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THE PYROMETALLURGICAL RECOVERY OF GOLD FROM  
LEACHED CALCINE RESIDUES

by

L.B. McRae  
and  
A.S.E. Kleyenstüber

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# THE PYROMETALLURGICAL RECOVERY OF GOLD FROM LEACHED CALCINE RESIDUES

by L.B. McRae\* and A.S.E. Kleyenstüber\*\*

## Synopsis

Attempts were made to recover the gold contained in leached calcine residues that originated in the Barberton area, and that contained between 10 and 20 g/t of gold. The process involved the smelting of the residues in a transferred plasma-arc furnace, and the recovery of the liberated gold in a suitable collector held in the hearth of the furnace.

Iron was used as a collector in the first instance and, while discard slags with a gold content of 1 g/t could be produced, the gold recovery in the collector was poor, being about 20 per cent and less. The overall accountability of the gold was poor, being about 30 per cent. The low recovery and accountability were ascribed to volatilization losses of the gold due to the high temperature that is needed to keep the iron collector in a molten state.

Gold was successfully collected in matte and speiss produced by the smelting of a portion of gold-bearing flotation concentrate with the leached calcine residues. In one campaign, where the collector consisted mainly of speiss, 90 per cent of the gold was recovered in the collector. Some 4,7 per cent was recovered from matte and speiss that had soaked into the lining of the furnace.

\*Chief Engineer, Pyrometallurgy Division  
\*\*Principal Scientist, Mineralogy Division  
Council for Mineral Technology (Mintek),  
Private Bag X3015, Randburg 2125, Transvaal.

## Samevatting

Daar is pogings aangewend om die goud in geloogde kalsienresidu's, afkomstig uit die Barberton gebied, wat tussen 10 en 20g/t goud bevat, te herwin. Die proses het die uitsmelt van die residu's in 'n oordraplasmaboogoond en die herwinning van die bevryde goud in 'n geskikte versamelaar in die herd van die oond behels.

Yster is aanvanklik as 'n versamelaar gebruik en hoewel daar afvalslak met 'n goudinhoud van 1 g/t verkry kon word, was die goudherwinning in die versamelaar swak, naamlik ongeveer 20 persent en minder. Die algehele rekenskap van die goud was swak, te wete ongeveer 30 persent. Die lae herwinning en rekenskap is toegeskryf aan vervlugtingsverliese van die goud vanweë die hoë temperatuur wat nodig is om die ysterversamelaar in 'n gesmelte toestand to hou.

Goud is suksesvol versamel in mat en speiss wat verkry is deur die uitsmelting van 'n gedeelte van die goudhoudende flottasiekonsentraat saam met die geloogde kalsienresidu's. In een kampanje waar die versamelaar hoofsaaklik uit speiss bestaan het, is 90 persent van die goud in die versamelaar herwin. Ongeveer 4,7 persent is herwin uit die mat en speiss wat in die oond se voering ingetrek het.

## Introduction

Calcined cyanide-leach residues from gold mines in the Barberton area still contain appreciable amounts of gold (10 to 20 g/t). The re-leaching of this material in cyanide, even after pre-treatment with sodium hydroxide, nitric acid, sulphuric acid, or hydrofluoric acid has been unsuccessful and only 20 to 26 per cent of the gold that remained in the residues could be recovered. However, one company found that by digesting the leach residues in hot 40 per cent sulphuric acid, the major

portion of the gold could be recovered. Unfortunately, equipment for use in this operation was not available in the Barberton area, and the residue had to be sent to another mine for treatment. High transport costs eventually led to the termination of this practice. It is only recently that re-leaching after ultra-fine milling has shown some promise<sup>1</sup>.

The idea of a possible pyrometallurgical solution to the problem was sparked off by the success of some small-scale experiments conducted at Mintek<sup>2</sup>. In these tests, the residual gold was collected in iron that was produced in situ by the carbothermic reduction of the iron oxide in the residue itself. The slags produced from these residues contained less than 1 g/t of gold. Sufficient carbon (about 4 per cent of the mass of the residue) was added so that iron, with a mass equal to about 10 per cent of the mass of the residue, was produced.

#### Practical Use of Iron

If sufficient carbon is added to a leach residue that contains 15 g/t of gold so that iron with a mass of 10 per cent of the residue will be produced, the final gold content of the iron will be only 150 g/t at best. This low gold content means that the subsequent recovery of the gold by a leaching process would not be very practical.

A commercial practice for the recovery of the platinum-group metals (PGMs) from autocatalysts into a metallic collector (iron, nickel, copper, etc.) involves the retention of a molten pool of collector in the bottom of the furnace. Residue and fluxes are charged to the furnace, and are melted. Slag is usually tapped continuously, but the metal collector that contains the precious metals is removed from the furnace only once

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its precious-metal content has risen to the desired level.

At the plant of Texasgulf Minerals and Metals Inc., at Anniston, USA, PGMs are recovered from scrapped auto-exhaust catalysts, which contain 500 to 1100 g/t of platinum, and palladium. The PGM content of the iron, which is used as a collector, is allowed to rise to 10 to 15 per cent before it is tapped<sup>3</sup>. The molten alloy is granulated, and is then leached in a mixture of sulphuric and hydrochloric acids. Another charge of metallic iron is made to the furnace at the beginning of the next smelting cycle.

### Electric Furnaces

Slag-resistance furnaces can be used for the treatment of residues that do not contain any reducible metal oxides. However, if reducible oxides are present, a form of electric heating in which there is no contact between the charge and the graphite electrodes must be used. If the electrodes are allowed to make contact with the molten charge, the metal oxide will be reduced to the metallic form by the carbon (graphite) of the electrodes, and this metal will dilute the precious metal contained in the collector. The excessive consumption of electrodes would be a further factor to consider. Furthermore, slag-resistance furnaces are also very sensitive to the electrical characteristics of the slag.

The use of an open-arc furnace may seem to offer a solution. However, the energy radiated by the arcs, which flare towards the sidewalls of the furnace because of electrical interaction, may cause excessive refractory consumption in that area. Transferred plasma-arc d.c. furnaces, by the nature of their operation, have been preferred. Various patents held by Johnson Matthey & Com. Ltd. and Engelhard Minerals &

Chemical Corporation describe their use<sup>4,5</sup>. The d.c. plasma-arc furnace offers a number of advantages over the three-phase open-arc furnace. Because of the presence of a single electrode, no skewed arc flare occurs and, because the electrode is in the centre of the furnace, radiation towards the side walls is reduced. Transferred plasma-arc furnaces are electrically stable, and this results in a level of electrical and acoustic noise that is far lower than in a.c. furnaces. Their design is also simple, and their electrical characteristics are easy to control.

#### Distribution Coefficient of Gold in Iron

It was felt that, in a gold-collection operation, a residual gold content of 1 g/t in the slag should be aimed for. Hence, the gold content of the iron used as a collector would need to be limited to a level that can be determined from the distribution of gold between the slag and the iron.

A series of three crucible tests were carried out. In these tests, iron, in an amount approximately equal to 20 per cent of the mass of the leached calcine residue, was reduced in situ from the iron oxide contained in the residue. Sufficient gold was added to the charge for each test so that the collector produced would contain approximately 2,5, 5, and 10 per cent gold respectively.

The slags were analysed for gold, the buttons were dissolved in acid, and the gold content of the residue was determined, after which the gold content of the buttons was calculated.

The Nernst distribution coefficient,  $\underline{D}$ , is defined as

$$\underline{D} = \frac{\text{Concentration of gold in iron, \% (by mass)}}{\text{Concentration of gold in slag, \% (by mass)}}$$

$\underline{D}$  was calculated from the experimental data as being about 40 000 for lower levels of gold in the collector,

viz up to 5 per cent, but was 28 000 at a gold content of 10 per cent.

Fig. 1 illustrates the relationship between the gold contained in the collector and the gold contained in the slag. It can be seen that, if a discard slag that contains only 1 g/t of gold is desired, the gold content of the collector will be limited to about 5 per cent.

[Insert Fig. 1]

### Preliminary Iron-collection Tests

Samples were obtained of residues from two mines in the Barberton area. The range of the compositions of these residues is shown in Table I.

[Insert Table I]

The initial experiments were run on a batch basis with single charges being made to the furnace. The reaction was carried out by use of a 100 kVA d.c. transferred plasma-arc furnace, which was operated at about 25 kW. A schematic representation of the furnace that was used, which had a crucible diameter of 180 mm, is shown in Fig. 2. The objective of these preliminary tests was to compare the effectiveness of iron generated in situ as a collector for gold, with that of iron added in metallic form.

[Insert Fig. 2]

In these tests, residue was fed to the furnace in a pelletized form at a controlled rate over selected periods of time. The energy required for smelting was calculated as being 0,7 kWh/per kilogram of residue. When char was added, it was mixed with the residue being charged to the furnace. When iron was added, it was introduced into the furnace in the form of steel washers at the beginning of each smelt. At the end of each

experiment the furnace was allowed to cool, and the slag and the button of collector was recovered.

A summary of the operation, as well as the gold content of the slags produced, is shown in Table II.

[Insert Table II]

On the basis of the gold content of the slag, it was concluded that iron generated in situ was more effective as a collector than iron added in the metallic form. The addition of a greater proportion of collector resulted in a lower gold content in the slag. The feed rate (which can be considered in this case to be almost equivalent to the retention time) did not clearly influence the amount of gold that remained in the slag.

#### Further Iron-collection Tests

In this series of tests, attempts were made to simulate the procedure that would be used on a full-scale plant. The furnace that was used is a modification of that shown in Fig. 2, in that it was fitted with trunnions, and was mounted in a stand so that it could be tipped for tapping. First, about 2 kg of iron was smelted with some residue from the first batch. When a molten pool had been established, the feeding of the first batch, which had a mass of 10 kg, was started. Once this had all been fed to the furnace and was molten, the melt was held in the furnace for five minutes so that any dense particles could settle. The power to the furnace was reduced during this time to a level that was just sufficient to make up for heat losses. About 5 kg of slag was then tapped, and a further 5 kg of residue was melted down. The process was then repeated. After the last charge had melted, as much as possible of the molten material that was contained in the furnace was tapped.



For one of two possible reasons, metal was not tapped in any of the tests: either the metal could have frozen between the time that the power was turned off, and the furnace was tilted, or the metal could have collected in the deep circumferential trough cut into the furnace lining, owing to the aggressive nature of the slag towards the refractory. The angle through which the furnace could be tilted was limited, and this angle could have been insufficient to allow the metal that was trapped in the trough to be poured out. The gold content of the slags produced is shown in Table III.

[Insert Table III]

On the basis of the gold content of the slag, it appeared that the process was effective in removing the major portion of the gold from the residues. However, up to that stage, the gold content of the buttons of collector had not been determined. During campaigns 5 and 6, a sample of fume was collected which was found on analysis to contain 52,6 g/t of gold. This, of course, gave rise to doubts as to whether the collection was as good as had been deduced from the slag analysis alone. The buttons of collector had a mass of about 2 kg, and were shaped roughly like mushrooms. The stem-like appendage to the 'mushroom' was the result of molten iron dissolving part of the graphite anode, and filling the space that was originally occupied by it. Because there was no certainty that the buttons had been completely molten during the whole of the collecting operation and, in consequence, were not homogeneous, they were not sampled and analysed for gold.

#### Large-scale Extended Test

The fact that metal was not tapped during any of the previous series of tests gave rise to fears that there might be a problem in the maintenance of a proper heat balance at that scale of operation. It was also felt

that the collection of gold might be improved with a higher mass ratio of collector to residue per batch. It was therefore decided to conduct a smelting campaign using a larger furnace and operating at a higher power.

The furnace used had a hearth diameter of 300 mm, and could be operated at up to 70 kW. A baffled drop-out box was interposed in the off-gas extraction line between the furnace and a small bag filter. This system permitted the collection of the major portion of the dust that was coming from the furnace.

Batches of 15 kg of residue were used for this campaign, and 15 heats were done. The depths of the slags, and the molten heel were monitored continuously, and the power to the furnace was controlled in such a way that at least some molten collector would be present. Approximately 15 kg of slag was tapped each time. The amount to be tapped was determined before each tap from assessments of the depth of molten slag in the furnace.

At the end of the campaign after the last slag tap, 5 kg of silicon in the form of minus 6 mm fines was added to the collector. This was done so that the silicon would form an alloy with the iron collector, and in so doing would allow brittle phases to form on solidification. This would make the collecting metal amenable to crushing and sampling.

Once the silicon had been charged, and was considered to have melted and alloyed with the iron, the furnace was tapped. The attempt at forming an iron-silicon alloy was not completely successful. Some metal reported to the tapping tray, and the rest was recovered from the furnace when it had cooled. The tapped metal was too hard to be crushed. However, it was presumed that, because it had been molten, it was homogeneous, and it was sampled by drilling. The metal that was recovered from the hearth was found to have a very large

grain size, and could be broken with a hammer. This material was then remelted in an induction furnace, so that it would homogenize, and was cast into two ingots. These ingots were then also sampled by drilling.

Samples of all the products of the smelting tests, including the fume, were submitted for the determination of gold.

The average gold content of the slags was 2,2 g/t. It was found that the gold content of the slag was dependent on the feed rate, which ranged between 5 and 25 kg of residue per hour. At the lower feed rate, the average gold content of the slag was 1,2 g/t. This rose to 3,5 g/t at the higher feed rate. The gold balance was calculated and is shown in Table IV.

[Insert Table IV]

The fume that was collected has a mass equal to 0,4 per cent of the residue that was smelted, and was analysed for both arsenic and gold. It was found to contain 4,5 per cent arsenic and 95 g/t of gold. A further sample was submitted for X-ray-diffraction analysis. It was determined that the fume consisted mainly of magnetite (estimated at 90 per cent) with only a trace of the phase  $As_2S_2$  (i.e. less than 5 per cent).

The accountability for arsenic over the campaign was 20,4 per cent.

#### Recovery of Gold in Iron

Because of the poor recovery and accountability of gold obtained during the extended campaign, it was decided to determine the actual recovery obtained during some of the preliminary tests. The buttons-of collector that had been obtained were dissolved in hot 6M hydrochloric acid, and the remaining residues were analysed for gold. The gold balances for campaigns 5,

6, and 7 are shown in Tables V, VI, and VII respectively.

It can be seen from the results that although the gold content of the slags produced during the preliminary campaigns had been satisfactorily reduced the recovery and accountability was poor.

[Insert Table V]

[Insert Table VI]

[Insert Table VII]

A further campaign (campaign 9) was carried out to shed further light on the poor accountability of gold. During this campaign an attempt to assess the amount of gold lost as a result of evaporation was made by melting the residue down without the presence of any collector. A steel anode was used so that no iron would form by in situ reduction. The operation was conducted in a similar manner to that for the other campaigns, except that the feed rate was increased in an attempt to limit the temperature of the bath to a level that was just sufficient to maintain a fluid melt.

The gold balance for this campaign is shown in Table VIII, and it can be seen that there is an improved accountability for gold. This is ascribed to the lower temperature used during this melting operation. During the other campaigns, the temperature was kept high (i.e. in excess of 1550°C) to ensure that as much of the collector as possible remained molten.

[Insert Table VIII]

#### Loss of Gold by Volatilization

The indications were that gold was lost by volatilization. The dust samples that were collected had a high gold content. The gold content of a

composite sample of dust from campaigns 5 and 6 was 52,6 g/t, and, from campaign 8, in which residues containing 15,3 g/t and 12,4 g/t of gold were smelted, the gold content was 93,0 g/t.

Only 0,98 kg of visible fume was collected during campaign 8. At a gold content of 93 g/t, some 19,1 kg of fume should have been collected so that the mass balance could be closed. This indicates that the gold leaving the furnace was so fine that it was not collected in the bag plant.

Pure gold melts at 1063°C, boils at 2808°C, and has a vapour pressure of more than 0,92 kPa at temperatures over 2000°C. During the operation of the plasma furnace, nitrogen is passed down the cathode at a rate of 6 l/m. The atmosphere in the furnace is therefore constantly being replaced - a factor that encourages the evaporation of any volatile species. Table IX shows the amount of gold that could be swept away by a nitrogen stream of 6 l/m if pure gold were heated to selected temperatures under ideal conditions and at an atmospheric pressure of 82,9 kPa (623 mm Hg).

[Insert Table IX]

There is evidence that the gold is present in the residue as discrete particles locked in the iron oxide product from the roasting of arsenopyrite<sup>6</sup>. As such, its activity would be unity, and the evaporation would be at its maximum rate at any particular temperature if ideal conditions apply. When iron is used as a collector, the melt must be maintained at a temperature in excess of 1550°C, so that the pool of collector can be kept molten. Gold that is still present as discrete particles in the melt can evaporate. This rate of evaporation will be greatly accelerated if these particles move into the arc-impingement area, where the temperature is probably far in excess of the mean temperature of the bath.

## Collection of Gold in Sulphide

It seemed essential that the smelting operation should be carried out at as low a temperature as possible, so that the evaporation of gold could be kept to a minimum. This would necessitate the use of a collector with as low a melting point as possible. The use of iron sulphide was considered, because the gold-bearing concentrates produced on the mines would be a readily available source of sulphide.

Collection in sulphide had the added advantage that the sulphide would be amenable to crushing prior to further hydrometallurgical treatment for the recovery of gold. A metallic iron collector would probably have to be granulated before further treatment.

Some preliminary tests were carried out to determine a suitable recipe. These tests showed that carbon in a sufficient quantity must be added to reduce the hematite, contained in the residue, to wustite. This decreased the viscosity of the slag, and minimized the oxidation of the matte.

The campaigns in which sulphide was used as a collector were run in a similar manner to that expected on a full-scale plant; the collector was kept in the furnace while additional amounts of residue were added and allowed to melt down, and the resultant slag was then tapped. The collector was tapped only after 5 or 6 heats had been completed.

### Sulphide Collection Test

In this test, carbon, which represents 4,8 per cent (by mass) of the residue, was added to each charge. Batches of 5 kg of residue were treated in each heat. An initial amount of 6 kg of sulphide concentrate was added with the first charge, and a further 6 kg of concentrate was added with the residue charged for heat

7. The furnace had been drained after heat 6 had been completed and the process had recommenced.

When the furnace was tapped after heat 6, a quantity of a metallic-looking phase, which was identified as a speiss, (nickel, iron-arsenide) was found in the mould into which the molten material had been cast. When the furnace was tapped after heat 11, matte was found in the mould, and speiss was recovered from the hearth once the furnace had cooled.

The gold balance for this campaign is shown in Table X. It can be seen that, in contrast to the campaigns in which iron was used as a collector, the accountability of gold was very high. The main difference between this campaign and the earlier campaigns that used iron as a collector, was the lower tapping temperatures (and, by inference, the mean temperature of the bath), which were on average 1450°C; the average tapping temperature for campaign 8, for example, was 1630°C.

It can be seen from Table X that the gold content of the discard slags was low (except for Tap no. 6, when the speiss and the slag were tapped together).

[Insert Table X]

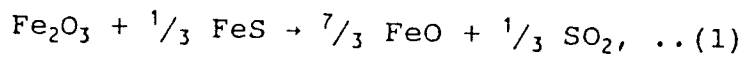
Of the gold that was contained in the products of the smelting campaign, 86,29 per cent reported to the collector. The potential recovery, i.e. the sum of the amount of gold in the collector and the gold contained in the collector-soaked refractory in the furnace hearth, was 94,7 per cent.

It can also be seen that the speiss was the major collecting phase, and accounted for 98,7 per cent of the total gold that was collected.

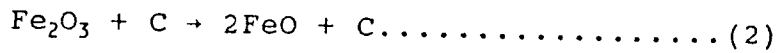
One of the striking features of the campaign was the low recovery of matte. Only 0,5 kg was recovered,

although about 4 kg was expected. The collector that was recovered consisted mainly of speiss.

The loss of sulphide may be the result of conditions inherent in this type of smelting operation. While hematite can oxidize pyrrhotite by the following reaction:



the free energy for the reduction of hematite to wustite by the reaction



is more negative at the smelting temperature, and this reaction should be favoured, i.e. pyrrhotite should not be oxidized. However, the conditions in the furnace may not have been favourable. It is presumed that, shortly after the start of the smelting operation, a pool of molten pyrrhotite formed at the bottom of the furnace, and that this was covered by a layer of molten oxides. The coke that was added as the reducing agent floated on top of the slag. The oxidation reaction could take place at the interface between the matte and the molten residue, and sulphur dioxide would be formed. At the interface between the coke layer and the molten residue, the reduction of ferric oxide to ferrous oxide can take place. It is presumed that, because a better contact interface existed between the slag and the matte than between the slag and the granules of carbon floating on top of the slag, reaction (1) would have been favoured, and the sulphide collector would have been consumed.

A further campaign was run in two cycles: the furnace was drained after heat 5, and again after heat 10. A 5 kg quantity of flotation concentrate was added to the first charge in each series, with 5 kg of residue and 0,33 kg of coke. A further 1 kg of concentrate was added with the charges to the subsequent heats of each series.

When the furnace was tapped after heat 5, 1,265 kg of matte was obtained. A further 2,955 kg of matte was



obtained after heat 10. When the furnace was dug out, a solid lump of speiss was found, with a mass of 0,965 kg. An intergrowth of speiss, slag, and magnetite was also recovered from the hearth. This was retained as a separate sample, and is referred to as 'high-grade' furnace remains.

The gold balance for the campaign is shown in Table XI.

[Insert Table XI]

#### Generation of Fume

During the course of campaigns 10 and 11, a sample of the fume, that was emitted from the furnace, was taken. This was assayed, and was shown to contain 156 g/t of gold. Unfortunately, there was no way of assessing the total amount of fume emitted during the tests. However, even though the overall accountability based on the gold input against the gold output (Tables X and XI) is over-optimistic, since it is in excess of 100 per cent, the results indicate that collection in sulphide-speiss results in a much lower loss of gold through evaporation, than does the collection in iron.

#### Demand of Collector

In the experimental campaigns, the amount of concentrate required for the generation of collector was high. For campaign 10, it was in the ratio of 1 part of concentrate to 4,6 parts of residue and, for campaign 11, in the ratio of 1 part of concentrate to 2,8 parts of residue. The yield of collector was less than expected, presumably as a result of oxidation of the pyrrhotite that was formed by the hematite contained in the residues. It is possible that stronger reducing conditions would have decreased the loss of collector.

It is believed to be unlikely that the addition of extra coke fines to the charge would radically increase the reducing conditions, since unreacted coke was seen to be floating on top of the charge during the smelting operation. More success would probably be obtained if milled coke fines were blended with the residue, and these were then fed to the furnace in a pelletized form. Some prior reduction in the solid state would then occur before the pellets melted in the bath.

### Formation of Speiss

The smelts in which flotation concentrate was used to form a collector gave promising results. The formation of a speiss phase with an even higher collecting capacity than that of sulphide was unexpected. Practical experience has shown that the presence of carbon in the charge is necessary for the formation of speiss. In some preliminary laboratory studies, during which flotation concentrate was melted with lime as a flux in an inert atmosphere, only a sulphide phase was produced. A similar result was also obtained during a preliminary test using the 100 kVA furnace, when no carbon was included in the charge.

The reduction potential during the course of the campaigns did not totally prevent the oxidation of the pyrrhotite. However, this could be put to good use in a plant-scale operation in which either a proportion of, or all the gold contained in the concentrate is to be recovered pyrometallurgically. If, at the end of a smelting campaign, only residue is charged to the furnace, the matte will be oxidized and the gold will be collected in a smaller volume of speiss.

### Conclusions and Recommendations

A transferred plasma-arc furnace of the design described in the present work was found to be unsuitable in

recovering gold from leached calcine residues by the collection of the gold in iron. Although the slags produced were largely stripped of their gold content, and contained typically 1 g /t or less of gold, the recovery of gold by the collector was poor, being typically 15 per cent. The overall accountability of gold was also poor, being typically less than 30 per cent. This lack of success is ascribed to the high temperature that is needed to keep the collector molten, which causes volatilization of the gold. Samples of fume collected from the furnace during these tests were seen to contain up to 90 g/t of gold.

The collection of gold in matte and speiss formed by the co-melting of leach residues, gold bearing sulphide concentrates, and a carbonaceous reducing agent was successful. In one smelting campaign, in which it was possible to recover most of the matte and speiss from the furnace, the recovery of gold in the collector was 90 per cent. A further 4,7 per cent of the gold was found to be present in the collector which had soaked into the furnace refractory and which, in practice, could be recovered when the furnace is broken down prior to relining. The refractory could be fed to the mill circuit, and the gold recovered by cyanidation.

It is recommended that further work be done in the area of the pyrometallurgical recovery of gold from leach residues. This work should concentrate on a few avenues. An attractive method would be the recovery of gold from the off gas stream when the leached calcined residue is merely smelted without the use of a collector. This would have the advantage of an improved cash flow, since the gold would be recovered on a continuous basis. More work should be done to improve the control of the collection process when sulphide flotation concentrate is used. The objective of this work would be to reduce the amount of concentrate that is required to effect collection. In its initial

stages, this work would involve some fundamental studies of the distribution ratio of gold between molten leached calcine residues, matte, and speiss.

#### Acknowledgement

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TABLE I  
RANGE OF COMPOSITION OF THE RESIDUES

$\text{Fe}_2\text{O}_3$ %	MgO %	$\text{Al}_2\text{O}_3$ %	$\text{SiO}_2$ %	S %	As %	Au g/t
44,7 to 64,6	1,2 to 3,2	4,1 to 6,3	21,3 to 38,0	0,43 to 0,80	0,43 to 0,67	11,0 to 16,5

Table II  
SUMMARY OF OPERATION AND GOLD CONTENT  
OF THE SLAG IN CAMPAIGN 1

Run no.	Residue kg	Char kg	Iron kg	Time min	Heel, mass recovered kg	Gold in slag g/t
1	10	1,0	0,0	90	1,661	0,10
2	10	0,8	0,0	99	1,291	0,24
3	10	0,0	1,5	60	1,681	2,36
4	10	0,0	1,5	96	1,979	2,40
5	10	0,0	1,0	89	0,908	3,38
6	10	0,0	1,0	59	0,824	3,08

Table III  
GOLD CONTENT OF THE SLAGS IN CAMPAIGNS 2 TO 7

Campaign no.	Gold content of residue g/t	Average gold content in slag g/t	Gold still contained in slag as of total
2	11,0	2,81	19,80
3	11,0	2,38	21,39
4	11,0	1,87	12,03
5	15,2	1,15	8,20
6	15,2	1,01	6,79
7	15,2	1,46	11,40

Table IV

## THE GOLD MASS BALANCE IN CAMPAIGN 8

<u>Input, g</u>	
Gold in residue	2,728
<u>Output, g</u>	
Gold in slag	0,523
Gold in fume	0,091
Gold in furnace residue	0,008
<hr/>	
Total	0,622
<hr/>	
Gold in collector, g	0,330
Accountability of gold, %	34,9
Recovery in collector, %	12,1

Table V

## THE GOLD MASS BALANCE IN CAMPAIGN 5

<u>Input, g</u>	
Gold in residue	0,562
<u>Output, g</u>	
Gold in slag	0,005
Gold in furnace residue	0,005
<hr/>	
Total	0,050
<hr/>	
Gold in collector, g	0,068
Accountability of gold, %	20,1
Recovery in collector, %	12,1

Table VI  
THE GOLD MASS BALANCE IN CAMPAIGN 6

<u>Input, g</u>	
Gold in residue	0,410
<u>Output, g</u>	
Gold in slag	0,029
Gold in furnace residue	0,003
<hr/>	
Total	0,032
<hr/>	
Gold in collector, g	0,067
Accountability of gold, %	24,1
Recovery in collector, %	13,3

Table VII  
THE GOLD MASS BALANCE IN CAMPAIGN 7

<u>Input, g</u>	
Gold in residue	0,456
<u>Output, g</u>	
Gold in slag	0,087
Gold in furnace residue	0,008
<hr/>	
Total	0,094
<hr/>	
Gold in collector, g	0,036
Accountability of gold, %	28,4
Recovery in collector, %	7,8

Table VIII

## THE GOLD MASS BALANCE IN CAMPAIGN 9

<u>Input, g</u>	
Gold in residue	0,532
<u>Output, g</u>	
Gold in slag	0,257
Gold in furnace residue	0,029
<hr/>	
Total	0,286
<hr/>	
Accountability of gold (based on gold in against gold out), % 53,8	
<hr/>	

Table IX

THE EVAPORATION RATE OF PURE GOLD AT  
SELECTED TEMPERATURES

Temp. °C	Vapour pressure of gold, kPa	Gold removed g/min
1450	0,003	0,0014
1550	0,011	0,0053
1650	0,036	0,0170
1800	0,164	0,0800
2000	0,920	1,1900
2427	15,68	8,8600



TABLE X

THE GOLD MASS BALANCE IN CAMPAIGN 10  
(RATIO OF CONCENTRATE TO RESIDUE 1:4,6)

Item	Mass kg	Gold concentration g/t	Gold contained g
<b>Gold in:</b>			
Sulphide concentrate	12.0	114.00	1.368
New dam residue	55.0	13.00	0.715
Total			2.083
<b>Gold out:</b>			
<i>Oxide products of smelts</i>			
Slag 1	5.168	0.59	0.0030
Slag 2	5.770	0.62	0.0093
Slag 3	5.516	1.72	0.0094
Slag 4	5.136	0.70	0.0036
Slag 5	4.131	0.24	0.0010
Slag 6* (top)	4.181	4.14	0.0173
Slag 6* (bottom)	2.001	24.9	0.0498
Slag 7	3.910	1.15	0.0045
Slag 8	5.382	0.59	0.0032
Slag 9	5.703	0.95	0.0054
Slag 10	5.016	0.55	0.0027
Slag 11	5.696	1.02	0.0058
Furnace remains	11.200	16.3	0.1826
Total gold in slag			0.1150
Total gold in slag and furnace remains			0.2976
<i>Collector products</i>			
Tap 6: Tapped speiss	0.2389	1012.00	0.2923
Tap 11: Tapped matte	0.5010	47.70	0.0238
Tap 11: Furnace speiss	1.6840	925.00	1.5577
Total gold in collector products			1.8738
Total gold out			2.1714

\* Slag tapped with speiss

Total gold in 2.083 g  
 Total gold out 2.1714 g  
 ∴ Accountability of gold, based on gold in versus gold out = 104.2%

*Distribution of gold (based on gold contents of products)*

Gold	Content, %
In slag	5.29
In collector	86.29
In furnace remains	8.40
Total	99.98
Potential recovery (gold in collector + gold in furnace remains)	94.69

TABLE XI

THE GOLD MASS BALANCE IN CAMPAIGN 11  
(RATIO OF CONCENTRATE TO RESIDUE 1:2,8)

Item	Mass kg	Gold concentration g/t	Gold contained g
<b>Gold in:</b>			
Flotation concentrate	18.0	114.00	2.052
New dam residue	50.0	13.00	0.650
<b>Total</b>			<b>2.702</b>
<b>Gold out:</b>			
<i>Oxide products of smelts</i>			
Slag 1	4.300	1.09	0.0047
Slag 2	4.100	0.78	0.0032
Slag 3	4.432	0.39	0.0017
Slag 4	4.414	0.64	0.0028
Slag 5*	7.890	2.29	0.0181
Slag 6	4.317	0.48	0.0021
Slag 7	4.804	0.29	0.0014
Slag 8	4.832	0.18	0.0009
Slag 9	4.449	0.42	0.0018
Slag 10*	7.069	2.88	0.0204
Furnace remains			
'High grade'	17.29	90.50	1.6030
'Low grade'	33.20	2.69	0.0893
<b>Total gold in furnace remains</b>			<b>1.6923</b>
<b>Total gold in slag</b>			<b>0.0601</b>
<b>Total gold in slag and furnace remains</b>			<b>1.7524</b>
<i>Collector products</i>			
Tap 5: matte	1.265	51.50	0.0651
Tap 10: matte	2.955	104.00	0.3073
Furnace speiss	0.965	680.00	0.6562
<b>Total gold in collector products</b>			<b>1.0286</b>
<b>Total gold out</b>			<b>2.781</b>

\* Slag tapped with matte

Total gold in 2.702 g  
 Total gold out 2.781 g  
 ∴ Accountability of gold, based on gold in versus gold out = 102.9%

*Distribution of gold (based on gold contents of products)*

Gold	Content, %
In slag	2.16
In collector	36.98
In furnace remains	60.85
<b>Total</b>	<b>99.99</b>
Potential recovery (gold in collector + gold in furnace remains)	97.83

## Figure Captions

Fig. 1. Gold content of collector versus gold content of slag

Fig. 2. Schematic representation of the 100 kVA water-cooled batch furnace.

- (1) Argon supply to hollow graphite electrode
- (2) Electrode support mechanism, clamp, and power cable
- (3) Water-cooled electrode seal
- (4) Gass offtake
- (5) Steel roof with magnesia lining
- (6) Ceramic fibre insulating gaskets
- (7) Water-jacket
- (8) Replaceable refractory crucible
- (9) Anode connection
- (10) Graphite block
- (11) Feed port.

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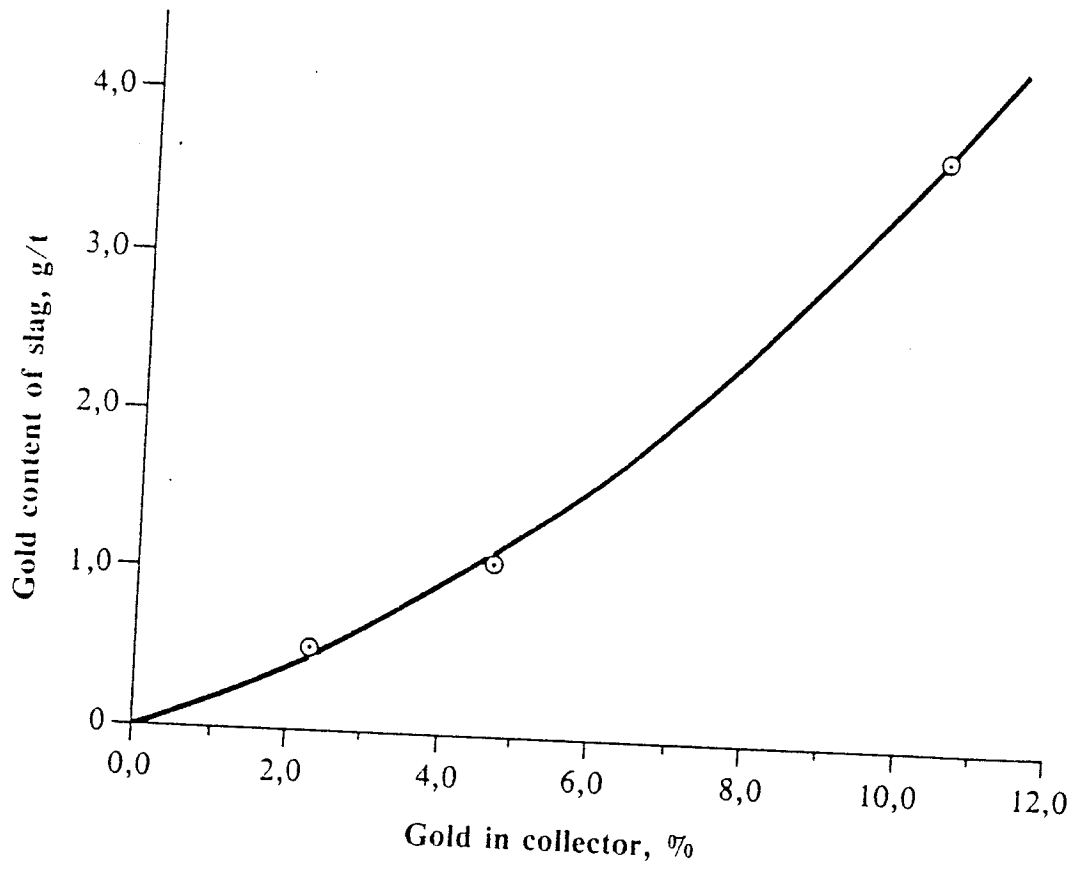
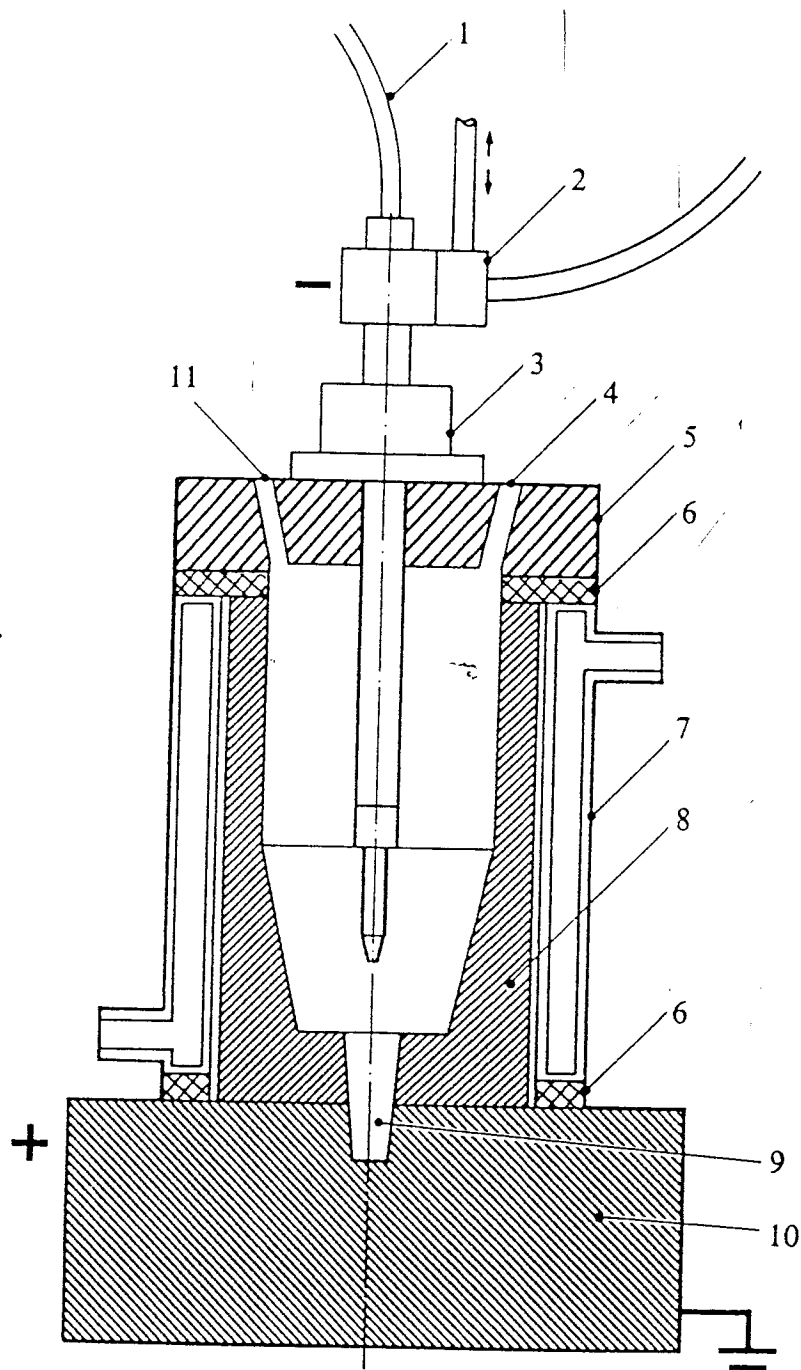


Fig. 1—Gold content of collector versus gold content of slag



- (1) Argon supply to hollow graphite electrode
- (2) Electrode-support mechanism, clamp, and power cable
- (3) Water-cooled electrode seal
- (4) Gas offtake
- (5) Steel roof with magnesia lining
- (6) Ceramic-fibre insulating gaskets
- (7) Water-jacket
- (8) Replaceable refractory crucible
- (9) Anode connection
- (10) Graphite block
- (11) Feed port

Fig. 2—Schematic representation of the 100 kVA water-cooled batch furnace

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