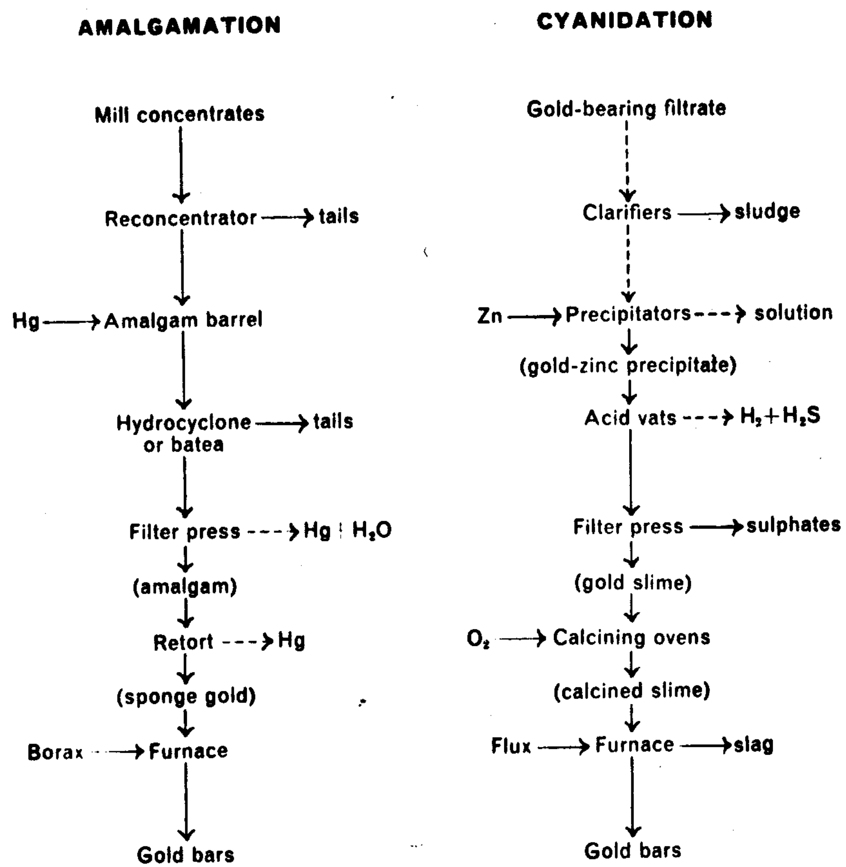

Figure 6. Flow diagram of recovery plant

SMELTING

Development

From the inception of mining in 1886 on the Witwatersrand, after the gold content of the banket reef had been concentrated, either as an amalgam or as a zinc precipitate, the final stage of recovery was to convert the concentrate to gold bullion. The bullion was cast into "bars", i.e. ingots, of a suitable size and shape for handling and transport. In this form it could be sampled and weighed and thereby the fine gold content could be accurately assessed. In line with the world-wide experience that native gold invariably contains some proportion of silver, South African deposits follow suit and in general comprise about 90% gold and 10% silver. Silver amalgamates with mercury as readily as gold,

dissolves concomitantly in cyanide solution and is precipitated as effectively by zinc. A slight absorption in the slag occurs during smelting but in the main the silver content of the ore reports with the gold and so the final metal smelted is a gold bullion containing 10% silver and 2% to 3% of copper, iron and other base metals. The subsequent separation of the gold and silver and the removal of the minor amounts of base metals are conducted in gold refineries. Consequently smelting practices on the gold mines are limited to the production of a good grade of bullion that can be accurately sampled and assayed for its gold and silver content and can be easily transported.

When gold recovery on the Witwatersrand was originally confined to amalgamation, the amalgam was retorted in tubular cast iron retorts set horizontally in coal-fired furnaces. The distilled mercury passed into a water-cooled condenser and was collected in suitable receptacles for re-use on the amalgam plates. The gold remained as "sponge" in containers inside the retort cylinder. After cooling the sponge gold was removed, its mass measured, after which it was melted in plumbago crucibles and finally poured into bar moulds. Retorting occupied from 3 to 4 hours and so simple and efficient was the operation that it has endured for 75 years. The only improvements have been the substitution first of oil and then of electric heating in place of coal or coke firing.

When the zinc shavings method of precipitating gold from cyanide solution was introduced in 1890 the production of gold bullion became more complicated and still more so when the zinc-lead couple was accepted as the most efficient precipitating agent. When the concentrate from the zinc boxes was "cleaned-up" it was reconcentrated by dissolving most of the surplus zinc in sulphuric acid and discharging the zinc sulphate solution through a filter press. The re-concentrated material contained insoluble lead sulphate and other impurities which could only be separated from the bullion by pyrometallurgical means. This was effected in two stages by calcining and fusion. The moist filter cake was distributed in flat trays which were placed in ovens and roasted at a bright heat for several hours. The lead content, residual zinc and any other base metals present were converted to their respective oxides. After cooling the calcined material was mixed with a suitable flux which in its simplest form consisted of borax and sand. The mixture was then placed in plumbago crucibles which were heated in a reverberatory furnace for two to three hours at a temperature of about 1200°C. The base metal oxides, principally PbO and ZnO_2, combined with the molten sand as borosilicates to form a fluid slag. While the fluxes adopted were capable of separating lead and zinc oxides from the gold bullion their effect was largely dissipated by the reducing action of the graphite that was the main constituent of the crucibles used for smelting the fluxed calcine. Once this reaction was appreciated it was countered by the introduction of inert clay liners inside the crucibles. After fusion the molten mass in each crucible was poured into a suitably shaped iron mould in which the liquid gold and silver settled to the bottom while the less dense slag formed a supernatant cover. When the slag and bullion had solidified the moulds were tipped upside down to discharge their contents. The bullion was separated from the slag by tapping with a hammer and then re-melted in suitable amounts to form bars with a mass of approximately 1000 Troy ounces (31 kg).

Prior to the introduction of zinc dust as a precipitant the amount of zinc shavings required in the course of precipitating an ounce of gold was considerable and consequently a relatively expensive quantity of sulphuric acid was consumed in dissolving surplus zinc derived from the "clean-up". Before acid treatment was generally adopted, an alternative method of smelting the zinc lead-gold precipitate was introduced. It was developed to obviate the use of sulphuric acid and embraced the use of two furnaces—one a reverberatory and the other a cupellation type. Known as the Tavener smelting process it virtually duplicated on a production scale the fire assay procedure of separating gold from associated gangue by fusing the materials with powdered litharge and carbon. The litharge was reduced by the carbon to metallic lead which drained through the molten mass in fine droplets. These collected the gold and silver particles and eventually a molten lead bullion containing about 10% gold settled in the bottom of the furnace. This bullion was tapped from the furnace into moulds where it solidified into bars known as "pigs". The pigs were taken to the cupellation furnace where they were heated to a sufficient temperature to melt the lead which was converted by an air blast to litharge. The litharge was removed for further use in the reverberatory furnace and the residual gold and silver was collected for re-melting into bars. The Tavener process had two advantages—no acid treatment

was involved, incidentally removing a gas hazard, and the residual slag was lower in gold content than the slag from crucible smelting. It was, however, much slower in operation and required more labour. Lead smelting therefore was not generally adopted and even where first practised was gradually replaced by crucible smelting. Nevertheless the Tavener process was retained as the most efficient method of treating smelthouse by-products. These comprise mainly slag from the crucible smelting and, in lesser amounts, discarded crucibles, crucible liners, furnace brickwork, floor sweepings, wood ash and any other material in the smelthouse that may have become impregnated with gold. Large mines treated their by-products in a Tavener furnace for many years but ultimately the facilities offered first by By-Products Ltd. and subsequently by the Rand Refinery rendered the localised operation less profitable.

Thus the standard procedure for the production of gold bullion from the cyanide process on the Witwatersrand for the first two decades of the century consisted of treating the zinc box clean-up material with sulphuric acid, calcining the residual precipitate in coal-fired ovens, fluxing the calcine and fusing the mixture in clay-lined plumbago crucibles which were charged by tongs into coal-fired reverberatory furnaces. When fusion was complete the crucibles were removed one by one from the furnace, again by tongs, and the molten contents poured into cast-iron moulds. After cooling the slag was separated from the refined bullion which was then remelted in suitable quantities to produce bars of approximately 1 000 oz Troy mass. The whole clean-up and smelting procedure lasted from four to five days and was usually conducted in the day time, the concentrates being locked in a strong room overnight. At the end of the final day of smelting the gold bars were cleaned, sampled and their mass determined, preparatory for dispatch to the purchasing agency.

When zinc dust came into use from 1918 onwards the clean-up procedure was simplified and shortened as the amount of metallic zinc to be removed was considerably reduced. Nevertheless calcining and smelting continued to be conducted in fire-brick furnaces fired by coal supplied from Transvaal collieries and as this fuel was both inexpensive and of high quality there was little incentive to alter the process. However, when the increase in the price of gold during 1932 was followed by the production of considerably greater quantities of bullion, attention was given to improving the clean-up and smelting methods, particularly by replacing hand-firing of the furnaces by electrical heating. The outbreak of war in 1939 acted as a brake on these developments and a further set-back occurred after the war owing to the shortage of electric power supply. Thus smelthouses erected during the nineteen-fifties in the Orange Free State commenced operating with retorts, calciner ovens and reverberatory furnaces all fired by coal. Eventually power supplies improved and electrically heated furnaces were introduced and generally accepted.

Although fuel oil was not particularly cheap in South Africa, experiments had been conducted with oil-fired furnaces in the Orange Free State during 1961, but the pilot installations did not prove acceptable. Difficulties arose in developing suitable fire clay liners to meet gold smelting requirements in existing types of oil fired tilting furnaces. Unlike the electric arc furnaces which were specially designed for gold smelting, the oil fired type was adapted from brass melting practice and was equipped with an unlined graphite crucible. Before a suitable clay liner could be developed electric arc furnaces had become established.

Once the availability of adequate electric power was assured, the conversion from coal firing to electric heating progressed steadily and electrical retorting, calcining and smelting furnaces were installed in most smelthouses. The only exceptions were on the older mines where their declining life did not warrant the capital expenditure involved in replacing the existing coal-fired furnaces. Well designed electrically heated and controlled units of the Barnes or Keegor types resulted in a streamlining of the smelthouse procedure. Both for retorting and calcining, temperatures could be set to very close limits, power could be switched on and off by automatic controls and labour requirements were considerably reduced. Operating conditions improved with the elimination of smoke and dust and except in rare cases of power failure the period required for a smelting campaign was markedly reduced.

In the case of smelting calcined slime, electrically heated reverberatory furnaces gave satisfactory performance but the large gold output of newer mines involved both additional time and labour for smelting and re-melting into bars. Therefore the high capacity three electrode submerged arc tilting furnaces proved more acceptable. Their large volume crucibles can accept sufficient fluxed calcine

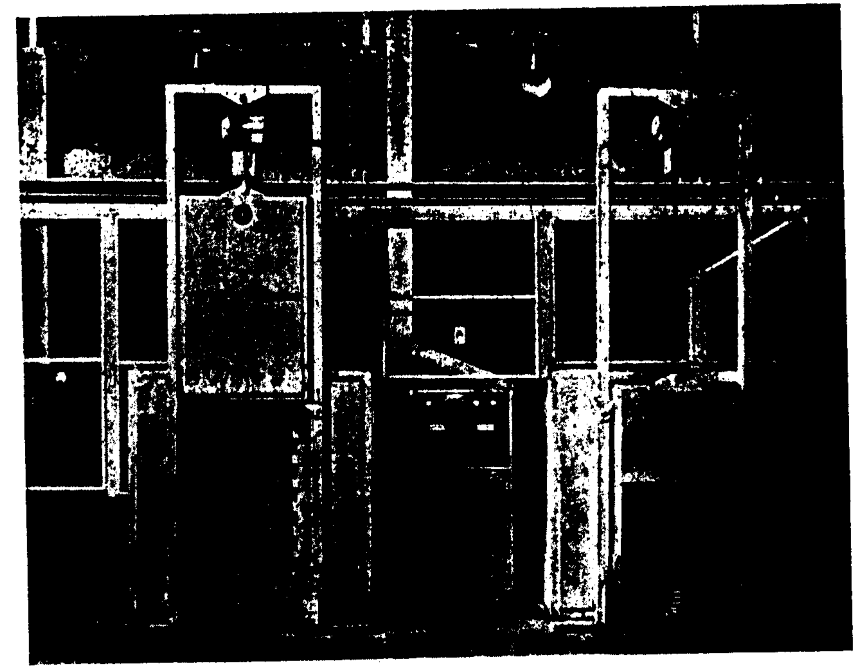

Figure 7. Electrically heated six-tray calcining ovens

to produce fifteen 31 kg bars from each fusion. They also have more than adequate capacity for melting retorted sponge gold into bars, since up to 600 kg of bullion can be melted in a single fusion.

The regulation of the smelting operations by electrical means combined with the automatic control provided by candle precipitation filters, presents obvious possibilities of introducing a continuous operation of cleaning-up the precipitation units and immediately smelting the precipitate. In the most recent practice the Stellar precipitate is dewatered on a small rotary filter and then transferred to pallets which pass slowly through a furnace with overhead heating elements. The calcined slime is collected in tared hoppers, the total mass determined and then the slime mechanically emptied into an arc furnace followed by a suitable flux addition.

Until 1922 the gold bars were purchased by one or other of two commercial banks and dispatched overseas (usually London) for final refining and sale on the world goldmarkets. However, when the Rand Refinery was established in 1922 all producing mines in South Africa which were members of the Chamber of Mines contracted to forward their gold output to the Rand Refinery where the gold bullion was not only refined to the market requirements of 99,5% fine gold but was cast into "good delivery" bars of 400 ounces Troy mass and sold by the Refinery on behalf of the mines. Similarly the silver separated from the bullion was refined to 99,8%, cast into 1 100 ounce Troy ingots and sold either to the South African Mint or to buyers overseas.

Clean-up and Smelting Procedures

The methods employed on South African gold mines naturally have minor variations but in general the following description is typical of modern practice.

The clean-up operation is carried out at regular intervals depending on local conditions or requirements and varies from 3 to 14 days. The gold slime from the Stellar or Merrill precipitation plant is pumped into wooden acid treatment vats fitted with a stirring mechanism and a gas removal system. Sulphuric acid is added in order to dissolve as much zinc and other superfluous constituents as possible. A considerable volume of inflammable gas is generated—a mixture of hydrogen and hydrogen sulphide—and therefore a hood and exhaust fan over the acid vat are essential to extract and disperse this effluent to atmosphere. To ensure maximum digestion of the gold slime it is usual to have about 1% of free acid left in the solution prior to filtration. On completion of the acid treatment the gold slime is pumped into a 0,76 m square Johnson filterpress containing 30 frames. Filter paper is used over the cloths of the press to ensure a clean separation of gold slime and thus obviate scraping precipitate from the canvas. After the contents of the acid vats have been pumped into the filter press and thoroughly washed, compressed air is blown through the frames to remove as much moisture as possible.

Where Stellar filters are used for precipitation, backwashing takes place either once or twice per week and the gold slime is discharged into a storage tank situated in the smelthouse. The gold slime is pumped from the storage tanks to an acid vat for sulphuric acid treatment and then into the Johnson filter press. Alternatively the gold slime may be pumped directly to a filter for dewatering without prior acid treatment. After filtering and air drying, the gold slime containing about 40% moisture is removed manually from the filter press and placed into flat trays preparatory to calcining. The quantity of gold slime handled during a

clean-up depends upon several factors such as time cycle, grade of pulp and tonnage treated, as illustrated by the following examples.

	Tons solution treated	Head value g/t	Clean-up period days	Gold slime recovered
Mine A.	88 300	3,34	10	2 720 kg*
Mine B.	44 300	6,04	7	2 000 kg
Mine C.	83 900	8,28	7	3 550 kg

* Gold slime not subjected to acid treatment.

Calcining the Merrill or Johnson press slime on stainless steel or iron plate trays takes place in electric calcining furnaces which accommodate either six, nine or twelve trays of gold slime per charge. Each tray, containing approximately 60 kg of moist gold slime is retained in the calcining furnace for a period of 16 hours, the temperature of the calciner being maintained between 600° and 700°C.

After calcining, the gold slime is mixed with a flux. The composition of the flux depends upon the nature of the calcined gold slime but mainly comprises borax and silica. In the following four examples the proportions are expressed as percentages of the calcined gold slime.

	Acid Treated			Not Acid Treated
Borax	15	13	13	35
Silica (sand)	16	10	12	15
Nitre	—	—	—	3
Soda	—	1	3	5
Fluorspar	—	7	7	—
Manganese dioxide	1	2	2	—

As already stated, the smelting of calcined gold slime with its accompanying flux is performed in electric submerged arc furnaces. The three carbon electrodes are lowered and raised mechanically but under electrical control. An hydraulic system, again electrically controlled, allows the furnace to be tilted forwards when the electrodes have been withdrawn from the interior of the furnace. At the commencement of the operation the furnace is charged with 90 to 100 kg of calcine and periodic additions are made until the furnace is completely charged with approximately 750 kg. Flux can be introduced as a mixture with the calcine or can be added separately. The total period of

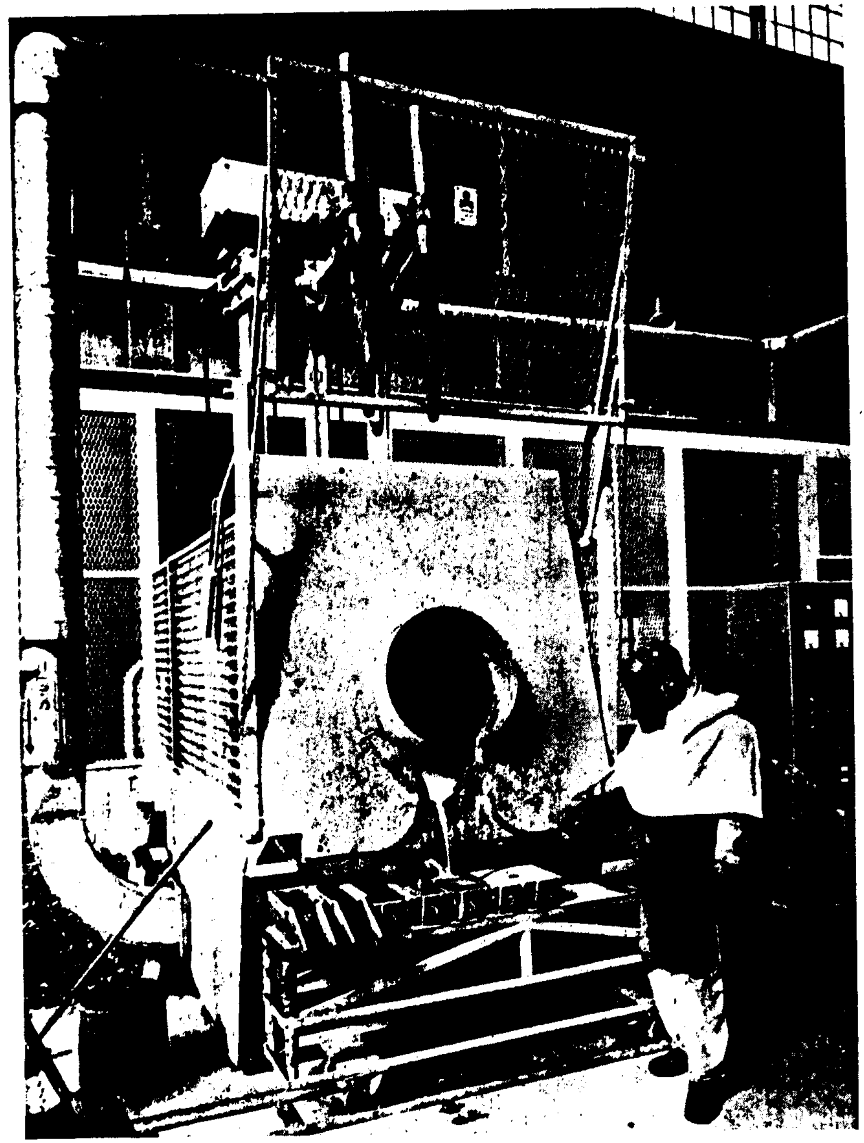

Figure 8. Pouring gold bars from tilting furnace after the three electrodes have been raised out of the charge

a smelting cycle occupies about three hours, comprising one hour for charging the furnace, one and a half hours for fusion and half an hour for slag removal and pouring. Smelting temperatures range from 1 200° to 1 400°C. When fusion is complete the three carbon electrodes are withdrawn and the furnace tilted from the vertical towards the horizontal position to pour the molten contents through the discharge spout.

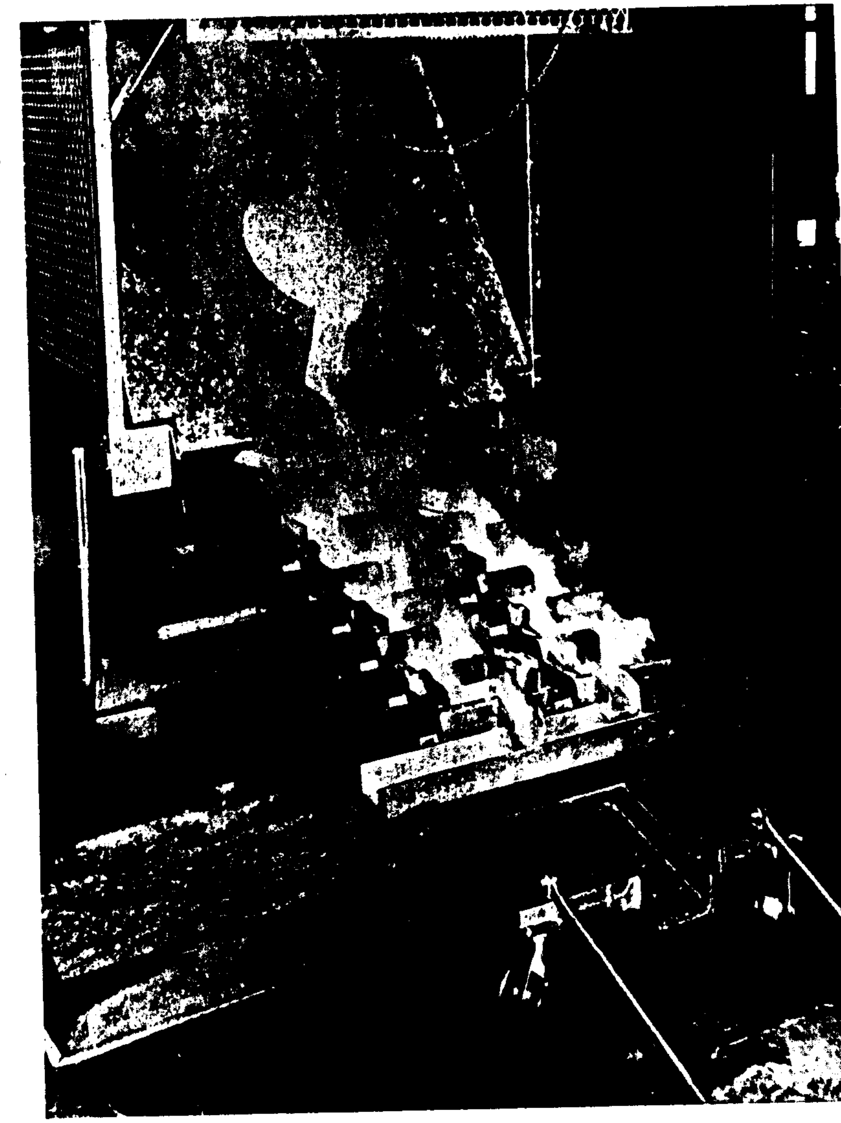

Figure 9. Cascade pouring of gold bars

The gold bullion is poured into standard moulds which are designed to contain bars of 31 kg mass, a figure based on the previously demanded quantity of 1 000 oz Troy. The metal may be poured into single moulds or cascade pouring can be practised. Where single moulds are employed, they are placed on a trolley and moved in turn under the spout of the tilted furnace and filled to the correct level. In cascade pouring the molten contents of the furnace are poured into a receiving mould with an overflow lip from which the gold flows progressively down a series of moulds each equipped with a similar overflow lip. A double series of moulds is usually employed in order to reduce the length of the cascade and so obviate solidification of some of the gold in the crucible before the last mould of the cascade is reached. By this means a total of 14 bars can be poured from a single furnace charge. In both methods, once the gold has solidified the bars are quenched in water and cleaned by hand scrubbing.

With large scale fusion in the arc furnace it is usual to sample only two bars per pour by hand-drilling in contrast to sampling every bar, as was the case in earlier practice when separate crucibles were employed for each individual bar. The arc furnace is normally operated with electrodes set at 600 amperes and the life of the carbon electrodes ranges from 6 to 10 fusions. The average life of the furnace liner is 35 fusions but this figure can be extended if repairs are made between fusions. A ventilation hood is installed above the furnace and all fumes and dust emanating from the furnace are withdrawn through a fan and passed into a filter. All fine particles trapped in the filter are collected and returned to the ensuing furnace charge. Thus any gold escaping as fume from the furnace is duly accounted for.

Retorting

As stated earlier, electrically heated retort furnaces have displaced the coal-fired types but otherwise the time-honoured method is still practised. The retort consists of a cast iron cylinder 0,3 m diameter × 1,22 m long × 50 mm thick and is set horizontally in a fire brick container. It is charged with pressed amalgam in semi-cylindrical moulds known as "boats" and then closed with a circular cast iron door. Heating commences slowly but steadily and the mercury is distilled as a vapour which liquifies on passing into the condenser. The condensed mercury is collected in buckets and is available for further amalgamation. The temperature inside the retort is maintained at approximately 600°C, considerably in excess of the boiling point of mercury which is 357°C. The completion of retorting is indicated by the cooling of the tube entering the condenser, usually after a period of 3 hours. At this stage heating is stopped and the retort allowed to cool. It is then opened and the boats containing the sponge gold are removed by tongs. Mercury losses during retorting normally should not exceed 0,02 gram per ton milled.

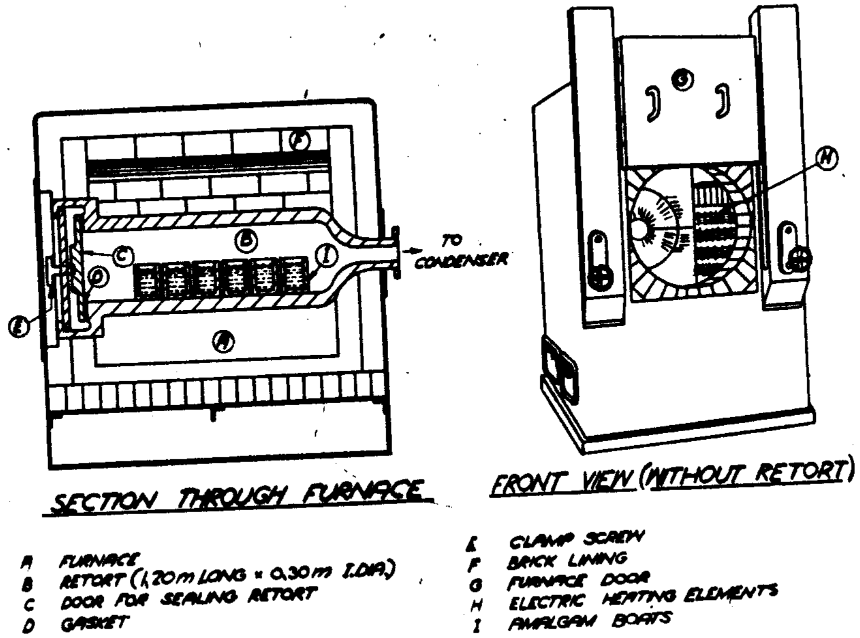

Figure 10. Amalgam retort

The melting of suitable amounts of sponge gold to form 31 kg (1 000 Troy) bars used to be and to some extent is still performed in graphite crucibles which can be heated singly in a Keegor single-crucible resistance furnace which represents a modernisation of the Cornish fire. Otherwise batches of sponge gold are melted as a single charge in an electric arc furnace and cast into 31 kg size moulds in a period of 40 minutes.

By-products

The slag resulting from the smelt is disposed of either by returning it to the mill circuit or forwarding it to the Rand Refinery for gold recovery. The furnace liners when rejected are similarly treated. It is a regular practice that all discarded equipment and material used in the smelthouse be subjected to a treatment process for gold recovery. The wooden structures, clean-up overalls and other combustible material are burnt, the ashes being dispatched to the Smelting Section of the Rand Refinery. Worn-out smelting tools, trays, linings and castings made of metal are soaked in acid solutions and the resultant sludge treated for gold recovery, generally at the Refinery. Fire bricks and furnace liners, when replaced, are crushed, passed over a shaking table to recover gold beads, and then also sent to the Refinery. Floor sweepings, dust from the smelthouse walls and the deposit in the ventilation filters are handled in the same way. In general, all material leaving the smelthouse is either returned to the milling circuit or conveyed to the Rand Refinery for gold recovery and final disposal.

Roasting

Flotation concentrates from the gold mines in the Barberton and Pilgrims Rest areas have to be subjected to a roasting operation to render the gold content amenable to extraction by cyanidation. This is achieved by converting the auriferous sulphides to porous calcines which permit the release of occluded gold particles. The main constituent of the sulphidic concentrate is iron pyrites, but copper, arsenical and antimonial sulphides are also present in varying degrees. Care has to be exercised in the roasting procedure to ensure that maximum extraction of gold is achieved. When the sulphur content of the float material is sufficiently high the roasting reaction is exothermic and therefore control of the reaction temperature has to be maintained to ensure that a suitable calcine is produced. This can be effected by the volume of air admitted. If however, the sulphur content should be insufficient for the material to be self-roasting, a supplementary fuel addition has to be made in the form of powdered coal. Equally important to temperature control is the provision of an adequate supply of oxygen to ensure primarily that all sulphur is oxidised, secondly that copper minerals are converted to water soluble sulphates and thirdly that iron pyrites is converted to hematite (Fe_2O_3) and not to magnetite (Fe_3O_4).

Reactions with pyrite:

if sufficient oxygen is available,

$$4\,FeS_2 + 11\,O_2 \rightarrow 2Fe_2O_3 + 8\,SO_2;$$

with deficiency of oxygen,

$$3\,FeS_2 + 8\,O_2 \rightarrow Fe_3O_4 + 6\,SO_2$$

Hematite (red oxide) is more amenable to cyanide treatment as it constitutes a finer grained product in which gold particles are more freely exposed than in the case of magnetite (black oxide). In order to obtain maximum gold extraction through the production of a porous calcine, roasting should be conducted slowly and at controlled

temperatures. Above 600°C, iron oxides recrystallise, change physically and lose porosity. However, to avoid the formation of basic ferric sulphate and to eliminate arsenic oxide the final temperature should be raised to 700°C. Ferric sulphate is a strong cyanicide which is not easily eliminated by simple washing procedures.

Roasting is conducted either in simplex or duplex Edwards roasters or alternatively in fluidised bed roasters.

Edwards roasters are controlled by the feed rate, amount of fuel addition (if needed), and the volume of air flow. The roaster output is quenched, milled and subjected to gravity concentration before cyanidation.

In the case of fluidised bed roasters the air is supplied by a blower carefully regulated to ensure an excess of oxygen. Temperature control can be augmented by means of the moisture content of the flotation slurry fed to the roaster. After roasting, the calcine and sulphur dioxide gas pass in a dry state through a cyclone where the bulk of the calcine is discharged through the cyclone spigot into a quench tank. The cyclone overflow is treated in a wet scrubber to collect fine particles of calcine while the sulphur dioxide and other gasses pass on to a sulphuric acid production plant. The total calcine is collected in a thickener preparatory to cyanidation.

Conclusion

It is of interest to note that it is estimated that by the end of 1972 the fine gold smelted in South Africa since 1886 will reach a total of one thousand million Troy ounces (31 105 000 kg) obtained from approximately 3 150 000 000 metric tons milled.

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