Recovery of Precious and Secondary Metals from Electrolytic Copper Refining

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Most modern electrolytic copper refineries have found it expedient to build and operate a complete secondaries plant for the treatment of the electrolytic slime accumulated in the cells during the corrosion of the copper anodes. In the earlier years this was done for the sole purpose of promptly recovering in marketable form the silver and gold contained in the slime. Later, improvements in refining methods made possible the recovery of the more precious metals such as platinum, palladium, osmium and iridium, and in recent years, due largely to an increasing market, many secondaries plants have undertaken the commercial production of selenium and tellurium. The complete up-to-date secondaries plant, or silver refinery as it is usually called, is therefore equipped for the production in marketable form of from six to eight metals. The physical plant and operations may be separated into six main divisions:

1. Slime treatment, in which the raw slime is treated by wet methods for the removal of its high copper content and portions of other impurities such as arsenic, selenium and tellurium.

2. Furnace refining, in which the remainder of the base-metal content of the slime is removed by fluxing and fire-refining.

3. Electrolytic parting and refining, in which the silver and gold are parted and obtained in the pure state.

4. Melting of silver and gold into marketable form.

5. Recovery and refining of the more precious metals.

6. Recovery and refining of selenium and tellurium.

The Silver Refinery of the Raritan Copper Works at Perth Amboy, N. J., is a complete secondaries plant and the following description of its equipment and processes is, in a general way, typical of other plants of its kind.

SLIME TREATMENT

The raw slime, pumped with a minimum amount of cell solution from the electrolytic copper refinery, is discharged into the center of circular settling tanks 15 ft. in diameter by 8 ft. deep, where the slime settles out and the solution overflows peripherally to return to the copper refinery by gravity. The slime is then dropped through a bottom outlet to a

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mixing tank where it is kept at the proper consistency, about 30 per cent solids, by mechanical stirring, and fed to an Oliver continuous filter.

This filter, which is mainly of hard lead construction and completely proof against sulfuric acid-copper sulfate solution, has a drum 3 ft. in
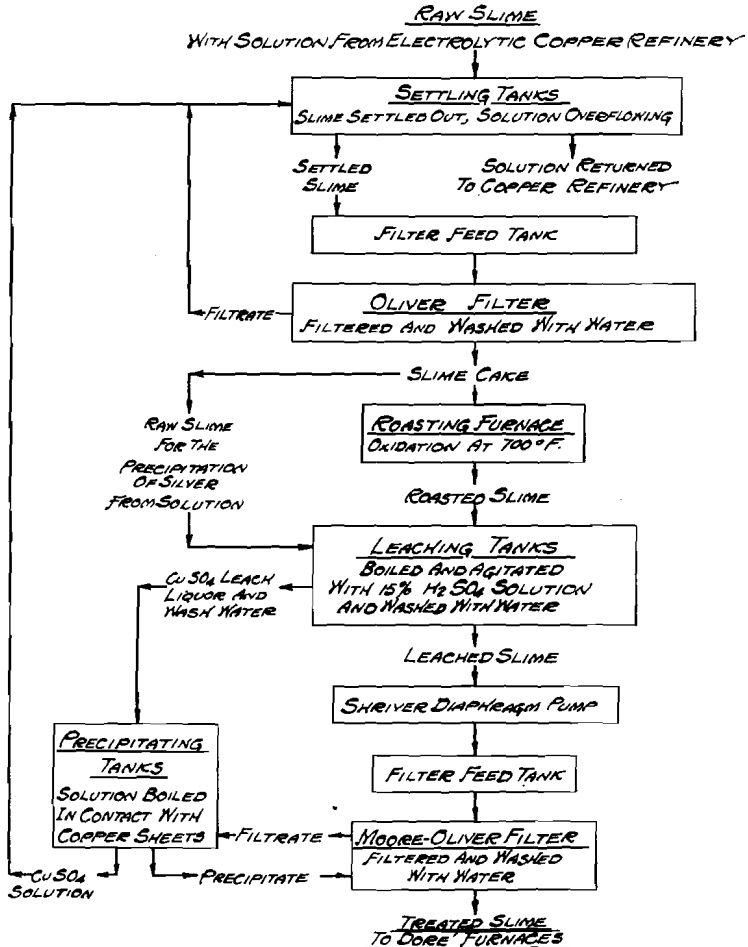

FIG. 1.—FLOW SHEET OF SLIME TREATMENT IN ELECTROLYTIC COPPER-REFINERY SECONDARIES PLANT.

diameter with a 4-ft. face.   The cake, which attains a thickness of $\frac{1}{2}$ in., is washed with water from a spray pipe as the drum revolves, and discharged with a moisture content of less than 35 per cent into lead-lined trucks.

The slime cake is then taken to a pan roasting furnace 27 ft. long by 9 ft. wide, transferred to sheet-iron pans 48 by 22 by 4 in., which are then stacked in sets of three.   One of these sets is introduced on rollers

into each of the 10 doors opening on either side of the furnace. Heating is obtained by means of two oil burners at one end of the furnace, the pans being protected from the flame by the brick roof over the divided firebox, which extends the full length of the furnace under either row of pan racks. At the rear end of the furnace the hot gases are conducted up and back over the pans to the gas stack at the burner end. The slime is left undisturbed under a temperature of about 700° F. for a period of 8 hr., ample air being admitted to the combustion chambers to insure an oxidizing atmosphere. The pans are then removed from the furnace and elevated by means of an electric hoist to the leaching-tank platform where the entire roaster charge of approximately 3000 lb. of oxidized slime is dumped into one leaching tank containing a boiling 15 per cent sulfuric acid solution.

These tanks, circular in shape and 5 ft. in diameter by 6 ft. deep, consist of a channel-iron skeleton lined with 10-lb. hard lead, and stirring is obtained with a slow-moving two-bladed cast-lead paddle, gear-driven from above. Agitation is continued for several hours until the copper is leached out and a test of the liquor shows silver in solution. A quantity of raw slime, having sufficient metallic copper content to precipitate the silver from solution, is then added and after a short period of settling the leach liquor is siphoned off. Water is run into the tank and boiling and agitation resumed in order to wash the entrained leach liquor out of the slime. Five washings are usually sufficient and the slime is then dropped through a bottom outlet into a sump from which it is pumped by a Shriver diaphragm pump to the mixing tank, preparatory to dewatering on a Moore Oliver-type filter. The cake is given a final wash on this filter and discharged into lead-lined trucks ready for furnace refining.

Any leach liquors or wash waters that upon test show selenium or tellurium in solution are pumped to precipitating tanks, where these elements are removed by boiling in contact with copper. The precipitate thus formed is later treated in the doré furnaces for the recovery of any values it may contain.

All boiling of solutions is done with steam introduced through open-end lead pipes or coils, dilution from condensation being of no importance. Leach liquors and wash waters are removed from the leaching tanks by means of large gravity siphons into a sump tank from which they are pumped wherever desired with a 2-in. vertical centrifugal pump. All solutions free from selenium and tellurium are pumped back to the settling tanks, from which they return to the electrolytic copper refinery by gravity. Solution lines are all of hard lead and fitted with lubricating duriron plug cocks.

The standard dry vacuum system is used for both Oliver-type filters, each being equipped with a 9½ by 8-in. vacuum pump and a 2-in. Olivite centrifugal pump to remove the filtrate against the vacuum.

Typical analyses of the electrolytic slime before and after this treatment are given in Table 1.

TABLE 1.—*Typical Analyses of Electrolytic Slime*

	Raw Slime	Treated Slime
Silver, oz. per ton..........................	5130.88	7684.00
Gold, oz. per ton...........................	101.52	146.20
Copper, per cent...........................	18.10	1.74
Lead, per cent.............................	3.19	7.18
Arsenic, per cent...........................	5.63	3.60
Antimony, per cent.........................	4.66	13.97
Selenium, per cent..........................	5.31	4.51
Tellurium, per cent.........................	6.14	3.49

The percentage difference unaccounted for in each analysis is made up of sulfates, oxides and silica.

FURNACE-REFINING OF SLIME

The treated slime, containing less than 2 per cent copper and from 25 to 30 per cent moisture, is charged into a small reverberatory furnace and melted down, a complete charge consisting of 8 to 10 tons of wet slime introduced into the furnace one ton at a time over a period of approximately 20 hr. When completely melted, a vitreous slag, consisting of the lead and a large portion of the antimony combined with the silica content of the slime, rises to the surface and is skimmed off. Two air pipes are then introduced into the melt, and working of the charge with rabbles is continued for about six hours until the remainder of the antimony has been volatilized. The arsenic and a part of the selenium and tellurium are also driven off during this period. Several hundred pounds of fused sodium carbonate is next introduced to break up the matte, after which the removal of the remaining selenium, tellurium and other base metals in the form of a water-soluble slag is carried on with the use of sodium nitrate as a flux and oxidizing agent. As this slag forms in sufficient quantity it is skimmed off and more niter worked in, this procedure being continued until the bath of doré (silver and gold) remaining in the furnace is of a high degree of purity, averaging 980 fineness silver and 15 fineness gold. The total time of the furnace refining cycle is from 48 to 72 hr., depending on the base-metal content of the slime.

This silver-gold metal is then ladled out into water-cooled, double-face, reversible molds to produce anodes for the subsequent electrolytic parting. Two types of anodes are cast, one $19\frac{1}{4}$ by $21\frac{1}{2}$ by $\frac{1}{2}$ in. for the Thum electrolytic process, and the other $6\frac{1}{4}$ by 9 by $\frac{1}{2}$ in., with a small lug at one end, for the Moebius process.

The furnaces have a hearth area 6 by 8 ft. The bottom and sides are lined with magnesite brick and the roof is of clay brick. They are oil-fired with a single Quigley-type burner through a combustion chamber at the rear, the oil consumption being from 15 to 20 gal. of light fuel oil per hour. The working of the charge is done through a front door above which the uptake carries the gases into the main flue, while the charging of slime is done through a door midway in the side.

A motor-driven draft fan 6 ft. in diameter midway in the main flue system furnishes the draft for the furnaces and keeps the gases moving through the entire flue system. Ample cooling area and chambers for settling out the dust are provided before the gases enter the base of the first of three steel-supported lead towers 25 ft. high by 4 ft. 6 in. in diameter. As the gases rise they are scrubbed and moistened with water falling from the top of the tower. This treatment is repeated in the downward passage of the gases through a second tower, and in rising in the third tower they drop their excess moisture before entering a Cottrell electrical precipitator.

The lower ends of the three lead towers in the scrubber system are immersed in a water-seal tank from which the scrubber water overflows into a settling tank. The clarified discharge from this settler is conducted to a 3-in. duriron centrifugal pump of capacity ample to deliver the water again at the top of the first two towers into reservoirs through the perforated bottoms of which it falls as a spray through the columns of gas. Adequate scrubbing and moistening is obtained in this manner without the use of baffles or tower packing.

For final treatment the gases are passed through a standard tube-type three-unit Cottrell electrical precipitator. Each unit consists of 26 vertical lead tubes 8 in. in diameter by 12 ft. long equipped with a carefully centered lead-covered electrode wire on which the high-tension electric current is impressed. The four corner tubes of the unit are 11 in. in diameter and have a correspondingly larger lead-covered iron-rod electrode sufficiently strong to carry an insulated grid, which keeps the other wire electrodes centered in the 8-in. tubes. Even distribution of the gas is obtained in a chamber beneath the tubes while another chamber over the top of the tubes protects the process from the weather, at the same time providing for the exit of the gases through a vertical stack in which tests of the efficiency of precipitation can be made. One of these units ordinarily is capable of clearing the gases from the operation of two doré furnaces for a period of 12 to 16 hr. The high-tension unidirectional current for the precipitation is obtained by transforming 220-volt alternating current to 50,000 volts and then rectifying mechanically.

Operation of the three units is so arranged that there is always available a clean unit that may be put into service while either or both of the other units are shut down for cleaning. Every approach to any unit in

operation is guarded by one of a system of Robinson interlocks which enables operators or repairmen to work on adjacent units without fear of injury from the high-voltage current.
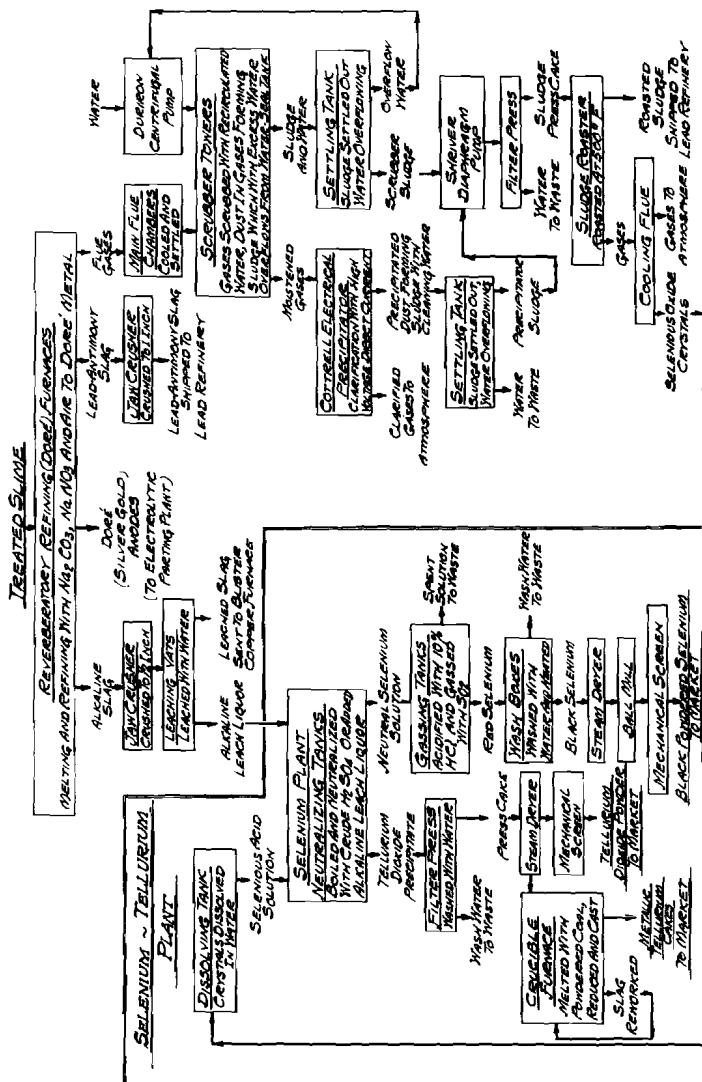
FIG. 2.—FLOW SHEET OF FURNACE REFINING AND SELENIUM-TELLURIUM RECOVERY AT ELECTROLYTIC COPPER-REFINERY SECONDARIES PLANT.

The precipitate of sludge accumulated on the walls of the tubes and electrodes is daily washed down by hand with water from a high-pressure hose and sluiced to a settling tank. This sludge, together with that accumulated in the settlers of the scrubber system, is filter-pressed with a Shriver diaphragm pump and plate and frame-press installation.

The sludge-press cake is then delivered to a reverberatory roaster, where it is given a slow low-temperature roast for the removal of selenium and arsenic. In the flue chambers taking the gases from this roaster the selenium is deposited as selenious oxide crystals, which are periodically removed to the selenium plant, while the arsenic passes out into the air through a high stack. The material remaining on the furnace hearth consists mainly of lead and antimony oxides together with the silver and gold values contained in the original sludges. This roasted sludge is packed in bags and shipped to a lead refinery for final treatment.

The vitreous lead-antimony slag skimmed off early in the furnace-refining of the treated slime is reduced in a jaw crusher and also shipped to a lead refinery for final treatment.

In order to insure the subsequent removal from the cycle and the recovery of the selenium and tellurium, every precaution is taken during the fluxing stage of the furnace refining operation to make the slag as completely water-soluble as possible by means of the proper use of both sodium carbonate and sodium nitrate. This alkaline slag is reduced to $\frac{1}{2}$ in. in the jaw crusher and dumped into concrete leaching vats 15 by 15 ft., which are equipped with burlap and sand filter bottoms. Ten to fifteen 24-hr. leaches with water remove the selenium and tellurium contents in the form of a highly caustic solution of sodium selenite and sodium tellurite which is pumped from beneath the filter bottoms direct to the selenium department for treatment. The insoluble slag residue remaining after leaching is dried and returned to the blister-copper furnaces.

Electrolytic Parting and Refining of Silver and Gold

The standard processes employed for the electrolysis of silver-gold anodes do not lend themselves to the elimination of any appreciable amount of base-metal impurities, since the latter may be rendered soluble as the anode corrodes and then be plated or cemented out at the cathode with the silver. The anode metal is brought to the highest possible purity therefore in the doré furnace, 995+ parts of silver plus gold, and the electrolytic treatment becomes simply a parting process to separate the silver from the gold and any other precious metals the anode may contain. Two types of electrolytic equipment are commonly employed, the Thum or horizontal process, and the Moebius or vertical process, each having its advantages and disadvantages, which will be discussed later.

In the Thum process the anode is placed within a wooden frame lined with muslin and covered with a close-fitting canvas sack. This frame in turn is placed in a heavier frame which, resting on the sides of the electrolytic cell, is partly immersed in the electrolyte and equipped with slats across the bottom to support the weight of the anode. This anode frame assembly covers about one-half the solution area of the cell, leaving

the other half open. The cells themselves, measuring 48 by 24 by 7 in., were formerly of vitrified chemical stoneware or porcelain but are now being replaced with asphalt-lined concrete tanks. The bottoms of these tanks, which must serve as cathode surfaces, are lined with slabs of graphite and carbon ½ in. thick. The current enters the cell through a
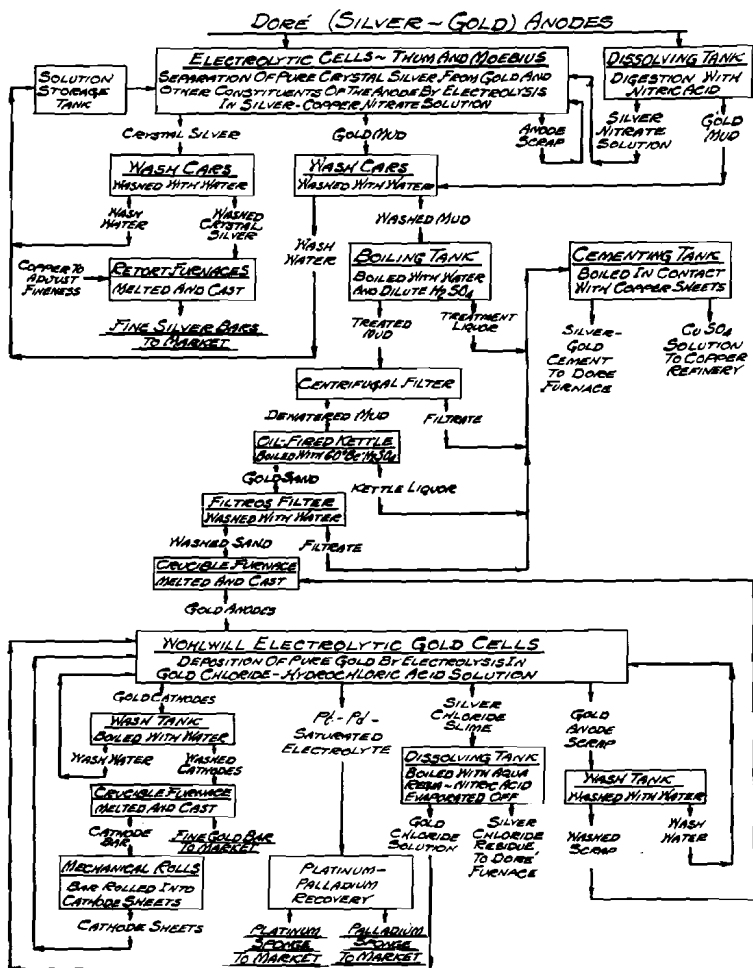

FIG. 3.—FLOW SHEET OF ELECTROLYTIC PARTING AND REFINING OF PRECIOUS METALS IN ELECTROLYTIC COPPER-REFINERY SECONDARIES PLANT.

silver contact piece which rests on the submerged anode in the tray, and is conducted from the cell through another silver contact which rests on the graphite slabs underneath the solution at the open end of the cell. The silver corroded from the anode passes through the muslin and canvas diaphragms, deposits on the carbon-slab bottom underneath the tray, and is scraped forward periodically by hand to the open end of the tank to

make way for continued deposition. The accumulated deposit is removed from the open end of the tank to a filter-bottom wash car once each shift. As the anode corrodes away, another is placed on top of it so that each anode in turn is completely electrolyzed, leaving no scrap to be remelted.

The anode current density is 50 amp. per sq. ft. with a maximum current per cell of 150 amp. at 3 to $3\frac{1}{2}$ volts, the cells being arranged in series in sections of 21 tanks. The electrolyte is a neutral silver-copper nitrate solution analyzing 60 grams per liter silver and 40 grams per liter copper.

In the Moebius or vertical electrolytic parting process the same solution and current density are employed, but the anodes and cathodes are suspended vertically in a cell measuring 24 by 29 by 22 in. The anodes, 9 by $6\frac{1}{4}$ by $\frac{1}{2}$ in. with a lug for hanging, are suspended in rows of three in canvas bags, the anode bar across the tank carrying the sack and its frame as well as the hangers supporting the anodes themselves. Cathode sheets of chromium iron 12 by 20 by $\frac{1}{16}$ in. are suspended from the bus-bars at the sides of the tank. The cathode surfaces are continually scraped free of the crystal silver deposit by wooden blades attached to an oscillating rack which rests on the framework over the tanks and is mechanically operated. The crystal silver scraped from the cathode sheets falls through the solution into a wooden tray, which is suspended from the electrode assembly frame over the tanks. As the anodes corrode away new ones are hung in their places and the scrap lugs, which otherwise would have to be remelted, are placed in the Thum anode trays for complete corrosion.

Each tank contains five anode groups and six cathode sheets, connected in parallel. The cells are connected in series in sections of six with a maximum current density of 450 amp. at $2\frac{1}{2}$ to 3 volts.

After a full producing day the entire operating equipment of a section is mechanically raised as a unit to a position 1 ft. above the cells. The trays, which have hinged bottoms, are then emptied of the accumulated crystal silver deposit into filter-bottom wash cars.

The advantages of the Thum process are: (1) Mechanical and current conductor simplicity, which make for low cost of installation and maintenance; (2) complete corrosion of the anodes, which may be of any size, shape, or weight within the capacity of the anode tray; (3) no necessity for shutdown while the crystal silver production is being removed.

The disadvantages are: (1) Corrosion from but one face of the anode, resulting in low production per unit of floor space and volume of electrolyte; (2) dependence on and cost of hand labor for scraping the deposit from the cathode; (3) lack of regular stirring or circulation of electrolyte in the cell.

For the Moebius process the advantages are: (1) Low labor cost due to mechanical scraping of the deposit from the cathode, with the added benefits of regular stirring of the electrolyte, a more uniform deposit, and a current efficiency of 98 per cent, which is 5 to 10 per cent higher than that obtained in the Thum process; (2) close spacing of electrodes, together with corrosion of both faces of the anodes, resulting in high production per unit of floor space and volume of electrolyte; (3) no necessity for muslin diaphragms, because the anode does not rest on the accumulated gold mud.

Its disadvantages are: (1) Mechanical and current conductor complexity, resulting in a high installation and maintenance cost; (2) complete shutdown necessary for the removal of the crystal silver production.

In both the Thum and Moebius processes the gold accumulates around the anodes in the sacks as a black slime or mud contaminated with particles of silver and basic copper salts. This mud is removed from the Thum trays after every three days of operation by taking out the muslin tray lining and washing it into filter-bottom wash cars. In the Moebius system the anode sacks are removed from the assembly and washed out after the same period. The solution is filtered off the mud in the wash cars and the mud transferred to a wooden boiling tank 3 ft. in diameter by 3 ft. deep, where it is boiled up with water to which is then added a small quantity of sulfuric acid to break up the basic copper salts. After settling has taken place the wash liquor is siphoned off and the mud is transferred to a Tolhurst centrifugal filter to be finally washed with water and dried. The dewatered mud is charged into an oil-fired cast-iron kettle 4 ft. in diameter, where it is boiled for about 3 hr. in 60° Bé. sulfuric acid. This boiling removes the copper and most of the silver from the mud and leaves a residue of brown sand which assays about 970 parts gold. This sand is thoroughly washed on a Filtros-block filter ready for melting down for the production of gold anodes.

Silver-copper nitrate wash waters from the washing of crystal silver and the preliminary washing of the gold mud are filtered and elevated by means of a small hard-rubber pump to a chemical stoneware storage tank from which hard-rubber distribution lines run to convenient sections of the parting plant. This wash water is used to make up electrolyte evaporation and withdrawals resulting from the removal of crystal silver and gold mud. Sufficient electrolyte is entrained in the latter and destroyed in its subsequent treatment to obviate the necessity of special withdrawals to maintain the purity of the electrolyte. Silver nitrate solution for replenishing the electrolyte is made by digesting doré anodes in nitric acid.

Gold mud wash water containing sulfuric acid and the strongly acid boiling kettle liquors, as well as the water resulting from the washing of the gold sand, are collected in a sump tank and pumped to a special

cementing tank in the slime plant, where the precious metals are pre-cipitated by boiling the solution in contact with copper sheets. The gold-silver cement thus produced is periodically refined and cast into anodes in the doré furnace.

Melting and Casting of Silver

The crystal silver, after being thoroughly washed with water in the wash cars, is charged into clay-graphite retorts and melted. These bottle-shaped retorts, 38 in. high, are mounted in oil-fired tilting furnaces of the Faber-du-Faur type and hold a charge of about 30,000 troy ounces. The consumption of light fuel oil with a small Quigley-type burner is 8 to 10 gal. per hour. The charging and melting of a full charge requires 5 to 6 hr., after which the molds are brought up on a carriage and the standard-size bars are poured by tilting the furnace, about 45 min. being required for the casting of the usual 25,000-oz. lot in 1000-oz. bars. The fineness of the silver is adjusted to 999, the standard for the fine silver market, by adding to the melt, just before pouring, a few ounces of pure copper.

Electrolytic Refining of Gold (Wohlwill Process)

The gold sand is melted in No. 25 graphite crucibles in a cylindrical oil-fired crucible furnace, the Quigley-type burner consuming about 8 gal. of light fuel oil per hour. The metal is cast into vertical closed molds to pro-duce gold anodes averaging 980 fineness and 140 troy ounces in weight, the dimensions being 9 by 5 by $\frac{3}{8}$ in. The electrolysis of these anodes is accomplished by the Wohlwill-process, in which from six to eight of them are suspended in pairs from silver anode bars in the electrolyte in a small stoneware electrolytic cell measuring 20 by 12 by 12 in. Cathode starting sheets of fine gold rolled to a thickness of 0.003 in. and having an immersed area 12 by 2 in., are suspended in rows of four from silver cathode rods spaced between the anode bars. A current of 450 amp. is put through the cell, giving a voltage of $1\frac{1}{2}$ volts and an anode current density of 125 amp. per square foot with six anodes. The electrolyte is a gold chloride solution containing from 80 to 100 grams per liter of gold and 10 per cent free hydrochloric acid. Heat to maintain the minimum operating temperature of 140° F. is obtained by means of a steam coil placed around the stoneware cell and enclosed by a lead-lined wooden box, sand being packed between the lead and the stoneware cell to give even distribution of the heat from the coil. Circulation of the elec-trolyte during operation is maintained by means of a Pohlé air-lift. The anodes are corroded to 5 per cent scrap in from 16 to 18 hr., the pure gold plating out on the cathode sheets in a rough, somewhat nodularized, crystalline deposit which adheres firmly.

The cathodes are removed from the cell, thoroughly washed, dried and melted down in a clean crucible from which the gold is cast into an open mold to produce a bar weighing from 700 to 800 oz. and assaying 999.7+ parts per thousand. The anode scrap is washed, dried and remelted with the next batch of gold sand.

Silver, the chief impurity in the gold anodes, forms silver chloride during the electrolysis and falls to the bottom of the cell as a slime together with varying amounts of gold. This slime is removed periodically, leached with aqua regia for the removal of gold and the production thereby of gold chloride solution to be used in replenishing the electrolyte, the gold content of which is gradually depleted during electrolysis. The silver chloride remaining after this leach is returned to the doré furnace.

Strong wash waters are used to take up electrolyte evaporation, while dilute waters are sent to the cementing tank. Auxiliary equipment used in this process consists of two electric drying ovens and power-driven 6-in. rolls for the rolling down of a small bar of fine gold into cathode starting sheets when needed.

Fine silver bars are first weighed on a springless scale and finally weighed with careful checking over a bullion balance of 3000-oz. capacity and 0.005-oz. sensitivity. Fine gold is weighed in the same careful manner.

PLATINUM AND PALLADIUM, SELENIUM AND TELLURIUM RECOVERY AND REFINING

Any platinum and palladium present in the original raw slime is finally concentrated in the gold anodes and during their electrolysis goes into solution in the electrolyte, the platinum and palladium content of which can be permitted to go as high as 60 grams per liter. The electrolyte is then removed, oxidized with nitric acid, and the combined salts of platinum and palladium precipitated by the addition of ammonium chloride. Separation and eventual refining of the platinum and palladium are carried out by the usual chemical methods, both metals being produced in the sponge form and of 99.5+ per cent purity.

The raw material for the selenium and tellurium plant is the caustic solution obtained from the leaching of the soda-niter doré-furnace slag, together with a water solution of the selenious acid crystals collected in the flue of the sludge roaster. This combined solution is brought to the point of neutrality while boiling in two lead-lined tanks, 10 by 6 by 6 ft., either by the addition of crude sulfuric acid or additional slag liquor, and complete precipitation of the tellurium as tellurium dioxide is thereby obtained. The precipitate is allowed to settle and the neutral solution siphoned off into one of the two large wooden gassing tanks, 10 by 6 by 8 ft., where it is acidified with 10 per cent of hydrochloric acid and 5 per cent of sulfuric acid by volume and gassed with sulfur dioxide

generated in a horizontal water-cooled sulfur burner. Complete precipitation of the selenium in the red amorphous form is obtained with from 24 to 72 hr. of gassing. After settling has taken place the solution is siphoned off to waste and the selenium dropped through a bottom outlet to a wash box having a burlap filter bottom, where it is washed with water. The washed red selenium is converted to the black form by the application of steam and then transferred to pans in which it is dried in a steam oven. The dried product is ground in a ball mill for several hours and mechanically screened through 100 mesh for the market. This powdered selenium, analyzing 99.5+ per cent, is packed in containers for shipment or, if required in vitreous form, is melted in iron pots and cast into cakes or sticks.

For the recovery of tellurium the tellurium dioxide precipitate from the neutralizing tanks is pumped to a filter press by a Shriver diaphragm pump. The cake is thoroughly washed and then transferred to pans and dried in a steam oven. If it is to be marketed as tellurium dioxide, it is screened through 100 mesh and packed for shipment. To produce metallic tellurium the dioxide is mixed with powdered coal and reduced in a clay-lined graphite crucible mounted in a Monarch oil-fired tilting furnace. The metal thus formed is then poured into cakes or sticks. If powdered tellurium is required, the metallic cakes, which are brittle and show a pronounced crystal fracture, are broken up and ground in a ball mill. The product is screened to obtain the desired grade of powder and packed in containers for shipment.

SUMMARY OF MAJOR EQUIPMENT AND PLANT CAPACITIES

The slime plant, with 10 slime-leaching tanks, can handle 120 dry tons of raw slime per month and turn out from 60 to 90 dry tons of treated slime per month, depending on the percentage of silver it contains.

The furnace plant, with two of the three furnaces for the refining of slime to doré in continuous operation, can refine 150 dry tons of treated slime per month, there being ample capacity to handle, in addition to the output of the Raritan slime plant, any treated slime or crude doré bullion received from other plants of the Anaconda Copper Mining Co. The capacity production of two furnaces working on slime varies, according to the silver content of the slime, from 1,250,000 to 2,000,000 oz. of doré anodes per month.

The capacity of the parting plant, with the three 6-cell Moebius sections and six 21-cell Thum sections in operation, is 2,250,000 oz. of crystal silver per month. The melting of this amount and the casting into fine silver bars can be taken care of with one of the two retort melting furnaces in continuous operation.

Facilities for refining gold mud are ample to handle an amount containing 35,000 oz. of gold per month. The Wohlwill electrolytic plant

with one of its two cells in operation will produce 25,000 oz. per month, so that its capacity is greater than that required to refine the gold resulting from full operation of the parting plant on doré running as high as 15 parts per thousand gold.   Platinum and palladium refining facilities are adequate for the production of 200 oz. per month of the two metals. The selenium plant is capable of a production of 3000 lb. of powdered selenium per month and the tellurium plant can turn out 1500 lb. of metallic tellurium per month.