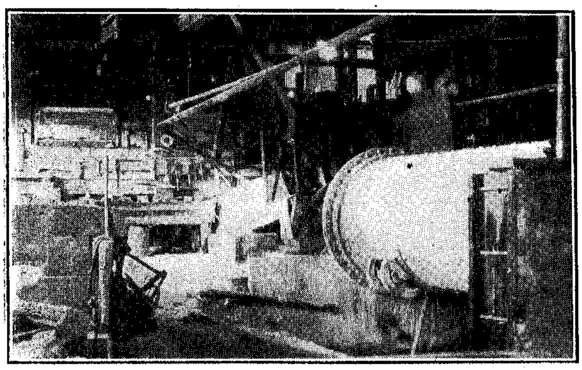
Foundations for the two batteries are of reinforced concrete. The mortars (Chalmers & Williams, No. 16-E) have a heavy base, and stand on sheets of rubber; while the battery-posts are set in cast-iron sole-plates, also standing on rubber. The cam-shafts are 6½ in. diam. The ordinary Blanton cam is used, and none have been broken so far. Stems are 4 in. diam., working in Pacific guides made by the Demarest company, and none have been broken. The Pacific guide consists of a metal frame into which are fitted cylindrical metal shells that are not held by keys or the like. The screens are fitted 3½ in. from the stamp-shoes, and through a No. 12 ton-cap square-slot screen the average duty per stamp was 7 tons of ore per day, but recently coarser screens have been used, and the duty has risen to over 8 tons. Crushing is done in weak cyanide solution having a temperature of about 70°F. Feeders of the improved Challenge type, which may be called a double monkey-wrench grab, deliver the ore to the stamps.



MACNAMARA MILL.

The pulp from the stamps goes direct to a Dorr classifier, working at 12 strokes per minute, the coarse material being fed into a 5 by 16-ft. tube-mill revolving at 26 r.p.m., through a spiral feeder. Pebbles are fed in at this point, amounting to 250 lb. per day, consumption being about 3½ lb. per ton milled. Slacked lime equal to 2½ lb. per ton is added in the classifier. The tube-mill discharge is returned to the classifier by a 5-ft. Frenier pump, which gives no trouble, the coarse material again passes through the tube-mill and so on, forming the usual closed circuit, in which only the overflow from the classifier can get away. In the batteries, 25% of the ore is slimed, while the final pulp, which is pumped to a Dorr thickener, shows 73% through 200-mesh screen. This is fairly coarse when compared with other mills, but is found to be fine enough for good results.

All the classifier overflow is pumped to a Dorr thickener, 12 by 26 ft., the gear of which travels at one revolution in eight minutes. Clear solution overflows to the battery storage-tanks, and when its silver content becomes high, it is decanted off to the 'silver' tank. The thickened slime, specific gravity 1.22, flows by gravity to three 15½ by 25½-ft. Trent agitators for 48 hours' agitation in a 2-lb. KCN solution. The agitators hold 80 tons of dry slime each, and cyanide and lead acetate are added here, as well as live steam, bringing the tempera-



STAMPS, CLASSIFIER, AND TUBE-MILL.

ture up to 115 to 120°F. This has been found beneficial, as with cold solutions extraction falls off considerably. This type of agitator gives satisfaction, and even after an enforced shut-down, little trouble is experienced in starting. Air for the agitators is produced by a motor-driven compressor with 6 by 10-in. cylinders, the working pressure being 25 lb. per square inch. Each agitator has a 4-in. Campbell & Kelly centrifugal pump, direct-driven by a 7½-hp. motor, for circulating the slime through the agitator arms. From the Trent vats, the slime is pumped to a 19½ by 27½-ft. stock or storage-tank, which has a chain-driven pump for circulation.

From the stock tank, pulp flows by gravity to the vacuum plant, which consists of 50 leaves, each 5 by 10 ft., and treating 20 tons per charge. Forming cakes takes one hour, and washing two hours, with a 5-minute final water-wash at times, while the whole cycle consumes about four hours. The cloths last well, and have been on about 12 months so far. They are given a 24-hr. bath in 2% hydrochloric acid, about two leaves being treated per day. The washed slime-cakes are blown off with high-pressure water into a full tank of water, and the pulp pumped away by a 4-in. pump to ponds. Although the slime from the storage-tank goes to the filter-vat by gravity, it is helped by a belt-driven Campbell & Kelly pump, which also pumps excess back to that tank, and does the circulating in the filter-tank. In October last the filter-tank was struck by lightning and completely destroyed, save the leaves; but in a short time a new one was built.

Solutions from the filters are clarified and run through four precipitating-boxes charged with zinc shavings. The precipitate is dried to about 10% moisture in a steam-drying pan, mixed with fluxes, and melted in an oil-burning tilting-furnace. An extraction of 93% is made. The operating cost per ton in November 1912 was as follows:

Crushing and conveying	$0.176	Assaying	$0.071
Stamp batteries	0.340	Superintendence and foreman	0.233
Dorr classifier	0.145	Heating solutions	0.253
Tube-mill	0.523	Water service	0.178
Dorr thickener	0.030	Compressed air	0.017
Agitation slime	0.776	General expense	0.073
Filtering and discharging slime	0.237	Surface and plant	0.050
Precipitation	0.116	Total cost	$3.299
Refining	0.097		

New York Metal Market

The report of the Copper Producers' Association for December, issued January 8, not only showing an increase in stocks of over 19,000,000 lb., but permitting the statistics of the year to be viewed as a whole, on the heels of which came news from abroad that the Balkan situation was not mending, marked a turning point in the copper market. Immediately following the report prices softened, then declined. Future delivery metal was especially hard hit. March and April 'delivered' was in a few days offered at close to 16c., and on January 13 sales were made around 16⅛c. for these months. Both domestic and foreign business was eagerly sought, even some of the large producers directly offering concessions, or doing so indirectly by asking for tenders. Some of the trade declared that an absolutely open market existed. On January 13 an offer of electrolytic at 17⅛c., immediate delivery, was met with the statement that a large producer would sell for 17c. On January 16 the market continued unsettled and irregular, although slightly firmer because of a more pronounced disposition on the part of consumers to buy. Some buying was reported on a basis of between 16.25 and 16.40c. for delivery in Europe, with domestic sales at 16⅜c. for February and March. The situation abroad was weak from similar causes, and between January 10 and January 15, standard or speculative copper (which is distinct from copper metal, but which has a reflective effect on the commodity) dropped £6. Despite the break, consumers who have stayed out of the market except when they could not avoid buying, have continued to hold off, apparently in the hope of still lower prices. Many of them are emphatic in their assertion that 15c. is enough to pay for copper. While a good domestic buying movement will stiffen the market, it will not resume the hardness characteristic of so many months until European buying in good volume is resumed.

23

Continuous Agitation at the West End Mill, Tonopah

By Jay A. Carpenter

Pachuca vats have been synonymous in technical literature with continuous agitation. Perhaps this is not so much due to the special adaptability of the Pachuca to continuous agitation as to the fact that the endeavor to find a more satisfactory method of agitation led to the introduction into common milling practice of the Pachuca vat, and continuous agitation at nearly the same time. The Pachucas have been sharply criticized for the large initial cost for a given capacity, and for their awkward height, which prevents either gravity flow into them, or a symmetrical design of a mill building to contain them. Their simplicity of operation is without dispute, but the statements of low power consumption have been sharply questioned. Continuous agitation has been called a fad, and in several cases tried and abandoned. Still the fact remains that companies such as the Santa Gertrudis and Tonopah Belmont, with ample capital and with the best of engineering talent, employ Pachucas with continuous agitation in their modern mills. It is evident from the study of the files of the technical magazines, that the unstinted praise given to the Pachuca in 1910 and 1911 has gradually changed to the sentiment that it holds its supremacy, not through its vastly superior work, but from the fact that no one of the many less costly types of agitators has as yet proved to the general satisfaction that under varied conditions it is the equal of the Pachuca for reliability and low operating costs.

Classification of Agitators

The agitators in use may be classified as follows, giving a well known example of each type.

1. Using a cone-bottomed vat, the motive power being: (*a*) compressed air, the standard Pachuca and the variations from it; (*b*) solution under pressure, the Patterson hydraulic agitator; (*c*) mechanical appliance, the Hendryx.

2. Using a flat-bottomed vat, the motive power being: (*a*) compressed air, the Parral; (*b*) solution under pressure, the Trent; (*c*) mechanical appliance, the old arm agitator; (*d*) mechanical appliance and compressed air, the Dorr agitator.

The variations from the standard Pachuca consist in using lower and wider vats with a flatter cone bottom. The higher first cost of the standard Pachuca is thus reduced, but it is an open question if at the same time the best features are not minimized. However, such an eminent authority as Philip Argall speaks very highly of this type of agitator in use at the Independence mill, at Victor, Colorado. In the hydraulic agitator, the form is the same as with the Pachuca, but solution under pressure from pumps is substituted for compressed air. This type is in successful use at the Portland mill at Cripple Creek, but like many other types of agitators convincing data are not presented by those who sing its praises. The Hendryx, using a mechanical device where the above two use air and solution, gives excellent agitation, but is handicapped by the small units used, and high power costs.

In all of the above, a form of vat is used that is more costly to buy and install than the flat-bottomed ones used in the other forms.

The Parral vat is an adaptation of the Pachuca air-lift principle to a flat-bottomed vat. It has much to commend it in cheapness of first cost and low cost for power and repairs, but to the average millman, it does not seem that the rotary action is positive enough to guarantee successful agitation under the varying conditions to be met with in practice. The Trent agitator is an adaptation of a form of hydraulic agitation to a flat-bottomed vat. The action is more positive than in the Parral, and although simple in design there are many wearing parts and the agitator requires careful attention to details to get successful results. Its apparent simplicity of operation and cheapness of first cost has won at many mills, but the fact that it has been abandoned in several cases shows that its simplicity is more apparent than real. The old style arm agitator that was abandoned in such haste for the newer designs is being returned to in a few cases as an old and reliable friend. Its efficiency has been increased by the addition of air lifts on the periphery, and it is probable that in new plants where the speed of the arms is reduced with the idea of depending partly on the air lifts for additional agitation, it is no longer to be sharply criticized for its power consumption. The Dorr agitator carries the change in the old arm agitator one point further by using the slanting arms of the Dorr thickener moving at a faster speed than in the thickener for the arm agitation, but depending to a greater extent upon the air lift which is centrally placed and fed somewhat thickened pulp by the arms. This type has won upon its apparent merits at several plants, the results of which are awaited with much interest.

Requirements for Continuous Agitation

Only part of the agitators mentioned above are well adapted for continuous work. For such work the pulp should be nearly the same specific gravity and the slime of the same screen test at all points in the vat, and the currents in the latter set up by the agitation should be of such a nature that they will aid in insuring the new pulp a definite length of time in the agitator before any part of it reaches the outlet. The standard Pachuca, when it is given from 75 to 100 cu. ft. of free air per minute at 30 lb. pressure, probably best fulfils these conditions. Cases can be cited in which it was necessary to increase the air to this amount before continuous agitation could be made a success. The variations from the Pachuca cannot be well adapted to continuous work.

The old style arm agitators, with the aid of peripheral air lifts, are in successful use for continuous agitation at the old Belmont mill in the Tonopah district, yet the same method was tried and discontinued at its neighbor, the Desert mill of the Tonopah Mining Co. Why such opposite results should be obtained is difficult to understand. An accurate description of this method of continuous agitation at the Belmont would be of great interest.

The Parral and Trent agitators can both be used successfully for continuous agitation, providing, as in the Pachuca, that sufficient power be used to give the proper conditions in the vat. Bernard MacDonald has thus operated the Parral alongside the Pachuca, claiming for it a better extraction with less power consumption. It is unfortunate that with such an excellent opportunity for comparison, and with the excellent detail so ably presented by Mr. MacDonald, he did not use a more accurate method of measuring the air consumption than by comparing valve openings. From the description of the Dorr agitator, it would appear well adapted for continuous agitation if sufficient air be used. As to the Trent agitator, it is the purpose of this article to describe a successful installation with continuous agitation.

Each Has Its Place

In fairness to all these agitators, it may be said that each may be best adapted to some special ore and particular operating conditions, and since they are rarely found working under exactly the same conditions, and when they are these conditions may favor one of them, it would seem that the millman should not base his conclusions too much upon his own personal experience. There is a lamentable lack of accurate and detailed data upon the use of these various agitators. In searching the files of the technical journals for such data, one finds many general conclusions expressed upon the subject, or just enough data to make it easily possible to draw wrong conclusions. It is the purpose of this article to go into careful detail concerning the operation of one of these agitators, the Trent, with the hope that it will lead to a discussion of the subject of slime agitation based upon accurate data. Since the Trent is doing excellent work in many mills, and in others it has been thrown out, it is an appropriate subject for such an article.

The Trent agitator was invented by L. C. Trent of Los Angeles, modified from the overfeed to the underfeed type at the Montgomery-Shoshone mill at Rhyolite, Nevada, and brought into prominence by its successful operation at the Tonopah Extension mill, where the necessary changes and details of operation were worked out under the direction of the mill superintendent, J. P. Montague.

The principal features of this agitator are perhaps sufficiently familiar. The solution is agitated by means of a centrifugal pump delivering to a revolving arm agitator in the bottom of the vat through a submerged grit-proof bearing in an air-sealed chamber and with provision made for the small amount of slime pulp escaping downward through the joint, to flow out into the vat through the ports without rising up in the ball-race chamber. The bearings consist of the top and bottom castings. The bottom casting is provided with a renewable ball race and a ring to hold down the top casting in case it tends to rise. The top casting is one solid piece provided with a head into which are screwed the arms and upon which is bolted the mast. The method of operation is simplicity itself.

Importance of Pump

The centrifugal pump, which is not furnished with the agitator, should be considered an integral part of it, for the choice and operation of the pump has had more to do with the success and failure of the agitator than the agitating device itself. Upon the pump mainly depends the power consumption, the cost of renewals and repairs, and the behavior of the agitating device. To make the installation a success, the pump must be able to handle a gritty pulp day in and day out without undue wear, attention, or power consumption. The use of centrifugal pumps for transferring slime pulp from the bottom of cone-bottomed vats to the top was abandoned as a method of agitation, because of the power and repair costs. With this in mind, it was the intention of the inventor of the Trent agitator that the pump used on the agitator should only pump the nearly clear solution from the top of the agitator, thus avoiding the wear on the pump. This entails operating the agitator so that the pulp shall be thick on the bottom and thin on top. With the centrifugal pump made with chilled iron or manganese steel liners and runners and operated at a comparatively slow speed, it has been found that the wear on the pumps is only nominal, and that this fact in the case of the Trent agitator makes it advisable to pump a large volume of slime pulp, which, although it causes somewhat greater wear and greater power consumption, keeps the tank of nearly the same specific gravity top and bottom, thus giving a more thorough agitation and preventing the chance for quick settling of heavy pulp on a shut-down, making starting difficult or impossible.

There are many standard makes of pump not adapted to this work, either because of the speed necessary with the runner used, or of the wear and cost of the rather intricate repair parts. Again, the same make of pump will give varying results at different plants, due to the speed at which it is driven, or to the number or size of nozzles on the agitator.

Why Wear Varies

A study of the table, 'Average Performance of a Centrifugal Pump,' on page 886 of the 'American Civil Engineers' Pocket Book,' will aid in understanding why the power used and the wear on the pump varies so greatly at different plants employing the Trent agitator. This table is based on the fact that for a given centrifugal pump there is one speed against one head that gives the maximum efficiency, and that as this head is increased or decreased, the volume pumped decreases or increases, but the efficiency decreases rapidly in both cases. This decrease is due mainly to the slippage of the pulp in both cases, which increases the wear.

In the Trent agitator, the pulp must issue from the nozzles with sufficient force to cause the arms to turn. With a given pump running at a fixed speed, the area of the nozzles determines the head or pressure against which the pump works. By chance this may just coincide with the maximum efficiency of the pump for the given speed, in which case the Trent agitator will be praised for its low power consumption. If the pump is working against only 50% of the correct head for the driven speed, its efficiency is but 66% of the best conditions, or if it is pumping against a 40% greater head, its efficiency is likewise only 48% of the best conditions. Since the efficiency of a centrifugal pump at its best performance is relatively low, such a drop as those indicated above make the pump an inefficient machine. Perhaps this partly explains the widely varying published figures of the horse-power consumption of Trent agitators. These range from the Goldfield Consolidated figures of 7½ hp. for 200 tons dry slime in a 1½:1 pulp, to the Hollinger figures, as given by Herbert Megraw, of as high as 18 hp. for a vat of 30 ft. diameter and 15 ft. high, filled with pulp heavier than 3:1, and probably containing 110 to 140 tons of dry slime.

Number and Size of Nozzles

When the agitator is revolving in good running condition, that is, revolving freely on the ball race when spun around in the empty tank, and all nozzles cleaned, it may be turned readily by hand when filled with pulp over the arms, showing that the force at the nozzles required to turn it must be small. It should continue to turn easily as long as enough pulp is introduced through the nozzles to counteract the natural tendency of the slime to settle, and with sufficient velocity to turn the agitator. Since the number and size of the nozzles plays such an important part in determining the head against which the pump works and the volume pumped, it is evident that a change in the size of the nozzle openings will play an important part in increasing or decreasing the efficiency of the pump. For instance, if a pump at a given speed is not turning the agitator fast enough, instead of running the pump faster to increase the agitator speed it is quite possible to obtain the same result by removing a part of the nozzles from the nipples, or increasing each nozzle area. Possibly the correction may be made the opposite way, by decreasing the size of the nozzles, depending upon whether for the greater speed of the pump head should be less or greater to get the maximum efficiency. At the Tonopah Extension, for an experiment, the agitators were run for a few weeks without nozzles at all, the greater volume pumped making up for the lower pressure and effecting the same result.

When the agitator is driven at a speed of about 3 r.p.m. by a slime pump working under an economical speed and head, it will successfully agitate week in and week out without any trouble or repairs, a slime pulp from 1.16 to 1.30 specific gravity with the slime varying from 60% through 200 mesh, to 90%, as long as the pulp does not contain wads of waste and wood pulp, broken bottles, tin cans, or other such material. Such an agitator can be stopped for several hours and started without trouble when working on thin pulp and coarse slime, and for a day or more when on thick pulp and fine slime.

A Few Difficulties

The following remarks concerning the operation, wear, and repair on this type of agitator are the result of watching the operation in several other mills, talking with operators from other districts, and of operating them for eighteen months in the milll under my charge. Having in the first few months often poked with a long stick at the mast and fished with a grab hook for the arms in a vain effort to make the agitator turn, and having waded in slime pulp pushing on the arms, I agreed with Whitman Symmes that the agitator hardly justified the effusive statement of the inventor, that with it "cyaniding, formerly a complex nightmare, becomes a delightful experience," but I am now sure that if the proper study and care be given the agitator, such performances are unnecessary and it will give service as satisfactory as the Pachuca.

The most frequent complaint against the agitator is the stopping of the nozzles by waste material. Since this same material gives trouble, but to a less degree, in launders, pipes, cones, thickeners, and pumps, it should be screened out at every available place; such as at the end of the classifiers, in the launders, over the tops of cones, and settlers. If this precaution be taken, with footboards on all walks, and the millmen interested in the subject, this complaint will be of minor importance.

Wear of Ball Race

The wear on the noozles and pipe arms is next to nothing, but the ball race will wear in spite of its protection. This wear varies greatly, depending on the amount of slime reaching it, which in turn depends upon how smoothly the agitator is operating. If the escape parts in the bottom casting become choked with settler slime, and a sudden jolt of air be given the agitator, or the agitator wobbles as a result of choked nozzles, then an extra amount of pulp passing the joint may flow over the ball race. The balls running in slime cut both the top and bottom race about equally and wear down in size. Since the clearance is made entirely too small on the original castings, the ball race of the upper casting may soon be rubbing on that of the lower casting, and the friction generated will gradually slow and finally stop the agitator. The remedy for this is to raise the top casting and chisel down the edges of the lower ball race, which can be done in an hours time, insuring at the West End mill several months run before repeating. The ball race in both the top and bottom castings should be made separate to be easily renewable. However, it will not be necessary to machine the castings and fit in new races until after two years or more of wear.

Another source of trouble met when an excess of air is suddenly applied or the pumps started with the nozzles pointed too steeply toward the bottom, is that the top casting rises and shifts so that the casting is suspended or hung up on the joint. The quickest way to right it is to empty the tank. In one case, to my knowledge, a raft was built and launched upon the cyanide sea in a vain attempt to right the submarine. In the latest designs a holding down ring prevents this distressing accident. In

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the older type, J. W. Harcourt, master mechanic at the Desert mill, prevented this occurrence by extending the joint to 4 inches in length. Before the top could rise this high, it would bind on the joint and settle back in place as soon as the pressure was relieved. This joint should also be made with two removable bushings so that they could be easily replaced after a couple of years service, when the small stream of pulp has cut them enough to allow too large a stream of pulp to escape for the parts to handle. The expense and labor of inserting two each of the new cast iron ball-race rings and bushings would be but a small item and would make the agitator as good as new. A set of balls will wear from one to two years, and as they grow smaller, the race should be kept filled by addition of balls of a size.

Starting the Agitator

If, after a power shut-down, or pump repairs, the agitator fails to turn, it is due to the accumulation of thick pulp or sandy material back of the arms where the nozzle streams cannot reach them. Often starting the pump several times or rocking the mast will aid the arms to pass over or through this mud, and the nozzle then cuts it away, or a grab-hook caught on one of the arms and carried around a half turn may be used. Where the platform runs over the centre of the tank, as at the Rawhide mill, the men easily turn the agitator by hand power, turning opposite the usual direction of rotation, which causes the nozzles to clean the bottom of the tank in advance of the arms. After a part of the turn, the agitator, upon release of pressure, will reverse and the arms will then rotate freely and there will be no more trouble.

Gaining by Experience

In the first few months of operation at the West End mill, many of these troubles were met, but we had the advantage of our neighbor's experience, and at the end of six months the agitators were running so smoothly and with so little attention needed, that I devised a simple and effective method of continuous agitation and put it in operation. The essentials of this system are: (1) that the pulp be drawn from each tank of equal specific gravity and fineness as to that which enters the vat, in order that there will be no gradual accumulation of thick or coarse pulp; (2) that the chance for a quick passage of new slime through any one of the agitators into the next of the series be reduced to a minimum.

With the Pachuca, the average agitation given for continuous work is sufficient to keep all parts of the tank of the same specific gravity. The greatest trouble with its rapidly moving centre column of pulp the danger of carrying new pulp to the next vat, with but a short contact. After considerable discussion of this subject, the general conclusion seems to have been that this danger became of little consequence when 8 to 10 agitators were used continuously.

A Record of Tests

Tests upon the Trent agitators before installing continuous agitation, showed the following specific gravity and screen tests:

Record of Test on Trent Agitator

Depth of sample, feet.	No. 220, 90% through 200.	No. 222, 2 hr. agitation after 4 hr. shut-down.
4	1.208—1.190	1.242
9	1.192—1.208	1.240
15	1.206—1.208	1.242

On No. 220, 500 c.c. of pulp was screened through 200 mesh.

4 ft.	gave	27¾	gr. on	200 mesh
9	"	27	"	"
15	"	28¼	"	"

There being as great differences between samples at the same depth as at different depths in the agitator, it was evident the agitation was as thorough as that of the Pachuca.

Having but four agitators, the second point was of great importance. Observations were made to determine the currents set up in the agitator. There seemed to be no definite rotary current, probably due to the fact that the arms moving in an opposite direction to the nozzle streams destroyed all currents. There was no rising current at the side of the vat, probably due to the force of the nozzle streams being exhausted before they had reached the side. It was evident that pulp delivered evenly to the bottom of the vat would not be subject to the action of currents bringing portions of it quickly to the surface.

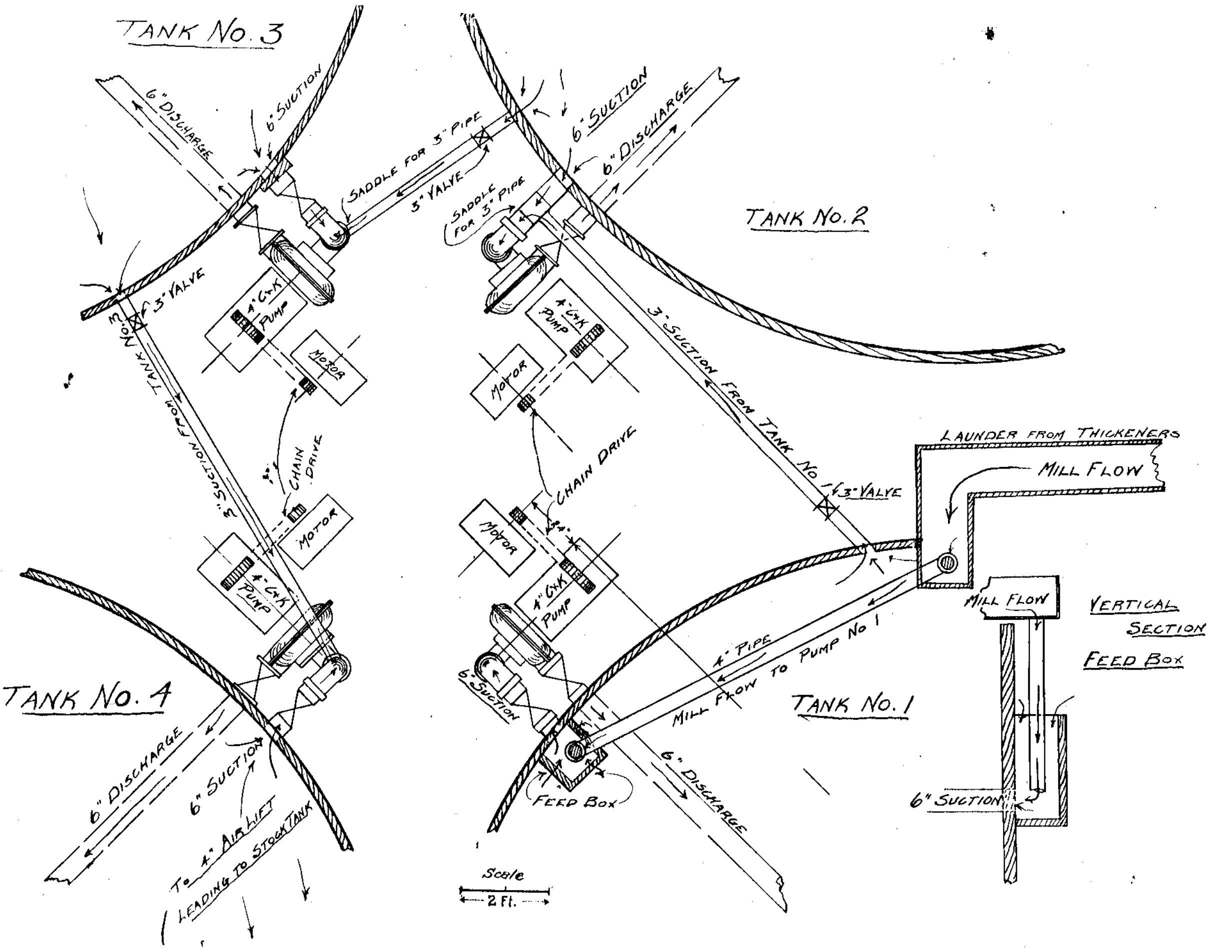
Arrangement for Continuous Work

Since the total pulp in the Trent agitator is not handled but once in three hours by the pump, it was evident that if the new pulp could be introduced in the pump suction and thus distributed over the bottom of the vat and the old pulp withdrawn at the top of the vat there would be little likelihood of any new pulp short circuiting through the vat. The accompanying illustration shows how this is done, using the agitator pumps as the means of transfer. The pulp from the crushing department is lifted from the bottom of Dorr thickeners by diaphragm pumps into a launder which delivers it at the edge of agitator No. 1 into a 4-in. pipe, which in turn delivers it to the bottom of a box over the suction of the pump. Since this is but a part of the pulp necessary to feed the pump, it is all drawn into the suction pipe by the current of pulp flowing into the box from the top of the vat.

The 6-in. pump suction of agitator No. 2 has a branch 3-in. suction to agitator No. 1 entering it at 3½ ft. from the top. Through this suction it draws pulp from No. 1 agitator fast enough to keep No. 2 level with No. 1, and if the flow into No. 1 is stopped, No. 2 will build up to 4 in. higher than No. 1. The pulp from No. 1 thus is distributed over the bottom of No. 2 tank and must rise nearly to the top of the vat to pass into the branch suction of No. 3 pump, thence to the bottom of No. 3 vat. Practically the only regulation required for this continuous system is that of the air lift taking the pulp from No. 4 agitator to the large Dorr thickener. In the design of the mill the thickener floor being limited, sufficient agitators were installed so that each agitator in turn was used as a collector until filled, agitating the pulp meanwhile. By changing to the continuous system, the present 150 tons daily capacity of the mill is given 54 hours continuous agitation, whereas the 120 tons under the charge system received 58 hours.

After operating the continuous system for a year's time it was found that after two to three months of steady running the mast of about one of the four agitators would begin to describe a small circle, showing that part of the nozzles are choked. This does not mean that it needs immediate attention. When convenient an extra filter charge is run, making room to transfer No. 4 agitator to the large Dorr thickener. A rope is stretched across the vat and the top casting is raised by chain-block high enough to expose the ball race. The nozzles are looked into and cleaned, the ball race washed out if slimy, and additional balls added, and the race chipped if necessary. Then No. 3 is transferred to No. 4, partly by gravity through common suction to the transfer pump, and No. 3 given the same attention. Then 2 to 3 and 1 to 2, and then No. 1 is allowed to fill. The arm needing attention the worst will be found with many of the nozzles choked, and often most of the nozzles choked will be on one arm, which has caused the wobbling of the mast. The bottom of the vat needs no attention whatever, as there is no settlement of slime or sand. It takes about 10 to 16 hours to go around the circuit; about half the time is taken by the mechanic for his inspection and work, the rest being taken for transferring. The labor of transferring and inspection at such periods is small compared with the weeks of steady work insured.

The assumption upon which the Trent agitator was placed in the mill in preference to the standard Pachuca was that although it would cost more to operate, it would require three years of operation before the extra expense added to the original cost would equal the capital outlay for Pachucas. The comparison of cost was based upon figures of costs for the Pachucas at the Goldfield Consolidated and for Trent agitators elsewhere, with a liberal allowance for the power and maintenance cost of the Trent. After two years' experience I believe this statement is nearly correct. The cost of the Pachuca vats is now lower than at any time, but the cost of operating and maintaining the Trents is lower than was estimated. The comparison of cost is as follows:



ARRANGEMENT OF TRENT AGITATOR FOR CONTINUOUS WORK.

TRENT AGITATORS (F.O.B. TONOPAH)

Four 24 by 18-ft redwood tanks, each 7000-cu. ft. capacity	$2120
Four Trent agitators with piping	2044
Four 10-hp. motors	760
Four 4-in. C. & K. pumps	720
Four 10-hp. chain drives	140
One air-compressor for 80 cu. ft. at 20 lb., with 10-hp. motor	400
Erection	500
Total	$6684

PACHUCA VATS

Four 15 by 45-ft. Pachucas, 6800 cu. ft. each, weight 29,000 lb., $883 each f.o.b. Milwaukee	$3532
Freight to Tonopah, $573 each	2292
Erection at estimate of 2c. per lb	2320
Air-compressor for 320 cu. ft. of air at 30 lb., with motor, erected	1500
Total	$9644

The cost of operating the Pachucas at 6.5 hp. per vat, I believe is moderate, especially since this is nearly the minimum figure based on 80 cu. ft. free air at 30 pounds.

Power Consumption

After 18 months the cost of operating the Trent agitator is found to be as follows: With the agitators filled with the average charge of 88 tons of ore in a 1.25 specific gravity pulp, the horse-power consumed by the 10-hp. motors, being the horse-power input taken with a polyphase wattmeter, was 6.8, 5.9, 6.8, and 6.5 hp. respectively, or an average of 6.5. This reading includes all motor and transmission losses. Power data as given by the technical press are generally published without giving the method by which they were figured. Often the millman bases his figures on whether the given motor runs cold or hot, irrespective of the fact that motors vary so greatly as to make this method a joke. Also, very often, the assumption is made that a 10-hp. motor takes 10 hp. when running full load, while a wattmeter will show considerable excess, depending on the motor efficiency. Others who take a wattmeter reading, figure the horse-power consumed by a unit, such as the Trent agitator or a compressor for Pachuca work, as the horse-power delivered by the motor, thus deducting motor losses as a separate item. Others, over-zealous people, may figure the transmission loss as a separate item. The air used by the agitators amounts to 80 to 100 cu. ft. piston displacement at 16 lb., estimated at a total consumption of 6 hp. This makes the total consumption for each agitator 8 horse-power.

Pump Repairs

The cost of maintenance for the agitators also has been less, due to a choice of Campbell & Kelly pumps; for the greatest item of maintenance is in the repair of pumps. This pump, made at the local foundry and machine shop, has an enviable record for service and has replaced many pumps of standard makes in this district. Its advantages are in the ease with which repairs or changes can be made to any part, in the simplicity of design, and the durability of the wearing parts. Our records show that the pump on No. 2 agitator was started September 20, and ran practically 24 hours per day until June 20, 9 months, without a repair or any mechanic's time. At that time it required a new shaft and runner, and on October 1, over a year, it required a new liner for the shell. The average life of the repair parts necessary, has been a new shell liner every 10 months, a runner and shaft every 6 months, and the main bearing re-babbitted every 6 months, the wear on shafts and bearings being due to slime in the solution fed to the packing gland. The yearly cost for these repairs is $50 for materials, and $18 for labor. Renewals for chain drives and agitator parts, with all labor for maintenance, added to the $68 for the pumps bring the total for maintenance and repairs for each agitator to about $100 per year. The maintenance cost of the Pachuca system complete would easily total $60. For the four units the power consumption is 6 hp. greater than for Pachucas, or $630 per year. The cost of maintenance is $340 greater. On this basis the Trents would cost $970 per year more to maintain and operate, but the installation would be $2900 less, or nearly the amount of three years' operating costs.

Accurate comparison of the two agitators as to extraction and cyanide consumption is impossible in this case. However, the perfect aeration of the Trent has much in its favor, the air being minutely subdivided and slowly rising through the charge. The blanket of foam on the vat acts as a nonconductor, preventing radiation from the heated pulp, in sharp contrast to the centre column of the Pachuca. With the continuous system on the coldest days there is no mist over the agitators. When solutions are heated to 90 to 120°F., the absorption and radiation of heat by compressed air becomes an important consideration. Since the extra cost of the operation of Trent agitators above Pachucas, in our case, amounts to less than 2c. per ton, it is probably easily paid back in extra extraction and heat saving.

In Other Mills

The description above of the satisfactory work of Trent agitators is by no means an isolated case, for in Tonopah alone, the Tonopah Extension and the MacNamara mills, while not attempting continuous agitation, obtain equally and in some particulars better results with these agitators. At the Extension mill, the superintendent, J. P. Montague, gives the motor horse-power as 6, and the air consumption as 25 cu. ft. of free air at 18 lb. for the 24 by 16-ft. agitators. He has discarded the round nozzles for flattened nipples which allow waste materials to pass through more readily. Shell liners and runners last six to eight months, and shafts and bearings, due to using water on the packing glands, last over a year.

At the MacNamara mill, the superintendent, J. B. Tregloan, gives the following figures: A charge containing 97 tons of ore is agitated with a total of 9.5 hp. A charge of 1.26 specific gravity, with the slime screening but 70% through 200 mesh will not vary over 0.01 in specific gravity between top and bottom, and screen test within 5% through 200 mesh. At this mill the ore is almost entirely pure quartz without kaolin, and the sulphides show no oxidation. The ore has a specific gravity of 2.64 and the pulp is white in appearance and contains a small amount of colloid as distinguished from fine crystalline matter. In spite of the quick settling of this ore, these agitators have been stopped for days and started in a few minutes. There are other installations of Trent agitators in the Tonopah district that often grow weary and stop for the lack of a kind word. When Trent agitators give results as satisfactory as in the three mills mentioned above, they surely compare favorably with the standard Pachuca. On the average millsite they will reduce the costs over a Pachuca installation in (1) the excavation and concrete work on the millsite, (2) the building cost to cover them, and (3) the outlay to buy and install them. The small extra cost per ton to operate them is probably balanced by the saving in chemicals, heat, and elevating costs. When properly erected and operated they closely approach the Pachuca in its best feature, that of reliability of operation.

Operation of the West End Mill, Tonopah

By Jay A. Carpenter

*While costs might be lower if the mill were situated at the mine, where certain expenses could be shared in common, they are at present at a figure that compares favorably with mills of an equal or greater tonnage. The extraction of silver and gold for the year, while nearly the average obtained, falls short of the best results in the district. During the first quarter of the year it was 88%, due to treating over $17 ore, containing coarsely crystallized ruby silver, without concentrating. Unexpected mine developments giving this grade of ore after belt sorting for smelter shipments, caused prompt action to be taken in adding the concentration addition, the excavation for which was started May 20, and the unit completed and put in operation July 1, 1912. With the increased tonnage and improvements made during the past year, the current term should show a further decrease in costs and an increase in extraction.

Treatment Operations

Results may be stated as follows:

Ore treated, tons	44,756
Gold content, ounces	9,357
Silver content, ounces	93,337
Gross value of ore	$764,854
Gross value per ton	$17.09
Gross value recovered	$690,932
Gross value per ton recovered	$15.44
Gross value in tailing	$73,922
Gross value per ton, tailing	$1.65
Recovery before shipment, per cent	90.3
Recovery after deducting transportation and treatment charges, per cent	88

The total recovery of metals was:

Gold, ounces	8,817
Silver, ounces	830,893
Gold extraction, per cent	94.24
Silver extraction, per cent	89.02
Total metallic extraction, per cent	89.91

There was obtained from the metallic contents of the ore:

	Gold, %.	Silver, %.
By cyanidation	87.38	73.49
By concentration	6.85	15.53
Lost in tailing	5.77	10.98

Each stamp worked 94.7% of the total time, with a stamp-duty of 6.12 tons per day. One per cent of the lost time was due to the lack of power, leaving 4.3% for repairs and holidays. This is a satisfactory record, considering that all of the batteries are driven from a single motor drive, and the tube-mills likewise. During the last quarter of the year, each stamp worked 97.4% of the time, with a stamp-duty of 7.26 tons per day; the total extraction for the same period being 91.16%. The direct milling costs for the year were $3.288 and the indirect 34.1c. per dry ton. During the year the tonnage treated per day was increased from 102 tons for the first quarter to 145 tons for the last quarter. To accomplish this increase, and to treat the higher grade of ore, it was necessary to add a concentrating addition, and to make many minor changes and additions to the equipment of the mill.

*Abstract from annual report of the West End Consolidated Mining Company.

Plant Improvements

The largest item of this expense, $9965, was that of the concentrating addition, built in May and June, containing 12 No. 3 Deister slimers, with the necessary cones, sumps, pumps, tanks, and the floor space for handling the concentrate. The flow of pulp and concentrate from the tables to their respective sump tanks being in concrete launders below the floor-level, gives a neat and labor-saving installation.

The cost of handling supplies and products to and from the cars on the new spur line is, in cents per ton: Concentrate and zinc, 15; cyanide, 27; lead acetate, 30; pebbles, 50; shoes, 55; and lime, 60.

In April, by additional piping, at a cost of only $60, the Trent agitators were connected for continuous agitation, described in the *Mining and Scientific Press* of May 3, 1913, by a simple and reliable method, resulting in a longer agitation for the pulp, a saving of power and heat, and less labor for operation. Another beneficial result of this change was that it gave a small continuous feed to the Dorr thickener, and by giving the arms of the thickener a steeper slope, it is possible to decant half of the pregnant solution direct to the silver-tank, and deliver a thick pulp of 1.36 or greater specific gravity to the stock tank. From this thick pulp a 1½-in. cake can be made in less than 40 minutes that is uniform and porous to the washing solutions.

Slag Grinder

A slag grinder and jig were installed to remove the metallic shot from the slag preparatory to sacking it for shipment, and a Case low-pressure fan and oil burner was adapted to the Faber du Faur refining furnace, resulting in but one-third of the oil consumption, faster melting, and less heat and noise in the refinery. These changes aided materially in lowering refining costs. Larger clutches and a stronger chain drive were placed on the tube-mill drive, at a cost of $1150, allowing the pebble load in the mills to be increased to the maximum capacity of the 100-hp. motor, resulting in a greater output from the mills and an increase of ore treated from 102 tons to 120 tons per day. The speed of the stamps was increased from 98 to 104 drops per minute, with 7-in. drop, by a change of pulleys. This resulted in greater tonnage, with no greater repairs. In 18 months only one stem has been turned. Experiments on one tube-mill proved that if the pebble load could be increased still further beyond the point used in general practice, the resulting greater tonnage would many times repay the greater power and pebble consumption. Consequently, a 150-hp. motor was installed to replace the 100-hp. machine, resulting in a marked daily increase of tonnage, averaging 146 tons per day for the last quarter. Two Campbell & Kelly 4-in. centrifugal pumps were purchased and installed at a cost of $425 to replace Butters pumps, used on Trent agitators, since the two pumps of this make already in service have proved remarkably well suited to the work, with a minimum expenditure for repair work. The Butters pumps were transferred to solution work. With the greater tonnage treated, it became necessary to increase the zinc-box capacity to precipitate the metals in solution, and especially to lower the value of the wash solution used in the filtering operation, in order to reduce the loss in soluble metal from the mill. To carry this saving of solution content to a further point than usual in common practice, where an incomplete replacement of the wash solution in the filter-cake can be made by wash-water, due to dilution of mill solutions, two steel zinc-boxes were installed which were 3 ft. wide, 12 ft. long, and 140-cu. ft. zinc capacity each. One of these is used to give thorough precipitation of the silver solution for wash-solution purposes, and the other to precipitate the long water wash given in the cake, this wash-water staying in a circuit by itself. By this method the filter-cake is washed with wash-solution until the effluent solution from the vacuum-pump is as low in value as the wash solution, then the cake is again washed with wash-water until the effluent solution is nearly as low as the precipitated wash-water. The loss in soluble content before the change was 20c. or greater per ton of ore, which, while excessive, was no greater than the average in the district. This loss was not determined by the inaccurate method of washed and unwashed filter-cake, but by assaying the filtered solution from the sample taken from the discharge launder. This loss is 7c. per ton of ore, or a reduction of 13c., which amounts to a saving of over $500 per month.

Zinc Lathe

A No. 3 Hampton zinc lathe was installed in the concentrating department, at a total cost of $233, and the concentrator men cut the zinc necessary. This has resulted in a saving of 2.8c. per pound of zinc shavings, or $154 per month, besides giving a fresher product and of the right thickness for the boxes. Of this saving, $135 is credited to the concentrating department for cutting the zinc.

In concentrating pulp after tube-milling it is difficult to obtain a good percentage of the value in the ore in the concentrate, without also including the heaviest particles of quartz, thus making a low-grade concentrate. At the West End, with the heavier tonnage obtained by other changes enumerated above, the tables were overloaded, and to obtain 15% of the value of the ore by concentration the concentrate contained 60% silica. After careful tests, a Wilfley table of half the standard area was installed to reconcentrate the concentrate from the Deisters; also a 1-in. centrifugal pump to raise the concentrate from the 12 tables. These were flattened to deliver the maximum concentrate along with considerable silica. The Wilfley, handling but two tons per day, makes a much cleaner separation, resulting in making but one-half the previous amount of concentrate of but 35% silica, which contains 15% of the value of the ore in 0.5% concentrate. The cost of the installation was $227, and the saving made in freight and treatment charges amounts to $170 per month. After this change was made it was not necessary to enlarge the concentrating plant.

Change in Character of Ore

In November the ore in the mill began to change from one entirely quartz and sulphides to one containing a large percentage of oxidized material. The clay content of this ore made settling very difficult, filtering slower, the previous method of discharge impossible, and the method of pumping away the residue expensive for additional water. An installation of 6-in. pipe connecting the filter pump with separate valves to the filter-hoppers made it possible by the use of the pump to discharge from the hoppers the entire filter residue in a well mixed pulp. In February, at the slime pumping station, where the slime is lifted 30 ft. over a ridge to join the Montana-Tonopah tailing, a 6 by 20-ft. tank at the end of the flume and pipe was cut out. The pump was also lowered in the excavation to pump from the bottom of the tank instead of from the 24-in. pipe. By this method the slime pulp is not held back in the pipe, but with the swirl in the tank it is kept in constant motion until discharged, resulting in a marked decrease of water necessary for discharge, which is now about 0.9 ton of water per ton of ore.

With this content of oxidized ore, the settling capacity of four 16-in. Dorr thickeners was not great enough, resulting in a slimy overflow, the solids of which accumulated in solution-tanks and pipes. To correct this, a fifth Dorr thickener was installed in one of two tanks previously used for return battery-solution, and a fifth diaphragm pump was added to take the thickened pulp from the new thickener.

Detailed costs were as follows:

	March 1913.	Fiscal year.
Superintendent and foreman	$0.056	$0.062
Crushing and conveying	0.080	0.103
Stamp batteries	0.224	0.248
Tube-mills	0.398	0.492
Concentrating	0.113	0.130
Agitating	0.823	1.033
Filtering and discharging	0.171	0.218
Precipitating	0.155	0.260
Elevating and settling	0.057	0.103
Motors and lights	0.040	0.036
Refining	0.072	0.097
Drayage	0.008	0.010
Assaying	0.028	0.042
Steam heating	0.136	0.130
Water	0.164	0.214
General expense	0.065	0.110
Total direct	$2.590	$3.288
Salaries	$0.070	$0.074
Office expense	0.016	0.010
Legal and traveling	....	0.002
Insurance	0.030	0.036
Taxes	0.034	0.040
Liability insurance	0.018	0.021
Depreciation	0.265	0.158
Total indirect	$0.433	$0.341
Total operating	$3.023	$3.629

Danish Tube-Mill Pebbles and Substitutes

By JAY A. CARPENTER

Since the advent of the tube-mill for fine grinding, the Danish pebble has been the main grinding medium. The reason for this is that these pebbles are extremely hard, tough, nearly spherical, and the cost of production is limited to the cost of gathering them from the seashore, and then sizing and grading them for shipment. The cost of such flints, sacked ready for export from Denmark, is about $11 per long ton. The true Danish flint is remarkably homogeneous, and free from all cleavage planes or lines of weakness. On being fractured, the faces are sharp and irregular, and vary in color from a brownish black to a jet black. Occasionally the faces show irregular patches, a brown to gray color, which, while softer are apparently an integral part of the pebble.

The pebbles, from 1 to 7 in. diameter, are divided into about eight sizes, from size 0, 1 to 1½ in., to size 7, 4¾ to 7 in. diameter. The sizes most in demand are No. 3, 2⅝ to 3¼ in.; No. 4, 3⅛ to 3¾ in., and No. 5, 3½ to 4 in. These are sorted into different grades, varying from the best spherical or 'special hand picked' pebbles to the culls. This grade has not been given much attention by the millman, consequently most pebbles are purchased on the specification of size only, without reference to grade.

While the Danish pebble sells around $11 per ton ready for export, the price quoted by dealers *ex* steamer at New York is about $15, and the price delivered to the railroad station of the mines varies greatly with their situation. being in the case of Tonopah about $35 per ton. The same pebbles, upon reaching the mills at Manhattan and Round Mountain, 45 miles and 72 miles, respectively, from Tonopah by wagon road, cost $55 and $62.50 delivered. This nearly places the homely Danish flint in the class of its more fortunately colored brothers, the amethyst and opal.

Quantity of Pebbles

The quantity of pebbles used per ton of ore depends mainly on the hardness of the ore, the coarseness of the sand product fed to the tube-mills, and the fineness to which it is to be ground. At the West End mill, during the year 1913-14, the ore was crushed through battery screens with 0.19 to 0.27-in. openings, and the final discharge from the tube-mill circuit was 80% – 200 mesh. The ore is a hard quartz, not tough, but breaking into sharp fragments. The consumption of pebbles per ton of ore milled was 7.1 lb., varying from 4.82 lb. during one month to 8.73 lb. in another month. The higher limits were with the coarsest screens or with French pebbles. At $34 per short ton, this gives an average cost of 12.1c. per ton for pebbles. This is a high figure due to the coarse crushing in the batteries, and to heavy pebble loads in the tube-mills. The pebble consumption at three other mills in the district, all using finer screens, is 4.2 lb., 4.4 lb., and 4.7 lb., respecticely, giving a cost from 7.1 to 8c. per ton of ore. At the Goldfield Consolidated mill the consumption of pebbles is about 4 lb. per ton, and the amount used in 1913 was approximately 660 tons. These figures show that the Danish pebble is an important item of cost in milling, and is a large tonnage item in the freight of a milling district like Tonopah.

It is primarily this cost per ton of ore for Danish pebbles that makes the millman interested in substitutes. It is also the carload substitution of French pebbles at Danish prices, and the receipt of inferior grades of Danish pebbles that sharpens his interest. There is an open question if the Danish supply can meet the demand coming from the increased number of tube-mills installed, and the increasing coarseness of feed to them. At least an extra cost or a decrease in grade can be expected. At the present time it is questionable whether the quantity exported will be seriously affected by the war. Prices soared in the early fall, but are nearly normal again with, however, the supply rather limited.

French Pebbles

A substitute for the Danish pebble in common use is the French pebble. This pebble on the exterior has much the appearance of the Danish article, and is as free from fracture planes; but on fracture its color is light brown to gray, in contrast to the dark brown to jet black of the Danish pebble. While pebble dealers may argue that Nature would not be mean enough to toss up a softer article to the Frenchman, the millman who compares pebble consumption knows it for a fact, even if the pebbles come to him under a Danish bill of lading, or with numbers on the sacks instead of letters.

The cost of French pebbles ready for export is considerably under the Danish, but delivered at Tonopah with the same high transportation charges added to each, the French is but little cheaper, and wearing faster, due, I presume, to being somewhat softer, its cost per ton of ore ground is higher. Where transportation is not such an element in the cost, the French pebble might give an equal or lower cost per ton.

The Newfoundland pebble has a limited use. This pebble is hard enough, but in those received at Tonopah there were distinct parallel fractures or bedding lines across them, along which they would fracture when broken. The reject from the tube-mills showed that this action took place to some extent in the mill, making their use uneconomical in comparison with the Danish pebble.

The substitute most commonly in use is that of pieces of mine ore. Primarily the great advantage of this is the low cost of the material and the saving of the purchase price of the Danish pebbles. Even if the ore only carries metal equal to the cost of milling, it more than pays its own way in the subsequent steps of milling. On the Rand, according to 'Rand Metallurgical Practice,' the mine ore has entirely supplanted the imported pebble. This in spite of the fact that on a test run of one month, as recorded in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, with similar mills, it required 171.5 tons of banket against 10 tons of Danish pebbles. The quantity fed to a mill per day is from 5 to 10 tons, or an average of 8 tons. This is 60% of the normal load of the mill. This figure is quite impressive of tonnage to those American operators who must rely on a small variation of meter reading to check up a failure to add the daily pebbles. This tonnage added to the mills during the year makes an addition of approximately 2½ to 3%, at the same or lower grinding costs.

The adoption of the banket for this purpose by all the mills is conclusive proof of the saving made over Danish pebbles. Although, in the test referred to above, the mill filled with banket gave slightly better grinding than the one filled with Danish pebbles, others cannot have agreed with this, as Alfred James, in his annual review[1] of 1913, states that authentic tests are still to be made to prove that the loss in grinding efficiency does not offset the saving in pebble expenses. Since it is generally conceded that a large portion of the grinding in the tube-mill, especially to my mind, extreme fine grinding is done by the rolling and sliding action of the pebbles, the rectangular edges on the banket must be a detriment to the best work possible. These edges may soon become rounded off, but in so doing a great deal of oversize material is introduced into the mill, being below 1½ to 1 in., which size material is spoken of in 'Rand Metallurgical Practice' as "of no further use for crushing purposes, and during this period it occupies valuable space—and arrangements should be made to remove them." In the extreme case of a square being reduced to a sphere the loss of volume is nearly 50%, and the oversize added to the mill would equal the volume of the pebble formed.

Cost of Power

Of the total cost of tube-milling, the cost of pebbles, though quite an item, is not the main one. Power is the heaviest charge, probably in most cases exceeding that of pebbles two to four times, and the cost of pebbles is probably about one-fourth to one-seventh of the total cost of tube-milling. If the use of ore as pebbles should entirely eliminate the cost of pebbles, it is evident that before adopting its use a careful study should be made to see if it impairs the efficiency of the other three-fourths or six-sevenths of the cost of tube-milling, for it is evident that the impairment would have to be very small to overbalance the saving made.

A. G. Quartana[2] gives data on the use of ore in place of pebbles in Mexican practice. In one instance the daily feed is 1600 kg. of mine ore, as equivalent to 250 kg. of Danish pebbles. He advocates a mixture of the two to avoid handling such large quantities of mine rock, and to furnish Danish pebbles to fill El Oro linings, for which purpose he says the mine rock is not adapted. He also commented on the destructive effect of the mine rock on smooth iron liners. Undoubtedly the use of mine ore or rock as a substitute for Danish pebbles will increase in use as the cost of pebbles increases. In the Tonopah district at the present time, the Tonopah Mining Co.'s mill is using mine rock or low-grade ore exclusively in its two tube-mills with good results.

Another substitute for Danish pebbles are rounded pebbles picked up from stream beds in the vicinity of the mills. These offer a marked advantage of cheapness, and an excellent spherical or rounded form. They have the disadvantage of being softer than Danish pebbles, as they are usually silicified igneous rocks. In a cyanide mill there would be a debit charge against them in the remainder of the plant, but on the contrary in a cement plant, I would presume they would add their quota to the cement. About four years ago flint pebbles from the Colorado river were used at the Tom Reed and Gold Road mills. Several of the cement mills in southern California use the pebbles of silicified igneous rocks found in their vicinity.

Steel Balls

Another substitute for the Danish pebble is a steel ball. Although the steel ball is in use in ball-mills for coarser crushing, it has found no application for fine crushing in tube-milling. The reason for this is probably that where the added weight in striking a blow or in crushing, weight is needed with coarser material, this very fact is against it in tube-milling, where grinding surface is vastly more important than weight. A steel ball would have nearly three times the weight, but no greater surface than a spherical pebble of equal diameter, and all added weight below the centre line of the tube-mill means greater power consumption. This statement is quite at variance with a circular letter recently issued by a reliable conveying machinery firm which has taken up the manufacture of steel balls for tube-milling. A test is quoted in which the capacity of a 5 by 22-ft. mill was increased from 14 bbl. of cement per hour to 23 bbl. by the use of 'manganoid' steel pebbles, and "power requirements not increased." If this be so, their use will become common. Undoubtedly these balls can be made harder than the Danish pebble, closer grained, tougher, and more spherical, which advantages should more than compensate for their greater first cost, especially in distant fields where transportation is the greatest item of final cost.

There might be raised an objection to the steel pebble on account of the metallic iron introduced into the pulp to be cyanided. Several years ago there was a strong prejudice against the cast iron liner as a substitute for silex on this same ground. Not so long ago, with a

[1]*Mining and Scientific Press*, January 3, 1914.
[2]*The Eng. & Min. Jour.*, May 20, 1911.

change of metallurgical engineers at a mill being constructed at a considerable distance from railroad, the new El Oro lining available was discarded on this assumption, and a hurry-up order for silex telegraphed out. Mills in the Tonopah district using smooth cast iron liners over long periods have never noted any harmful effect from this source. In one mill where the pulp was concentrated after tube-milling, the metallic streak of iron was quite noticeable on the final reconcentrating table, giving some idea of the quantity in the pulp. In this case it was shipped with the concentrate, but later, when concentration was discontinued, it went through to the filters with the pulp, as in the other mills. If this iron was acted on by the mill cyanide solutions, it should introduce appreciable quantities of ferrocyanide in the mill solutions. The regular and frequent tests of these solutions at the West End mill for impurities have never disclosed any ferrocyanide, in spite of the large amount of iron in the pulp, the high cyanide and alkali solutions, and the high heat and long agitation.

There is one drawback to experimenting with steel balls in tube-mills designed to carry pebble loads, that of placing too heavy a strain on the tube-mill heads and trunnions. W. A. Caldecott[3] gives the pebble load of a 5 by 22-ft. mill, 56 in. inside of liners, and filled to the centre line, as 9.88 tons. The weight of the pulp within the mill is about 4 tons, giving a total load of 13.88 tons. If steel balls of about three times the density of quartz were substituted, the pebble load would then be 29.64 tons, and the total load 33.64 tons, or 2.42 times as heavy. This load would subject the tube-mill to strains beyond which it was designed, and might exceed its factor of safety.

Artificially Rounded Flints

If, as the circular mentioned stated, the power requirement (interpreting it literally to mean the power requirement per ton of cement) was not increased, then the added weight of the steel pebbles must add wonderfully to the grinding efficiency of the mill. At least, the steel ball is worthy of more experimental work. It must be that large cement plants, having tons of reject steel balls from Lehigh-Fuller mills or ball-mills, have tried them out in tube-mills, but I know of no published data on the subject in the technical press.

Another possible substitute for the Danish pebble is an artificially rounded flint. This substitute is now being tried in the Tonopah district. Such a pebble will have limited application in tube-milling, due to the fact that there must be available a deposit of homogeneous flint, chalcedony, or onyx, and a consumption in the district large enough to justify a 'tumbling' plant.

Omer Maris, of Manhattan, found such a deposit near that place. The rock can be described as a banded and patchwork onyx, yet it has no tendency to break along the banded or patchwork lines, and all the various colored components seem to be of nearly equal hardness. Mr. Maris believed this rock could be used as a substitute for Danish pebbles, and has devoted all his time and energy to bringing this about. As the Danish pebble is worth $50 per short ton in Manhattan, the mill operators gave him hearty co-operation. The first experiments consisted of charging several tons of rough surface rock into the mills. The edges soon rounded on these rocks, and then they gave equal wear with the Danish pebble. After the Manhattan mills adopted the rock for pebble use, Mr. Maris began to mine the rock in order to get fresher material. Soon after this, enough rock was sent to the Belmont mill at Tonopah to charge a tube-mill and supply it with feed during a test. The net result was that the consumption of Manhattan rock was 6.44 lb. per ton of ore against 4.14 lb. of Danish, and its efficiency as a grinder, judged by the amount of minus 200 mesh in the discharge, was only 80% of the Danish. A small lot sent to the West End mill was charged into a mill, and the large amount of broken fragments in the discharge of the mill during the first day did not augur well for the rock, but several weeks later, on opening the mill, the size of the pebbles remaining bore testimony to their hardness.

Results at the West End

It was apparent that the rock must be made into a pebble before shipment, to save freight and to give it a better record for wear and efficiency. A tumbling barrel or mill was built for this purpose, and from the rectangular rock, after an hour or two of dry rolling, a fair pebble is produced. It is far from spherical, but all corners are rounded, and waste in the tube-mills is thus eliminated. This rounded pebble is now being tested in Tonopah mills and the Goldfield Consolidated at Goldfield.

The results at the West End mill have been quite favorable to the pebble. There are two 5 by 18-ft. tube-mills in the plant driven from a common pinion shaft at the same speed, and each taking the classified sands from 10 stamps with screens having 0.12-in. openings. No. 1 mill has smooth liners and No. 2 has Komata; the latter lining being credited with being more economical on pebbles. During August, 12,500 lb. of Danish pebbles was added to No. 1 mill. Beginning September 1, Manhattan pebbles only were added daily to the mill, totaling 11,640 lb. for the month. As the pebble load of the mill when carried at centre is about 8.1 tons, and the daily charge is about 400 lb., the average life of a pebble is 40 days. Calculating from this, the load should have been about one-half Manhattan on October 1. During October there was added 12,190 lb. of Manhattan pebbles, and when the mill was opened the load was at the centre line and all the pebbles except the smallest were Manhattan pebbles. During November there was added 13,490 lb., leaving the mill with a slightly higher load. Comparing the above months, it will be seen that the Manhattan pebble is practically equal, pound for pound, with the Danish pebble, as far as wear is concerned.

During November the load was entirely Manhattan

[3]*Mining and Scientific Press*, June 6, 1913.

pebbles, but different from charging the mill with all one sized pebbles, for by the method of adding them gradually there were all sizes of smooth worn pebbles in the mill down to those small enough to pass through the discharge screen. During November the fineness of the pulp at the discharge of the mill remained practically the same as during the preceding three months, being about 45% – 200 mesh.

In November No. 2 mill took 12,800 lb. of Danish pebbles. This gives a comparison of the Manhattan and Danish pebbles as follows: wear ratio, 1.05 : 1; cost per pound ratio, 0.95 : 1.73c.; cost per ton ground ratio, \$1 : \$1.75, or a saving of 40% on pebble cost by Manhattan pebbles. To offset this gain is an occasional delay or stop due to irregularly shaped pebbles sticking in the tube-mill scoop.

While this pebble appears to be as efficient a grinder as the Danish pebble in the smooth-liner mill where a large share of the grinding is done by slippage, and the pebbles tend to wear flat, it is a question whether they would be as efficient in the Komata lined mill, where the pebbles are cascaded; the more nearly spherical the pebble the more efficient the grinding. They are being tried with this type of liner at other mills in the district.

These pebbles are delivered at Tonopah with a wagon haul of 40 miles at \$19 per short ton. It is evident that at this price the Manhattan pebble has a marked advantage in price at Tonopah over the Danish, but with freight shipments from Tonopah toward coast points this advantage rapidly diminishes to a vanishing point, leaving this particular substitute for the Danish pebble to the limited field of local use.

The subject of the Danish pebble and its substitutes is an interesting one to the mill operator, and if many operators hold opinions different from those expressed above, they must remember that this article is based to a certain degree on personal opinion, since the literature and discussions on the subject are limited, which fault lies in their power to correct.

TRIBUTERS OR LESSEES at Kalgoorlie, Western Australia, mine a considerable tonnage of oxide and sulphide ores. The former is treated in a number of custom mills using stamps or Huntington mills; but the latter, since the Associated Northern dry-crushing, roasting, and cyaniding plant was dismantled after 10 years' run, has no suitable plant. The government was approached and asked to provide a mill, but in the meantime the metallurgical firm of Allsop & Don leased the Hainault mill of 40 stamps, Cobbe grinding pans, Wilfley concentrators, and sand, slime, and concentrate treatment processes, for the treatment of custom ore. The following charges are to be made for complete treatment of the sulphide ores supplied, 90% recovery being paid for: ore worth \$15 per ton and under, \$4.80 per ton; for each additional dollar up to \$20, 25c. per ton extra; and for each additional dollar over \$20 and up to \$60, 12c. per ton. This gives a minimum of \$4.80 and a maximum of \$10.80 per ton. The government is granting the mill lessees a subsidy of \$4800 to make necessary alterations to the plant. This should be a satisfactory arrangement, as the Hainault mill is complete. Thus the government at small expense aids a large number of miners, who otherwise would have difficulty in getting their ore properly treated.

Mining in the Belgian Congo

The following circular has been issued by the Union Minière, a subsidiary of Tanganyika Concessions, Ltd.:

"The estimated production of copper by the Union Minière du Haut Katanga for 1914 was 12,000 tons; for 1915, 25,000 tons; and for 1916, 40,000 tons. After the occupation of Brussels by the Germans, part of the Union Minière executive staff, including Mr. Velge, the secretary of that Company, was transferred from Brussels to the offices of the Tanganyika Concessions, Ltd., London, where legally constituted directors' meetings are now regularly held in conformity with article 16 of the articles of association of the Union Minière du Haut Katanga. The Company's smelting operations at Katanga have been kept running without interruption, although copper fell rapidly from £60 to £48 per ton, but has risen since, as will have been seen, to over £56 per ton. During the period that has elapsed since war started, and although considerable difficulties have been caused directly by the war, two furnaces have been kept in blast, producing 3730 tons of copper up to the end of November. The third furnace was blown in at the beginning of December in order to increase the output of copper, and, in view of the rising price of the metal, it is estimated that the total production for this year will therefore not be far short of the 12,000 tons anticipated. The copper produced from the beginning of the year up to the commencement of the war was realized at a price which shows a profit of approximately £20 per ton over the cost delivered in England. In spite of the adverse conditions affecting the metal market since the beginning of the war, the copper produced and landed in England since then has been sold at a price which shows a profit of over £10 per ton over the cost delivered in London. At the present price of copper a considerable increase in this profit will be secured, and as the price of copper is improving, the Company is now considering the immediate erection of two further blast-furnaces which have already arrived at Beira, on the east coast, and it is hoped that the Company will be able to erect these within a short period."

COKE-OVEN ACCIDENTS in 1913 were as follows, according to Albert H. Fay, of the U. S. Bureau of Mines: plants operating, 227; men employed, 24,345; killed, 46; seriously injured, 342; and slightly injured, 2172. The rate of fatalities was 1.89 per 1000 employed, against 1.72 in quarries and 2.06 at metal mines, surface only. There are over 100,000 ovens in 575 plants in the country. About 50% of the deaths were due to transportation systems.

What is probably one of the largest deposits of this earth in California is found along the banks of the Pitt river, in Shasta county, California. It is reported to be 16 miles long and several hundred feet thick. Unfortunately, it is too inaccessible and too far from rail to be commercially important. In the Owens River valley, Inyo county, California, a little has been mined by John Black in the vicinity of Aberdeen, six miles from the Owenyo narrow-gage railroad. A shaft sunk on this claim shows the earth for a depth of 40 ft. It can be followed for many miles along the foothills of the eastern slope of the Sierra Nevada in Inyo county. In Los Angeles county there are a number of beds of diatomaceous earth, and attempts are now being made to explore these. In Sonoma county, two carloads were mined and shipped in 1913 from a point about five miles from Mark West station. The deposits are small and the earth is somewhat discolored. In Nevada, a few carloads are occasionally shipped from points along the Tonopah-Goldfield railroad, and some has been produced from deposits 12 or 15 miles from Miller. The wide distribution of infusorial earth is fairly indicated by the fact that deposits have been noted in 16 counties of California.

MINERALS EXPORTED from Broken Hill in January were only worth $720,000, against about three times that amount in normal times.

Cost of Mining and Milling at Tonopah

By JAY A. CARPENTER

*At some future time when the United States Bureau of Corporations requires a fixed form of detailed report from all metal-mining companies, and such reports are given by the companies to their stockholders, such a compact statement as given in this article will be accurate in detail and easily secured. At present, to illustrate the contrast, the Tonopah Extension divides all operating expenses under the four headings, 'mining,' 'milling,' 'general expenses,' and 'administration expenses,' while the Montana-Tonopah divides its disbursements into 48 items, and a table on the cost of operations has eight headings, as follows: 'mining,' 'development,' 'milling,' 'general expense,' 'maintenance,' 'marketing,' 'indirect costs,' and 'depreciation.' In most of the reports the total costs of operations per ton of ore include only the Tonopah office vouchers, and to get the total cost per ton the financial statements must be studied to find the head or Eastern office expenses; but this is not the case with the Montana or Nevada Hills companies, with head offices at the mines. This contrast exists through all the reports. In one report the total information given concerning mill costs is in the one item 'milling'; in a second, the annual cost of 61 items under the head of 'milling' is given; and in others, complete milling costs for every month of the year are to be found.

After a careful study of these many different reports, table No. 2 is a conscientious attempt to take all these varying methods of cost-accounting and group them under a few definite heads for a direct and fair comparison of cost data for the year 1913.

It will be noted that the Nevada Wonder and Nevada Hills companies are also included, since these companies treat ores similar to the Tonopah ores and the wagon haul per ton to these properties from Fallon is no greater than the Hazen-to-Tonopah railroad freight rates.

*From *The Tonopah Miner*, March 27, 1915.

In table No. 1 is given a description of the reduction plants of the mining companies at the time the cost figures in table No. 2 were obtained. The data for table No. 1 were taken from articles in the technical press and from my personal knowledge of these plants. The tables are given on the next page.

The following general observations on table No. 2 bring out points of special interest. The grade of ore milled varies from $8.41 to $21.09, the daily tonnage produced from 52 to 473 tons, yet the gross cost per ton of those mines having reduction-plants varies only from $7.42 to $8.93.

The gross extraction varies within the limits of 87.2% and 94.45%, while the net extraction, which is the important extraction figure since marketing costs are not included in the total working costs and must be deducted from the gross extraction, varies from 84 to 91.95%. The variation in the cost of marketing depends upon whether bullion is the only product made, in which case about 1.6% is the cost of marketing, while if any concentrate is shipped out, with its heavy freight and smelter charges, the cost of marketing goes up in proportion to the tonnage shipped.

The book charge of depreciation, varying from nothing to $4.65 per ton, illustrates the varying importance attached to this charge by the different companies. The tons of ore for one foot of development work is an indication of the progress made in blocking out ore reserves, or the search for some to block out, and has a direct effect upon the total cost of mining.

If the "total of all costs" be divided into the two heads 'mining' and 'milling', and the important factor of net extraction be considered, a summary of table No. 2 would be as follows:

No.:	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.
Mining	4.68	3.58	5.87	4.19	4.46	0.00	4.20	4.62	4.35	5.57
Milling	2.92	3.84	3.06	3.26	3.59	3.94	3.24	3.53	3.32	7.27
Net ext., %	92.0	85.8	86.8	88.6	87.5	84.4	84.0	91.9	91.1	90.0

TABLE NO. 1. REDUCTION PLANTS OF THE MINING COMPANIES

Company.	Year Built	Crushers.	Grinders.	Regrinders.	Type.	Concentration	Classification.	Agitators.	Filters.	Precipitation.	Daily Tonnage.
Desert Power & Mill Co. Tonopah Mining Co.	1906	Gyratories 1 No. 7½ 2 No. 4's	100 Stamps 1050-lb.	2 Huntingtons 2 Chileans 2 Tube Mills 1 5-ft.x18-ft. 1 5-ft.x20-ft.	Changing to All-Sliming	Wilfleys	Dorr Classifier and Thickener	Arm Agitators and Dorr Agitators	190-leaf Butters	Zinc Thread	400
Tonopah Belmont D. Co. 1. Millers Plant ...	1906	Gyratories 1 No. 6 2 No. 4's	60 Stamps 1050-lb.	2 Tube Mills 1 4-ft.x16-ft. 1 5-ft.x18-ft	Mostly Sliming Some Leaching	Wilfleys	Dorr Classifiers and Thickeners	Arm Agitators	120-leaf Butters	Zinc Thread	Custom Mill about 150
2. Tonopah Plant	1912	1 No. 7½ 2 No. 4's	60 Stamps 1250-lb.	8 Tube Mills 5-ft.x18ft.	All-Sliming	Wilfleys	Dorr Classifiers and Thickeners	Pachucas	200-leaf Butters	Zinc Dust	500
Montana-Tonopah M. Co.	1907	1 No. 5 2 No. 3	40 Stamps 1050-lb.	2 Tube Mills 5-ft.x22-ft	All-Sliming	Wilfleys	Dorr Classifier Cone Settlers	Hendryx & Trent	144-leaf Butters	Zinc Dust	150
West End Con. M. Co..	1911	10x20 Blake 1 No. 3 Gyratory	20 Stamps 1250-lb	2 Tube Mills 5-ft.x18-ft	All-Sliming	Deister	Dorr Clasf'r, Dorr & Trent Thick'nrs	Trent	100-leaf Butters	Zinc Thread	150
Tonopah Ex. M. Co..	1909	Gyratories 1 No. 4	30 Stamps 1050-lb	2 Tube Mills 5-ft.x18-ft	All-Sliming	No Concentration	Dorr Classifier Cone Thickeners	Trent	100-leaf Butters	Zinc Thread	160
MacNamara M Co.	1912	Gyratories 1 No. 3	10 Stamps 1350-lb.	1 Tube Mill 5-ft.x16-ft	All-Sliming	No Concentration	Dorr Classifier and Thickener	Trent	50-leaf Butters	Zinc Thread	70
Nevada Wonder M. Co.	1911	Blake 10-in.x16-in	10 Stamps 1400-lb.	1 Chilean 1 Tube Mill 5-ft.x22-ft	All-Sliming	No Concentration	Dorr Classifier and Thickener	Pachuca	2 12-ft.x8-ft. Olivers	Zinc Thread	110
Nevada Hills M. Co.	1911	Blake 12-in.x20-in.	20 Stamps 1250-lb.	2 Tube Mills 5-ft.x18-ft.	All-Sliming	Deisters	Dorr Classifier and Thickener	Pachuca	2 12-ft.x12-ft. Olivers	Zinc Dust	120

TABLE NO. 2. MINING AND MILLING COSTS IN 1913

Company.	TONNAGE. Year		Month	Day	MINING Without Depreciation Charges					MILLING Without Depreciation Charges.			Depreciation.			Total Mining and Milling Costs at Local Office Without Depreciation.	Eastern or Head Office.	Total of All Costs Without Marketing Depreciation or Exploration.	Extraction of Values.			Tons of Ore for 1 ft. Development.
	Dry Tons Ore......	Value Per Ton			Without Develop'nt	Develop'nt	Including Develop'nt	Indirect Charges	Total Mining	Direct	Indirect	Total Milling	Mining	Milling	Total				Gross	Marketing Cost	Net ..	
(1) Tonopah Belmont Tonopah Plant ...	172,398	$21.09	14,360	473	$3.01	$1.10	$4.11	$0.407	$4.517	$2.56	$0.20	$2.76	$.087	$0.29	$0.377	$7.367†	$0.318‡	$7.685	% 94.45	% 2.5§	% 91.95	6.7
(2) Tonopah Mining ..	163,387	$17.79	13,615	448			2.96	312	3.28	2.59	(.22 *(.74	3.55	.00	.00	.00	6.83	.59†	7.42†	89.00	3.26	85.75	9.1
(3) Tonopah Extension	58,022	$12.00	4,835	159			5.40	.38	5.78	2.58	40	2.98			.30	8.76	.17	8.93	88.4	1.6	86.8	3.6
(4) West End	52,838	$18.70	4,403	145			3.47	.64	4.11	2.97	21	3.18	.138	.35	.488	7.29	.15	7.44	91.3	2.6	88.6	7.7
(5) Montana-Tonopah .	52,362	$12.70	4,363	143	2.62	1.36	3.98	.47	4.46	3.12	47	3.59			.757	8.056	.00	8.05	91.15	3.62	87.52	5.1
(6) Tonopah Belmont Millers Plant	48,088	$18.90	4,007	132						3.67	27	3.94			.00		.00		87.2	2.85	84.35	00
(7) Nevada Hills	41,919	$13.78	3,493	115	2.45	1.35	3.80	40	4.20	2.84	40	3.24			4.65	7.44	.00	7.44	88.4	4.46	84.00	?
(8) Nevada Wonder .	39,118	$14.63	3,260	107			4.10†	22	4.32	3.09	16	3.25‡			1.06	7.57	.58§	8.15§	93.6‡	1.67	91.93	5.2
(9) MacNamara	28,098	$ 8.41	2,341	77			3.93	42	4.35	2.96	36	3.32			00	7.67‡	00†	7.67	92.7	1.60	91.1	8.5
(10) Jim Butler	18,900	$19.80	1,575	52			5.20	37	5.57	6.23	*1.04	7.27			.00	12.84	.08	12.92*			90.00	?

(1†) Checks total on page 7 of the annual report, minus depreciation and marketing costs, and "total direct mining" of page 26
(1‡) Checks from page 20 includes U. S. corporation tax.
(1§) Includes from page 7 "marketing bullion," "marketing concentrates and slag," and "smelting losses in concentrates."
(2) All figures except those marked (†) taken from "costs per ton" table page 12
(2†) The figure $7.44 is derived from page 19 by adding all disbursements for the year except that of $12,549 for patent litigation, $75,007 for exploration, and those for capital investments. The difference between this $7.44 and the $6.33 shown on page 19, is here shown as eastern office expense. However in the "income and surplus account," page 20, the total expense per ton of ore figures $7.50, thus including the patent litigation but not exploration, and is divided into mining, shaft sinking and general expenses, $3.68, milling costs and general expenses $3.08, and freight $0.74. This throws all eastern office expense directly into mining and milling.
(2*) Freight.
(3) All figures taken from cost per ton and "profit and loss account" statements. The figure $8.93 does not include construction account which totaled 29.2 ct. a ton.
(4) All figures taken from mining and milling operation tables. The item "marketing costs" shown in the table as a percentage figure is in most all of the reports shown as a money item in the cost per ton statement but does not show as an expense in cash receipts and disbursements because the freight, express and treatment is all deducted by the smelting and refining companies before remitting to the companies.
(5) Figures taken from cost of operation table in which the items "general expenses," "maintenance," and "indirect costs" are divided equally between mining and milling as the total indirect charges. This leaves nothing for eastern or main office as this is at the mine in Tonopah.
(6) Operated as a custom plant on Jim Butler, Tonopah Merger and other ores and on sweepings and tailings.
(7) As in the case of the Montana-Tonopah all expenditures outside those given for direct mining and milling are divided equally between mining and milling as indirect charges. Current construction of 2.1 ct. is not included.
(8†) Includes item "Ore on Dump at Cost of Mining" from "assets" table as 3,517 tons dump ore was milled.
(8‡) From mill supt. report.
(8§) The figure $8.15 is from cash receipts and disbursements table, page 25. The difference of 58 ct. thus falls to main office expense. It probably contains considerable mine construction expense, but this item does not show anywhere in the report. It is not in plant and betterments for that asset shows a decrease for the year greater than the depreciation charged off.
(9†) All indirect expenses, including San Francisco office, are given as one item which is divided into mining and milling.
(9‡) This total does not include interest payments on loans.
(10) This company has no railroad spur or reduction plant of its own, but sends it ore to the Belmont's Millers plant.
(10*) Transportation.

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President of the Pennsylvania Salt Manufacturing Company. He told me that he was in London endeavoring to secure a contract for 250,000 tons of Rio Tinto 'pyrites' partly for his own works and partly for the Standard Oil subsidiaries, which he represented. He said that he feared his contract would fall through because the Rio Tinto company insisted that he take the 'pyrites' free on board at Huelva; that the Spanish government had permitted the entrance to the port to become filled up to such an extent that only boats of comparatively small draft could cross the bar; that he could pay the price asked for the pyrite if he could get large ships into the harbor, upon which the freight-rate was somewhat less than upon smaller vessels. I took Mr. Armstrong to New Court and introduced him to Mr. Alfred de Rothschild. I knew that he was interested in the Rio Tinto and thought possibly the introduction might lead to a somewhat lower price being quoted, under the circumstances. Mr. de Rothschild, upon learning that the contract was not to begin for a few months, said that Mr. Armstrong could sign it with confidence and that the difficulty would be removed. I never heard what steps were taken, but a few weeks afterward I saw in a Paris newspaper the statement that the government of Spain had ordered immediate steps to be taken to dredge the bar at Huelva to permit the free entry of large vessels. Evidently Mr. de Rothschild had spoken the timely word.

Apex Litigation. Jim Butler v. West End

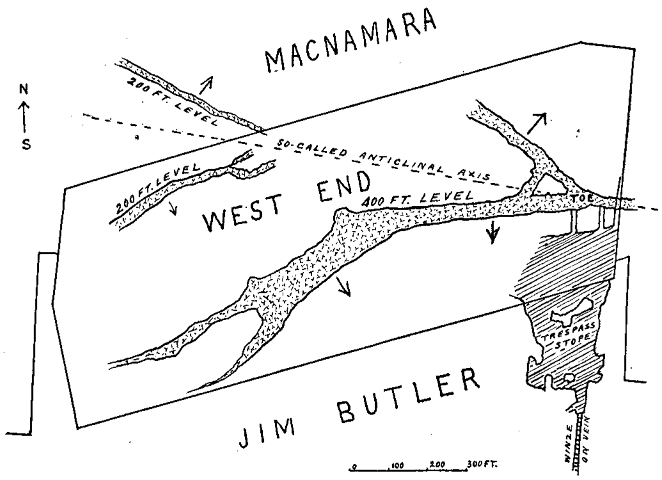
By H. V. WINCHELL

The elaborate opinion accompanying the decision handed down on April 30 by Judge Mark Averill of Tonopah in the above entitled cause is in many respects of unusual interest. Noteworthy because of the array of legal and technical talent engaged, it is also important in establishing a precedent as to the law in exceptional geological conditions and in deciding questions that have not heretofore arisen in mining litigation.

Although apparently simple in its intent and wording, the apex law of the United States, designed by its framers to apply to an ideal condition, has been so stretched in the endeavor to cover all classes of metallic mineral deposits that it is now in a state of great complexity and confusion; and not even the most learned and experienced legal practitioner can say in advance what will be the ruling of the courts under conditions slightly different from those which have been already adjudicated. The present law was enacted in 1872, and it is not surprising that the statement is often made by those who are poorly informed that the interpretation of the law is now complete, and that no more new questions remain to be fought over and settled. That this statement is not true, and that there is in sight no limit to the number of new questions which may arise for interpretation is known to many, and is illustrated in the case now to be discussed.

Generally speaking, the owner of a 'lode claim' under the Federal mineral-land law is entitled to follow and mine his veins downward upon their dip into the earth from their tops or apices within the said claim even though he passes beneath the surface of land owned by others. This right is, however, subject to certain restrictions and limitations:

1. Its direction depends upon the relation of the apex of the discovery-vein to the boundaries of the claim.
2. The 'extra-lateral right' can only be enjoyed between the planes of the end-lines of the claim extended in their own direction.
3. If the apex of the discovery-vein crosses both side-lines of the claim as located, these side-lines become end-lines for the purpose of determining the direction and extent of extra-lateral rights, but
4. In either case, in order that there may be any extra-lateral rights, the end-lines (*de facto* or constructionary) must be parallel or at least not diverge in the direction of the dip.
5. Where the apex of the discovery-vein crosses one located end-line and one located side-line of the claim, extra-lateral rights are allowed, as the law has been judicially interpreted, a new theoretical vertical boundary being erected for such rights, parallel with the located end-lines and passed through the vein at the point where it departs from the claim through the side-line.



PLAN OF CLAIM AND OREBODIES.

6. Because they are thus important in fixing the direction of extra-lateral rights the end-lines must be continuous and unbroken.

There are many other principles which may be regarded as now definitely established by court decisions, but there are still many points of uncertainty, and among them are at least two which have been for the first time passed upon in the Jim Butler-West End case. In the first place, Judge Averill has decided that although a claim was located on a discovery-vein which dips north it may also have extra-lateral rights upon other veins apexing within its boundaries which dip south. In the second place, he has held that, although a claim has a broken end-line, yet, where its discovery-vein crosses a located end-line, and then passes out through one of its side-lines before it reaches the other end-line, the defect in the location is not material, and the claim has extra-lateral rights for the extent which it has apex in the same manner as a claim with parallel end-lines. There is also in the present case a decision that a top or apex exists in the West End claim under the rather unusual conditions hereafter described.

The plaintiff, Jim Butler Tonopah Mining Co., was represented by its chief counsel, Judge Curtis H. Lindley, the author of the standard treatise 'Lindley on Mines,' and his associate, Mr. William E. Colby, and by Mr. Hugh Brown, of Tonopah. As expert witnesses they presented Prof. Andrew C. Lawson, of Berkeley, Prof. John W. Finch, of Denver, Mr. Fred J. Siebert and Mr. Fred Searls, Jr., of Nevada.

Chief counsel for the West End Consolidated Mining Co. was Hon. W. H. Dickson, of Salt Lake City, who was assisted by his partner, Mr. Ellis, and by Messrs. Peck, Bunker & Cole, and Horatio Alling, Jr., of Oakland, Judge S. S. Downer, of Reno, and Mr. Harry Atkinson, of Tonopah. As expert witnesses appeared Messrs. W. H. Wiley, of Los Angeles, E. C. Juessen, of San Francisco, John W. Chandler, of Tonopah, and H. V. Winchell, of Minneapolis.

In the course of its mining operations, the West End company had followed downward upon a vein from beneath its own surface under ground owned by the Jim Butler company. Ore of considerable tonnage and large value had been extracted, and although the West End company, in the desire to avoid expensive litigation, offered to divide the ore and the proceeds already accrued, the offer was rejected and suit was instituted to recover triple damages for wilful trespass. The amount thus sued for was in excess of $3,000,000, and the suit took on serious aspects.

The geology of Tonopah is not simple. There is a series of volcanic rocks which intrude and overlap each other like a pile of batter-cakes unevenly arranged, and of varying thickness. The productive veins are found to a large extent within what is called the Mizpah trachyte formation. This trachyte is believed to be older than the other volcanics, some of which have been intruded into it, and others of which have flowed out upon it, and still others of which are in the nature of overlying tuffs or volcanic breccias. One important rock is the West End rhyolite. It has been intruded into the trachyte in the form of an irregular sheet. Another widely spread eruptive is the Midway andesite. The latter rests upon the trachyte and conceals it from view throughout the area in controversy in this case. The veins do not enter the Midway andesite. They are supposed to have been formed prior to its extrusion; and it is even suggested that there was a time when the veins outcropped on the surface of the ground, and that the andesite flow covered these outcrops. There was a still later volcanic breccia, called the Brougher Mountain or Fraction dacite-breccia, whose relation to the veins and to the other rocks was a matter of some discussion in the case.

The veins of Tonopah do not commonly outcrop on the surface. They generally terminate upward against the Midway andesite, and are thus buried to varying depths, from nothing to 1200 ft., depending on the thickness of this 'cap-rock.' Nor do they all lie in one and the same country-rock. They are found cutting the trachyte, extending downward through it to the West End rhyolite, following along the contact for a distance, then cutting through the rhyolite to another intrusive rock which underlies the rhyolite, and again taking up a position on this 'lower contact.' They thus have the characteristics of fissure-veins which for a considerable distance lie upon or in the planes of contact between eruptive rocks.

As shown by the plan, there appear to be two veins on each level, except in the vicinity of the 'toe.' One of these veins strikes northwesterly and dips northerly. This is the vein which was encountered in the discovery-shaft of the West End claim. The other vein strikes somewhat north of east, and dips southerly, and it was on this south-dipping vein that the alleged "trespass stopes" were made. It will appear from the plan that, on each level, these veins approach each other on strike, coming together along an axis or line of junction which inclines or pitches to the east.

It was shown by the testimony that these two veins had been usually considered as one vein, called the West End-MacNamara vein, and that there were stopes in the mine upon the north-dipping vein which had been carried up to a point where the vein became first flat and then assumed a dip to the south. The vein was therefore termed an 'anticlinal' vein, and the question was raised whether it was all one vein, and whether as a matter of fact being completely buried by andesite, and having no outcrop, it had any "top or apex" within the meaning of the statute.

No plane of cross-section will show precisely the same conditions as to overlying and underlying rocks for any considerable vertical distance. The section shown herewith is, however, substantially one which represents the structure near the eastern end of the West End claim. In this part of the ground, although there is a great amount of quartz beneath the crest of the so-called anticline, the arch is either broken, and one side thrust over the other, or else there is evidence of two different veins, one slightly subsequent to the other, and overlying its present top or terminal edge. That portion of the vein shown on the section between the point marked *A* and the Midway andesite cap-rock was the chief battle-ground of the contest. The Jim Butler witnesses claimed that the quartz in certain rises upon that part of the vein was not vein-matter in place, and at most nothing more than a part of the "halo" or "stringer zone" of the vein. The West End witnesses maintained that these rises show the full thickness of the north-dipping vein, containing ore with characteristic structure, and that this vein not only overlay the south-dipping vein, but had moved on it so as to be cut by it, as shown in the section.

The West End company claimed to be the legal owner of the stopes in controversy by virtue of its patent from the Government, alleging (1) that it had an apex more than sufficient to cover the area from which ore had been extracted as alleged in the complaint; (2) that although its discovery-vein dipped to the north, while the alleged trespass was on a south-dipping vein, yet that its rights extended also in that direction; (3) that its lode-claim was correctly laid out and patented and that there was no impediment to extra-lateral rights in the shape of its west end-line.

To these claims the Jim Butler company objected on the following grounds:

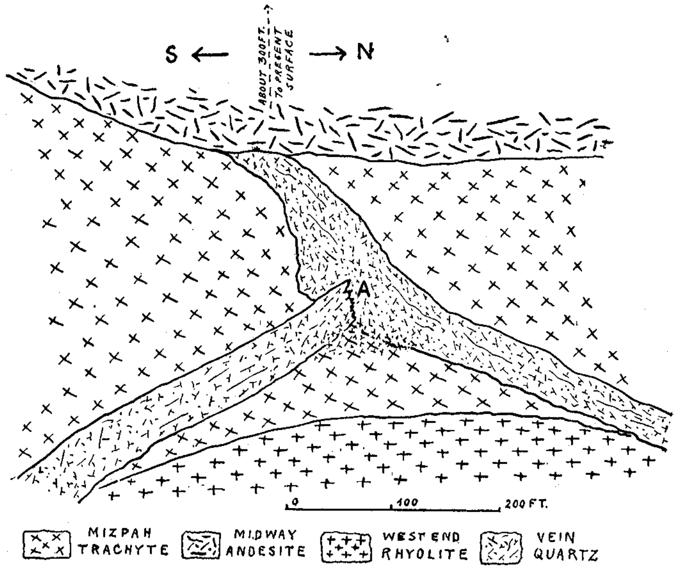
(1) The vein has no "top or apex" such as is contemplated by the statute. That it is a 'blanket' or contact vein, which not only does not reach the surface but which has no terminal edge unless it be one which extends in a general northerly and southerly direction near the west end of the claim, and which crosses both side-lines and thereby constitutes them really end-lines. On this point Judge Averill finds that "this contention is without merit. * * * Its termination, as that now exists in the westerly portion of the West End claim, is the result of the eruption of the volcanic neck known as Mt. Brougher." He finds abundant evidence of apex, showing a diagram of his own making on which there is 600 ft. of apex in one place, 360 ft. in another, and remarks further that "if the halo is to be included in the vein, and geologically it should be, this vein then reaches now and did reach all the surfaces to which apical facts are referable. It has, therefore, along its crest and extending above the anticlinal axis of the united main quartz bodies, a condition of things that may well serve as a terminal edge under the many peculiar circumstances and facts of the case."

(2) That the vein is one of rolling type or habit, and that other similar 'anticlines' are found in it which might likewise be looked upon as having apices. As to this theory, the Judge finds that "the vein is very irregular in size and very much distorted and broken, but it is almost free from rolls, having only two worthy of notice, one easily accounted for as a phenomenon of junction with the Fraction vein, and the other as a result of a number of small faults lying close together."

(3) That the strike of the vein or veins (which the plaintiffs claimed to be but one vein, and in which view they were supported by the findings of the Court) is not easterly and westerly, but northerly and southerly, and hence the true direction of its dip is neither north nor south, but to the east. On this point the Court finds that: "Much time and attention was given to the question of strike. As a matter of common sense rather than technical mathematics the strike of the vein as a whole is easterly and westerly, and the strike of its two slopes, assumed for the purpose to be separate veins, is as claimed by the West End." It is almost incredible that two opposing sets of geologists and engineers could differ so radically as to the direction of strike of such a large and well developed structural feature.

The questions of law which were decided in favor of the West End have already been referred to, and the learned Judge concludes as follows:

"If the conclusion reached herein seems revolutionary, that from the apex or apices of one vein extra-lateral rights flow downward across both of its side-lines, attention should be given to words of former Chief Justice Beatty of Nevada:



CROSS-SECTION OF SO-CALLED ANTICLINE.

" 'We are willing to admit that cases may arise to which it will be difficult to apply the law; but this only proves that such cases escaped the foresight of Congress, or that although they foresaw the possibility of such cases occurring, they considered the possibility so remote as not to afford a reason for departing from the simplicity of the plan they chose to adopt.'

"It is easy to realize that the condition described in this opinion is one that escaped the foresight of Congress and is also exceedingly remote from the simplicity of the plan they chose to adopt; yet the law must be applied to it.* * *

"This branch of the case is decided in favor of the defendant corporation. That is to say, the West End Consolidated Mining Company is the owner of the ore-bodies in dispute."

This case, like many others, may be appealed; and the present decision may be only the first of many yet to be rendered over the facts here so briefly sketched. Indeed, it is often many years before the final word is said and all doubts as to the final outcome dispelled. In view of such perplexities, is it any wonder that mining men are clamoring for a more rational and easily solved mineral-land law?

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Precipitation With Zinc-Thread

By JAY A. CARPENTER*

The purpose of this article is to give data on precipitation with zinc-thread at a Tonopah mill and to show that this method, although second in efficiency to the use of zinc-dust, gives excellent results when operated under favorable conditions. There seems to be a tendency in articles published in the technical press to extoll the virtues of any new method, to the disadvantage of the old one. At the time of the growing popularity of the Pachuca agitator it would have been heresy to publish the fact that they filled gradually with solidified pulp, and it was not until their popularity was on the wane that this was asserted in print. Likewise, zinc thread or shaving, once so popular because it won in competition with electric precipitation, is now only mentioned for its disadvantages. Even such an excellent paper as that of G. H. Clevenger on precipitation methods (presented at the International Engineering Congress) contains the inference that zinc-thread is greatly outclassed by zinc-dust. It is true that zinc-thread precipitation has its bugbear, namely, the white precipitate, but it is also true that with precipitating-presses, a bad cask of zinc-dust, or some other reason safely locked in the iron press, may cause the metallurgist just as much worry. Occasionally muddy solutions give trouble in zinc-boxes, but they play havoc in zinc-presses. Zinc-boxes may be subject to petty theft, but the bulk of the precipitate is not in a frame waiting the hold-up man. It has been said that, "comparative costs of zinc-dust and zinc-shaving show considerable advantage in favor of zinc-dust," but special mention should also be made of the fact that by the time the mill is provided with the heavy metal presses necessary to clarify and precipitate the solution with zinc-dust, the investment is so greatly in excess of the simple zinc-boxes that the saving made with the dust must extend over a long period to balance the greater outlay of capital. When the metallurgist considers the large number of mills that never pay back the capital invested, this point should cause him to consider the less efficient but cheaper methods, to see if their efficiency cannot be improved sufficiently to warrant installation.

In designing the West End mill in 1911, from data on silver cyanide-plants, I decided that five tons of solution should be precipitated for each ton of ore treated, or 600 tons for the 120-ton mill-capacity, and that this 600 tons required 0.75 cu. ft. of zinc-shaving for each ton, or 450 cu. ft. in all. Six redwood boxes were built, each with six compartments 30 by 30 by 24 in. deep. Below the screen of each compartment was a double hopper, which discharged through an 1½-in. plug-cock into a gravity system of launders leading to the precipitate vacuum-tanks. At the end of the first three months' operation the consumption was 1.7 oz. troy of zinc for each troy ounce of refined gold and silver recovered. About four tons of solution had been precipitated per ton of ore. This solution was made up as a 6 lb. KCN solution, with 1 lb. protective alkali, expressed as NaOH. The cyanide titration is made for total cyanide without any additional alkali, and we titrate for the protective alkali without the addition of ferro-cyanide. This high zinc consumption was probably due to the fresh strong solution, but it resulted in a local hunt for comparative data, which were recorded as shown herewith:

	Montana. 1908.	Extension. 1911.	Desert. 1910-'11.	Belmont. 1910-'11.	West End. 1911
Zinc used per oz. of bullion, lb.	0.108	0.065	0.084	0.09	0.11
Zinc per ton solution p'p'ted, cu. ft.	Dust	1.57	0.81	0.75	0.87
Solution p'p'ted per ton of ore, tons..	2.75	4.0	5.5	4.5	4.0
Bullion per ton of ore, oz.	8.5	17.0	21.0	20.0	15.5
Troy oz. zinc for troy oz. Au and Ag in bullion	1.67	0.945	1.22	1.31	1.7

Believing that the high consumption of zinc at the West End was partly due to buying zinc already cut (about a 1/600-in. shaving), a zinc-lathe was installed, and fresh zinc was cut as needed, about 1/300 in. thick. Results in 1912 were much better, there being a zinc consumption of 1.18 unit-weights of zinc for each unit-weight of refined bullion, but with only about 0.8 cu. ft. zinc per ton of pregnant solution, averaging 4 oz. in silver and 0.04 oz. gold; the precipitated solution still contained 0.10 oz. to 0.30 oz. silver. This precipitated solution, used as filter-cake barren-wash, resulted in high soluble losses.

In 1913, two 120-cu. ft. iron zinc-boxes, each having seven compartments 36 by 24, and 32 in. deep, were added; one to precipitate solution to a low value for filter-cake wash solution, the other to precipitate the wash-water circuit on the filter. The quantity of zinc-shaving in this second box, and the tonnage of wash-water precipitated by it, are not considered in the following table, but its zinc consumption and the bullion produced by it are included. This box precipitates about 2000 tons per month of wash-water, drawn through the filter-cake by a vacuum-pump and carrying about 0.4 oz. silver, giving a precipitated solution containing a trace to 0.02 oz. of silver. These boxes resulted in impoverishing the precipitated solutions used as filter-washes, and allowed the wooden boxes to be used for incomplete precipitation. An increase in ore tonnage nearly offset the increase in zinc-box capacity. However, for the year 1913 the consumption of zinc was 1.48 unit-weights of zinc for each unit-weight of refined bullion. This was due to an increase in alkalinity of the pregnant solution, and to a

*Mill superintendent, West End Con. Mining Co., Tonopah, Nevada.

Ore Treatment at the West End, Tonopah

By Jay A. Carpenter

A considerable amount of interesting work has been done at the West End mill, as the following notes from the annual report for the past year will show:

There was 56,976 tons treated, with a gross value of $958,657.

The value is calculated at $20.67 per ounce for gold, and 50.35 cents for silver. The average metallic content of the ore was 0.245 oz. of gold and 23.42 oz. of silver. Gold extraction of 94.80% and silver of 90.88% are the highest recoveries yet made by the West End mill. The combined extraction was 90.92% of the metallic content, and 92.10% of the money value of the ore. (This latter figure of 92.1% is the accepted method of reporting extraction.) It is to be further noted that this recovery was made by cyaniding only. Concentration was abandoned in 1914, as the additional gross extraction made by its aid was overbalanced by the 8 to 10% marketing cost on the value removed in the concentrate. The cost in 1915 of marketing the products of cyanidation was only 1.7% of the gross value of the ore, giving the high net extraction of 90.4%. This is also the highest annual figure yet attained, and is considerably above the average 1915 figure of 89.75% for the mills of the Tonopah district.

The total recovery of metals was 13,219 oz. of gold and 1,210,037 oz. of silver.

The daily amount treated was 156 tons, which required 22.6 stamps dropping continuously, giving an average of 6.9 tons per stamp-day. Of the daily tonnage, 39 was custom ore, and 47 was the disputed Jim Butler *v.* West End ore which was treated as custom. The custom ore was irregular in quantity, varying from $6 to $96 per ton, and ranged in character from oxidized-ore screenings to the hard flint of the Kernick ore. These variations made it impossible to keep the consumption of chemicals and power to as low a point as could be done with a fairly consistent tonnage and grade of ore, and it required great attention to details of plant operation to obtain equal extraction. The sampling of all ores, with the control assays and necessary supervision, introduced a new expense in milling charges. However, if the treating of custom ores added to the expense of milling, it was far over-balanced by the direct profit from the milling of these ores. In addition to the direct profit there was an important indirect profit accruing from the increased daily tonnage milled.

During the last six months of the year the average value of the ore treated was $20.10. This was a composite of Tonopah ores, being mainly sulphide ores. A gross extraction of 92.1% was obtained, giving a net extraction of 90.5%. Although there was no concentration, the average titration of the strongest solution in the mill, being that of No. 1 agitator, was 3.5 lb. of potassium cyanide and 0.7 lb. of protective sodium hydrate. Sodium cyanide consumption was 2.9 lb. and 1.16 lb. of zinc shavings. These results are worthy of mention, as they mark a high point in the treatment of Tonopah ores without concentration.

The year 1915 was notable for its advance in price of nearly all materials used in milling, but such increases are usually met in all industries by greater economy and a closer study of costs. We are pleased to state that in our case, with a reduction in daily tonnage from 179 tons to 156, and with an increase in the gold and silver contents of the ore from 18.7 to 23.66 oz., the total direct milling costs show a decrease of 3c. per ton. With the decreased tonnage the total cost of labor and power per ton was held at the same figure, and with a 25% increase in the metallic content of the ore the cost of supplies actually decreased 3c. per ton.

The following table shows the main items of cost for 1915, and those of 1914 for comparison:

	Amount Used per Ton		Cost per Ton	
	1915	1914	1915	1914
Labor (average of $4.50 per shift), shift	0.156	0.161	$0.704	$0.724
Power, kw.-hour	36.0	35.0	0.526	0.506
Sodium cyanide, pounds	2.6	2.83	0.567	0.600
Cut sheet zinc, pounds	1.21	1.34	0.179	0.126
Lime, pounds	3.88	3.95	0.028	0.033
Lead acetate, pounds	0.66	0.43	0.051	0.037
Fuel-oil, gallons	3.19	3.10	0.122	0.121
Water, gallons	226.0	246.0	0.170	0.185
Pebbles, pounds	7.39	6.05	0.087	0.105
Tube-mill liners and supplies			0.042	0.050
Shoes and dies			0.030	0.026
Other supplies			0.235	0.255
Total direct milling costs			$2.741	$2.768
Indirect costs without depreciation			0.211	0.202
Total direct and indirect without depreciation			$2.952	$2.970

Special features that held the interest of the mill-crew during 1915 were as under:

Six tons of manganoid-steel balls were substituted for the six-ton load of Danish pebbles in the 5 by 15-ft. tube-mill. There was a sharp increase in the power required for the mill, but less power per ton ground. The saving in power was over-balanced by the greater cost per ton for the steel balls. Later, the mill was reduced to 3 ft. diam., and charged with steel balls, resulting in a considerable increase of tonnage and decrease in the power required for the mill over the use of Danish pebbles in the 5-ft. mill. On account of a 33% saving in power per ton ground, the test is being continued over a long period to determine the consumption of steel balls under the favorable conditions of the 3-ft. diam. mill. In 1915, using mostly Manhattan, Nevada, chalcedony pebbles in the 5 by 18-ft. tube-mills, the consumption per ton of ore was 20% greater, but the total cost per ton of ore ground was 20% less than in 1914 when Danish pebbles were mostly used. The Komata shell-liner placed in one of the 5 by 18-ft. mills in 1913, with the idea of reducing the cost of shell-liners 50%, has given the following results: life of plates, 22 months; life of angles, 6 months; while filler-bars were not worn at all. All this material was purchased from the local foundry, and the cost per ton ground was 2c., which is 33% of the cost when using the smooth liners.

In order to hasten the settling of light slime that lies close to the surface of the Dorr thickeners, an experiment was tried in one thickener of placing ½-in. square sticks 4 in. apart horizontally, and projecting 3 ft. vertically below the surface of the solution-level. Due to the daily change in character of the ore milled, accurate settling data have not yet been obtained, but the originator of the idea demonstrated a great improvement in settling in his experimental tank thus equipped.

During the year one of the Trent agitators was equipped as a replacer, to determine the feasibility of replacing the rich solutions of the agitator pulp with barren solution and then with water without recourse to the canvas leaf-filter. It was shown that it was possible to maintain a barren zone in the bottom of the replacer and thus replace solutions by this method; but, like the continuous-decantation system, although to a lesser degree, the extra tankage required did not justify a change from the present leaf-filter.

The most interesting and profitable experimental work of the year was a study of the mill solutions to determine what saving could be made in the use of cyanide and zinc without lowering extraction.* The result was an increased extraction, as shown in the figures given above for the last six months of the year. The saving in chemicals is shown by the fact that during 1915 we used 0.126 lb. of sodium cyanide per fine ounce of gold and silver bullion, compared with 0.170 lb. in 1914, a saving of 26%; and in 1915 we used 0.056 lb. of zinc shavings per ounce of bullion, compared with 0.081 lb. in 1914, a saving of 31%. While the 1914 results compared favorably with standard practice of this district, the 1915 consumption of cyanide and zinc would have been about $16,000 greater had the 1914 consumption figures prevailed. This would have increased milling costs 28.1c. per ton.

During the year considerable testing was carried on to find out if the flotation process could be used instead of cyaniding on West End ore, or as an adjunct to cyaniding. Although flotation can make an equal extraction with cyaniding on sulphide ores, it was decided that there was no field for flotation at present in the mill, as some of the ore is oxidized, and there is at present no satisfactory method of treating silver concentrate locally.

It is to the credit of the mill-crew that, due to their observance of Safety-First the year passed with only minor injuries, and no call was made on the State insurance fund.

*M. & S. P., Dec. 11, 1915.

Gold in Bolivia

A gold nugget weighing 14 lb. was recently found in the placer mines of Benedicto Goytia at Chuquiaguillo. This nugget has a maximum diameter of 13 centimetres (1 centimetre = ⅓-in.) and a maximum and minimum thickness of 8 and 3 centimetres, respectively; it is valued at $4,000. In the gravel of this same river another celebrated gold nugget, worth $4500, was found by an Indian miner in the seventeenth century, and was placed on exhibition in the Museum of Natural History at Madrid, Spain. Much of the gold in the possession of the Incas at the time of the invasion of the Spaniards 384 years ago is said to have been obtained from the Chuquiapu and Chuquiaguillo rivers. The name of the latter river means in the Aimara Indian language 'inheritance of gold.' In Bolivia gold is widely distributed in veins and placers. Along the rivers which run in a north-easterly direction following the eastern slopes of the Andes, there are extensive deposits of auriferous sands of great richness. The Chuquiapu, or river of gold, upon which La Paz, the capital of Bolivia, is situated, contains within the city limits auriferous gravel washed down from the slopes of the Andean range, and during the colonial period the gold-placer deposits of this river in the vicinity of La Paz were profitably exploited by the Spaniards.—*Bulletin* of Pan American Union.

Chromite

The only commercially important ore of chromium is chromite, which is an oxide of chromium and iron, $FeCr_2O_4$. In California the iron is likely to be replaced by magnesium, and the ore is found in serpentine. Chromite is usually sold on the basis of 50% of chromic oxide ('chromic acid'), Cr_2O_3, on which basis it brings $12 to $30 per ton at New York. The wide variation in price is due to the irregular and uncertain condition of the market, which is affected spasmodically by imports.

The production of chromite in the United States during 1915 was 3281 long tons. The quantity imported was 76,455 tons; this came from New Caledonia, Rhodesia, Portuguese Africa, Quebec, and Greece. The domestic production is largely from California, which uses most of its output locally, although a little is shipped to the Eastern States at a freight rate of $11 per ton. The Quebec deposits have attracted attention since a revival of mining during the past two years. During 1915, Quebec shipped 10,087 tons of chromite to American buyers in Pennsylvania steel districts. New Caledonia, the French island in the South Pacific between Australia and Fiji, has been the world's greatest producer of chromite, having mined as much as 82,806 metric tons in one year. Of late years Rhodesia has been an important producer.