such rows. Into this tunnel the gases resulting from the roasting are drawn through perforations in the back walls of the stalls, and are led away to the sulphuric-acid chambers. The combustion is started by lighting a small fire in a grate outside the stall; this kindles the wood, the heat evolved by the combustion of the sulphur in the regulus being afterwards sufficient to continue the process without the addition of any other fuel. This roasted second regulus is then treated as before, until a regulus is produced which contains about 35 per cent. of copper.

Instead of this repeated roasting and treatment with the slag-charge, the regulus, after having been roasted, may be treated in a cupola, together with lead-slags and other fluxing and reducing additions. The product is a comparatively rich regulus, the lead originally present being reduced to the metallic state, taking with it the silver, the enriched regulus retaining but little of the precious metals. When enriched to the degree mentioned above, the regulus is brought, by a single roasting in an ordinary Welsh reverberatory furnace, to a "white" or "pimple" metal containing about 75 to 80 per cent. of copper. This is not further treated at the Müldenhütte, the principal works at Freiberg, but is sent to the Halsbrücke works, where it is roasted sweet, and the copper converted into copper sulphate by treating the roasted material with sulphuric acid. Any lead and gold that may have been present remain undissolved, as does also the greater portion of the silver, the small quantity that passes into solution being reprecipitated by metallic copper. The undissolved lead residue is added to the blast-furnace treatment.

Speise.—This is a comparatively rare product. The nickel and cobalt it contains are concentrated by a process resembling that for the concentration of the copper in the regulus, the speise being first roasted and then re-melted with fluxes, such as lead-slags, and with reducing agents, in a small blast-furnace. Metallic lead is obtained, which contains the greater portion of the precious metals originally present in the speise, and the latter is much richer in nickel and cobalt after this treatment than it was before. In this concentrated state it is sold to outside works, where it is treated by the wet process, the residues, which contain the gold and silver of the speise, being bought back by the Government works.

The *ore-slag* is treated in the manner that has been described, the resulting slag, poor in lead, being thrown away. This contains about $2\frac{1}{2}$ per cent. of lead and 0·0045 per cent. of silver, the zinc-oxide also present occasionally reaching some 20 per cent.

The *fume* contains large quantities of lead-oxide, together with zinc-oxide and arsenious anhydride, which is collected and sold after a further sublimation. The oxides of lead and zinc remain on the bed of the furnace, and are added to the blast-furnace charge.

Thus, the general scheme of treatment is as follows:—The ores are mainly those of lead, copper, zinc, and silver. The products are lead, zinc, bismuth, silver, gold, arsenic, as metals, and copper, mainly recovered as sulphate. Besides these, there are also incidental products, such as sulphate of iron and sulphate of manganese. Nickel and cobalt are recovered as arsenides—that is, in the form of speise; and the arsenic is recovered partly as metal, but mainly as powdery arsenious anhydride, and as red, yellow, and white arsenical glass.

The ores come partly from surrounding mines, and partly from foreign sources, the latter being smelted mainly for the copper, silver, and gold they contain. Almost all the ores are argentiferous. The principal portion of each charge consists of lead ores, and these are divided into galenas with 30 per cent. or more of lead, and poor lead ores with 15 to 20 per cent. In the mean they contain generally 40 per cent. of lead, and 0·015 per cent. of silver. The lead in the ore is only paid for when it reaches 15 per cent., additional payment being made for every 5 per cent. above 20 per cent.—that is, for 25, 30, 35, and so on. Those ores in which only the silver or gold is of a value sufficient to cause them to be paid for are called "Dürrerze," and the class is subdivided into—pyritic, 20 to 40 per cent. sulphur; quartzose, 10 to 19 per cent. sulphur; and spathose, 0 to 9 per cent. sulphur.

The sulphur is paid for when it exceeds 24 per cent., and every additional 5 per cent. is paid for. Each per cent. of copper is paid for, but fractions of 1 per cent. are not taken into account.

The zinc ores must not contain more than 5 per cent. of lead, and in ores that are not described as zinc ores, only 3 to 4 per cent. of zinc may be present.

The copper ores contain from 1 to 15 per cent. of copper, arsenical ores from 10 per cent. arsenic, zinc ores from 30 to 40 per cent. zinc, sulphur ores from 25 per cent. sulphur.

The arsenic in an ore is paid for when it reaches 10 per cent., and fractions of 5 per cent. are accounted for.

Gold is paid for from 0·0005 per cent. and upwards, and silver 0·004 per cent. and upwards.

The weighing takes place in the presence of representatives of the mine and of the works. About 5 cwt. is weighed at a time. The accuracy of the weighing depends on the richness of the ore; 0·01 to 0·05 per cent. silver accurate to 1 lb., 0·50 to 5 per cent. silver accurate to 0·1 lb., 5 per cent. and upwards, accurate to 0·02 lb. Ores rich in gold are always weighed to 0·02 lb.

Sampling.—A small scoopful is taken from every 2 cwt. and is thrown into two or more wooden dishes, according to the amount weighed off. The ore is then put into the ore-house, and a board is inserted in the ore-heap, showing the character of the ore, its quality, etc. The moisture is determined by heating 75 grammes in a copper shovel; the calculation of the moisture is accurate

to $\frac{1}{2}$ per cent. The mixing takes place in the ore-house. The various ores are spread out in layers, the thickness of which depends on the nature of the material, layer over layer. Then a definite portion is cut off one end of the heap, and the ore so removed is thrown up into a conical heap, which is then considered to be sufficiently mixed.

Roasting.—Heaps of from 6000 cwt. to 7000 cwt. supply four roasting-furnaces for a week and a half; one furnace will treat 180 cwt. a day. The roasting-furnace is a single-bedded reverberatory, upwards of 40 feet long and about 10 feet broad; it has eleven doors on each side, and one, which is used for roasting pure galena, has fifteen doors on each side. It is worked from both sides at the same time. The fire-bridge is hollow, and the bed consists of a layer of fire-bricks, made from two parts of raw and one part of burnt clay; they contain 60 to 70 per cent. of silica. These bricks rest on common bricks supported by an iron plate, resting in turn on pillars. If the hearth, or fire-bridge, becomes worn the work is not stopped, but, by means of a long iron ladle, a ball consisting of one part of clay and two parts of poor quartz ore, is inserted. This is beaten down so as to repair any defective place.

Each charge of ore consists of 34 cwt., and there are five such charges on different parts of the furnace-bed at one time. The charge near the fire-bridge is allowed to clot, and is then removed. The other charges on the furnace-bed are then advanced towards the fire-bridge the distance of two doors at a time. The fireplace is divided in the centre, so that really there are two fireplaces, or grates, to each furnace, as the breadth is too great to permit one fireman to stoke the whole.

These long-bedded calcining-furnaces are in connection with long brick chambers for the condensation of the fume, the collection of which takes place every six months, when some 2700 cwt. to 3000 cwt. of material is collected, containing—0·01 to 0·02 per cent. silver, 10 to 28 per cent. lead, 40 to 50 per cent. arsenious anhydride. The sulphurous anhydride from these flues cannot be used, as the gases are far too dilute and impure.

The charge for roasting generally consists of—galena, 30 to 40 per cent.; poor lead ore, 20 to 30 per cent.; poor quartzose ore, 10 to 15 per cent.; and residues, 5 to 10 per cent.

The roasted ore still contains from 5 to 7 per cent. of sulphur when the semi-fluid charge is withdrawn from the furnace into barrows of sheet-iron. The roasted charge is tipped out when solid, and broken into pieces about the size of the fist, and sorted by sight into (1) well roasted, (2) ordinary, and (3) badly roasted, according to whether much or little undecomposed galena is seen to be present. The men are paid accordingly.

There are several large Pilz furnaces, each with eight tuyères, and one small one with four tuyères. The following is a charge

for the ore-smelting:—Ore, 450 cwt.; pyritic ore, 80 cwt.; roasted residues, 40 cwt.; slag, 550 cwt.; quartzose ore, 20 cwt.; total, 1140 cwt. This gives 88 cwt. of argentiferous lead, containing 0·4 to 1 per cent. of copper, and 20 cwt. of regulus, and very often a speise forms as well. In charging, the coke is thrown towards the centre, and the larger pieces of ore to the side.

If an obstruction or "bear" forms in the furnace, it is usually a ferruginous one; but it may contain much zinc-sulphide, and it generally forms at the top of the boshes. In order to remove it, bricks are removed below the bear, and the obstruction is knocked away while hot. The bear often has the form of a ring. The hearth of the furnace is built upon an iron plate. Then follow three layers of common bricks placed flat, and then two layers of fire-bricks, these together being about 16 inches thick. Then follows another layer, 16 inches thick, of the fire-bricks already described. The pressure of the blast is about $10\frac{3}{4}$ to $13\frac{1}{4}$ inches of water when smelting ore, and 7 to 10 inches when smelting slag.

The hearth is allowed to fill until some of the regulus is seen to come out with the slag, which is tapped continuously from one or other of two tap-holes.

The lead contains .	.	0·5 to 0·6	per cent. of silver.
The lead regulus .	.	25·0 ,, 30·0	,, ,, lead.
		6·0 ,, 15·0	,, ,, copper.
		0·2 ,, 0·25	,, ,, silver.

The slags contain 0·3 per cent. of copper, 4 to 5 per cent. of lead, 0·003 to 0·004 per cent. of silver.

Smelting the Rich Slag.—The slags may be viewed as three equivalents of monosilicate with one equivalent of bisilicate, and slags that do not contain more than 0·001 to 0·002 per cent. of silver and 1·5 per cent. of lead are thrown away.

The charges in slag-smelting are very varied. The following are examples:—

(1) Slag, 900 cwt.; plumbiferous material, 45 cwt.; broken-up hearths, 12 cwt.; which yielded 52 cwt. of poor lead, 85 cwt. of coke being required.

(2) For every 2 cwt. of a mixture of 5 cwt. roasted regulus from slag, 12 cwt. roasted ore-regulus, 2 cwt. roasted copper-regulus, and 5 cwt. roasted liquation-residues, is added $4\frac{1}{2}$ cwt. of slag from galena-smelting, and 4 cwt. of common slag.

(3) Three cwt. of ore-slag, 3 cwt. of various residues rich in lead, $1\frac{1}{2}$ cwt. once-roasted ore-regulus, 0·5 cwt. of coke. To this charge there is added, in the working-day of twenty-four hours, 12 cwt. of marl, 12 cwt. of hearth, and 12 cwt. of lead-skimmings.

(4) Twelve cwt. of slag, 25 cwt. of twice-roasted regulus, 40 cwt. of slags from the smelting of the litharge.

Products of the Slag-smelting.—Slag-lead containing 0·4 per

cent. of silver, regulus containing 10 to 20 per cent. of lead, or regulus containing 20 to 30 per cent. of copper, poor slag containing 0·0015 per cent. of silver and 1·5 to 2 per cent. of lead.

Sometimes a speise of nickel and cobalt is also formed. The regulus is only about one-third of the amount of lead produced. The slags often contain 9 per cent. of oxide of zinc. The regulus is broken up and roasted in kilns, or stalls, and is then added to a smelting charge similar to that by which it was formed, or is concentrated in a Pilz furnace, or goes to a reverberatory for concentration with silica and barium-sulphate.

Smelting the Speise. — The charge is 150 cwt. of speise, 675 cwt. ore-slag, 4 cwt. impure litharge, 44 cwt. lead residues, 5 cwt. hearth, 40 cwt. fluorspar, 75 cwt. coke. The products are 54 cwt. of lead, 12 cwt. concentrated speise.

The speise is concentrated by continuous and alternate roastings and smeltings, until it contains 20 per cent. of nickel, when it is sold. The poor lead goes to the liquation-furnace. The regulus is roasted in kilns and stalls, then fused to concentrate it, until it contains 30 per cent. of copper; it is then called "copper" regulus. It is afterwards treated in a reverberatory to form pimple metal. Incidentally, large quantities of impure lead and substances containing lead-oxide are produced.

The process employed for the reduction of the lead from such substances consists of a reducing-fusion. One such substance, the litharge derived from the rich argentiferous lead, contains about 78 per cent. of lead. The furnace used is a small Pilz furnace with four tuyères. The charge in twenty-four hours is 1500 cwt. litharge, 450 cwt. lead-slags, 50 cwt. slags from a previous smelting, 130 cwt. coke. The products are (1) a variety of lead, which, according to its purity, is either first liquated and then refined, or is taken to the Pattinson pots, and (2) a slag containing 10 per cent. of lead. Part of this slag goes to the next charge, and part to the first ore- and slag-smelting.

Liquating the Impure Lead. — This is done at Freiberg on the inclined bed of a reverberatory furnace, 500 cwt. being treated daily. The copper, with some lead, remains solid in the form of liquation-residues. The lead, before the liquation, contains 0·5 per cent. copper, afterwards it still retains 0·07 per cent. The residues are about 5 per cent. of the total lead; they consist principally of lead with 15 to 18 per cent. of copper, and are added to the slag-smelting in the Pilz furnace.

The scum from the Pattinson pots is also treated in this reverberatory, in admixture with lignite. When antimonial lead is liquated the temperature must be kept low, but a little wood must be kept kindled in the well of the furnace to prevent the lead solidifying.

If the lead contains 1·5 per cent. of silver, it is cupelled at once—that is, it is added to lead that has been already enriched

by the Pattinson process. If it is impure, the lead is refined and Pattinsonised.

In refining, the lead is fused in an oxidising atmosphere on the bed of a reverberatory furnace. The products are a variety of different kinds of litharge to which the name of *Abstrich* is given. They are nothing but impure litharge, and, as the nature and proportion of the impurities vary greatly, the appearance is very dissimilar; some are dark-grey and stony, others are light-yellow and are crystalline in structure:—(1) Powdery tin *Abstrich;* (2) fused tin *Abstrich;* (3) arsenical *Abstrich;* (4) antimonial *Abstrich;* (5) impure litharge *Abstrich;* (6) pure litharge *Abstrich.* There is a fractional oxidation of the various impurities, the tin and arsenic going first. The time required to refine 400 cwt. of lead is about fifty hours.

The products, in the case of lead smelted from ore, are as follows:—10 cwt. of tin *Abstrich*, or about 3 per cent. of the total lead, containing 11·3 per cent. of tin, 14·4 per cent. of arsenic, 2·8 per cent. of antimony; 20 cwt. arsenical *Abstrich*, or about 5 per cent. of the total lead, containing 8·5 per cent. of antimony, 8·9 per cent. of tin, 8·7 per cent. of arsenic; 10 cwt. of antimonial *Abstrich*, about 3 per cent. of the total lead, containing 6·8 per cent. of antimony, 1·3 per cent. of tin, 4·4 per cent. of arsenic; 10 cwt. impure litharge, about 3·2 per cent. of the total lead, containing 3·1 per cent. of antimony, 0·5 per cent. of tin, 2·0 per cent. of arsenic.

In refining lead produced from the smelting of slag, very similar varieties of *Abstrich* are obtained, except that they are not so rich in tin, as when lead from ore is refined. At the Halsbrücke works, a hollow pipe through which steam is passed is sometimes used as a mechanical stirrer when refining. In desilverising the tin *Abstrich*, about 90 cwt. of slag-lead is placed on the bed of the refining-furnace and then, on that, 15 cwt. of the *Abstrich*, mixed with 4 per cent. of coal; the charge is melted down in three hours, and then another 15 cwt. of *Abstrich* and 4 per cent. of coal is added, and so on until the furnace is full. The desilverised *Abstrich* is taken off (the silver passing into the lead) and run down in a small Pilz furnace.

The lead is refined and Pattinsonised. The antimonial *Abstrich* is desilverised in the same manner, except that 24 cwt. of it is added instead of 15 cwt., as it melts easier. The arsenical *Abstrich* and impure litharge are directly revivified by being run down in a small Pilz furnace for metallic lead, the lead obtained being then refined. The impure lead is added in part as a plumbiferous addition to the Pilz furnace smelting ore.

The desilverised tin *Abstrich* is smelted in a Pilz furnace, the charge being 100 parts of *Abstrich* to 100 to 150 parts of poor lead-slag, 50 per cent. of slags from the same working, and 20 per cent. of limestone. The lead obtained is revivified and poled in

a Pattinson pot. The various kinds of *Abstrich* obtained are desilverised over and over again, until they contain comparatively no silver. The composition of the hard lead obtained from the stanniferous *Abstrich* is very varied. One variety contains 18 per cent. of tin, 10 per cent. of antimony, and 2 per cent. of arsenic.

The desilverised antimonial *Abstrich* is smelted in a Pilz furnace with slag and 10 per cent. of limestone. In twenty-four hours 100 cwt. to 150 cwt. are sent through. The hard lead is liquated and poled, and contains about 10 per cent. of antimony, 3 per cent. of arsenic, and 1 per cent. of tin. Each refining-furnace is worked by one refiner and two assistants.

Pattinson Process as conducted at Freiberg.—In this process, the silver is concentrated in fluid lead by straining off crystals of lead which separate from the bath. The pots have a thickness at the bottom of 2·36 to 2·76 inches, and 1·9 inch at the sides. They are 5 feet in diameter at the top, and 2 feet 9 inches at the bottom. They hold about 15 tons, and last about 500 crystallisations. Each pot has a separate fire; there are sixteen pots, sometimes worked in two batteries of eight pots each. They are worked by the one-third system. Each pot is kept at a determined percentage of silver, and assays are made daily. When fresh lead is added, it is introduced into the pot containing the same percentage of silver. At Freiberg it is usually the third pot from the left. The rich lead contains from 1·5 per cent. to 2 per cent. of silver, and the poor 0·0018 per cent. The scum from pots 1 to 4 is liquated by itself, and similarly those from pots 5 to 8, and from 9 to 15 by themselves. In a case in which the enrichment had not been carried far enough, the following were assays of the contents of the pots :—(1) 1·07 silver, (2) 0·80 silver, (3) 0·42 silver, (4) 0·33 silver, (5) 0·30 silver, (6) 0·26 silver, (7) 0·18 silver, (8) 0·11 silver, (9) 0·08 silver, (10) 0·04 silver, (11) 0·02 silver, (12) 0·012 silver, (13) 0·007 silver, (14) 0·0035 silver, (15) 0·0015 silver, (16) 0·001 silver. The poor lead never contains more than 0·05 per cent. of copper, 0·2 per cent. of iron, and traces of arsenic and antimony. The fuel used is a mixture of lignite and small coke. To each battery of sixteen pots there is one fireman, and to every two pots there are two men. There is also one lead ladler. The men work for eleven hours, but the fireman works for twelve hours.[1]

Cupellation at Freiberg.—The hearth is formed of 48 cwt. of fresh marl mixed with ½ cwt. of clay. The marl is of three kinds :—

	I.	II.	III.
Calcium carbonate . .	50	68	66
Magnesium carbonate .	13	27	6
Iron carbonate . . .	2	2	2
Clay	14	3	25

[1] Pattinson's process is now supplemented by Parkes' process, a joint-process being used.

Any pyrites present is carefully removed.

The greatest depth of the hearth from a line on a level with the tuyères ought not to be more than 7·8 inches. The fire-resisting material with which the roof is lined consists of one part of clay and two parts of silica; 100 cwt. to 200 cwt. of lead is placed on the damp hearth, and this is covered with sawdust and chips; the roof is then put on, the wood lit, and the roof luted on with clay; the fire is kindled, and the blast turned on; the lead melts down in sixteen to eighteen hours. The temperature is gradually raised, and lead is added, 700 cwt. being the total charge. The litharge, as it forms, is removed until the remaining lead on the bed contains 60 to 80 per cent. of silver. This is taken out, and the extraction of the silver completed on a similar but much smaller hearth. If the red litharge obtained contains less than 0·02 per cent. of silver, it is sold. It is said that if the lead contains 0·2 per cent. of bismuth no red litharge will be produced. At the end of the process, where the concentrated lead and silver settle on the hearth, there is a dark spot which indicates the presence of bismuth, and which is broken away and the bismuth subsequently extracted, this portion of the hearth having been previously hollowed to collect the rich lead. There are two men attending to each furnace, and it requires 120 to 140 hours to cupel 700 cwt. of lead. The silver obtained is granulated by running it into water. It is about 990 fine, and is sent to the Halsbrücke works to have the gold parted from it.

Solution of the Regulus.—The concentrated copper regulus is sent to the Halsbrücke works, where it is roasted nearly sweet. It still contains about 1 per cent. of sulphur. It is then passed through a rather fine sieve, and the larger pieces are crushed and re-roasted. The roasted regulus is dissolved in sulphuric acid in wooden vats lined with hard lead. The sulphate of copper is crystallised out, and, after purification by re-crystallisation, is sold. Such copper sulphate as does not crystallise out but remains in solution is removed by scrap-iron. The solution of copper sulphate before crystallisation is made to pass over metallic copper, in order to remove any silver it may contain. The residue, after the treatment of the regulus with sulphuric acid, contains the lead, and most of the silver originally present in the regulus treated. This is sent back to the Pilz furnace, with a view to concentrate the silver in the lead produced from the lead ores.

Wet Processes for Treating Argentiferous Copper Ores.—In certain cases copper pyrites is converted into copper sulphate by the action of air and moisture. The sulphate is dissolved in water, and the copper precipitated from the solution by iron. When, however, argentiferous ores are dealt with, containing much copper but not much lead, the processes are more complicated. These processes are illustrated by the following examples :—

1. Ziervogel's process, which consists in submitting the regulus to an oxidising roasting, yielding cupric oxide and silver sulphate.
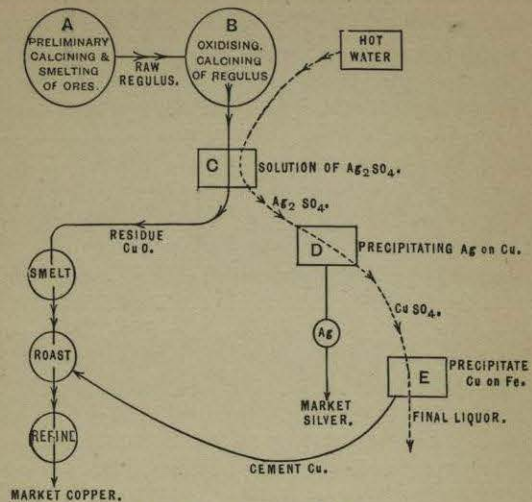

FIG. 184.—Ziervogel Process.

The latter is dissolved in water and precipitated on copper. The cupric oxide is recovered from the residues by smelting (fig. 184).
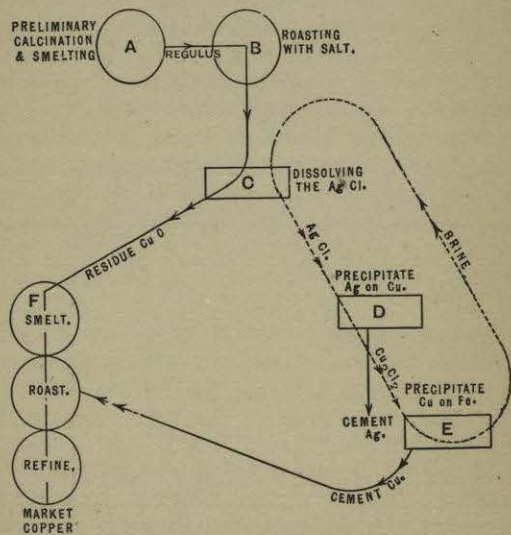

FIG. 185.—Augustin Process.

2. Augustin's process, which has now been superseded by other methods, consisted in submitting the regulus to an oxidising and subsequently to a chloridising roasting, yielding cupric oxide and

silver chloride. The latter was dissolved in hot brine and precipitated on copper. The cupric oxide was recovered from the residues by smelting (fig. 185).

3. Longmaid's or Henderson's process, which is applied to the ore after roasting, and which consists in dissolving the copper and silver from the ore roasted with salt by water and dilute hydrochloric acid. The copper is precipitated by scrap-iron, and the silver is recovered by the Claudet process (fig. 186).

4. In Claudet's process the silver is precipitated as silver iodide from a solution of copper and silver chlorides by zinc iodide regenerated during the process, zinc being used to precipitate the silver.
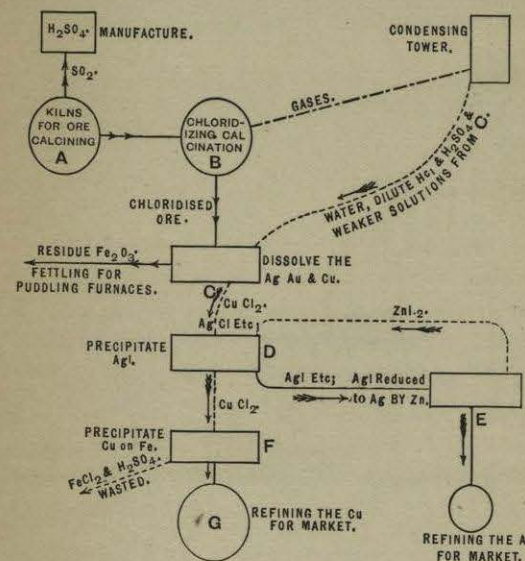

FIG. 186.

These processes are illustrated by the accompanying diagram, Claudet's process forming part of the scheme (fig. 186).

The plant used in the Ziervogel process is illustrated in fig. 187. The charge of roasted material is placed in tubs A, provided with false bottoms, and hot water is introduced through the pipe b until the liquors begin to flow through the tap c. The pipe b is then closed, and acidulated hot water is admitted through the pipe a. The silver sulphate solution flows into a long tank, divided into two compartments B, C, whence it is distributed into precipitating tubs D, provided with false bottoms on which cement copper is placed, with copper bars above it. Most of the silver is precipitated, and the liquors pass to a trough E, on the bottom of which is a layer of pieces of sheet-copper, and thence

to the tubs F, also containing a little copper. The desilverised liquors are conveyed by the gutter g to a leaden pan, and again utilised. The precipitated silver is treated with sulphuric acid in the tubs H, and finally washed with hot water. The liquors rise through L, and are conducted by the trough M over metallic copper into tanks containing scrap-iron. The water from the final washing is run off at N, and conducted to a lead-lined tank. The silver is moulded into blocks and refined.

Treatment of Gold Ores.—Plattner's method of extracting gold from its ores by means of chlorination is based on the fact that chlorine gas transforms gold into soluble gold chloride without sensibly attacking the earths or metallic oxides with which the gold is mixed.
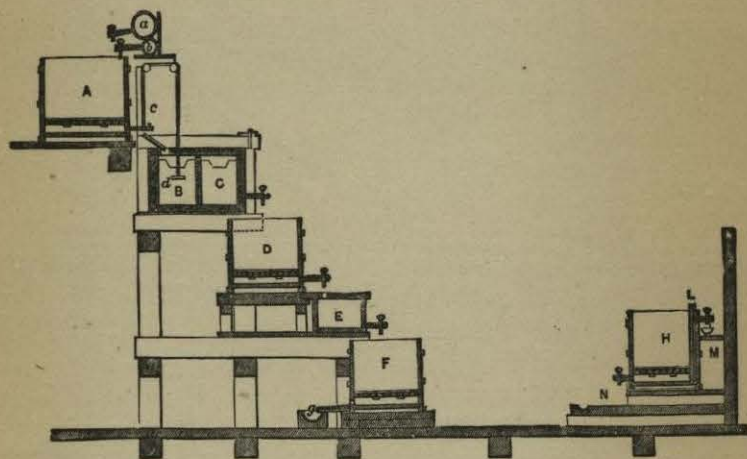

Fig. 187.

The ore is crushed and roasted with salt, 5 to 9 lbs. of salt per ton of ore having been added. Any lead, bismuth, or silver present is thus converted into chloride. The iron sulphide present, however, will first be converted into iron sulphate, and this must be decomposed to form ferric oxide by raising the temperature of the roasting-furnace. The base chlorides are soluble in water, and are removed by treating them in wooden vats. The residue is then acted upon in a moist state in wooden, pitch-lined vats by gaseous chlorine. The gold is thus converted into chloride, but the ferric oxide remains unchanged. The gold chloride is dissolved in hot water, and precipitated by ferrous sulphate solution in another set of wooden vats. The insoluble residues left in the "gassing" vat are now treated with sodium hyposulphite solution in order to remove the silver chloride, which may be precipitated by means of calcium sulphide and reduced by heating in air.

Numerous modifications of the chlorination process[1] have been introduced, and large sums of money have been sunk in the formation of companies to work the processes. In many cases very satisfactory results have been obtained, more especially where the process has been applied to the treatment (after roasting) of the sulphurets constituting the "concentrates" from the Frue Vanners, over which the tailings from the stamp batteries are allowed to pass.

Dr Mears, of Philadelphia, was the first to lay stress on the influence of pressure in increasing the solvent action of chlorine. With certain ores, however, it has been found that as good results are obtained without pressure as with the highest pressure.

The largest chlorination plant in the world in 1890 was at the Mount Morgan mine in Queensland. According to M'Dermott and Duffield,[2] the ore averages 5 oz. of gold per ton, and 1500 tons is worked per week, while the tailings contain only 3 dwt. per ton. The process consists in drying, crushing between rolls, roasting, and working in revolving barrels by the aid of chloride of lime and sulphuric acid. The ore is roasted for two and a half hours, and worked in wooden barrels $5\frac{1}{2}$ feet in length. Each barrel takes a charge of 1 ton of ore, with 30 lbs. of chloride of lime, 33 lbs. of sulphuric acid, and 80 gallons of water. The gas-pressure does not exceed 20 lbs. per square inch at its maximum development. The barrels are slowly revolved for two hours, and the charge is then filtered on gravel and sand beds, and the solution precipitated by filtration through charcoal filters, which are subsequently burnt in a reverberatory furnace. A method of leaching with chlorine water has more recently been introduced at Mount Morgan.

Of late years the MacArthur-Forrest process and its modifications have come into much prominence for the recovery of gold from the "tailings" from amalgamation plant working upon ores of all grades, for the direct treatment, in some cases, of low-grade ore, and also for the treatment of silver ores. It consists in the treatment of the mass of finely divided material with a very dilute solution of potassic cyanide for a period of about twenty-four hours. Cyanide of gold is formed, soluble in the excess of reagent, and the solution is run over metallic zinc upon which the gold is precipitated. By this means very large quantities of gold are now annually recovered that would, owing to the cost of treatment by the older methods, have otherwise been lost.

The development of this process, which has almost completely replaced the chlorination process, has assumed gigantic proportions in South Africa, Australia, New Zealand, and America, where plants have been erected capable of dealing with enormous quantities of material. A full description of the plant and

[1] Rose, *Metallurgy of Gold*.
[2] *Losses in Gold Amalgamation*, London, 1890, p. 30.

methods of working is given by Rose,[1] but a few general remarks may be made. As already stated, the "Cyanide Process" has been applied chiefly to the treatment of "tailings" from stamp batteries. This material, which has generally passed a 25- to 30-mesh screen, is first sorted by Spitzluten and Frue Vanners, the fine sands and slimes being conveyed to separate vats for special treatment. The concentrates from the Vanners, consisting chiefly of pyritic material, are also treated separately.

The main bulk of the tailings is conveyed to vats often 40 feet in diameter and 7 feet deep. These vats are of wood, concrete, or steel, and are provided with false bottoms on which rest filter-beds of cloth and pebbles, suitably supported to bear the weight of the "tailings." Before the leaching operation proper, which is effected with dilute solutions of potassium cyanide usually containing from 0·05 to 0·35 per cent., care is taken to neutralise any acidity in the "tailings" by means of soda or lime, and so prevent undue losses of cyanide. After leaching, which usually occupies from twelve to twenty-four hours, the cyanide solutions containing the gold, dissolved as the double cyanide, KCy,AuCy, are drained off from the vats and conveyed to the precipitation plant. The particular form of this plant varies considerably. The original method consisted in allowing the solution to flow through a series of boxes containing zinc shavings on which the gold was deposited. This method, with certain modifications to admit of its use with very dilute solutions, is still largely employed. The chief rival to this method is the electrical precipitation on lead cathodes by Siemens and Halske, who claim for it many advantages over the original method of precipitation by zinc. Precipitation by zinc dust and charcoal have also been employed.

In modern practice the whole of the material from the stamp batteries is slimed by passing through tube mills, then agitated with the cyanide solution, and passed through filter-presses. Vacuum filter-presses of various makes are being largely used at the present time. In order that ores may be successfully treated by the cyanide process, certain essential conditions must be fulfilled, such as:—

1. Sufficiently fine grinding to allow the solution to come into intimate contact with the gold.

2. The gold itself should be in a fine state of division, for coarse gold dissolves in the cyanide solution extremely slowly.

3. The absence of interfering bodies, some of which may occur owing to the oxidation of pyrites, etc., present in the ore. A preliminary wash with water takes away interfering soluble salts, whilst a preliminary wash with alkali neutralises the effect of acid present.

4. The absence of gold in combination with various elements. When the gold is present in combination with tellurium, bismuth,

[1] Rose, *Metallurgy of Gold.*

etc., simple cyanide treatment is not efficient, and sometimes a preliminary roast is found to improve matters.

5. The character of the gangue is also of importance, as difficulties are met with in treating certain ores, containing kaolin, talc, clay, etc. For kaolin and talc sliming is good, and for clay a preliminary roast is often used.

Purification of Platinum.—The methods of treating the metals of the platinum group present features of much interest; the use of both "dry" and "wet" operations being involved. The crude native metals are first treated in a reverberatory furnace, with an equal weight of galena. Some of the lead will be reduced to the metallic state by the iron present in the charge, and will form an alloy with the platinum, rhodium, palladium, and some of the iridium present. Any osmiridium present will not alloy, but will settle down to the bottom of the fluid mass; whilst the earthy impurities are fluxed off by the addition of glass and borax. Litharge is then added to oxidise the remaining sulphur, the slag is skimmed off, and the metal is run into ingots. The metal is then cupelled, and yields platinum and the other members of its group. To purify the mass, it is melted with six times its weight of pure lead, is granulated, and treated with dilute nitric acid. The insoluble black powder, forming a residue, contains all the platinum and iridium present in the lead, together with small quantities of the other metals, and it is treated with dilute *aqua regia*, which dissolves out the platinum and the residue of the lead, but does not attack the iridium. The solution of chlorides is filtered off, evaporated to a small bulk, and the lead precipitated by sulphuric acid. A mixture of sodium and potassium chlorides is added to the filtrate, which is then heated to 80° C., and left to stand for several days. Most of the rhodium present remains in solution, and the precipitate of platinum-potassium chloride is collected and washed with a solution of ammonium chloride, and then with water. As some rhodium may still be present, the chloride is dried, and mixed with potassium bisulphate and a little ammonium sulphate; the whole is then heated in a platinum vessel until the platinum is completely reduced. The mass is thoroughly washed with water, and the rhodium removed as soluble bisulphate, the pure platinum sponge is finally melted under the oxyhydrogen blowpipe upon a lime crucible.

Wet Process for Treating Nickel and Cobalt Ores.—The wet methods for the extraction of nickel and cobalt from a complex regulus or arsenide consist, in the first place, of a roasting operation having for its object the volatilisation of the sulphur and arsenic, and, it may be, antimony, and the conversion of the iron, nickel, cobalt, and other metals present into oxides. Ferric oxide formed in this manner at a high temperature is but little soluble in acids, whilst the other oxides may be readily dissolved. On treating the roasted material repeatedly with hydrochloric acid or with

dilute sulphuric acid, a residue is obtained containing but little or no cobalt or nickel, and consisting mainly of ferric oxide. Some iron will, however, have passed into solution. Should the solution contain lead, bismuth, or copper, these metals may be precipitated by sulphuretted hydrogen; but it is customary to precipitate the copper at a later stage of the operations. The bismuth, too, may be precipitated from a hydrochloric acid solution by dilution with water.

The next operation consists in the precipitation of the iron. Any ferrous oxide which may have passed into solution is converted into ferric oxide by careful addition of chloride of lime, followed by the addition of lime, which precipitates the iron. Arsenate of iron is, at the same time, also precipitated if arsenic is present. Should the temperature of the solution exceed 40° C. some nickel and cobalt are precipitated, as also is some copper.

Instead of an addition of lime as a precipitant, caustic soda or sodium carbonate is occasionally employed to prevent the precipitation of calcium sulphate when working with sulphuric acid solutions. Care must be taken to avoid using an excess of the precipitant, as the precipitation is a fractional one, and, as soon as the iron has been precipitated, oxides of the other metals present begin to be thrown down.

The next stage of the process consists in the precipitation of the copper. This is effected by raising the temperature of the solution to 70° C., and then precipitating the copper by the careful addition of either calcium carbonate, milk of lime, or a solution of soda. If an excess of the precipitant is employed, nickel will be thrown down. When a test with potassium ferrocyanide shows that the whole of the copper has been thrown down, the cobalt is precipitated from the filtered solution by the careful addition of a solution of chloride of lime to the perfectly neutral, hot, and not too dilute filtrate. If too much chloride of lime is added the precipitate becomes nickeliferous, and this must be carefully avoided. The nickel is next precipitated, either by calcium carbonate, milk of lime, or soda. The nickel hydrate is filtered, dried, heated with sodium carbonate, to decompose any calcium sulphate that may be present, washed with acidulated water, and finally dried and reduced by carbonaceous materials to the metallic state.

This process, being dependent on the fractional precipitation with the same precipitants of the several metals present in the ore or metallurgical product under treatment, is frequently subject to slight alterations of procedure, and the following is a description of the process as carried out at a works in the United Kingdom:—

About 3 cwt. of fine ore or speise A, that has been thoroughly roasted, is charged with hydrochloric acid in granite jars, into which steam is passed. The mass is kept boiling for twelve hours.

WET PROCESS FOR EXTRACTING NICKEL AND COBALT.

A. Calcined ores or speise of nickel and cobalt.

B. Granite jar containing HCl. Steam passed in.

C. Precipitation of iron and arsenic by milk of lime after peroxidising the ferrous oxide by $CaOCl_2$. Steam passed in.

D. Filter.

E. Solution. Pumped up to F.

F. Precipitation of copper sulphide by sulphuretted hydrogen.

G. Filter.

H. Ni and Co solution. Pumped up to J.

J. Solution carefully mixed with metallic iron, chalk, and chloride of lime. Steam passed in. Sulphuretted hydrogen evolved.

K. Filter.

L. Stock tank. Solution pumped up to M.

M. Dividing tank. Heated to 40°. Co precipitated by $CaOCl_2$.

N. Filter.

O. Nickel ppt. by milk of lime.

Residue Co_2O_3 to market.

Ppt. reduced to metallic nickel.