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Notes relative to the ores occurring at Mercur, Utah together with a general description of the metallurgical practices applied to them at DeLaMar's Mercur Mines

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N O T E S

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D E L A M A R ' S M E R C U R M I N E S

BY

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The most notable increase in the gold production of Utah is due to the extensive operations of DeLaMar's Mercur mines in the Mercur district, which are at the present time producing more than two-thirds of the total tonnage of the entire camp. The nucleus of this great property consists of 143 acres, called the Golden Gate Group, by which name the property is still locally known, was purchased by Captain J. R. DeLaMar in 1895. This gentleman with his charcteristic, well-directed energy began immediate developments on an extensive scale, and in a short time was so well pleased with his initial purchase that he began acquiring surrounding territory, and continued to do so until the fall of 1897, by which time the area of the group had been increased to over 800 acres. At the time of DeLaMar's first purchase, the values of the ores from the district were being successfully extracted by the cyanide process, but the majority of the ores developed in the DeLaMar property were of a very different character from those coming from different other mines in operation, and were so refractory from various causes that the method of coarse crushing and direct cyandiding then in vogue in the mills of the camp was entirely unsuited for their treatment, as had been clearly demonstrated by the disastrous effects resulting from even small amounts of this base ore which were being occasionally encountered in small quantities in some of the other mines, gettinginto their tank charges, and entirely reversing otherwise satisfactory results. The Mercur ores in general cannot be characterised specifically with regard to their physical or chemical composition , for the reason that they are made up of widely varying proportions of several distinct varieties of gold-bearing material, viz, a very fine grained, minutely fractured and extremely hard quartz, a silicious limestone, also highly fractured, a compact ferrugineous clay, and small quantities of decomposed porphyry. These varieties of rock occur singly and mixed in all proportions. The values in the quartz and limestone lie largely in the numerous fracture seams. In the clays and porphyry they are very uniformyl dessèminated throughout the mass, and this class of ore is usually of higher grade than the others. Locally the ores are classified into three varieties, according to their mode of treatment.

1. Oxid ized Ores consisting of a mixture in which the calcareous and silicious varieties predominate, and in which the proportions of clay and talc are insufficient to interfere with percolation. This class of ore contains only an insignificant quantity of compound base elements, showing only very small fractional percentages of mercury, as cinnabar and arsenic oxidised compounds.

2. Talcy Ores; which are almost entirely clay and talc and soft, decomposed porphyry. These ores like the class above are almost free from base element compounds, but impossible of percolation, for the reason that, on contact with water, they disintegrate and settle to an almost impervious mass.

3. Base Ores; consisting of a mixture of the above classes with the calcareous and talcose varieties predominating, and containing large quantities of base metal sulphides. Arsenic is the chief of these, occurring as realgar, orpiment and mispickel, in quantities sometimes as high as 50%, but averaging not to exceed 2%. Realgar is by far the most plentiful of these arsenic bearing minerals, fully ~~Fuxl~~ three-quarters of the arsenic occurring in this way. Antimony is present as stibnite. Occasional small quantities of galena occur. Considerable quantities of iron pyrites are frequently encountered in minute crystals. Mercury is invariably present, but in less quantities than in the oxidized ores. Various hydrous sulphates of iron are present, as well as oxidization products of arsenic, both simple and in combination with lime and magnesia.

Some of the rarer elements, most notably tellurium, are also present, in traces only. The clays of this class of ore are invariably dark gray or black in color, due to a considerable quantity of carbon, frequently as much as four percent, and in these are sometimes found organ -ic compounds, which decompose potassium cyanide very rapidly. Silver is very sparingly distributed in all classes of ore, rarely exceeding one ounce of silver to ten ounces of gold. Not metallic gold is visible in any of the ores until after they have been roasted, when occasional minute, irregular particles may be discovered under the microscope. All the clean, base minerals are invariably poorer in gold than the gangue in which they are associated. The clear crystals of realgar and orpiment carrying none at all, or only traces, showing that the increased values of the base ore are not directly due to these base metal minerals. The gold in whatever ore found dissolves very rapidly and completely in solutions of potassium cyanide, indicating that it is very finely divided in whatever condition it occurs, and these various facts have led me to the conclusion that the gold in these ores is present in a finely divided, amorphous metallic state, having the black or brown color characteristis of the metal when in this condition, and consequently being unrecognisable under the microscope in its naturally occurring state.

Early in the development of the DeLaMar propeties, it was apparent that the larger proportion of the ores were of the base or talcose varieties above described, and which were at that time being left in the other mines when encountered, so that it became necessary to devise a plan for their successful treatment, and to this end laboratory experiments were began immediately after the purchase of the property in 1895, and continued into the spring of 1896, at which time sufficient data had been obtained to convince all concerned that the vexing problem would be solved along the lines followed in the laboratory work, and in May, 1896, an experimental plant to demonstrate these results on a larger scale was completed.

This plant, capable of treating five tons of ore at a time was equipped with complete crushing, roasting, leaching and precipitating machinery, so arranged that the details of treatment could be varied widely, and after continuous operation for nearly a year, and the expenditure of many thousands of dollars in equipment and investigation, it was closed down, having served its purpose in demonstrating that the refractory ores of the Mercur district could be successfully treated by the method hereinafter described,andn almost immediately afterwards (July, 1897) ground was broken for the erection of the magnificant plant which has now been in successful operation for over two years.

The mill is built in eight levels on the hill side above the mines, having a slope of 20 degrees. It has been enlarged twice since the completion of the original design in March, 1898, and will be described as it is at present.

The highest level is occupied by a steel ore bin, having a capacity of 2500 tons, on top of which is erected a steel gallows frame carrying the hoisting cables from the mine to the power house. Above the ore bins is also located the coarse crushing machinery, consisting of two No. 6, style D, Gates crushers. Into these the ore is dumped over grizzlies, direct from the skips, and passing through the crushers, falls into the storage bin below. The ore is delivered to the mill by 3-ton skips, running in balance direct from the mine through 300 feet of vertical and 400 feet of incline shaft, ad thence over 200 feet of inclined steel bridges connecting the mouth of the incline from the mine with the top of the gallows frame. The hoisting is done by a double drum, electric hoist, situated 100 feet up the hill back of the gallows frame and main storeage bin.

The next section below contains the necessary drying machinery, consisting of two straight line driers, with hearths 60 feet long by 12 feet wide, and two revolving cylindrical driers, 7 feet in diameter at the discharge end and 30 feet long.

The third level contains the fine crushing machinery, screens and elevators required to finish the crushing of the ore to the necessary fineness, and deliver it to the steel,-finished ore storage bin of 3000 tons capacity, also located on this level. The machinery of this section consists of four sets of 15 x 36" and five sets of 15" x 26" Gates high grade rolls, nine 24" x 8 ft Berthelot separating screens, four 48" x 8' revolving, hexaginal screens, six 12" elevators, and two motors, one of 150 h.p. and one of 100 h.p., giving the necessary power for driving the machinery of this and the drying section.

The next section, ebracing three levels, contains the roasting and calcining machinery, consisting of nine straight line furnaces, each with a roasting hearth 100 ft long and 12 ft wide; three of these are of the Jackling pattern; two are Holthoff-Wethey and the four others were originally built according to the Brown design, but have been entirely re-constructed and made to correspond with the Jackling pattern as far as possible, being supplied with Jackling mechanical parts throughout.

The next sections contains the leaching tanks, 26 in number, each 25 ft wide, 50 feet long and 5 ft deep, and having a capacity of 250 tons. Adjacent to and above this level are three solution standardizing tanks, having a combined capacity of 500 tons of solution. The next and last section contains three precipitating tanks, having a combined capacity of 100 tons of solution; two gold solution tanks holding 300 tons, and one barren solution sump tank, holding 150 tons; nine Stilwell-Bierce & Smith-Vaile all-iron filter presses, distance frame pattern, each having 36 sections, 24" square; two 600 gallon, cycloidal belt driven pumps, and 10 h.p. belt driven air compersser.

The plant throughout is driven by electrical power, derived from Provo River, 36 miles distant. The current is transmitted by the 3 phase system over three No 5 bare copper wires, at a pressure of 40,000 volts to the sub-station near the upper part of the mill, where it is transformed to a tension of 220 volts, for use by Westinghouse type C, two-phase induction motors, and distributed for power and light purposes for the mill and mine at this prssure. There are in use for every purpose twenty motors, ranging in size from 10 to 150 h.p., and giving for all uses the utmost satisfaction, and requiring a surprisingly small amount of attention and repairs. In stormy weather there has been considerable difficulty and delay occasioned by the breaking down of the long distance transmission system, and to overcome these delays in the future an 800 h.p. steam generating plant at the mill is just being completed. The plant is equipped with complete and commodious shops, in which all the repair and manufacturing work required about the place is done, with the exceptdon of making castsings, which are purchased in the rough. The extreme length of the mill throughout the leaching department (the longest section) is 720 feet along the face of the hill, with a width up the hill of 500 feet measured on the natural slope. The immensity of the plant can be better understood from the fact that the single floor space covered by the main and adjunct buildings is something over three acres. The total surface area necessary for the accommodation of the entire plant is nearly ten acres. The buildings are steel throghout, with the exception of the power house and refinery, which are brick, with corrugated iron roofs. The tanks and ore bins are also of steel, there being no wood anywhere in the plant except where it was desirable to use heavy timbers rather than steel for machine foundations.

The ores are roughly graded in the mine into the three classes above described, as follows: "oxidised", "Talcy", or as it is called at the works "mixed", and "base", and these classes are kept separate until the furnaces are passed, when they are all mixed together to go to the leaching tanks.

In order to keep the three classes of ore separate, the crushing machinery is necessarily arranged in three series, and both the large storage bins divided into three compartments. The crushers are so arranged that two classes of ores can be taken through either of them, but the rolls are operated each series for one class only. The crushers are set so that everything passing through them will pass a two inch ring. From the coarsely crushed ore storage bin, the ore is automatically fed to the driers and by them discharged to the heads of Berthelot screens, the fines going direct to the elevators, which discharge them into theirproper finished ore bin compartment. The coarse ore passes a set of large rolls, set with their faces about 1/2" apart, to anotherinclined screen, where the fines are again removed, the rejections passing on through the small rolls, which are kept set face to face. From the small rolls the ore passes to elevators, and is elevated to revolving screens over the finished ore bin. From these screens, the fines drop directly into the bin, and the rejections are returned to the small rolls for further reduction. It is seldom necessary to operate the driers, as the ore as it comes from the mine, except at unusually wet seasons can be rolled and screened to the proper fineness without previous drying. On this account belt conveyors have been provided to convey ore from the coarse ore bin to the rolls when it is not necessary to dry it. These also work entirely automatically. On account of using both inclined and revolving types of screens, it is not possible to [illegible] an absolutely uniform product, but it is intended to finish the oxidised ores so that they will pass a 3-mesh No. 16 wire cloth; the talcose, or mixed, to pass a 4-mesh No. 16 wire; and the base an 8-mesh No. 16 wire. About 90% of the pulp will pass these sizes, the remaining 10% of slightly coaser material having gone through the inclined screens, the mesh of which is necessarily coarser than the product desired, on account of the inclined position of the cloth.

From the finished ore bin, the oxidised ore is ready to go direct to the leaching tanks, the base and mixed classes going to the roasting and calcining furnaces. The four furnaces nearest the bins are automatically fed from it by means of an elevator, discharging into a small hopper connected by a six-inch standard pipe to each of the furnace automatic feeding hoppers. It is worthy of mention that all spouts in the mill are 6-inch or 8-inch standard black pipe, and that they serve the purpose admirably, from the fact that they are dustless, seldom clog, and when worn on one side can be turned one-third way over, thus affording three new wearing surfaces for one spout, and avoiding the troublesome operation of frequent repairing. The other five furnaces are automatically fed by two belt conveyors, one conveyor supplying two furnaces, and the other, which is over 200 feet long, the other three.

The base ores are roasted to a "dead" or "sweet" condition, or as nearly so as practicable, being finished at a bright, red heat. The raw ores, entering the furnace with about 2% to 5% sulphur, and xxe1% to 2 1/2% arsenic, have these constituents reduced to about 3/4% of 1% and 1/1o of 1% respectively. The high sulphur contents remaining in the roasted ore is almost entirely due to calcium sulphate, the sulphur remaining as sulphides rarely exceeding 1/10 of 1% and usually being lower. The arsenic sulphides, realgar and orpiment begin to volatilize with only partial decomposition, causing dense, orange colored vapors to arisebefore the ore reaches the first fireboxes in its travel through the furnace, and by the time it reaches a dull red heat, 90% of these minerals have been discharged, but the most difficult part of the operation follows in completely oxidising the pyrites and arseno-pyrites. This difficulty is partially due to the fact that a high heat is not permissible on account of the presence of from 5% to 10% of lime in the ore, producing a mixture which sinters easily, and is consequently ruined. It is worthy of note that the most careful investigation and checking of results on the roasting of over 100,000 tons of this arsenical ore, no appreciable gold losses could be detected that were traceable to other causes than ordinary dust losses, which were only a small fraction of a percent. The talcose ores are calcined, not so much for the purpose of ridding them of any deleterious compounds as to change their physical character -istics, although the small amount of arsenic present in them as arsenous acids and its compounds, is slightly reduced. The sulphur contents, all of which are present as sulphates of the alkaline earths, is not affected. The primary reason for the calcination of this ore is to de-hydrate the clay and talc, and deprive them of their properties of plasticity, thus rendering them amenable to percolation, and this is very effectively done. The effect on the clayey ore is very much the same as that of burning a brick clay, and the particles of ore which when raw would immediately disintegrate on contact with water, are, after calcination, unaffected by it. The ore is at the same time rendered extremely porous and permeable by solutions, and is thus transformed from an impossible leaching ore to an ideal one.

Both styles of furnace in use have been used as roasters and as calciners. As roasting furnaces for base ores, they have a capacity of 70 tons of roasted ore in 24 hours, this capacity, of coarse, varying somewhat with the varying characterof the ore. In roasting base ore, the furnaces are driven at about 40 feet per minute rabble speed, with a 4-inch bed of ore on the hearth. As calciners they have about double the above named capacity, which is obtained by speeding them up slightly and increasing the feed so as to run a much thicker bed of ore, usually about six inches. It requires a given particle of base ore about 8 hours to pass through the furnaces, while the talcose ores, which only require heating to a dull red heat, pass through in 6 hours. The Jackling furnaces have been found better adapted to roasting than the others, both in lower fuel consumption and greatercapacity, and are used for that purpose. The other furnaces, which require an excessive amount of fuel to maintain them at high heats, are run at a low heat as calciners. About the same amount of fuel is required for the furnaces whether runas calciners or roasters, viz, 6 1/2 tons of slack coal per day. All the furnaces are run with a light forced draft, Sturtevant blowers being used for that purpose. After being roasted or calcined, as the case be , the heated ore is elevated at the discharge end of the furnace, to a sheet iron floor above, and is conveyed by the rabbling mechanism in its return motion to the fed end of the furnace. During the 6 or 8 hours required for its return trip, it has become cool, and is sprayed with water until slightly moist, for the double purpose of giving a more porous charge in the leaching tanks, and avoiding dust in handling it to them. Just before reaching the feed end of the furnaces again, the now roasted and cooled ore is dishcraged off the side of the furnace, and automatically fed to a mixing and charging bin, from which it is drawn into cars for charging the leaching tanks. Four of the furnaces are in such a position that they discharge by gravity into this bin or pit, which is excavated in the middle furnace level, the ore being brought to it from the other furnaces by a system of belt conveyors. Into this pit is also drawn the oxidised ore from the finished ore binm, and it is essential that care be taken to get the most intimate mixture of all classes of ore before charging into the tanks, as otherwise greatly varying leaching rates will be maintained through the different characters of ore, crushed to different degrees of fineness in the same charge, and uniform leaching results be impossible.

The bottom of the charging pit is reached from the leaching section at the level of the top of leaching tanks by a short double compartment tunnel. Here for the first time since leaving the mine the ore is handled manually, being drawn out of the charging bin into one-ton, side dumping, basket cars, and dumped into the tanks, from four tracks running longitudinally over the top of them for the full length of the section.

The filters in the leaching tanks are of gravel, four inches deep. In constructing them a lattice ~~ma~~ work is mae of 1" x 4" pine strips, set on edge longitudinally with the tanks, spaced 5" centres, and held in place by 4" x 4" x 1" blocks, placed 18" apart and nailed in, thus making a stiff framework, with compartments 4" x 18" and 4" deep. These are filled with two inches of gravel sized to about 3/4"; one inch of gravel sized to about 1/2", and one inch of coarse sand. Between each of these layers of filling, is laid a strip of 12-oz. burlap, fitting the compartment neatly. On top of the finished gravel bed, a covering of 12-oz burlap is tacked, and over this asecond covering of 16-oz burlap. Iron strips are then fastened overthe covering, by nailing them through to each longitudinal strip of the frame. These strips are to protect the cloth in shovelling out tailings. This filter is very effectualy giving a perfectly clear solution under all circumstances. The top covering of 16-oz burlap has to berenewed about once a year, and in two years it is necessary to renew both covers, and replace the one inch of sand with fresh material, on account of it having collected enough fine slimes to become hard and slow of filtering. Two men on a 8 hour shift tram the pulp to a tank and fill it, and one additional man half a shift levels off the surface and prepares the charge for applying the first solution, and as this labor is paid $2.50 for 8 hours work, it will be seen that the ore is charged into the vats for less than 2 1/2¢ per ton. All tanks are connected bottom and top to one strong solution main, and at top only to weak solution and water main. the first solution is applied to the ore from the bottom of the charge, under a 14-ft head from the standardising tanks above. This solution contains 4/10 to 5/10 of 1% potassium cyanide, or 8 to 10 lbs of the solid salt per ton of solution, and is allowed to saturate the charge slowly until solution begins to appear at the surface, which requires about 8 hurs, when it is shut off at the bottom, and sufficient additional solution run on top to cover the charge about two inches. Thus it is allowed to stand about 16 hours, when the bottom outlet is opened and percolation started, strong solution being again run on at the top. This is continued for 24 hours, and is followed by a weaker solution of strength 3/10 to 35/100 of 1%, for from 48 to 72 hours, or until preliminary sampling, which is done every day, indicates that the charge is ready to wash and discharge. Washing and draining requires about 24 hours, making the total time from the application of the first solution to the discharge of the final wash 5 to 6 days, and from the time when the tank begins charging until it is again ready to be filled, 6 or 7 days. All solutions, whether strong, weak or wash, coming from the leaching tanks run into a common launder direct to the gold solution storage tank. Percolation is continued without interruption when once started, When a strong solution of 45/100 of 1% is being used, these solutions coming from the various leaching tanks have a strength of 3[illegible]/100 to 35/100 of 1%, and this solution, after being precipitated, is pumped back and used as weak solution, without the addition of any fresh cyanide. The leaching rates varies greatly with the character of the ore in the charge, being most rapid with charges containing a large proportion of the coarsely crushed , oxidised and calcined mixed ores. The practise is to allow three-quarter in to one inch depth of solution in the leaching tanks to pass off each hour, but with charges containing a large proportion of base ore, this rate is frequently not attainable, sometime dropping as low as 1/2" or 1/3" per hour, thus requiring that the charge be given longer time in order to get the necessary amount of solution through it, or that vacuum pumps be used, which is now sometimes done, although the standard practice is to leach by gravity only. The best results are attained when the total solution and wash passing through the charge is about two tons of solution to one ton of ore. A peculiar condition sometimes arises in tanks containing large proportions of roasted base ore, or all of this class, from the fact that the proportion of calcareous clays and silicious limestones are right to make a pretty good cementing material when burned, and in such cases the charge sets, and becomes very hard, after about two days contact with solution, rendering preliminary sampling with an augur very difficult, and greatly impeding percolation. When the percentage of limestone is exceptionally high in the ore, and is pretty thoroughly burned to lime in the furnaces, its subsequent hydration on contact with the solution will [redacted] cause a tank charge of ore of this class to swell perceptibly, and in such cases leaching becomes extremely slow and diffiult. The consumption of cyanide in leaching is about 9/10 of a lb per ton of ore, one-half of which amount is consumed from the first strong solution applied, and a considerable portion of the remainder is lost by imperfectly washing out the weak solution. The actual consumption from the weak solution during the several days of contact is inconsiderable.

The standard strong solution is made by adding the necessary cyanide in lumps to the weak solution formerly described. Before bringing cyanide strength up to standard, however, a sufficient quantity of cuastic soda is added to bring the solution to a fixed standard of alkalinity. About a pound of caustic soda per ton of solution is used for this purpose. By this means a very great saving of cyanide is effected, amounting, as found by practice to about one pound of cyanide saved for every pound of caustic soda used. On account of absence of sufficient water for flushing tailings out of tanks, these are discharged by shovelling through gates in the bottom, into cars below, holding 2 1/2 tons, and trammed out of the building by men. Each tank is provided with 8 discharge gates, 15" in diameter, located above four longitudinal tracks, running the full length of the building below the tanks. Discharging a tank takes 5 to 7 hours, and costs 6 to 8¢ per ton depending on the condition of the charge of tailings, i.e., whether they are more or less compact.

Precipitation of the gold from the solution is effected by means of zinc dust. The material used is the blue powder by-product obtained in zinc smelting. The dust used by the Company is imported from England or Germany, and contains about 90% metallic zinc. The solution is pumped from the gold solution tanks to the precipitating tanks, of which there are three. 30 tons of solution is a charge for precipitation in each tank. While the tank is filling, air at 10 to 15 lbs pressure is blown into the solution through a half inch pipe [redacted], keeping it in a state of violent ebullition. This is done to stir up the precipitate that has settled to the bottom of the tanks from former charges, and which contains a large amount of unconsumed zinc. Five pounds of fresh dust is used for every charge of 30 tons, this amount being seived in, beginning when the tank is half full, and continuing the additions at intervals until the tank is full and the zinc has all been added. The air pipe is then moved about the bottom of the tank for a short time, to thoroughly stir up all sediment again and then removed, and the suspended matter allowed to settle about half an hour, when the supernant solution is drawn off through an opening in the side of the tank, eight inches above the bottom, and as it still contains considerable quantities of suspended gold slimes, it is passed through the filter presses to collect these, and runs from them direct to the barren solution sump. It is seldom necessary to use pressure to force the solution through the presses, other than the head of 18-ft between them and the bottom of the precipitating tanks, but sometimes this has to be done, when the presses become well filled with slimes, and for this purpose two pressure tanks of 30 tons capacity each are suspended beneath the precipitation tanks, and

connected with them by large pipes, so that atank of solution can be discharged into them in a very few minutes, and then forced through the presses, and when it becomes neccessary, the solution can be handled in this way as rapidly as when the presses are free to work by gravity. The filtering medium used in the presses is two layers of light canton flannel between which is placed a sheet of heavy, unsized paper. The precipitating and filtering operation is continuous, as while one tank is filling another is discharging, and the third settling, and by this system as much as 2500 tons of solution has been precipitated in three tanks in 24 hours. The dust precipitation is almost instantaneous, being complete [illegible] a sample taken at once after the last zinc has been added. The precipitated solutions ~~xxx~~ after passingthrough presses rarely exceed 20¢ per ton in value and are usually lower. The consumption of zinc dust is about 1 1/3 lbs per oz of gold recovered.

Clean ups are made monthly and require about three days working on the day shift only. The precipitating tanks are allowed to drain and the wet slimes scooped into pans, after which the tank is washed out into the presses, at the same time the presses are opened and cleaned. The long time occupied in collecting the product is on account of having to clean one tank and three presses only at a time, so that the operation of the plant is only slightly inconvenienced. About half the total value is found in the presses, although the weight of product recovered from the tanks is three-quaretrs of the whole. The product is taken to a refinery in a separate building, and dried in shallow iron pans in a furnace having large cast iron muffles into which the pans are put. The cloths from the filter presses are also burned, and the product brought to a dull red heat by which most of the remaining metallic zinc is oxidised. After cooling it is pulverised througha 1/4" mesh screen and **sampled. It usually contains from 2 1/2 to 3 oz of** gold per lb of dry product. It is next treated in a lead lined, dissolving tank, with a mixture of dilute sulphuric and nitric acids, by which the remaining zinc and zinc oxide, arsenic and mercury are removed, as well as considerable quantities of lead, coming from the zinc, and complex cyanides, lime, etc, from the solutions. The nitric acid is used to more effectually oxidise these substances, as well as to, in a great measure at least, prevent the evolution of the deadly arsenuretted hydrogen gas. The dissolving tank is covered with a closely fitting hood, which is connected to a large exhaust fan taking the disagreeable and dangerous gases entirely outside the building, and discharging them from a tall stack. The action of the acid is very violent and it must be added in small quantities to avoid its boiling over the top of the tank. When the addition of fresh acid fails to cause further action, the tank is filled with water and allowed to settle. The supernant liquor is then drawn off through a pressure tank and filter press, and the slimes again agitated with fresh water, this latter operation being repeated twice, to free them as far as possible of soluble salts. The slimes are then flushed into the pressure tank, filter pressed into cakes, dried, coarsely pulverized, mixed with a flux of soda, potash and borax glass, and smelted in graphite crucibles, in a double oil-burning furnace, capable of receiving two No. 300 crucible at once. At this stage, before melting, the product contains 60% gold, the other 40% being largely silicious slimes. The resulting bullion is 950 fine, and is cast into bars of 1,000 oz. for shipment. This plant is the only one in the district refining its own product, all the other mills shipping to eastern refineries. The total cost of refining is about 15¢ per oz. of gold.

Nothing has been said regarding the capacity of the mill for the reason that this is a variable quantity, depending upon the proportions of the various classes of ores being treated. The plant is designed to handle 1000 tons, run with seven furnaces on base ore and two on mixed, which gives 800 tons capacity of roasting and calcining ores, and supplied with 200 tons per day of non-roasting or oxidised ore. This capacity it handles with the utmost ease, the crushing equipment being equal to considerably more. If more calcining and oxidised ores are delivered to the mill, the capacity increase by reason of more furnaces being used as calciners, and putting through double the tonnage they do when used as roasters. If the base ore delivered increases beyond the proportion stated, the capacity of the mill decreases correspondingly. The mill is designed to take the various ores in about the proportion they exist in the mine, but its operation is varied to suit condition in the mine with regard to the most economical and complete extraction of the ore as it occurs, and the ore bodies are very spotted as regards the different classes of ore.