GOLD ROOM PROCEDURE AT LAKE VIEW AND STAR, LTD.

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SUMMARY.

Recovery of gold by smelting zinc dust concentrate has been carried on at Lake View and Star, Ltd., since July, 1930, when the Merrill-Crowe precipitation plant was first put into operation.

Part I describes the development of current gold-room practice, commencing with the clean-up of the Merrill-Crowe plant, a procedure that has not materially changed since its inception, and then shows how modifications to straight roasting and smelting became necessary, following a gradual increase in the copper content of the ore treated by the mill.

Present smelting practice converts most of the copper into a mixed copper-lead matte, the sulphur being supplied by the addition of flotation concentrates containing about 35% sulphur as pyrite. The process yields gold bullion of a sufficiently high grade to send to the Royal Mint, but until 1943, suffered from the disadvantage of giving matte and slag by-products high in gold and silver, that were not recovered directly but had to be sent to a smelter.

Part II gives details of experimental work carried out since 1943 which led to a process for the satisfactory and economic recovery of not only the gold and silver, but also the copper from these, hitherto, troublesome by-products. Nevertheless, incidence of matte in a smelt has always been considered undesirable, and it is hoped that eventually it will be obviated by removing most of the copper from the zinc slime before smelting.

Part III indicates the methods by which this might be achieved, and outlines the experimental work that has been done up to the present.

PART I.

THE HISTORY AND DEVELOPMENT OF CURRENT GOLD-ROOM PRACTICE.

INTRODUCTION.

Gold is extracted from ore at the Lake View and Star, Ltd., by the following series of operations:—

Coarse crushing;
Fine grinding;
Straking for free gold;
Flotation;
Concentrate roasting;
Straking for free gold;
Cyanidation of concentrates;
Precipitation of gold from cyanide solution.

The last operation is performed in a Merrill-Crowe radial leaf type precipitator, to which zinc dust is fed continuously, together with a small amount of lead nitrate to form a lead-zinc couple that more readily precipitates gold than pure zinc alone. The resultant zinc slimes therefore contain as principal constituents; gold, silver, lead, copper, and zinc, in varying amounts, with minor quantities of other elements and radicals.

Part I describes the clean-up of zinc slimes from the Merrill-Crowe plant; then traces the history of the smelting processes used to separate the gold as far as possible from the base metals in the slimes.

Initially, in 1930, slimes were pressed in a leaf filter, acid treated with dilute sulphuric acid, roasted and smelted in Salamander crucibles using a simple silica-borax flux. By 1936, however, the copper content of ore had risen so much that the bullion contained relatively large amounts of base metal, and caused a large increase in refining charges at the Mint.

The bullion was then granulated and sulphured with moderately satisfactory results. Later, however, with increases in mine tonnages, the work of granulating and sulphuring 15,000 to 16,000 oz. of bullion per month became too heavy and exhausting. Pyrite concentrates from the flotation plant were then added to the silica-borax flux, producing a bullion of satisfactory grade, and mattes containing quantities of gold and silver that were only partially recovered by re-melting with 20% soda ash.

Current practice still adheres to the matte smelt, which is therefore fully described herein. The matte smelting process was extended after 1943 to include recovery of metals from the matte and slag, and forms the subject of Part II of this report.

It will be noted that in the general ore treatment scheme, free gold is separated from the ore during fine grinding, and again after roasting of the concentrates, by passing over strakes covered with corduroy cloths. Recovery of the gold by amalgamation and retorting, although not properly a smelting process, is included after the description of smelting because it is an essential and highly important part of gold-room procedure.

Brief reference will also be made to the smelting of raw zinc slimes from a second plant of the Lake View and Star, Ltd., that recovers gold from old flotation tailings by cyanidation, filtration and precipitation.

PAST PRACTICE.

Clean-up.

Zinc slime was collected in the Merrill tank over a period of about 10 days. On clean-up day, the inlet solution flow was stopped at 7.30 a.m. and the tank run out into a monteju that delivered to either of two Dehne filter presses. Slime collected on the cloths was hosed down with a spray of high pressure water, and passed through to the press until the leaves were full, indicated by a cessation of liquid flow from the filter. Air was then turned on the press until excess solution had been blown out, when the press was opened and the cake scraped out with wooden scrapers into rectangular galvanised tubs capable of holding about 220 lb. of moist slime, which contained approximately 33% moisture. It was then weighed and the net weight recorded.

In July, 1930, when the plant started, the filter cake was transferred to a lead-lined wooden vat containing warm dilute sulphuric acid of about 6% strength, and stirred by hand with a wooden paddle until action was complete. It was then filtered again through the same presses, washed with water, blown with air and again weighed before transferring to the muffle furnaces for roasting. Aciding of the slime was stopped in 1933, because the treatment did not result in any loss of weight in the slime, nor improvement in bullion grade.

Each clean-up day, when the tank had been emptied of solution and slime, one man used to enter the tank and chip off a segment equal to about one-third of the area of the bottom. This was necessary because a layer of zinc dust, about ¼ in. to ⅜ in. thick, accumulated and had to be removed. The zinc chips contained gold, although in much smaller amounts than in the main slime. It was roasted and smelted in a similar manner, and with the pressed slime.

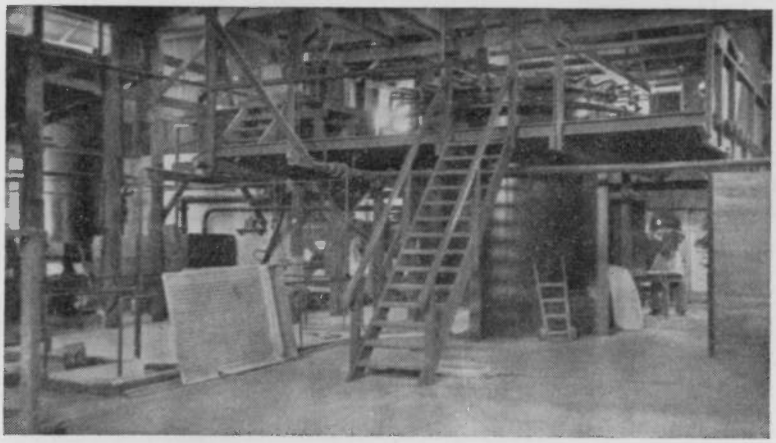


Fig. 1. Merrill-Crowe Precipitation Plant.

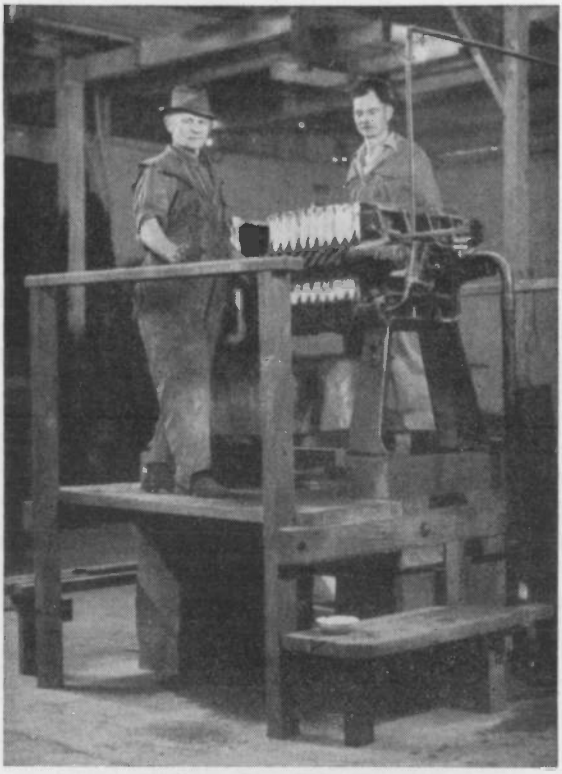


Fig. 2. Dehne Filter Press.

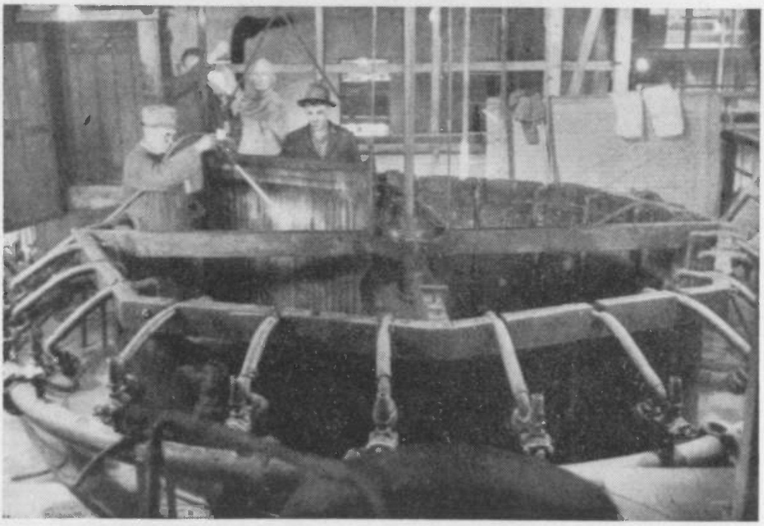


Fig. 3. Hosing Down Merrill Cloths.

Cloths from the Merrill and Dehne presses were removed when holed, or coated too badly with gypsum. They were burnt in the muffle furnaces after the main roasting of slime was completed, and fluxed with the last smelting charge for the day.

Clean-up of the Merrill tank took approximately four hours, but slime was available from the presses by 10 a.m. It was transferred, after weighing, to the muffles for roasting.

Roasting.

The first tub of slime was shovelled into and divided between, four muffles, thus each contained about 50 lb. of moist slime per charge. The charge was spread evenly on the muffle hearth in a layer 1½ in. thick, using a hand rake, and the door was then closed to conserve heat to drive off moisture before roasting proper commenced. No attempt was made to rabble the slime until it was dry, because clouds of dust would be raised at one stage during drying. When the slime glowed red in a number of places, it was carefully turned over until it was certain that the risk of losing dust was past, when it was rabbled steadily for about 10 minutes.

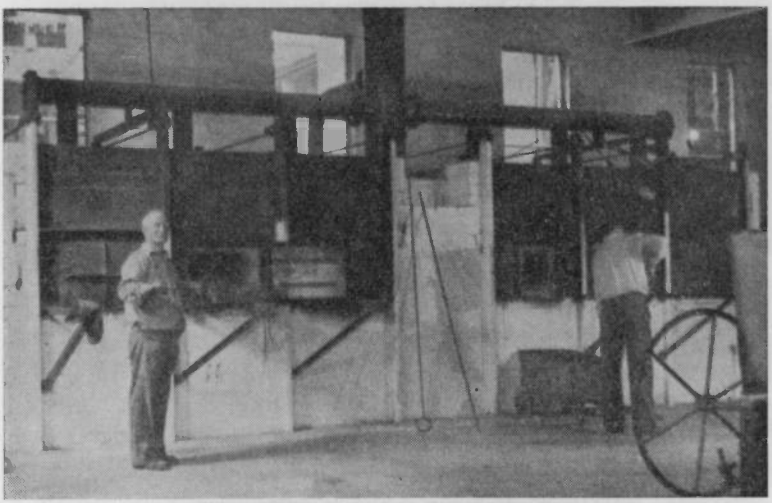


Fig. 4. Muffle Furnace for Roasting Zinc Slime.

The exact point at which roasting was deemed to be complete was determined by the appearance of the hot mass. Soon after rabbling commenced, the slime glowed brightly in spots, and the temperature of the whole charge gradually rose to a maximum. Once this peak had been reached, further rabbling produced little effect, the slime quickly cooled on the surface, giving it a dead appearance, and very little further oxidation of the base metals took place.

The roasted slime was then ready to be raked into a tray for weighing. It was weighed hot, tipped on to an iron mixing table, and flux in the appropriate proportion was weighed out and mixed with it, using a small household-type ash shovel. Roasting time for one muffle charge was about 1½ hours, half of which was used in drying off water.

Smelting.

Faber du Faur tilting furnaces, coke fired, with No. 9 Salamander retorts were used initially to melt down a charge of fluxed slime, consisting of:—

Roasted slime	120 lb.
Borax glass	36 lb.
Silica sand	24 lb.

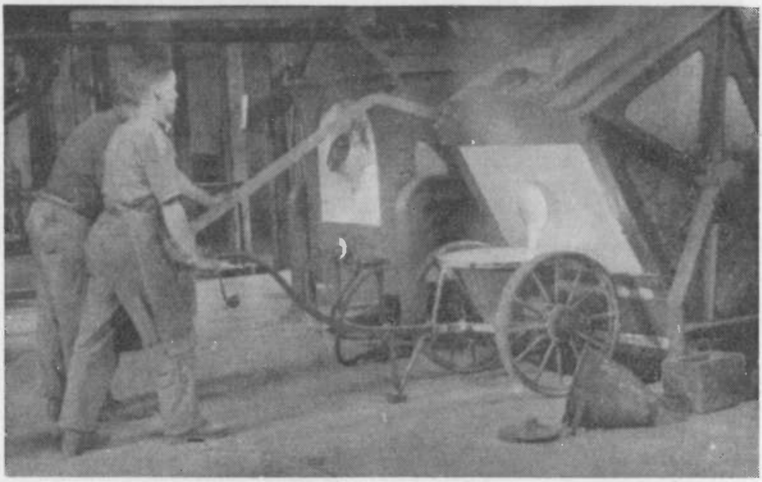


Fig. 5. Pouring a Faber du Faur Tilt Furnace. Small conical Mould shown beside the Furnace.

Coke firing was later replaced by oil firing, using Major 2 in. low-pressure air, crude oil burners. These burners were so readily manipulated and controlled that they have been installed on several other furnaces since, giving complete satisfaction.

For lighting a furnace, a fairly good wood fire was first set to prevent any danger of an explosion due to delayed firing of the oil, the air was turned on slowly, then the oil, which should take fire immediately and burn steadily.

Using the No. 9 tilt furnaces, about 60 lb. of fluxed slime at a time was charged into a red hot retort, and allowed to run down partially before adding the next charge. Three such loadings, containing up to 120 lb. of roasted slime, were usually as much as one retort would hold without any chance of slag boiling over. If a boil-up did occur, a handful

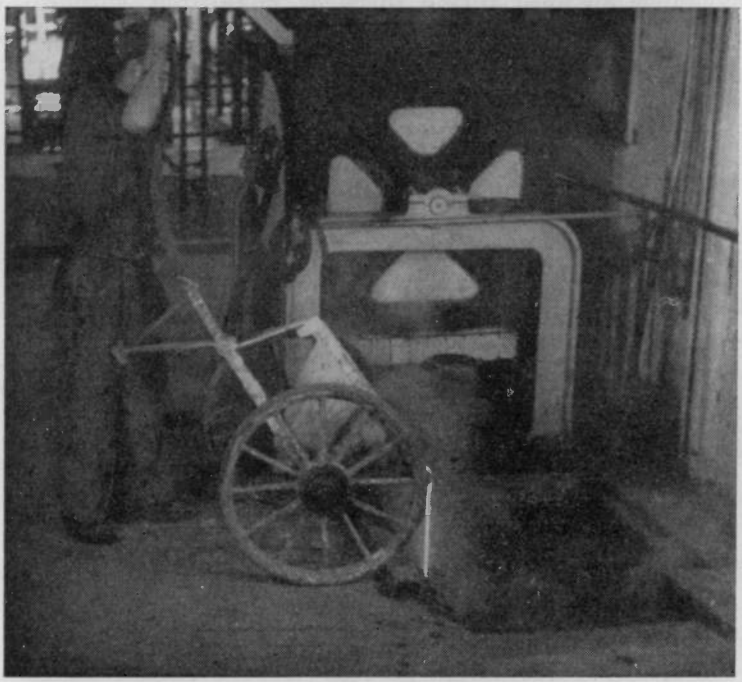


Fig. 6. Granulating Slag.

of common salt was thrown in to prevent slag overflowing.

Time required to run down a charge of 120 lb. of slime, from the commencement of charging was approximately two hours. Completion of melting was judged by stirring with a warmed iron bar, to feel whether the slag was free from lumps and sufficiently fluid; also by the quiescent appearance and colour of the top of the molten mass.

Slag was poured into a conical mould permanently mounted on wheels, and having a long handle for tipping purposes. This mould, as with all other metal equipment that might come into direct contact with molten slag or matte, was preheated with a light wood fire before use to evaporate moisture from the surface, and so remove any hazard of hot molten material spattering or exploding. Matte is particularly dangerous in this respect. To prevent slag sticking to the mould, it was given a lime wash immediately after use.

When most of the slag had been poured off, the large slag mould was removed and replaced by a smaller, deeper conical mould, which could be removed by hand from the trolley into which it was placed. This mould received the remnant of slag in the retort, together with the bullion. Oil poured into the mould just before pouring prevented sticking.

A new charge was fed into the furnace immediately after pouring. By that time, the slag in the large mould was ready to granulate by pouring slowly into a pit filled with water. No trouble was ever experienced in this operation because the slag was carefully watched for any signs of matte towards the end of the granulation. The slag remaining as a crust over the sides of the mould was easily broken out with a bar and pushed into the water pit. The mould was then given a lime wash while still hot, so that it would be coated and dry ready for the next pour.

The bullion pour was allowed to cool off for some minutes, to allow it to set, the actual time depending on the size and grade of the button. The mould was then lifted from the trolley, tipped over on the concrete floor and the bullion button separated as far as possible from slag before quenching it in acid and water. Remnants of slag were then chipped off.

Barring.

Bullion buttons produced in this way were melted again in a Salamander No. 60 lipped crucible built into a special small tilting furnace usually referred to as the "barring furnace." A borax cover only was used at first, but when small quantities of matte began to occur in the smelt, extreme difficulty was experienced in cleaning the final bar. Experiments resulted in the use of half a Battersea G assay crucible of soda ash on the button when it was just run down, followed, when the bullion was red hot, by a full Battersea G crucible of a mixture of soda ash and borax,

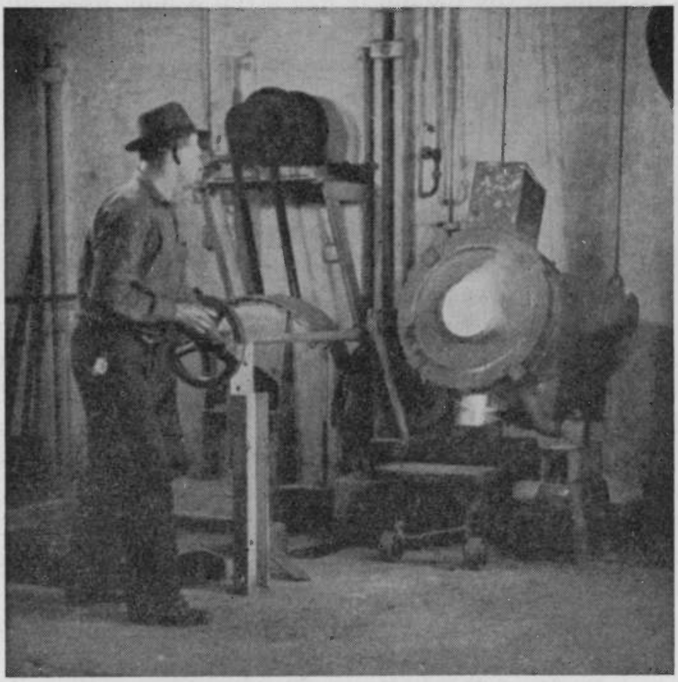


Fig. 7. Pouring a Bar from the Barring Furnace.

in the proportion of 1 to 1. Sometimes the half-crucible of soda was unnecessary, depending on the amount of matte adhering to the plug.

Fluxed bullion was poured into cast iron bullion moulds of 600 oz. capacity. The moulds were previously coated with carbon black internally, by burning kerosene underneath them. Bars so formed were taken, while still red hot, brushed with a wire brush to remove slag, rubbed over with nitre, which melted with the heat of the bar, then quenched by plunging into a small lead-lined tank fitted with a hood and stack, and half-filled with dilute sulphuric acid. The bar was then immersed in a wooden tub filled with water, to cool it completely before hand polishing by scraping and brushing with sand and water on a table in front of the water tub.

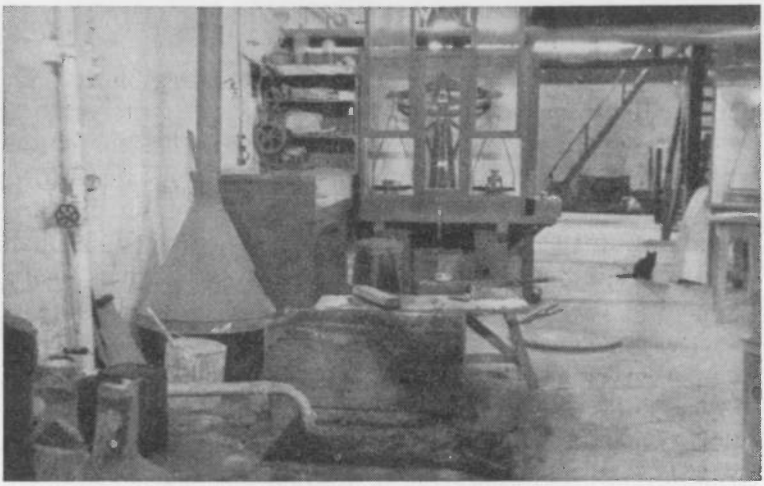


Fig. 8. Acid Tub with Stack, Wooden Wash Tub, Gold Scales and Drill for Sampling Bars.

Both the acid and water tubs were cleaned out at intervals of about three months. Prills and chips of gold could then be panned off from the sand and sludge, which were bagged and sent to the flotation or cyanide plant, the pannings going directly to the barring furnace.

Gold bars produced by this method averaged:—

Gold	840
Silver	100
Base metal	60

and were sent to the Royal Mint, Perth, for further refining.

About 1935, however, copper content of the mine ore began to rise, and in 1937 copper sulphate was introduced as a conditioning agent in the flotation plant. These two factors caused a marked increase in the copper content of the Merrill slimes. For a time copper was held in the slags by adding up to 20% of manganese dioxide to the fluxes, but in January, 1936, the bullion grade had fallen to:—

Gold	600
Silver	90
Base metal	300

Mint refining charges were too high and were increasing, so that efforts were made to prevent the copper entering the bullion.

Aciding of raw slime was reverted to for a short time, but was quite inadequate.

Small scale tests proved that using a silica-borax flux with or without an oxidising agent, carbon from the retorts reduced the copper out of the slag into the bullion; nothing but a clay-lined pot was found to give a satisfactory smelt in these circumstances. The quantity of slimes being handled at that time would have required large clay liners, which were considered impracticable for the No. 9 Salamander crucibles. Hence, in February, 1936, the practice of granulating and sulphuring all bullion from the primary smelt was introduced.

Granulation and Sulphuring of Bullion.

Plugs of bullion weighing up to 900 oz. were granulated as follows:—

The plug was melted in a No. 60 Salamander crucible in a well furnace, the temperature, just before pouring, being brought up to a bright red to insure that the metal would not freeze before all of it had been poured. The crucible was lifted from the furnace with a pair of tongs to which a rope block and tackle ("handy billy") was attached and raised to a height of about 4 ft. from the tank of water into which the bullion was poured slowly in a thin stream. Speed of pouring and grade of the bullion affected the size of granule produced, the slower the pour and the lower the grade, the finer the size. The tank used was 2 ft. in diameter by 4 ft. high.

After pouring, the water was emptied from the tank and the bullion tipped into a flat iron tray, 2 ft. 6 in. square with 2 in. sides, in which it was dried by heating over the well furnace. When cool, the granulated bullion was stored in a tin marked in chalk with its weight, ready for mixing with sulphur.

Not more than 900 oz. of granulated bullion was taken and mixed with 20% of its weight of flowers of sulphur for each sulphuring. Mixing was done as thoroughly as possible on a table, working the sulphur into the bullion with the two hands. Then the mixture was transferred to semi-cylindrical shaped scoops, 2 ft. 6 in. long, for pouring directly into the same No. 60 pot as had been used for the granulation.

The pot was in the furnace, red hot, with oil turned off just before charging. Oil was turned on again immediately afterwards and the charge began to melt down. Three quarters of an hour to an hour was required to complete the reaction. At the end of this time the mixture was stirred with an iron bar to make sure no lumps remained, when the furnace was stopped and the melt poured into four bullion moulds. When it was judged the bullion had set, the matte was poured away from it by tipping the mould gently on the floor, and adhering matte was quickly barred off the bullion while still hot, as it was most easily removed then and not later after cooling. Bullion so obtained varied in grade considerably, the best being:—

Gold	880
Silver	40
Base metal	80

but an average figure would be:—

Gold	840
Silver	30
Base metal	130

Effectiveness of the sulphuring operation appeared to depend greatly on the initial grade of bullion granulated; the lower the initial grade, the higher the finished product, within limits. Fineness of the granulation also appeared to play a minor part.

Matte from the sulphuring was fed back to the tilt furnace with the addition of 20% of its weight of soda ash and remelted, on the second day of the clean-up. Up to 200 lb. would be run in one pour, taking about one hour.

Three products were obtained: slag, matte, and bullion.

Slag was poured off into the large conical mould until just before matte appeared. The remainder of the melt was then run into the small conical mould. Slag was granulated as before, but the matte-bullion mould was allowed to cool until the bullion solidified, after which the still molten matte was carefully tipped on to the concrete floor. The bullion button was tipped out and quenched in water. Adhering matte was broken off with a 7 lb. hammer, and the button added to the bullion from sulphuring for barring.

Secondary matte from the foregoing operation, containing some slag, was melted again with a further 20% of soda ash, and yielded secondary matte bullion, tertiary matte and slag. Slag was granulated as before; the matte was set to one side for crushing and sampling before shipping in bags to the Electrolytic Refining and Smelting Company's works, Port Kembla, New South Wales.

Secondary matte bullion was allowed to accumulate until sufficient was obtained for granulation and sulphuring, after which the bullion was barred with bullion from the main sulphuring. Matte was returned to the secondary matte remelt.

Combined, the bullion from matte retreatments assayed:—

Gold	760
Silver	80
Base metal	160

so that final bars containing it approximated the assay previously given, viz.:—

Gold	840
Silver	30
Base metal	130

Slags from all the sources described were amalgamated after granulation. A fuller description of their treatment is given later under "Amalgamation."

PRESENT PRACTICE.

Clean-up procedure remains much the same to-day as in 1930, although some minor changes have been made. One of these is that chipping the Merrill tank has been rendered unnecessary by reversing the direction of rotation of the impeller in the tank, thereby throwing solution downwards and preventing any accumulation of zinc from occurring. Previous trouble in starting flow of solution and slime through the bottom valve also disappeared when this change was made.

After clean-up, the zinc and lead nitrate are set at an abnormally high rate for four hours to assist precipitation by coating the cloths quickly. Then it is reduced to slightly above normal for 24 hours, or until the assay of the tails solution shows 0.05 dwt. per ton, or less. Solution flow rate is held almost constant between clean-ups.

The only modification in roasting has been the building of an oil-fired roasting furnace with six muffles to replace the old four-muffle furnace that burned wood. The new furnace has a layer of diatomaceous earth, insulating firebrick between the inner refractory and outer ordinary red brick wall. The insulation has resulted in much cooler working conditions for the operators and lower heat losses from the furnace. Kyanite and silicon carbide bricks were used to line the fireboxes.

Granulation and sulphuring of bullion was practised for about two years. Towards the end of 1938, the amount of bullion that had to be so treated had risen to 15,000 to 16,000 oz. per month. The work became too heavy and exhausting, and attention was therefore directed towards a simpler and less laborious method.

Preliminary trials introducing pyrite-containing flotation concentrates into the flux gave highly promising results. Tested on a plant scale, the method overcame the problem of copper in the bullion and the work of granulating and sulphuring simultaneously.

Smelting Practice.

Two distinct classes of gold slime are treated at present in the gold-room at the Chaffers plant:—

1. Main ore treatment plant slimes that form the principal subject of this report;
2. Smaller quantities of lower grade slimes from the Chaffers re-treatment plant operated by the Lake View and Star Ltd. This plant recovers gold by cyanidation from residue dumps derived from a previous all-roasting and cyanidation plant, mixed with early flotation tailings from the present treatment plant. Their history makes clear that no complications arise from copper content. Hence they can be smelted simply, without roasting, or even drying, by using a flux containing nitre to oxidise the base metals.

   No details beyond the actual flux used will therefore be given; most attention will be focussed on the matte smelt of the main treatment plant.

Retorts for the Faber du Faur tilt furnaces were recognised as being expensive in first cost, as well as short-lived. With the possibility of replacing them with a furnace in which the slime could be smelted raw, i.e., without roasting, it was decided in 1937 to try a Wabi Monarch-Rockwell type semi-rotary furnace with silicon-carbide liner. Two were purchased originally, followed by two more at a later date.

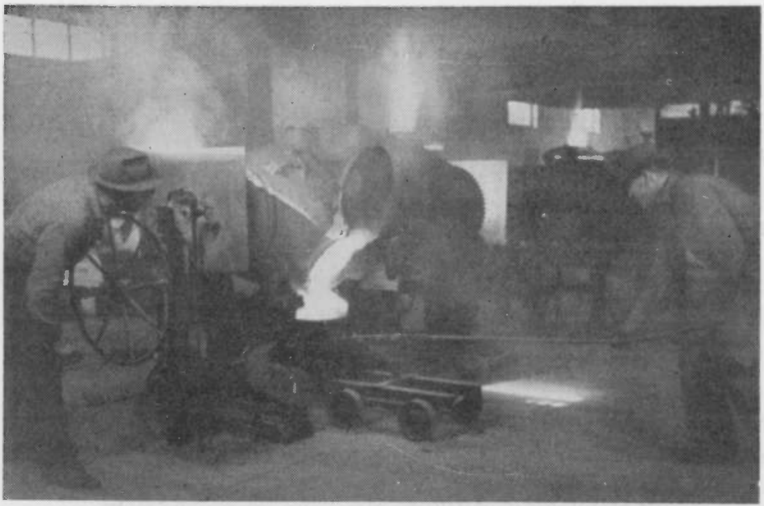


Fig. 9. Pouring Slag from a Wabi (Monarch-Rockwell Type) Furnace.

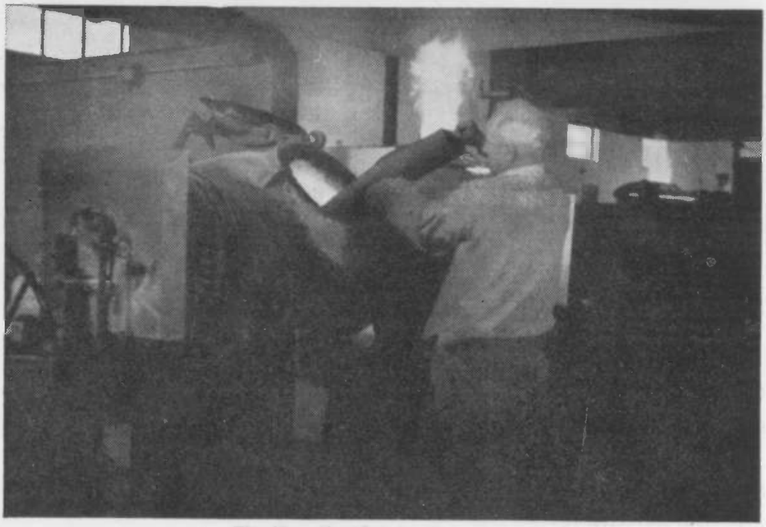


Fig. 10. Charging the Wabi Furnace.

Smelting of raw slimes was attempted for a short time, using the following fluxes:—

1. *Main Smelt*:	Parts
Raw slime	100
Nitre	30
Borax glass	30
Silica sand	20
Manganese dioxide	10
Fluorspar	5

Bullion, however, was low in grade, giving bars assaying:—

Gold	710
Silver	60
Base metal	230

The method was quickly abandoned for the main slimes, but was adopted and has been used ever since for the re-treatment slimes, which are fluxed as follows:—

2. *Re-treatment Slimes*:	Parts
Raw slime (moist)	100
Borax glass	30
Silica sand	20
Nitre	25

Fusion takes place readily, slag is very fluid, and bullion grade quite acceptable:—

Gold	570
Silver	340
Base metal	90

Matte Smelting: Slimes from the Merrill-Crowe are cleaned-up at intervals of about 10 days, as previously described, and roasted in the oil-fired muffle furnaces.

Table 1 gives the analyses of both raw and roasted slimes, together with analyses of the various products from the matte smelt.

	Primary Smelt Bullion	Primary Matte Bullion	Raw Slime	Roasted Slime	Slag	Primary Matte
Au	840-860 fine	720 fine	33.7	34.9	0.04	5.71
Ag	50-60 fine	120 fine	3.5	3.6 Base to Oxides	0.01	5.23
Cu			16.7	17.0-21.3	1.19	36.60
Pb			13.4	13.0-14.0	3.58	18.30
Zn			16.0	14.8-18.5	13.46	Trace
Fe			1.0	1.1- 1.5	6.00	14.79
Ca			1.6	1.7	1.30	-
Sulphides			1.7	Trace	0.19	18.20
Sulphates			0.8	1.7	0.51	-
Insol			0.1	0.1	29.10	Trace
			88.5	87.9-98.3	54.38	98.83

Table 1. Analysis of the various Gold Room Products. Figures are in percentages except where otherwise stated.

Each roaster charge when ready, is taken out, weighed, and fluxed while hot on the mixing table, using the following fluxes:—

	Parts
Roasted slime	100
Silica sand	15
Borax glass	30
Soda ash	5
Pyrite concentrates	30

Dry pyrite concentrates come from the flotation plant and contain approximately 35 to 38% sulphur. Formerly 5% of fluorspar was added to the charge, but as it appeared to have no effect on fluidity, it is not now used.

Although composition of the roasted slime does vary somewhat from time to time, it has not been found necessary to analyse it before each smelt, nor to alter the flux proportions.

A full charge for one Wabi furnace, containing between 100 and 115 lb. of roasted slime, is added to the red hot furnace. With the tilt furnaces, dividing the charge into three separate additions was found to increase the speed of melting, but no advantage could be obtained in this way with the Wabi furnaces.

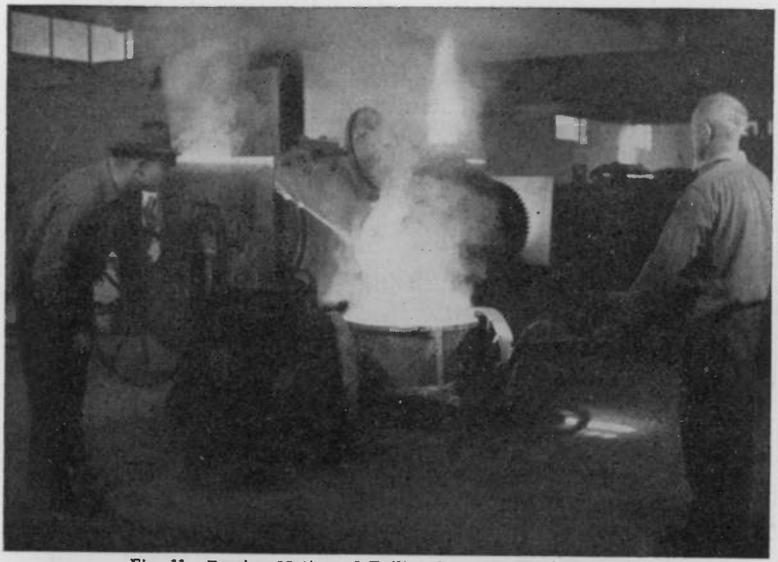


Fig. 11. Pouring Matte and Bullion from the Wabi Furnace.

Time required between charging and pouring is about 1¼ to 1½ hours. To begin, the furnace is left for ¾ to 1 hour in the upright position. Then it is rolled through 35 degrees to one side for five to 10 minutes, rotated back to a similar position on the other side for the same time, and the procedure repeated until the melt is ready for pouring, judged by the colour and quiescent appearance of the surface.

Pouring follows a similar course to that already described except that care is taken to ensure that all the matte goes into the second smaller conical mould, which is capable of holding 80 lb. of matte when full. After cooling sufficiently for the bullion to freeze, the still molten matte is tipped on to a cast iron tray, and the bullion button cleaned as far as possible before quenching in water. The bullion is later barred from the barring furnace, either with similar initial smelt bullion, or admixed with bullion derived from the primary matte.

Primary matte is melted with 12 to 15% by weight of soda ash, after the smelting of the slimes has been completed. From this melt are obtained: Primary matte bullion, primary matte slag, and secondary matte. A complete separation of primary matte slag and secondary matte is obtained by allowing the contents of the large mould to freeze overnight, when the two products break cleanly from each other. The small mould containing the bullion button is poured while the matte is still molten to recover the bullion, but no slag is present in this mould.

Formerly, the practice was to melt the secondary matte once more with 20% of soda ash, yielding secondary matte bullion, secondary matte slag, and tertiary matte. Fig. 12 shows diagrammatically the process in use up till this time. Even so, the tertiary matte still retained comparatively large amounts of gold and silver. Tertiary matte could not be considered as a satisfactory end-point to the matte smelting process, because it had to be sent to a smelter for final recovery of the precious metals.

Since 1943, however, by melting the secondary matte with 7% of flour and 50% by weight of soda ash, a base bullion is obtained in which is concentrated nearly all the gold, silver, lead, and some of the copper. Whether the final mattes are worth treating or not depends largely on the value of the copper they contain compared with the cost of recovering it. The value in any case is small.

The precious metals are readily separated from the base bullion by cupellation, yielding high grade gold-silver bullion, and oxides of lead and copper. The copper is recovered, together with any small amounts of precious metal retained by the oxides during cupellation, by smelting with a flux to slag off the litharge, the lead not being considered worth recovering.

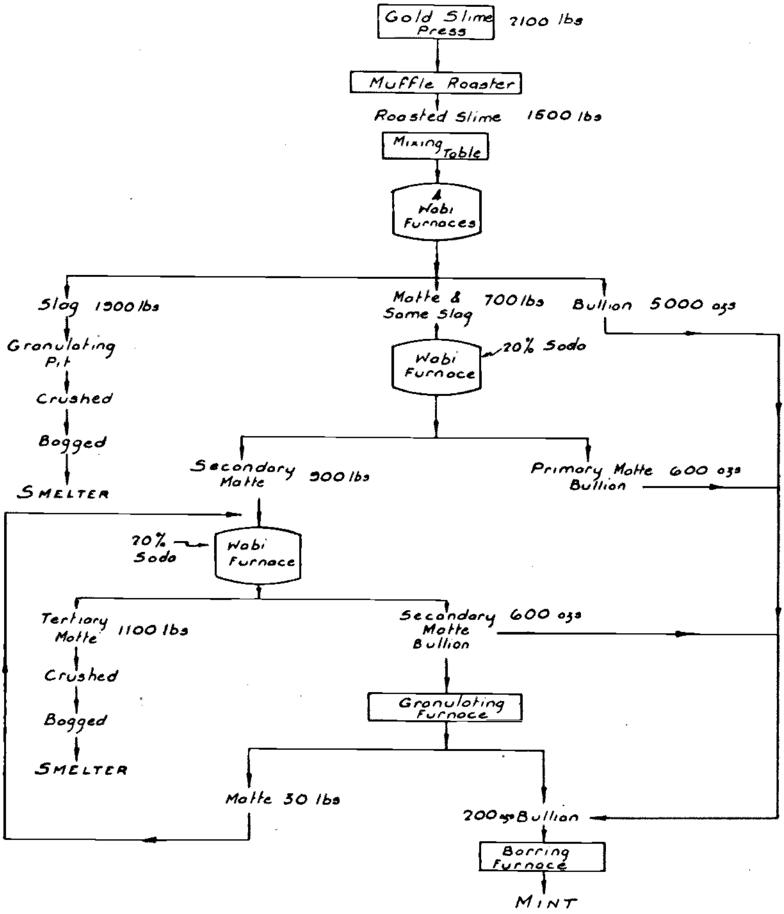


Fig. 12. Gold Room Practice prior to any attempt being made to treat Gold Room products. (Figures show the approximate weight per month of each product.)

Fig. 13 shows diagrammatically the process being used at present, and Fig. 14 the distribution of values in the various stages.

Full details of this recent addition to the matte smelting process, which brings it to a satisfactory end-point, are given in Part II, where treatment of slag is also discussed.

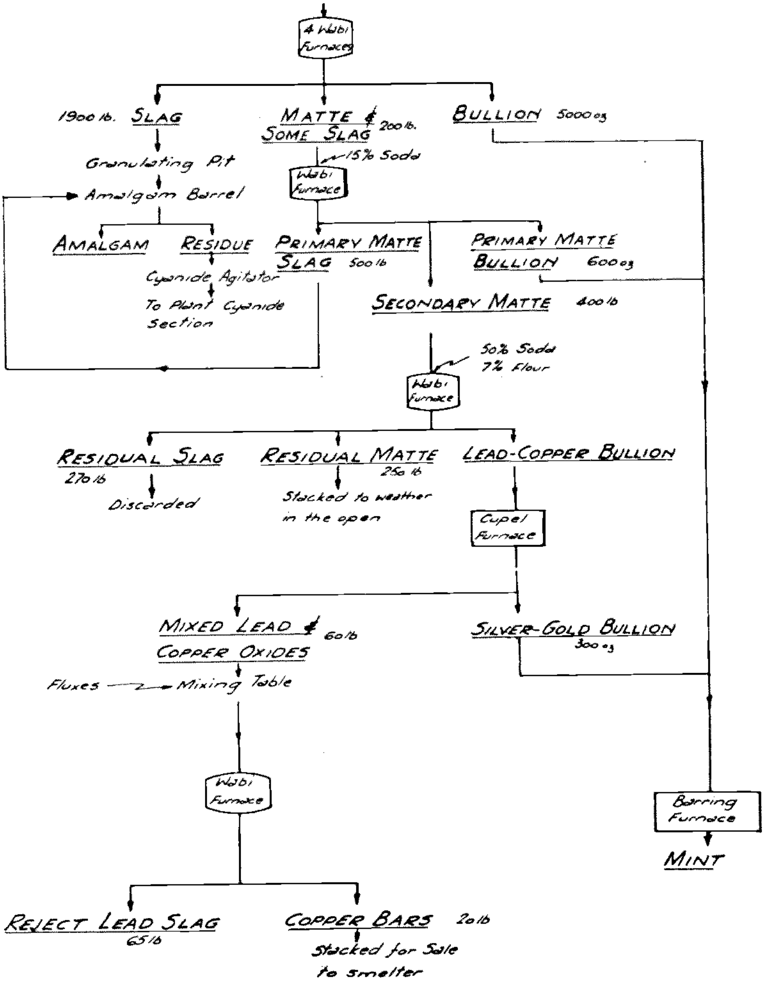
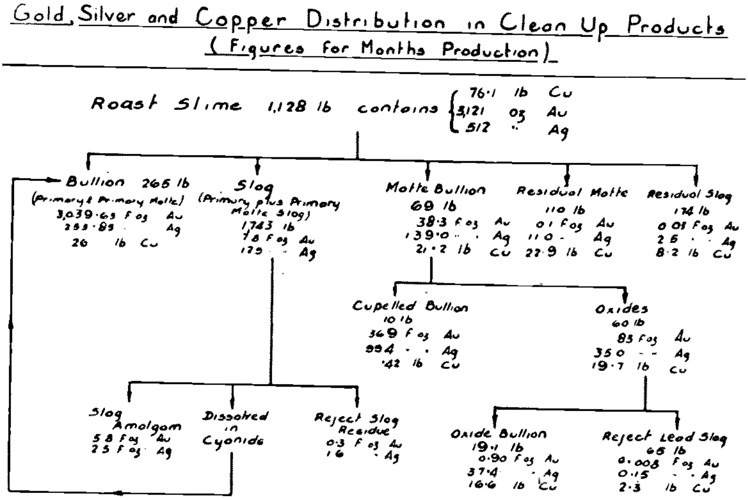


Fig. 13. Present Gold Room Practice.

Until 1943, subsequent treatment of slag was to recover coarse gold by amalgamation in a barrel, after which it was dried and allowed to accumulate. It was then mixed in one lot, sampled by coning and quartering, and finally bagged ready for shipment to Port Kembla.

Present practice retains the amalgamation step, but thereafter the ground slag is agitated with cyanide solution for eight days, or until assays show that solution is practically complete.



All figures are based on actual assays of the various products
The calculated figures for roasted slime based on the smelt products in the top branch of the above tree are :- 3,085·86 oz Au
419·25 oz Ag
78·3 lb Cu

Monthly Loss (as from above table)

	Au (F oz)	Ag (F oz)	Cu (lb)
Reject Slag Residue	0·3	1·6	—
• Residual Matte	0·1	11·0	22·9
" Slag	0·03	2·5	8·2
Reject Lead Slag	0·008	0·15	2·3
Total Loss	·438	15·25	33·4
Percentage loss using Calculated Head Value of Roast Slime	0·015 %	3·64 %	42·66 %

• Stocked for possible further treatment

Fig 14

Gold, Silver and Copper Distribution in Clean Up Products.
(Figures for Months Production.)

Barring follows former practice closely, but one improvement has been made in the bullion moulds used. For some years past, cast iron bullion moulds developed an indentation at the point where the molten bullion first touched the mould during pouring. This sometimes happened during the first time of use. Surface cracks in some of the older moulds added to the imperfections on the bottom of the resulting bar, which made it difficult to clean, and handle during drilling, as well as detracting from its appearance.

Alloy cast irons were tried to overcome the trouble, but none was successful.

Eventually, the bottom of the mould was machined off, and replaced by a false bottom fixed to the sides with countersunk screws. A layer of Carbolox (silicon-carbide cement) about 1 in. thick was tamped in and smoothed off.

Bars poured in these moulds are now regularly smooth, and can be so maintained because the lining can be replaced as soon as defects occur.

Treatment Times and Quantities: On clean-up day, solution flow to the Merrill-Crowe plant is stopped at 7.30 a.m. The first tub of raw slime, approximately 200 lb. wet weight, is available for roasting by 10 a.m., and is roasted and on the mixing table by 11.30 a.m. The roasters are lighted at about 4 a.m., so that the temperature of the brickwork can be brought up slowly, thereby minimising stresses due to expansion.

Those two of the four Wabi furnaces to be used for the day's smelt have had wood fires in them since 8 a.m., oil is turned on at 10.45 a.m., and at 11.15 a.m. ashes and charcoal are cleaned out so that the whole furnace is red hot to receive the first charge by 11.30 a.m.

Wet weight of raw slime treated for a nine to ten days' run varies between 450 to 600 lb., yielding 300 to 400 lb. after roasting. These figures are for the present milling rate of 25,000 tons of ore per month.

Smelting proceeds continuously as slime comes forward from the presses and continues until all the slime and the primary matte have been smelted, usually about 3.30 p.m. Work for the day is then finished.

Smelting time for one Wabi furnace charge of 100 to 115 lb. of roasted slime is 1¼ to 1½ hours; for one primary matte charge of 200 to 240 lb. the time required is 1 to 1¼ hours.

Next day, bullion buttons produced on the first day, as well as gold derived from the retorting of amalgam on the second day, are cast into bars, one at a time, from the barring furnace. This operation commences at 8 a.m. and lasts till about midday. In all, four to five bars are made.

During the afternoon shift, slag is shovelled out of the pit on to the floor, allowed to drain, and stacked. Once a month, it is crushed, sampled, and transferred to an amalgam barrel.

Amalgamation.

Half a ton of slag is the maximum load taken by the amalgam barrel reserved for slag amalgamation. The slag

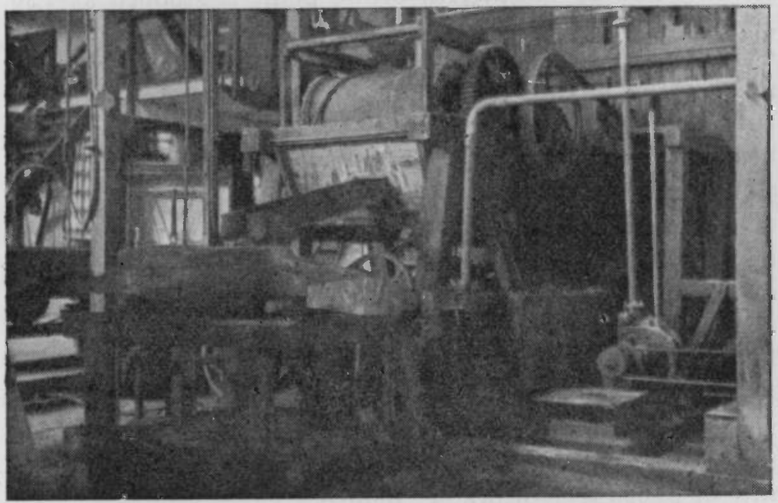


Fig. 15. Amalgam Barrel Batea and Pump, with Connecting Launders.

is charged into the barrel through the manhole in the top, sufficient water is run in to make a thin pulp after grinding, then 30 lb. of mercury and about 30 lb. of slaked lime are added. This amount of mercury is enough to ensure that the resultant amalgam will remain fluid and not form hard lumps that might not separate easily from the rest of the pulp after amalgamation. One barrel of slag usually yields about 100 oz. of amalgam after pressing, but the amount of mercury added could cope with 300 oz. and still fulfil the above-mentioned condition.

Billets of cast iron, 3½ in. dia. by 10 in. long, 60 in number, are fed in to grind the slag and work the gold into the mercury. The barrel is then closed up without further additions, and rotated for a period of two days. At the end of this time, the barrel is stopped, with the manhole on top, and the 2 in. outlet plug on the under side.

An amalgamator's enamel-lined bucket containing water, and about 20 lb. of retorted mercury, is placed centrally in a 16 in. panning-off dish, and the assembly set up immediately below the outlet plug.

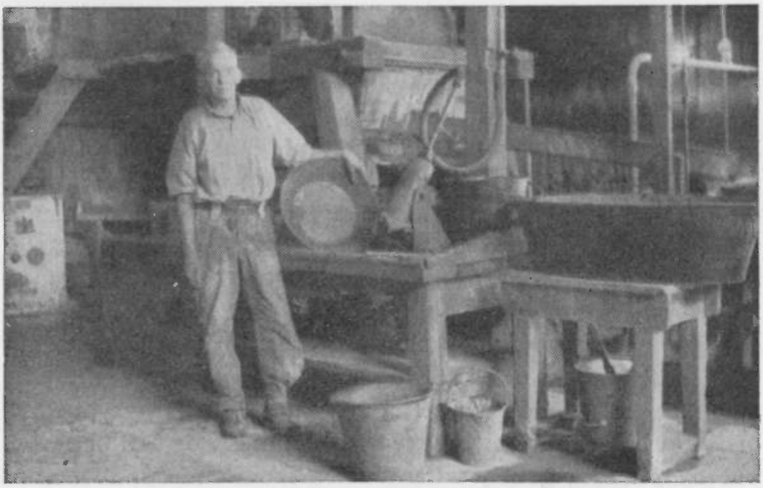


Fig. 16. Amalgamator beside his Panning Table, showing Equipment he uses for Separating Amalgam from Sand.

The amalgamator's mate opens the manhole and places a wooden plug on the end of an iron rod in the plug hole, so that he can regulate the rate of outflow when the amalgamator carefully unscrews the plug. While the barrel empties, the amalgamator lifts out any coarse material from the surface of the mercury and transfers it to another bucket. Fines overflow the bucket and panning-off dish, and run down a launder into a batea, thence to a pump which delivers into an agitator reserved specifically for cyanidation of slag residues.

Liquid amalgam in the enamel-lined bucket is cleaned off with a sponge, using fresh water to wash the surface clean from any particles of slag. The sponge is wrung out into the second bucket containing coarse particles. When the amalgam has been thoroughly cleaned, it is poured into a small amalgam press, and left there until the remainder of the amalgam has been separated from the second bucket. This is done by washing a portion into the panning-off dish, and panning until the mercury has collected at the bottom. Breaking the surface at the right place discloses a globule of mercury, which is then poured into a small enamel-lined bowl. Sand left in the dish is washed into a wooden trough over which the amalgamator works. This material is returned later to the barrel for further amalgamation.

Panning-off is continued until all the contents of the bucket have been treated. Liquid amalgam that is collected in the bowl is further cleaned by pouring into another bowl and washing the first in a tub of clean water. Five such washings from bowl to bowl are usually sufficient, when the amalgam is poured into the press with the main amount recovered earlier.

The amalgam press consists of an iron pipe about 4 in. dia., bored with a slight taper at the filtering end, so that the hard amalgam, after pressing, will either fall out of the press into a bowl placed in readiness, or can be eased out by pushing gently with a stick from the opposite end. Nine pieces of fine calico cut to shape and formed into a pad about $\frac{1}{8}$ in. thick, are used as the filtering medium and are held in position by a removable plate bored with 140, $\frac{1}{16}$in. holes. Two thumb screws hold the plate to the cylinder. The press is mounted on a copper table coated with mercury, which will take up any spillage of amalgam or mercury, above a funnel let into the table. Compressed air is supplied at 50 lb. or more, pressure to the top of the press through a rubber hose that is screwed to the press with a nut and tail having extended wings.

When the air is turned on, mercury filters through at a fairly fast rate into the funnel and is collected in an enamel-lined bucket below. Amalgam remains in the press.

On completion of pressing, the air is turned off, the press is tilted forwards away from the funnel so that when the plate and filter pad are taken off, the amalgam drops into an enamel bowl placed to receive it. Tilting the press is possible because it swings centrally between the two supporting plates, and can be locked in any position in a vertical plane.

Finally, the amalgam is weighed, using a counterweight for the bowl, on a pair of gold scales used for all rough weighing of bullion or amalgam, then it is transferred to the safe to be retorted later.

Amalgam weighing about 100 oz. per half-ton of slag, usually contains about 20% of fine gold.

Flotation Strakes.

It will be recalled that in the general flow sheet, strakes covered with corduroy cloths are used to recover gold liberated during fine grinding of ore by ball and tube mills. This gold is recovered by amalgamation in much the same way as the slag treatment after granulation.

Cloths are changed once per shift. They are folded inwards so that no pulp or gold can escape, and are carried in ordinary buckets to a special two-storey room for washing. In the centre of the upper floor is mounted a square tank with a square cone bottom terminating in a 3 in. pipe fitted with a plug cock. Directly below the pipe is a small amalgam barrel which receives the washings from the cloths.

Before washing begins, the tank is filled with fresh water, which may be warmed in winter for comfort, to within about a foot of the top. The tank has a removable wooden grid, made in two sections, that rests on angle irons welded to the inside of the tank, 2 ft. from the top. This supports the cloths during washing and prevents their being dropped into the cone.

The amalgamator and his mate wash each cloth once in the tank, then throw it over a wooden horse to drain into a wooden trough. This work is all done under supervision.

When all the cloths have been washed, the floor of the room is mopped by hand with sponges, using a bucket of water to clean the sponges, and the buckets are then emptied into the tank.

Pulp in the tank is allowed to settle for a few minutes, after which a drain cock, set in the side of the tank, is opened to a pump to drain off excess water. . The offtake for this drain line is welded to the tank at a point which allows all the contents, plus wash water subsequently used, just to fill the amalgam barrel without overflowing it.

While water is draining off, the amalgamator removes the manhole from the barrel, feeds in ten 5 in. steel balls, followed by 30 lb. of retorted mercury and about 30 lb. of slaked lime. This quantity of mercury is sufficient to cope with 350 to 450 oz. of pressed amalgam. The amount normally obtained on present tonnage is 90 oz. per day, which contains about 40% of fine gold.

The amalgamator then opens the 3 in. plug cock and his mate, on the floor above, breaks a hole through the sand with a piece of ½ in. rod, so that the contents of the washing tank run through into the barrel. The tank is carefully hosed out, using a minimum amount of water.

Later in the day, the amalgamator's mate returns, fills the tank again with water and rewashes the cloths. The washings are left in the tank until next morning, when the whole procedure is repeated.

After closing the manhole of the barrel, it is set in motion and rotated for about four hours. Cleaning out the flotation barrel is done in much the same way as the slag barrel, the pulp overflowing the bucket and panning-off dish directly to a Wilfley pump that delivers to the batea, thence to the flotation concentrates thickener. Recovery of amalgam from the grit and sand in the bucket is done in the gold-room by the amalgamator, the final amalgam being weighed and set aside in the safe for retorting.

Roasted Concentrate Strakes.

Concentrates from the roasters are pulped with fresh water, and pass over a set of strakes covered with long corduroy cloths, before being filtered and repulped with cyanide solution. Because fresh water and not cyanide solution is used on this section, there is seldom any trouble from cloths becoming coated with a hard gypsum layer. They are therefore comparatively easily washed in a wooden vat in the gold-room.

Washings are transferred with a bucket to a third amalgam barrel, to which 30 lb. of mercury and about 30 lb. of soda ash are added. Lime is not suitable for this class of material; caustic soda is the most effective alkali to use, but soda ash is chosen because it is less difficult to handle than the caustic. Alkali has the effect of cleaning dirty gold particles and hastening their amalgamation.

Mercury used is sufficient to cope with 200 oz. of pressed amalgam. The amount usually obtained at present is 150 oz. daily, containing about 30% of fine gold before retorting.

An important factor, not yet mentioned, that applies to all types of material in amalgamation, is that grease or oil must be prevented from entering the barrel. It causes the mercury to "flour" and in extreme cases may result in all the mercury being so finely divided that none settles in the bucket when draining the barrel, but overflows and is carried away to a point in the plant where it cannot be conveniently recovered.

Retorting of Amalgam.

Amalgam accumulated from the three sources is retorted twice a week. The weight of each bowl is checked against the weight recorded in an amalgam book when it was weighed before going into the safe, then placed in a rectangular-shaped mould according to its source. This separation is made as a check on the value of gold obtained from each set of strakes daily, so that the value can be entered on the metallurgical record sheet by calculation, in advance of the retorted results. Minor corrections to the sheet can be made weekly so that the final monthly figure recorded is correct.

Fig. 17. Lifting Retorted Amalgam from Retort Furnace.

Trays containing the amalgam are loaded into a special retort, the door is screwed into position, and oil turned on the burner. The retort is slowly brought up to a red heat. Mercury volatilises from the amalgam after about two hours' heating, and is condensed in a water-cooled coil, dropping into an amalgamator's enamel-lined bucket that has been weighed previously. When no more mercury distills, usually about three hours after lighting the furnace, the bucket is weighed and the net weight of mercury obtained is recorded. This weight, added to the weight of gold obtained from the retort, should equal the total weight of amalgam placed in the furnace. Shortages indicate cracks in the retort, which must then be replaced.

Retorted gold in the trays is lifted out with tongs, transferred to tared tin dishes, and weighed according to source. It is then ready for barring, the procedure for which is much the same as described previously, except that the cover used is borax only.

Bulked amalgam gold from the two sets of strakes usually averages:—

Gold	955
Silver	40
Base metal	5

It is barred separately from all other bullion, and is termed an amalgam bar.

PART II.

RECOVERY OF METALS FROM MATTES AND SLAGS.

INTRODUCTION.

Prior to 1944, mattes and slags, which were the end products of the matte smelting process described in Part I, were sent to Port Kembla for final recovery of their metal content. Approximately 15 tons of slag and eight tons of matte used to be shipped yearly. Shortage of shipping and rising charges for treatment of these products since the war began, caused the decision to be made to attempt their further treatment in the gold-room.

Slags were first considered, and an alternative to the normal method involving the use of mercury, which was in short supply, was sought. Although no alternative to amalgamation was found, ordinary cyanidation after amalgamation obviated the need for sending them to the smelter.

Matte treatment proved to be a much more complex and difficult problem. Much experimentation was carried out on what has been termed the "Lead-bath" method, in which lead is added to the matte. The two materials are melted together, the lead forming a layer below, and the matte is left in contact with the lead bath for some time in order to collect the precious metals. Silver was more difficult to extract in this way than gold, but for both metals, the number of contacts necessary to give best results made further investigations desirable.

Reduction of lead and copper from the matte itself appeared to be the most likely avenue. Small scale tests were, therefore, made along these lines, and when a promising result was obtained, experimentation was carried out on a larger scale in the gold-room. Eventually a method using soda ash was evolved that recovered nearly all the gold and silver from secondary matte in a lead-copper base bullion, which, by cupellation, yielded a precious metal bullion suitable for sending to the Mint.

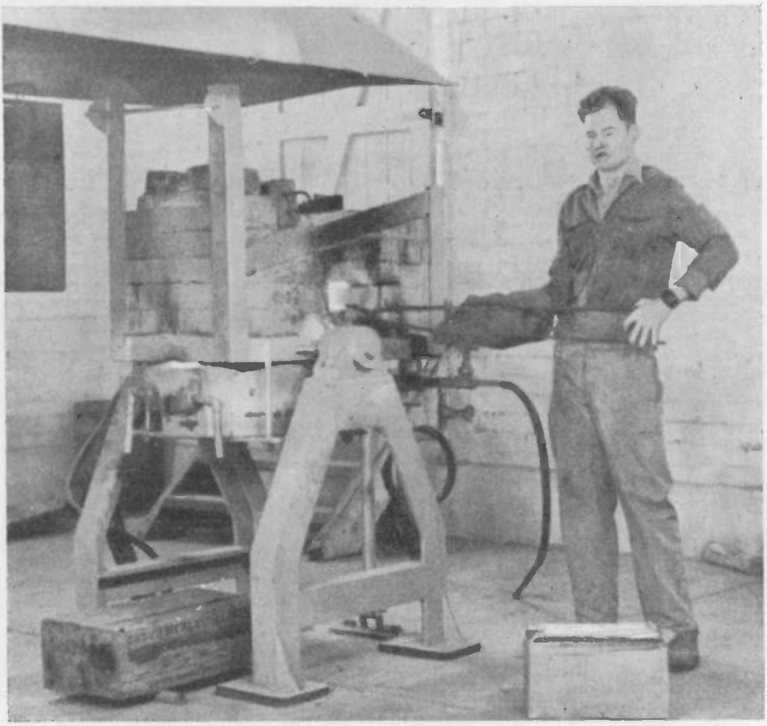


Fig. 18. Loading the Cupellation Furnace.

Products from cupellation were then subjected to investigation, and a process was developed for recovering all metals of value from them.

The reduction of secondary matte, with its auxiliary recovery processes, has been adopted as standard gold-room practice since January, 1944, and is likely to be in use until it is possible to smelt treated zinc slimes without production of matte.

In the report that follows, an account is first given of the various products of the matte smelt, in order to familiarise the reader with their composition and characteristics, before proceeding to the investigation work done on the recovery of their metal contents.

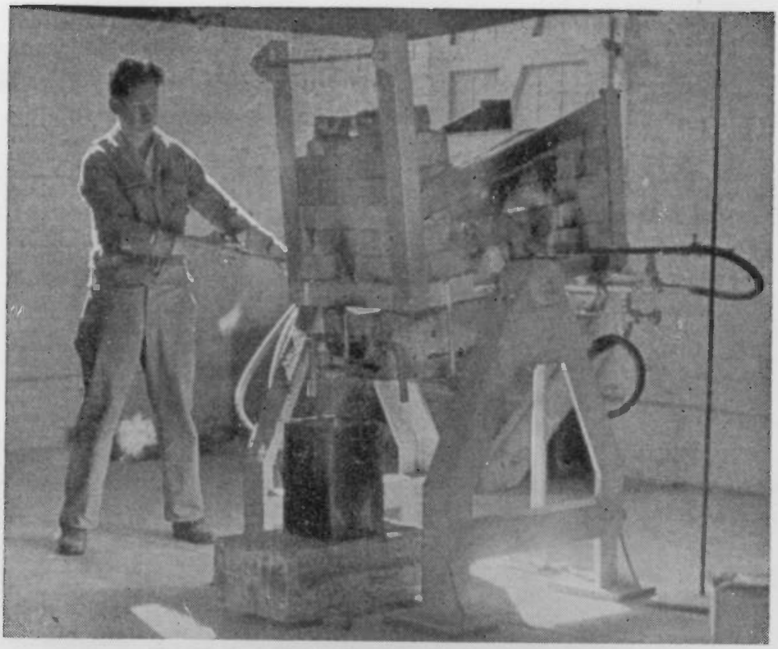


Fig. 19. Pouring off Oxides from the Cupellation Furnace.

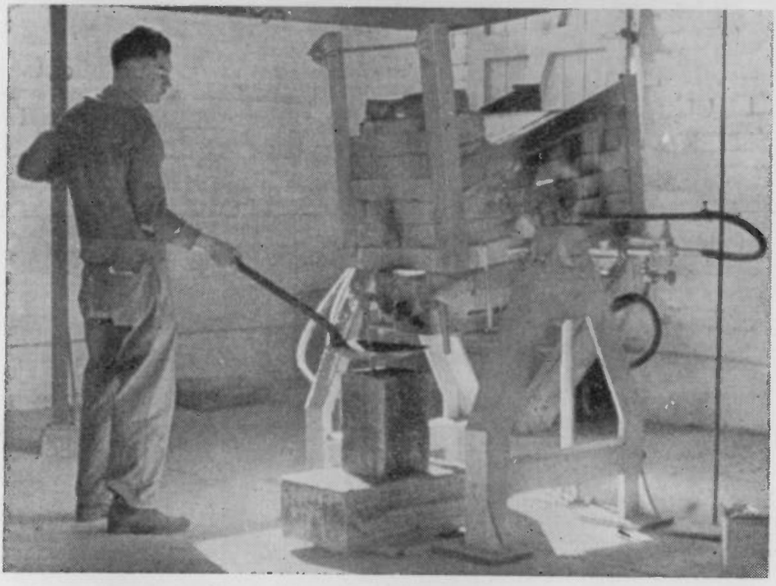


Fig. 20. Testing Oxides to make sure Metal is not also Overflowing.

GOLD-ROOM BY-PRODUCTS.

Slimes smelted by the matte process before 1944, yielded the following products:—

Primary slag:
  From the initial smelt.

Primary matte slag:
  From the first remelt of primary matte with 20% soda ash.

Secondary matte slag:
  From the remelt of secondary matte with 20% soda ash.

Primary matte:
  From the initial smelt.

Secondary matte:
  By remelting primary matte with 20% soda ash.

Tertiary matte:
  By remelting secondary matte with 20% soda ash.

Fig. 12 makes clear the source of these several products. Two of them, secondary matte slag and tertiary matte, are no longer made, because the new recovery process operates on secondary matte. They will be described, however.

Slags.

Preliminary trials for the recovery of gold from slags included:—

Remelting the slag, with or without added fluxes;

Grinding followed by straking and cyanidation of the tailings, with amalgamation of the strake concentrates;

Dry crushing followed by dryblowing twice, with smelting of the riffle concentrates;

Amalgamation of the whole of the slag in an amalgam barrel, then cyanidation of the resultant finely ground slag. The last method gave most satisfactory results.

Commencing from 7th April, 1943, a complete record of slag treatment has been kept, which includes the assay value of a head sample after each smelt and a calculated head value from the gold recovered by subsequent treatment.

Head values for the period 22nd September, 1943, to 30th May, 1944, are given in Table 2; figures are in oz. per short ton.

Period	Sample Granulated Slag	Sample Crushed Slag	Calculated Head Value	Recovery by Amalgamation Per Cent.
22.9.43 to 19.10.43	33.5	39.3	34.3	71.0
20.10.43 to 16.11.43	18.4	32.0	27.0	81.0
17.11.43 to 14.12.43	17.6	25.1	23.8	74.8
12.1.44 to 8.2.44	18.8	38.4	34.4	68.8
8.2.44 to 7.3.44	16.2	18.1	16.2	77.8
8.3.44 to 4.4.44	33.4	69.4	61.4	47.6
5.4.44 to 2.5.44	27.8	78.7	60.4	64.7
3.5.44 to 30.5.44	12.4	37.8	27.7	71.0
Averages	22.3	42.4	35.7	

Table 2. Chaffers Plant. Head Values of Slags in oz. per Short Ton and Recovery by Amalgamation.

The calculated head value is taken as the most nearly accurate, because it is based on actual gold recovered by amalgamation, and on the assay of finely ground slag, which should contain very little coarse gold, before and after cyanidation. The assays of granulated slag, and of slag dipped from the slag mould while still molten, as shown by previous work not reported here, are quite unreliable. A sample from crushed slag, however, checks fairly closely with the calculated head value and could be used as a guide if the calculated head value from actual treatment were not available.

Table 3 gives figures for the Chaffers Retreatment Plant slags.

Period	Sample Granulated Slag	Sample Crushed Slag	Calculated Head Value	Recovery by Amalgamation Per Cent.
22.9.43 to 19.10.43	25.5	48.5	48.7	93.7
20.10.43 to 16.11.43	29.0	36.7	38.1	93.6
17.11.43 to 14.12.43	26.4	36.1	38.5	91.8
15.12.43 to 11. 1.44	26.4	61.7	35.9	91.8
12. 1.44 to 8. 2.44	44.6	64.8	48.0	93.0
9. 2 44 to 7. 3.44	44.1	54.9	40.3	90.0
8. 3.44 to 4. 4.44	50.0	67.1	57.9	92.1
5. 4.44 to 2. 5.44	42.7	79.8	78.8	93.8
3. 5.44 to 30.5.44	32.0	43.4	39.4	92.8
Averages	35.6	49.3	42.5	

Table 3. Chaffers Retreatment. Head Values of Slags in oz. per Short Ton and Recovery by Amalgamation.

The percentage of gold recovered by amalgamation is given in both tables. The fine oz. of gold in the amalgam is computed with reasonable accuracy from the assay of the composite bar into which it is placed, or if this cannot be done, the retorted gold is melted, poured into a button, and sampled.

After amalgamation, a head sample of the finely ground slag is taken for assay. It is then cyanided in an agitator specifically reserved for this material, for eight days. At the end of this time, a sample is taken, filtered, washed and assayed. If the residue contains 0.5 oz. gold, 8.0 oz. silver per ton, or thereabouts, cyanidation is considered complete and the agitator is discharged through the filtering and precipitation plant with the current roasted concentrates in the main treatment plant.

Cyanide recovery is about 93%, making the overall recovery 99.1% of the gold in the slag from the furnaces. The overall silver recovery from slag is 83%. Chaffers retreatment slags give a recovery by amalgamation of about 92%, with an overall recovery of 99.6%.

Primary matte slag is treated concurrently with primary slag. It contains much less gold and silver than primary slag, and presents no problems when treated by the above method.

Mattes.

Primary mattes contain roughly 5% of gold and 5% of silver. On remelting with 15% of soda ash, the yield in bullion is approximately 0.6 oz. of bullion per lb. of matte.

Secondary matte carries, on an average, 350 oz. of gold and 2,000 oz. of silver per long ton. Bullion from remelting secondary matte was produced at a rate of approximately 0.6 to 0.7 oz. per lb. of secondary matte. Its grade was very low, and even after granulation and sulphuring with 30% of sulphur, assayed only:—

Gold	680 to 560
Silver	180 to 250

It was, nevertheless, barred without further refining. Matte from sulphuring of secondary matte bullion was treated with the secondary matte for the next month.

The residual tertiary matte, when it was being produced, assayed 30 to 50 oz. of gold and 2,200 oz. silver per long ton.

It will be observed that successive remelts with soda ash reduced the concentration of gold in the matte, but not the silver.

LEAD BATH TREATMENT OF MATTE.

Experimental work, first in the laboratory, using small test quantities, and later in the gold-room, on a plant scale, was carried out by A. C. McDonald, of the Experimental and Research Laboratory, Western Australian School of Mines, in 1943. Extracts from his report (private communication) will be given in this section because of their interest.

The lead bath method was not adopted because work on the reduction of lead and copper from the matte itself was proceeding concurrently with the experimental work by A. C. McDonald, and resulted in a method, involving less smelting, that extracted almost all the silver as well as the gold from the matte. The lead bath method was not successful in this respect.

The programme of work carried out by A. C. McDonald, and summary of results obtained on the lead bath method, follow:—

Programme.

"The programme of work in the gold-room was arranged:—

1. To check the possibility of recovering gold from any one of the several products produced during the smelting treatment of the calcined Merrill slime by remelting any one of these mattes and/or slags and holding it in contact with lead and/or antimonial lead while both are in a liquidus condition, the work to be carried out as a check on the results obtained on the laboratory scale.

2. To develop a method whereby a continuity of treatment might be obtained where the resultant gold-lead, and/or gold-antimonial lead bullion would be concentrated to a comparatively high degree prior to cupellation, and at the same time obtain a reject matte sufficiently low in gold value to discard.

3. To determine the interdependency of:—
    (a) Ratio of weight of lead bath and weight of matte treated in a batch treatment.
    (b) State of equilibrium of gold values in both the conjugate solutions, i.e., matte and lead bath.
    (c) Above (b), with varying gold values in the matte prior to treatment.
    (d) Above (b), with varying periods of contact from 15 minutes to 120 minutes.
    (e) Above (b), with a varying range of temperatures of the two conjugate solutions.
    (f) Other aspects of the process.

Unfortunately, the programme of work was far from being completed at the end of the five-week period."

Summary.

"The preliminary investigation carried out for the recovery of the gold from the matte compounds produced in the Lake View and Star gold-room, indicates a successful treatment for these products.

The laboratory work herein described has shown several interesting features, the main one being the stable property of the matte compounds when heated. It would appear that no sulphur can be removed with ordinary roasting conditions, and furthermore, no sulphur is given off, even when the compound is in a liquidus condition. Possibly only by blowing air through the liquid mass could the sulphur be removed.

From earlier experiences of the writer, it has been found that by bringing a gold-bearing material in contact (when liquid) with a bath of molten lead, antimony, antimony-lead, or copper, the gold transferred from the material to the metal bath. Several features controlled the transfer of the gold, viz.:—

(a) Temperature (which in turn controls the degree of liquidity of the two conjoining liquids).
(b) Time of contact.

(c) Ratio of gold in material and gold in the metal bath, i.e., the degree of gold concentration in the metal bath.

(d) Depth of gold bearing material (liquid condition) in the metal bath. (This has a direct bearing on the time of contact. Tests have been carried out where the liquid gold bearing material was 18 in. thick above the metal bath, but good results were obtained after 16 to 20 hours' contact.)

The matte compounds were found to have a melting point from 610°C. to 650°C. and were quite liquid at 800°C. Consequently this low melting point is a distinct advantage when applying the lead bath treatment for the recovery of gold and silver from the matte compounds.

Tests were carried out using Salamander crucibles, and the results indicated that very little corrosion took place. The treatment of the secondary mattes should be more favourable in Salamander crucibles since less sodium salts are present. The sodium salts have the tendency to react with the silica from the crucible mixture to form sodium silicate.

The following tests were carried out to determine the efficiency of the lead bath as a collector for the gold and silver contained in the high value matte compounds, since a considerable amount of work has been carried out by the writer with lower value materials.

The following tests were carried out in the laboratory:—

Test No. 2 was made on a tertiary matte; the approximate value of this matte being 19.0 oz. gold per long ton. The sample was submitted to four fusion treatments, being in contact with a new bath of lead for each treatment; that is, as the value decreased after each fusion, the reject matte was held in contact with a quantity of new low value lead.

Subsequent to the first fusion, the gold in the matte had been reduced from 19.0 to 6.2 oz. gold per ton, and the lead bullion concentrated to 30.83 oz. gold per ton. After the fourth fusion the gold in the matte was down to 0.26 oz. gold per ton and the lead contacting this matte contained approximately 1.0 oz. gold per ton.

Test No. 3 was made on a secondary matte with an estimated value of 510.6 oz. gold per ton. The method of using a new lead bath after each fusion was also used in this test and the matte was submitted to three fusions.

After the first fusion the gold in the matte had been reduced from 510.6 oz. gold per ton to 3.77 oz. gold per ton; subsequent to the third fusion the gold in the reject matte was down to 0.14 oz. gold per ton and the lead bullion contacting this matte contained 1.4 oz. gold per ton.

Both these tests were carried out for a 120 minute period of contact.

Further tests were carried out using clay crucibles which in turn showed very little corrosion with repeated usage.

Test No. 4 was made on a secondary matte, making a number of short interval fusions, i.e., eight fusions of 15 minutes, and using new lead for each fusion. After the first fusion the matte had dropped in value from 193.3 oz. gold per ton to 110.7 oz. gold per ton. Each successive treatment produced a lower value reject matte. The eighth and last treatment gave a reject matte having a gold value of 0.2 oz. gold per ton and the lead in contact with this matte assayed 1.9 oz. gold per ton.

Test No. 5 was similar to the fourth test except that the period of contact was 30 minutes in lieu of the 15 minutes, the matte being submitted to six fusions, and the results indicated that at least one more fusion was required. The gold in the matte was down to 1.9 oz. per ton, with the gold in the lead 16.3 oz. per ton.

From the above series of tests it has been established that the gold concentration in the lead bath for the final clean-up must not exceed a figure of approximately 2 oz. gold per ton of lead. With this concentration, the gold in the reject matte should be below 0.2 oz. gold per ton.

Test No. 6 was similar again to the previous test, and after the six fusions of 60 minute periods of contact the final matte assayed 0.9 oz. gold per ton with the lead bullion at 3.3 oz. gold per ton, which further confirms the above feature.

Test No. 7 showed that with the period extended to a 90 minute interval, three fusions were required to reduce the value to 0.5 oz. gold per ton in the reject matte, the lead bullion assaying 3.3 oz. gold per ton.

Test No. 8 was abandoned owing to the crucible cracking and the loss of the sample. It was to have been repeated. The time of contact was for three hours.

Test No. 9 was carried out to determine the extent to which the gold could be concentrated in the lead bullion, and also the collecting property of the high grade lead-gold bullion as a collecting agent. A secondary matte was used in this test and the same bullion used for six fusions with a new sample of secondary matte for each fusion.

The test indicated that as the gold became more concentrated in the lead-gold bullion, so the collecting power of the lead-gold bullion decreased, although with the bullion concentrated up to 1,700 oz. gold per ton, it had the power of collecting approximately 39% of the gold contained in the secondary matte (188.5 oz. gold per ton).

Test No. 10 was carried out on similar lines to the previous test, but a sample of primary matte was used, the estimated value being 1,066 oz. gold per ton.

The result of this test indicated a similar feature to Test No. 9, but with a much higher range of gold concentrations in the bullions. With the bullion concentrated to 15,601.7 oz, gold per ton it had the power to collect 58.3% of the gold from the sample of primary matte.

The last two tests were carried out using storage battery lead, which contains approximately 5% of antimony. At this stage the work was transferred to the gold-room. The antimonial lead recovered from the storage batteries was used for the metal bath.

It may be mentioned that the main consideration in this preliminary investigation was the recovery of the gold, as expeditiously as possible, from the matte compounds, the behaviour of the silver being under observation to a lesser degree. At some future time the copper contained in the matte should receive some attention, whereby it may possibly be removed by a leaching process subsequent to a fusion treatment, the solution yielding copper sulphate suitable for re-use in the treatment plant.

The results of the treatment on a larger scale in the gold-room confirmed the laboratory testing in every respect.

It has been established that the gold contained in the resultant mattes from the process is interdependent with the gold content of the metal bath, i.e., a definite state of gold and silver equilibrium exists between the two conjugate solutions. Possibly the copper enters this phase as well, but the behaviour of the copper was not considered at this stage.

The behaviour of the gold was examined with the least number of variables; consequently in this preliminary work the following features were held as nearly constant as possible, but each in turn requires further examination.

These features were:—

1. Ratio of weight of metal bath to weight of matte being treated. In all of these tests the ratio at the beginning of the test was one of matte to one of metal.
2. Temperature of conjugate solutions. (In the plant testing it was aimed to maintain a temperature of 900°C.)
3. Time of contact. (Plant tests were for 90 minutes of contact in most instances.)
4. Thickness of layer of top solution (matte) on the metal bath.

The behaviour of the silver and copper subsequent to the removal of the gold also requires investigating with conditions varying as follows:—

(a) Increased time of contact.
(b) Retreatment with a low value metal bath.
(c) Introducing a reducing agent with the charge, e.g., powdered coal, charcoal or sawdust.
(d) Introducing an oxidising reagent with the charge, e.g., manganese dioxide.

(c) and (d) have been suggested since it is fairly evident that the silver is present in solid solution, or as a chemical such as a sulphide or sulphate.

It is interesting to note that in one test an attempt was made to operate the de-golding process on a continuous basis, but the unit proved to be too small. However, an interesting phenomenon showed up with the silver.

The matte compound was one which had been treated on previous occasions, the gold and silver being reduced to 3.0 oz. gold per ton and 322.4 oz. silver per ton.

During the retreatment of a charge, No. 3, of this material, the furnace was using a direct fire, that is, the flame contacting the charge, and a low value metal bath was being used. The values were reduced to 0.05 oz. gold per ton and 22.0 oz. silver per ton.

These results have established a phenomenon which requires further investigation as outlined above, whereby the silver may be released from the matte compounds as well as the gold, and collected by the metal bath.

The following conditions may have had some effect on this result:—

1. Direct firing of the charge.
2. Increased ratio of metal bath to matte compound. (75 lb. lead bullion, 15 lb. matte.)
3. Increased time of contact with a low value metal bath. (Fired for a further 60 minutes.)
4. Thickness of liquid matte layer conjoining the metal bath, together with one or other of the above features. (In the above test, the layer of 15 lb. matte was not much greater than $\frac{5}{8}$ to $\frac{3}{4}$ in. thickness.)

One other important feature, which may have some bearing on the resultant programme of treatment, is that the final bullion for cupellation should have the gold concentrated to some high figure. Consequently a series of tests was made.

In this set-up, secondary matte added as three separate charges of 100 lb., 100 lb. and 96 lb. to 103 lb. of bullion, gave reject mattes of approximately 20.0 oz. gold per ton and 3.519 oz. silver per ton. This bullion was then used for running down with a primary matte sample. The residual matte from this treatment gave a value of 44.0 oz. gold per ton and 1,496 oz. silver per ton, and the bullion was further concentrated to 2,157.2 oz. gold per ton and 3,862 oz. silver per ton.

These several parcels of reject mattes were then retreated. The results show that subsequent to this treatment the reject matte from all of the above parcels averaged approximately 4.0 oz. gold per ton, whilst the bullion assayed 320.4 oz. gold per ton and 2,412.4 oz. silver per ton.

With the cyclic treatment, this bullion would be re-used with new secondary matte, and the reject matte, i.e., 389 lb. with a value of 4.0 oz. gold per ton, would be retreated a third time with new low value lead.

Of course these results were obtained with one set of conditions. With increased time of contact the gold and silver values may be reduced to much lower figures in the one fusion.

In the gold-room work, an interesting feature to note is the behaviour of the silver. When using 103 lb. of (value unknown) low-gold, silver-lead bullion, the first reject matte assayed 15.5 oz. gold per ton and 1,019.5 oz. silver per ton. The second reject matte assayed 24.4 oz. gold and 1,410.6 oz. silver per ton, and the third reject matte assayed 21.6 oz. gold and 1,763.4 oz. silver per ton.

The interesting feature of this test is that, although the gold content of the bullion increased to 1,161.0 oz. gold per ton, the gold in each of the reject mattes remained at about the same 20.0 oz. mark, although the silver value increased progressively in the reject mattes, e.g., as the silver concentrated in the bullion from an unknown figure to 2,981.8 oz. to 3,591.0 oz. silver per ton, the silver in the corresponding mattes varied from 1,019.5 oz. to 1,410.6 oz., to 1,763.4 oz. silver per ton.

A further interesting feature shows up in the second treatment of these reject matte compounds in this work. In this treatment new lead was used for the first fusion, the resultant bullion from each fusion being used for the next. The values showed up as follows:—

Gold	Silver
No. 1 fusion from 15.5 oz. to 3.0 oz. per ton	From 1,019.5 oz. to 407.0 oz.
No. 2 fusion from 24.4 oz. to 3.5 oz. per ton	From 1,410.6 oz. to 802.5 oz.
No. 3 fusion from 21.6 oz. to 4.5 oz. per ton	From 1,763.4 oz. to 1,152.2 oz.

The above shows that the collecting power of the bullion for the silver was fairly constant, although the bullion progressively became concentrated with both gold and silver. The amount of silver removed during each fusion was approximately 610 oz. silver per ton.

Since the first fusion was made with 87 lb. lead and 76 lb. matte, and 612 oz. silver per ton was collected, it may be reasonable to assume that the 600 oz. silver per ton is the amount that can be collected during the 90 minute time of contact. Further tests should be made with increased periods of contact."

REDUCTION OF MATTES TO METALS.

Although the lead bath method showed promise of good gold recoveries from the matte, given sufficient time to contact in a cyclic treatment, using low gold-value lead in the last cycle, the method did not accomplish the chief objective, namely, the recovery of all the metal values economically.

Reduction of Matte to Base Bullion.

Reduction of lead from the matte itself appeared to be the most likely avenue for further investigation. A series of tests was therefore made, adding several different reducing agents to a matte charge to produce a fall of lead.

Preliminary tests on a small scale were made on secondary and tertiary mattes to determine the best starting point for any process developed. On the results, secondary matte appeared the better of the two for plant practice, but there was an accumulation of tertiary matte already on hand that had also to be satisfactorily treated. Consequently a set of laboratory tests on each type of matte was made.

All tests were carried out in Salamander crucibles so that large scale tests in the gold-room, using No. 9 Salamander retorts or silicon-carbide lined Wabi furnaces, could be expected to follow the small scale tests closely. The results are tabulated for secondary matte in Table 4 and for tertiary matte in Table 5.

Deductions made from these tests are summarised as follows:—

1. Melting the matte with 50% by weight of soda ash, with added reducing agent, gave very good recoveries of gold and silver in the base bullion produced, particularly with secondary matte.
2. Time of contact with the soda, after fusion was complete, was important. This was appreciated only after a considerable number of tests had been made. Insufficient time after fusion accounts for the poor silver recoveries reported in the tests on tertiary matte, which were conducted first.
3. In all tests where the silver recovery was good, a large fall of base bullion was recorded, whereas gold recoveries were not affected so much by a reduction in bullion weight.

Test No.	Wt. of Matte Taken	Fluxes & Weight	Wt. of Button	Wt. of Matte (or slag)	Wt of Au & Ag in Button		Slag Assays (ozs/L.T.)		Equiv in Mgms in Slag		Calcd Head Value		Recovery in Button		Remarks
					Au mgms.	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	
	gms.	gms	gms.	gms.											
Preliminary	100	Soda 20	13.5	74	128.2	2860	1.40	1092.4	3.17	2465	41.4	1680	97.7	53.8	Old Graphite Crucible used Good fusion-residue looked like all slag. No matte Note big loss of weight.
1	100	Soda 20	3.6	80.0	112.0	791.0	5.35	2165.15	13.0	5285	39.4	1920	89.7	13.0	Good fusion mostly matte left
2	200	Soda 70	3.9	226.0	194.0	811.0	6.60	1484.4	45.6	10260	37.8	1742	81.1	7.33	Good fusion
3	200	Soda 150	10.1	Matte 156 Slag 134	Button	Lost	4.80 1.44	2042.2 215.56	22.9 5.9	9720 880	-	-	-	-	Good fusion-soda slag & matte separated & assayed separately
4	100	Iron 7	3.6	100	102.0	702.0	3.38	1817.12	10.3	5570	35.5	1980	90.8	11.2	Good fusion
5	100	Litharge 40 Flour 5	27.6	94	121.0	2048	1.70	1415.0	4.9	4060	39.8	1925	96.1	33.6	Good fusion soft matte produced. No slag apparent
6	100	Litharge 75 Flour 10 Borax 50	49.6	Slag 57 Matte 97	117.6	2546.4	0.64 1.7	91.66 1222.8	1.11 5.04	160 3620	39.2	2000	95.0	40.2	Good fusion
7	100	Cyanide 50 Borax 50	9.0	140.0	122.0	2553.0	1.5	492.7	6.4	2100	40.5	1472	95.0	54.8	All fluxes mixed then 50 gms borax added as cover Poor fusion-not molten just a frothy mass
8	200	Cyanide 60 Borax 60	11.1	240	221.0	2420	1.0	1250.0	7.35	9150	36.1	1822	97.0	21.0	Fluxes & matte in layers in pot. Mushy result not a fuse
9	100	Cyanide 100 Borax 50	16.5	229.0	104.0	4414.0	1.0	266.0	7.0	1862	35.0	1985	93.6	70.3	Fluxes mixed-"mushy" fusion
10	100	Cyanide 100 Borax 50	8.0	226.0	113.4	2158.6	0.40	565.60	2.77	3920	36.8	1925	97.5	35.5	Repeat of (9) but run at Low Temp. Max.650°c.

Table 4. Tertiary Matte Retreatment using Salamander Crucible.

Sample (Matte No. 6) for period 12/1/44 produced by normal Soda (20%) treatment in Gold Room. This sample cut out and crushed for assay

52.9 ozs./L.T. Gold
2184.1 ozs./L.T. Silver

(N.B.—This is higher than normal in Gold)

Test No.	Wt. of Matte taken (gms)	Fluxes and Weight (gms)	Wt. of Button gms.	Wt of Matte or slag. gms.	Wt. of Au & Ag in Button (mgms)		Slag (or Matte) Assay ozs/L.T.		Equiv in mgms in slag		Calcd Head Value		Recov. in Button		Remarks.
					Au	Ag	Au	Ag	Au	Ag	Au ozs/L.T.	Ag	Au on calcd values	Ag	
1	100	Soda 20	8.2	Slag 29.5 Matte 67.4	809.0	3083.0	1.66 18.6	168.14 1982.4	1.49 38.3	152 4090	268	2320	95.4	42.1	Good fusion 1 hour
2	100	Soda 50	33.4	Slag 61.6 Matte 45.3	749.0	4116.0	0.82 1.20	103.02 272.80	1.54 1.67	194.5 381	243	1480	99.4	87.8	Sticky pour 1 hr
3	100	Soda 50	33.5	Slag 46.5 Matte 45.3	705.0	4448.0	0.46 1.50	55.34 245.00	0.65 2.1	78.6 342	224	1540	99.6	91.2	Fair Fusion 1¼ hrs All show Graphite "scoria" on top from pot corrosion
4	100	Soda 50	34	Comb.98	855	4370.0	0.70	194.30	2.1	584	271	1566	99.7	88.1	" " " "
5	100	Soda 50	33	Comb 105	881	4494	0.40	226.10	1.3	725	279	1651	99.8	85.9	" " " "
6	200	Soda 100	64 10	Slag 96(out) Matte 107 Comb 122	1535 Back 2.9	7795 with 697.1	0.20 53.5 0.16	114.34 gms 51.84	0.59 Soda 0.60	336 194	244	1425	99.9	94.1	" " " "
7	200	Soda 100	68 33	Slag 93(out) Matte 102 Slag 42 Matte 86	1705 Back 2.3	9335 with 50 Pb 0, 10 Flour. 20 Borax 415.7	0.20 0.12 0.30	89.00 55.48 182.70	8.57 0.15 0.79	252.0 71 480	270	1670	99.9	92.3	" " " "
8	100	Soda 50 Borax 20	3.9	Slag 77 Matte 72	879	596	2.76 41.50	87.24 2001.0	6.5 91.5	205 4410	309	1648	90.0	11.4	Good pour. Borax appears to slag Lead off as it "falls
9	100	As above	5.1		765	911	Not Assayed								
10	100	Soda 100	40.0	Comb. 128	801	5064	0.20	44 80	0.78	175	254	1656	99.9	96.6	Difficult fusion. Lead very "coppery"
11	100	Soda 100 Borax 10	36.5	Comb. 138	762	4508	0.60	95.40	2.53	403	242	1555	99.7	91.6	Fairly fluid but still difficult fusion
12	100	Soda 100 Borax 20	32.5	Comb. 151	782	5408	0.60	122.40	2.77	561	248	1888	99.6	90.4	" " " " "
13	100	Iron 12.5	17.5	Matte 68 Slag 23.5	855	3570	4.00 0.60	1010.5 123.4	8.3 0.4	2100 88.7	273	1820	99.1	61.9	Fluid Haematite looking Matte Pb "Coppery"

14	100	Iron 31.5	22	Matte 77 Slag 22	624.0	3257	10.10 6.10	756.6 182.9	23.8 4.1	1776 123	206	1630	95.6	63.1	" " " " " " Not all Iron used up
15	100	Iron 12.5	16	Matte 66.5 Slag 28.5	850.0	3115	5.25 0.46	1234.75 98.54	10.65 0.4	86.0 2515	272	1810	98.7	54.4	Very fluid ¼ hr longer in furnace than 13
16	100	KCN-100 Borax 50	10.5	Slag & Matte 239	847.0	2523.0	0.86	324.34	6.3	2370	270	1545	99.3	51.6	After ½ hr low heat fluid: then got mushy
17	100	KCN-50 Borax 50	10.7	Slag & Matte 178	919.0	2492.0	1.16	427.64	6.3	2320	292	1524	99.4	51.7	In furnace ½ hr only Poured fairly well
18	100	Borax 100	No	Lead Button Obtained			Test rejected (see Test 8 & 9)								Very fluid
19	100	PbO 40 Flour 10 Borax 20	41.5	Slag 33 Matte 72	703.0	2888.0	7.52 5.00	125.28 1100.0	7.6 11.0	126.8 2420	228	1718	97.3	53.2	" "
20	100	Assay Slag 250	8.1	Slag 240 Matte 87	548.4	608.5	3.20 104.0	66.8 1870.5	23.8 278.0	491 4990	269	1922	64.3	10.0	Good fusion
21	100	Assay Sl.250 Silica 10 Flour 10	37.25	Slag 230 Matte 87	696.0	2725.3	8.60 25.00	157.0 969.0	30.7 66.5	1100 2580	250	2022	84.6	42.5	" "
22	100	SiO_2 10 Assay Sl 300 Flour 12	27	Slag 283 Matte 81	796.0	2018.0	0.68 10.8	10.56 1376.2	5.89 26.8	91.6 3410	262	1745	96.0	36.5	" "
23	100	As above	35 30	All slag & matte put back with 50 PbO & 5 Flour produced Slag 275 Matte 83	818.0 36.0	2700.0 1137.0	0.20 0.80	9.10 819.2	1.69 2.03	75.5 2080.0	271	1895	99.6	64 0	
24	100	As above	46.5 33.5	Slag 275(out) Matte 81 back with 50 PbO & 5 Flour " 101	857.0 22.0	3523 780.0	0.44 0.50	11.56 554.5	3.7 1.55	97.5 1695	280	1925	99.5	70.8	Both good fluid pours
25	100	Repeat of above test	44.0 42.5	Slag 283(out) Matte 78.5	890.0 28.0	3418 1213	0.46 0.50	16.94 504.5	4.0 1.20	147 1210	292	1890	99.5	77.3	" " " "

Table 5. Secondary Matte Treatment — Using Salamander Crucibles.

Sample of Secondary Matte for period 12/1/44 to 8/2/44. (The main bulk of this was treated in the Gold Room with 20% Soda to give Tertiary Matte No. 6 as used in Series I Tests on Retreatment of Tertiary Matte.)

Assay of Gold Room Tertiary Matte No. 6 = { 52.9 oz./L.T. Gold
2184.1 oz./L.T. Silver

Head Assay of this Sample. 1. 262.5 oz./L.T. Gold 1526.50 oz./L.T. Ag.
2. 259.0 oz./L.T. Gold 1761.65 oz./L.T. Ag.

4. In certain tests where a large lead fall was produced, by using litharge or assay slag, the silver recoveries were not high.
5. On the other hand, those fluxes that produced a large metal fall from the matte itself, gave the best results, particularly when the bullion produced was coppery in nature.
6. Silver was held closely by the copper sulphide portion of the matte.

Some confirmation of (6) was found in an article by Fulton and Goodner[1] on copper-iron and copper-iron-lead mattes. In their conclusions the authors state:—

> "Cuprous sulphide and silver sulphide form an unbroken series of solid solutions and there is little doubt that when cuprous sulphide is present, silver is in solution in this substance as sulphide. It is also a powerful solvent for gold.
>
> Metallic copper, usually present in copper mattes, had a powerful solvent action on both gold and silver. Lead sulphide has a considerable solvent action on silver sulphide but only a limited one on gold. Zinc sulphide has very little solvent action on silver or gold. Metallic iron has a powerful solvent action on gold but only a limited one on silver. Ferrous sulphide has practically no solvent action on silver sulphide or gold."

Confirmatory tests on a larger scale were next made in the gold-room. Table 6 summarises the results of the first 12 of these tests, which will now be discussed in more detail than the table permits.

Test No. 1 was a preliminary test done in a Wabi furnace with Carbolox lining and before the effect of a long contact with soda had been recognised as essential. As will be seen from Table 6, the time of contact was only 1¼ hours, and the recoveries were poor.

Test No	Description of Sample	Charge (lbs)	Furnace Used	Time of Contact	Products & Weights (lbs)			Assay of Products						Fine Ozs in Bullion		Calculated Heads (ozs/long ton)		Recovery %		Remarks
								Bullion (fine-ness)		Matte (ozs/ L.T.)		Slag(ozs/L.T.)								
					Bullion	Matte	Slag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	
1	Secondary Matte	150 Matte 7.5 Soda	Wabi	1¼ hrs	13	114	82	115.0	216.0	14.30	1756.5	1.26	68.7	21.3	40.0	330.0	1942	96.7	30.8	
2	" "	" "	Tilt	2¼ hrs	44.5	126	33	34.4	180.0	0.90	287.6	0.92	60.5	22.3	117.1	333.8	2002	99.7	87.3	Violent action & burning for 1st 1¾ hrs. Quiet but still gases burning for last ½ hr. Bullion coppery ringe.
3	" "	" "	Tilt	After 1 hr.50 mins tilt holed. Charge later gathered up & charged into Wabi Furnace for separation of Lead Bullion																Note:- By accident slag & matte samples for tests 3 & 4 were bulked Wabi furnace appeared to cause slagging of some Bullion
4	" "	" "	Wabi	2 hrs	26) 15.5)	218	) 140)	71.0 107.0	225.0 206.0	) 9.15)	1309.6	) 1.00)	35.2	26.9 24.1	85.5 46.5	) 388.0	) 1952	)98.1	)50.6	
5	Tertiary Matte	150 Matte 75 Soda 10 Flour	Wabi	2¼ hrs	29	130	46	8.0	249.0	0.80	508.9	0.25	30.3	3.4	105.2	51.5	2010	98.6	77.7	Flour added to supply additional reducer in the Wabi. Also burner run with minimum of air to avoid oxidizing atmosphere
6	" "	" "	Wabi	2⅓ Hrs	34	130	54	15.0	252.0	0.80	655.9	0.20	28.7	7.5	126.0	112.8	2330	99.4	80.7	Conditions as for Test 5
7	Secondary Matte	" "	Tilt	3⅔ hrs	48	46	80	22.0	163.0	1.50	139.5	1.00	72.8	15.4	114.0	232.0	1785	99.6	95.3	Ground to pass 10 mesh sieve All previous tests ½"
8	" "	" "	Wabi	2⅚ hrs	29	106	46	39.0	189.0	2.75	221.5	0.85	80.0	17.4	79.5	262.0	1370	99.3	86.8	Fine Ground
9	" "	" "	Wabi	2¼ hrs	47	91	73	40.0	220.0	1.80	501.4	0.72	29.5	27.4	151.0	410.0	2575	99.6	87.6	Fine Ground Air normal on burner
10	" "	" "	Wabi	2⅓ hrs	50	71	67	38.0	202.0	3.40	469.1	0.66	27.2	27.8	147.5	416.0	2450	99.6	90.6	Fine Ground and air normal
11	" "	300 Matte 224 Soda 75% 20 Flour	Wabi	2 hrs	102	210	210	28.0	174.0	1.50	225.8	0.60	18.7	41.7	259.1	312.0	2105	99.5	91.9	Fine Ground and air normal
12	" "	150 Matte 112 Soda(75%) 10 Flour	Wabi	2⅛ hrs	37	92	131	50.0	224.0	2.35	572.6	1.50	343.5	27.0	121.0	406.0	2460	99.4	73.6	Coarse ground (-½") and air normal

Table 6. Retreatment of Matte in the Plant.

Test No.	Date	Oxides	Combined Assay			Contents			Fluxes Used	Bullion							Slag							Calc^{d}. Heads			Recovery (%) in metal			Remarks.
											Assay			Contents				Assay			Contents									
			Au oz/L.T.	Ag oz/L.T.	Cu %	Au F.ozs	Ag F.ozs	Cu lbs		Wt.lbs	Au (fine)	Ag (fine)	Cu %	Au (f.ozs)	Ag (f.ozs)	Cu lbs.	Wt.lbs.	Au (ozs L.T.)	Ag (ozs L.T.)	Cu %	Au (f.ozs.L.T.)	Ag (f.ozs.L.T.)	Cu lbs	Au (ozs/L.T.)	Ag (ozs/L.T.)	Cu %	Au	Ag	Cu	
1.	3/6/46	L K Litharge G 174 lbs	17.6	1642	30.8	1.36	127.8	63.6	7 lbs Flour	118	2.6	86.0	64	4.31	148.2	63.7	26	1.00	8.20	-	.011	.092	-	66.6	1910	-	99.8	99.9	-	Smelt time 1½ hrs Bad Corrosion of lining. Lot of lead fuming after ¼ hr.
2.	14/6/46	1-108 J-64 172 lbs	41.1	1600	32.8	3.16	123	66.6	Flour 2 lbs Borax 34 Si O_2 17 Soda 3	67	3.0	146.0	77.3	2.93	142.8	61.8	136	0.3	12.0	6.0	.018	0.72	8	38.4	1870	34.7	99.4	99.6	86.7	Smelt time 1 hr Good pour. Light corrosion of lining.
3	19/6/46	G-97 G1-66 163 lb	30.6	784	32.2	2.22	67	62.4	Flour 1 lb Borax 33 SiO_2 18 Soda 3	44	6.0	86.0	83.2	3.21	66.2	37.0	162	0.2	6.6	7.3	.014	.40	11.8	44.6	768	29.9	99.6	99.3	76.8	As above
4.	19/6/46	H-80 H1-71 161 lbs.	64.0	1630	36.3	3.64	103	64.8	Flour 2 lb Borax 30 Sio$_2$ 16 Soda 3	62	6.0	138.0	78.6	3.79	104.7	40.8	138	0.24	9.26	8.7	.016	0.67	12.0	56.4	1660	36.0	99.6	99.6	77.3	As above Smelt time 60 mins.
6	12/7/46 & 17/7/46	A-177 Cuo1-24 F-116 E-10 (leached) 321 lbs	19.6	1087	28.3	2.70	161	88	Flour 4 Borax 66 Sio$_2$ 37 Soda 9	96	3.0	102.0	69.0	4.2	142.9	66.1	324	0.60	6.65	3.76	0.07	0.82	12.2	30.8	1033	26.2	98.4	99.4	84.4	Smelt time 60& 66 mins
6	17/7/46	2-106 3-231 E-10 (leached) 347 lbs.	4.6	1278	27.6	0.71	198	96.9	Flour 4 Borax 69 Sio$_2$ 36 Soda 7	107	1.0	148	67.1	1.66	231.2	71.8	336	0.60	6.70	6.03	.09	1.00	16.9	10.62	1600	26.6	94.6	99.6	80.9	Smelt time 60& 46 mins
7	9/8/46 & 14/8/46	1A-216 D-238 E-20 474 lbs.	12.4	1968	30.3	2.68	423.2	146.6	Flour 6 Borax 92 Sio$_2$ 48 Soda 10	149	1.6	20.4	72.6	3.26	443.7	108.0	486	0.20	14.8	6.0	.06	4.20	29.1	16.7	2116	29.0	98.1	99.1	78.8	Smelt time - 46, 60 & 46 mins.
8	6/9/46 & 11/9/46	C-194 M-64 N-62 310 lbs.	11.06	74.6	27.6	1.60	101	84	Flour 4 Borax 61 Sio$_2$ 64 Soda 6	87 49 38	2.0 2.6	81.0 108	64.6 74.9	1.43 1.39 2.82	58.0 60.0 118	31.7 28.6 60.2	164 180 334	0.19 0.32	2.16 7.24	2.4 4.4	.013 .026 .039	.148 .581 .729	3.7 7.9 11.6	20.6	866	23.2	98.6	99.4	83.8	Smelt time 60 & 70 mins Fluxes charge 1 Sio$_2$&Bor.16% Charge 2 Sio$_2$20% Bor.18% N.B. Calc^{d} Copper Assay below combined assay & fluxes .. insufficient.
9	29/11/46	O-46 P-76 Q-36 157 lbs.	48.2	144.4	32.6	3.38	101.2	61.2	Flour 2 Borax 31 Sio$_2$ 16 Soda 3	64	3.6	143	77.0	2.76	112.7	41.6	162	2.64	41.9	6.35	.184	3.02	10.3	42.0	1642	33.1	94.9	97.9	80.1	Slag contained some metallics due to splashing during pour. Smelt time 46 mins.

Table 7. Reduction Smelt of Oxides from Cupellation.

Test No. 2 was run in the Faber du Faur furnace with a Salamander No. 9 retort. This test, with the long contact of 2¼ hours, gave a result as good as any small scale test using the same fluxes. The charge could be watched. There was a violent evolution of gas, with much burning of the evolved gas, for the first 1¾ hours. The charge then became tranquil, but flaming from above the molten mass continued.

The bullion plug was distinctly coppery in colour.

All products were allowed to freeze after pouring, to ensure separation of the matte, slag, and bullion.

Test No. 3 was a repeat of Test No. 2, but after 1 hour 50 minutes the retort holed on the slag line. The charge was held until the time of contact was 2¼ hours and then poured. During the pouring operation the neck holed and practically the total contents of the charge ran through to the well below the furnace. Next day, this was cleaned up and a sheet of lead bullion was observed. The whole melt was recharged into a Wabi furnace in order to separate the bullion. Unfortunately the large sheet of lead which was recovered from the well was not roughly weighed before feeding into the Wabi furnace. Only 26 lb. of lead bullion was recovered on remelting, yet it was considered certain that there was more than this weight to commence with. This led to doubts as to the usefulness of a Wabi furnace in which the flame is in contact with the charge during a reducing smelt.

Test No. 4 was run simultaneously to Test No. 3, in another Wabi. Despite a 2 hour contact, only 15½ lb. of lead bullion was obtained.

At this stage, it appeared as if oxidising conditions prevailed in the Wabi furnace, in spite of the carbide lining. From the point of view of wear on the liner, however, the furnace was quite satisfactory; with the Salamander retorts, wear was excessive.

Tests Nos. 5 *and* 6. — By this time, all secondary matte on hand had been smelted, and it was decided to carry on further tests with accumulated tertiary matte. The sample taken assayed 52 oz. gold, 2,180 oz. silver per long ton.

Air was cut back to the minimum necessary for complete combustion of the oil, to avoid the oxidising conditions that appeared to be present in Tests Nos. 2 and 3. Ten pounds of flour was also added to the charge to give extra reducing power. Melting of the charge with the reduced amount of air was slow, and the lead fall was not as high as expected.

Tests Nos. 7 *and* 8 were later run on secondary matte, still using the reduced air on the Wabi furnace, but with normal burning on the tilt furnace. The lead fall in the Wabi furnace was again disappointing, in spite of the longer time of contact, using finely ground material. Results in the tilt furnace, on the other hand, were excellent.

Tests Nos. 9 *and* 10. — Air was restored to normal for Tests Nos. 9 and 10, conducted in Wabi furnaces, using finely ground material, with 10 lb. of added flour. A good lead fall was obtained, and lining corrosion was small.

This type of smelt at last appeared satisfactory.

The residual matte and the lead bullion from Test No. 10 were assayed for the base and precious metal contents. Results were as follows:—

Assay	Bullion %	Residual Matte %
Copper	30.0	32.0
Zinc	1.2	3.7
Iron	Trace	4.2
Lead	42.2	1.4
Silver	20.2	—
Sulphur	—	21.0
Gold	3.8	—
Insoluble	—	6.1

Test No. 11 was next run using 75% of soda ash, the matte being again finely ground. The silver recovery was slightly higher than in Tests Nos. 9 and 10. All three of the products were assayed:—

Assay	Residual Matte %	Residual Slag %	Bullion %
Lead	0.4	0.2	43.3
Copper	23.0	3.7	32.2
Zinc	3.8	2.7	2.5
Iron	4.3	3.6	Trace
Sulphur	19.8	4.8	—
Aluminium	0.4	4.1	—
Silica	7.4	23.3	—
Gold	—	—	2.8
Silver	—	—	17.4

The matte and slag contained considerable amounts of sodium salts, including the sulphide which was plainly noticeable by its odour. Water soluble tests were conducted and the matte contained 41.5% of water soluble salts and the slag 49.7%.

Test No. 12. — Recoveries were adversely affected, although 75% of soda ash was used, by not grinding the matte finely before smelting.

Little improvement had been shown in the two tests succeeding Tests Nos. 9 and 10. Accordingly, that procedure was transferred to the gold-room without alteration, that is, secondary matte was finely ground, mixed with 50% of soda ash and 7% of flour, and smelted for 2¼ hours in the Wabi furnace.

Results confirmed the laboratory tests. Nearly all the gold and silver was concentrated in a base bullion, which, when allowed to cool overnight, could be readily separated from the residual matte and slag. Furthermore, the slag layer could be identified and separated from the residual matte layer.

The amount of copper in the base bullion is important in two respects. Firstly, unless 20% or more is present, it is found that silver is not satisfactorily removed from the matte; secondly, an excess of copper up to 50% makes subsequent cupellation difficult. It has not been found necessary, however, to attempt the control of the copper content in the base bullion beyond the practice stated above.

Secondary matte is accumulated for one month before crushing and smelting, about 300 lb. yielding 70 lb., approximately, of base bullion. This is broken under a power hammer into pieces of a size convenient for inserting with a pair of tongs into the cupelling furnace.

Slag is of no value and is dumped. Residual matte contains between 20 and 40% copper, 1.5 to 2 oz. per ton gold, 100 to 200 oz. per ton silver, and is stacked in the open to "weather" for approximately six months. It is then washed four times by decantation with water, using a pulp ratio of about 1.5 water to 1 matte each time. Loss of weight on washing varies from 51% to 30% of original weight, so that the residue contains about 3 oz. gold, 300 oz. silver per ton.

The residues have not yet been further treated, but it is expected that by smelting it will be possible to recover the gold, silver, and copper as a copper base bullion, which can be sent to a smelter. In that case, the copper should pay for all shipment and smelting charges.

The value of gold and silver tied up in this way in the residual matte is very small, as shown in the summary of distribution of values given in Fig. 14, and for all practical purposes, the secondary matte may be considered to be cleaned of all payable metals by the reduction smelting process just described.

Cupellation of Base Bullion.

A cupellation furnace was built to refine the base bullion produced in both the lead bath and the reduction processes for secondary matte treatment. Perhaps the most important feature of this furnace is the refractory lining used to form the cupel test. It was developed by A. C. McDonald in earlier work on a larger scale, and was the only material found to resist satisfactorily the intense slagging action of the oxides formed. A description is therefore given here of its composition and method of forming the test.

Refractory Lining for Cupel Test.

Composition:

Magnesite, minus ¼ mesh	24 lb.
Cement (newly opened bag)	12 lb.
Mabor or Cupex	4 lb.
Water	4 lb.

Mixing:

The three constituents of the charge are thoroughly mixed dry by completely turning over at least six times. The well mixed dry charge is then spread on a clean plate to a uniform depth, not greater than 3 in., and the water added through a watering can rose over the whole area. It is then well turned until no dry portion remains.

Tamping:

All the mixed material is shovelled immediately after mixing into the cupel basket and levelled out, so as to contour the cast iron pan, the bed at this stage being about 3½ to 4 in. thick.

A 4 lb. gympie hammer with a short handle is used for tamping, commencing in the centre and working outwards. Force of the blows with the hammer is increased as tamping proceeds, until no further impression is made, when the 4 lb. gympie is replaced with a 2 lb. mason's gympie. Tamping is then done more carefully with greater force, each hammer mark being partly covered by the subsequent blow.

When tamping has been completed, the surface is finished off with a trowel. Usually sufficient water is present to float some mabor to the surface for this phase, but if not, a little can be sprayed on with a bannister brush. The mabor fills any interstices in the surface and produces a very smooth finish. The lip over which oxides are poured is formed during tamping and trowelled at the same time as the rest of the test.

Thickness of the test on completion is about 2 in.

Time taken for the whole operation is about four hours.

Tests have been successfully used after standing only 20 days, but as a test lasts about three to four months, the spare usually has had much longer than the minimum time to cure.

Treatment of Old Linings:

When a test cracks it is replaced with the spare, and the old lining is crushed in a small dry ball mill that is on hand for such work in the gold-room. The crushed material is disposed of by feeding into the reduction smelt of secondary matte, at the rate of 10 lb. lining per 150 lb. charge of matte.

Cupellation.

Base bullion from the secondary matte reduction smelt is assayed for silver and copper before cupellation. The course cupellation will take is guided according to the percentage of copper present. If this is low, cupellation can be conducted at a faster rate at a lower temperature than if the copper percentage is high.

Lighting the cupellation furnace differs from usual practice in that a few sticks of pine wood only are used to ignite the oil. It will be remembered that with all the other furnaces, the practice is to have a good wood fire burning for ¾ hour or more before turning on the oil.

With oil on, the furnace is brought up to a red heat as slowly as possible to prevent cracking the cupel test. Preparatory heating takes about 1½ hours.

Pieces of matte bullion, 40 to 50 lb., up to a maximum of 100 lb., are loaded into the cupel through the side door. Preheating the cold bullion, either outside, or within the furnace, before allowing it to come in contact with the lining, has not been found necessary.

During melting, metal runs in droplets away from each piece of bullion, until at one stage, lumps are left that are rather difficult to melt, particularly with high-copper base bullion. They are stirred in the molten metal with a bar until all is molten.

Air is next played on the surface through an adjustable ½ in. pipe. The pipe is directed to a point about 2 in. from the metal line at the burner end, and tilted slightly forward. The amount of air used is sufficient to cause a decided ripple in the surface of the molten metal, but not enough to cause splashing.

An oxide layer forms on the surface, and is allowed to grow until the metal is covered to a depth of about ¼ in., or until the "eye" below the air jet is just showing. Oxides are then poured off in a thin stream over the lip by tilting the furnace forward slowly. They are granulated directly by falling into a four gallon kerosene tin full of water.

The oxides overflowing the lip are tested at frequent intervals by collecting some on an ordinary dry shovel, moving the shovel while the stream falls on it to give a thin layer of material. Metal adheres to the shovel and, unlike oxide, is not readily dislodged by shaking up and down. Another indication is by observation of the stream. Oxides rarely run straight over the lip, but follow an uneven line, whilst metal falls in a straight stream. If metal should appear on the shovel or the eye of metal in the cupel below the air jet gets too large, the furnace is tilted back slightly to stop the flow until more oxide has accumulated, when pouring is recommenced.

Cupellation ceases fairly abruptly, a few circles of litharge race about on the surface and are soon absorbed by the liner, leaving a brilliant metal surface.

The refined bullion is finally poured into a conical mould. When it has frozen, the button is tipped on the floor, quenched in water, and is then ready for barring.

Recovery from Cupellation Oxides.

The simple cupellation procedure of current practice was preceded by much experimental work, because, although a good grade of precious metal bullion could be readily made, there remained the problem of recovering gold, silver, and copper from the cupellation oxides.

Recently it has been found that by analysing the oxides and calculating a flux to form a sesquisilicate slag with the lead only, it is possible to effect an almost complete separation of copper, gold, and silver in a copper base bullion from a lead silicate slag, by a reducing smelt, again a relatively simple process.

Preliminary Test Work: Before the reduction of a copper base bullion and the slagging off of the lead was adopted, a number of methods was tried on both laboratory and plant scales for the recovery of valuable contents. A description of these is now given, ending with the laboratory scale tests which led to the adoption of the reduction smelt to a copper bullion as standard plant practice.

When secondary matte bullion first came forward for cupellation from the reduction smelt of secondary matte, little was known of the course cupellation would take, although it was expected that with a high percentage of copper present in the bullion it might be difficult to cupel without the addition of sufficient new lead to lower the copper percentage to 15%.

Nevertheless, a trial cupellation was made on matte bullion alone, and by increasing the temperature as cupellation proceeded, bullion of the following grade was produced:—

Gold	141
Silver	813
Base metal	46

During the first cupellation, it was observed that a marked physical change occurred at a point about midway through the cupellation. Differential oxidation was suspected to be the cause, and in the next cupellation, samples of the oxides formed were taken for assay early, midway, and late in the cupellation. Assay results confirmed that lead was being oxidised sooner than the copper:—

Time of Sampling	Lead %	Copper %	Calculated to PbO %	Calculated to CuO %
Early	75.5	11.7	81.3	14.6
Midway	42.1	38.2	45.5	47.7
Late	24.9	50.7	26.8	63.5

Gold and silver were distributed unevenly in the impure litharge and copper oxide stages, as will be seen from the following table, which shows the amounts of the various products with their contents for the first 548 lb. of matte bullion cupelled:—

Product	Weight lb.	Gold Fine oz.	Silver Fine oz.	Lead Calculated to PbO %	Copper Calculated to CuO %
Matte bullion	548	231.5	1,511.9	—	—
Finished bullion ..	96	222.5	1,217.9	—	—
Impure litharge ..	171	0.5	23.4	72.8	22.4
Copper oxide	280	2.5	273.3	45.0	48.2
Totals	547	225.5	1,514.6	—	—

Neither product could be rejected because both contained precious metals and copper.

Further investigation work on the cupellation oxides was conducted with three main objectives:—

1. Recovery of the gold and silver.
2. Recovery of copper in a useful form.
3. Recovery of sufficient lead of low copper content for subsequent cupellations. Lead, in itself, was not considered worth recovering as a commercial product.

Impure litharge, when reduced, satisfied the last condition, the lead also collecting all the gold and silver. Copper content of the lead so reduced, was higher than hoped for, but no further work was done on this product.

The copper oxide product was treated by partial reduction only, when it was found that copper reduced before lead. Reduction was then carried out in two stages, first by adding sufficient reducer to convert 10% of the original weight to metal, second by completely reducing the residue. . The compositions, in percentages, of the two metals obtained were as follows:—

Assay	Partial Reduction %	Complete Reduction of Residue %
Copper	67.8	16.2
Lead	9.7	82.8
Gold	0.1	Trace
Silver	21.5	0.24

Coppery metal from the partial reduction, with its high content of gold and silver, was considered too difficult to treat, and the method was abandoned.

Next, a complete reduction was made. Sulphur was stirred into the resulting metal to remove copper as a matte, and the two products were assayed:—

Assay	Metal %	Matte %
Copper	2.1	45.3
Lead	93.8	—
Gold	0.02	10 oz. per ton
Silver	2.98	530 oz. per ton

This treatment method was also rejected, for two reasons:—

1. The lead, although suitable for breaking down the copper concentration before cupellation by addition to matte bullion, was produced in too great a quantity.
2. High values in the matte would necessitate returning it to the secondary matte smelt, thereby producing a closed copper circuit.

Liquation of the completely reduced metal was attempted without success. Liquation of the oxides was also considered as a possibility before any reductions were made, but separations were very incomplete, and the extremely corrosive properties of the molten oxides presented some difficult equipment problems.

Leaching with sulphuric acid to remove the copper was next tried. The product was ground to minus 35 mesh and leached with sulphuric acid of 5% maximum strength. Copper recovery, after 48 hours, was 72.3% as copper sulphate. Finer grinding increased the speed of copper dissolution, but recovery rose only slightly.

The resulting copper sulphate solution was evaporated and allowed to crystallise. Allowing costs for grinding, acid consumption and power, crystallised copper sulphate could be produced for less than the existing market price. The yield would be approximately 16 cwt. of copper sulphate annually.

The residue, after leaching, was much finer than the original feed, as disintegration took place during leaching. The residue was next partially reduced to give a 30% by weight fall of metal. Lead so produced contained 7.4% of copper. The residue carried only 0.20 oz. gold and 8.4 oz. silver per ton. Therefore, it was considered that if the copper leach could be improved sufficiently to give a lead lower in copper, the three objectives of the investigation would be attained.

The next cupellation was run, using 40% of new lead, reduced from the litharge of the previous cupellation. Although the cupellation was much more easily controlled after this addition, a copper oxide product was still produced in the later stages. Bearing in mind the results of the foregoing leaching tests, it was decided not to separate the two stages, but to granulate all the oxides into one bulk lot.

Details of this cupellation, showing the weights and distribution of values are given in the table below:—

Product	Weight lb.	Gold Fine oz.	Silver Fine oz.	Lead Calculate to PbO %	Copper Calculate to CuO %
Matte bullion and new lead	447	169.8	1,239.9	—	—
Finished bullion	97	168.7	1,152.4	—	—
Impure oxides	445	0.8	105.5	15.8	77.6
Totals	—	169.5	1,257.9	—	—

Leaching tests on these impure oxides were made, and were quite successful. Recovery of copper as sulphate was no higher than with the copper oxide product described earlier; on the other hand, lead reduced from the leached residue contained only 4.9% of copper. This was because of the lower percentage of copper in the original oxides.

Gold and silver recoveries into the lead were almost complete.

Leaching on a plant scale was not as successful as the small scale tests. An unexpected complication appeared after the first batch had been leached twice. On adding the third wash of dilute sulphuric acid, a metallic copper film formed over the oxide particles and dissolution of copper ceased.

Smelting tests were then made on a laboratory scale in an attempt to slag off the litharge, from the cupellation oxides, at the same time reducing the copper oxide to metal, which would act as a collector for gold and silver. The quantities and kind of flux used were empirical, until preliminary tests had shown that borax, with charcoal as reducing agent, gave the best results. To reduce the cost, borax was then partially replaced with an equivalent amount of silica to give a boro-silicate slag.

Finally, it was found possible to calculate the flux required from the analysis of the oxides. This is now done by calculating the amount of silica necessary to form a sesquisilicate slag, then replacing silica by its equivalent in borax, so that the final flux contains two parts borax to one part silica.

For example: Suppose a charge requires 30 lb. silica alone to form a sesquisilicate slag.

Let x be the amount of silica required.

Then 1.5 (30 - x) is the amount of borax required,

because 1.5 borax is equivalent to 1 silica in slagging action.

But as the amount of borax is to be twice the amount of silica in the final flux,

Therefore 1.5 (30 - x) = 2 x

Whence x = 12.9 approximately

= 13 „

The flux would then be:—

Silica	13 lb.
Borax	26 lb.

Soda ash, equal to 2% of the weight of oxides is added. No allowance is made for this in calculating the silica-borax flux. Its effect is not only to increase fluidity, but to hasten melting, so that the charge is in contact with the reducing carbide liner of the furnace for the minimum time.

Flour, 1.33% of the weight of oxides being taken, is added as the reducing agent.

A peculiar effect is noticed when pouring this particular charge. When the slag enters the large conical mould, there appears to be a large evolution of gas, and the whole charge froths up, filling the mould. It subsides again after a few minutes to a normal-looking slag of a deep chocolate colour.

Table 7 gives details of plant tests conducted on the cupellation oxides in a Wabi furnace, using the method of reduction smelting described above. It will be noted from the figures that the recovery of gold and silver into the copper base bullion is almost complete. Recovery of copper is not as high, some being lost into the slag, but its value is slight.

Bars of copper base bullion produced to date, total 16 in number. They have been sampled in groups for assay, the results and weights being shown in Table 8. Other base metals, not analysed, mean principally lead, which has been reduced during the smelt, and carries a penalty when the bullion is treated by a copper smelter. Again, its negative value is low, so that when this copper base bullion is eventually shipped, the return should be approximately the full value of the gold and silver contents, the copper paying freight, smelting charges, and penalty for lead content.

SUMMARY AND FLOW SHEET.

Gold and silver are recovered successfully from secondary matte by a series of smelting operations set out diagrammatically in Fig. 13. Fig. 14 shows the gold, silver and copper distribution in the various stages.

Taken in order, the separate treatment processes are:—

1. Secondary matte is ground in a dry ball mill before mixing with 50% of soda ash and 7% of its weight of flour. Charges containing 150 lb. of matte at a time are smelted for 2¼ hours in Wabi, silicon-carbide lined furnaces, yielding three products, residual slag, residual matte, and a copper-lead base bullion that collects nearly all the gold and silver, with about half the copper from the secondary matte.

2. The lead-copper base bullion is cupelled in a special furnace, producing high grade gold-silver bullion suitable for shipment to the Royal Mint, and a mixture of cupellation oxides which still retain small amounts of gold and silver, that are, nevertheless, worth recovering by further treatment.

3. Cupellation oxides are smelted at intervals, using a calculated silica-borax flux with a small percentage of flour added as a reducing agent. Copper base bullion produced by this smelt contains the remnant of gold and silver in the

Bar Mark	Weight (lbs.)	Assay				Contents		
		Au(fine)	Ag(fine)	Cu %	Other % Base Metals	Au(f.ozs)	Ag(f.ozs)	Cu(lbs)
L.V.S. 1	37½	2.0	161	71.5	12.2	6.73	541.77	164.8
" 2	38½							
" 3	39							
" 4	39							
" 5	37½							
" 6	39							
	230½ (3365 ozs)							
L.V.S. 7	38½	3.9	108	69.85	19.0	17.22	476.93	211.3
" 8	40½							
" 9	40½							
" 10	40							
" 11	40							
" 12	37							
" 13	39							
" 14	27							
	302½ 4416 ozs							

Table 8. Barring of Metal from Reduction of Oxide for Shipment.

cupellation oxides, as well as most of the copper, with only a small percentage of lead, so that the precious metal values can be recovered at low cost by shipment to a copper smelter.

4. Gold and silver retained by a cupel test is recovered, after the test is of no further use for cupellation, by grinding and feeding back to the reduction smelt of secondary matte at the rate of 10 lb. per 150 lb. charge of matte.

5. There are two by-products only — residual matte and residual slag — the former of which still retains gold and silver in sufficient concentration to consider further treatment; even so, the quantity of these materials produced is very small.

6. The original objective of the investigation, namely the recovery on the mine of all gold and silver values from mattes and slags, has been practically achieved.

PART III.

REMOVAL OF COPPER FROM GOLD SLIMES.

INTRODUCTION.

Prevention of copper entering the final bullion has been the principal objective of all experimental work performed on zinc slimes in the Lake View and Star Ltd. gold-room. Above all other metals encountered during treatment, it is the most difficult to separate from gold and silver.

Broadly, two methods are possible for elimination of copper. The first is, to prevent its entry into the Merrill-Crowe precipitation plant in the cyanide solutions. Practically no experimental work has yet been done towards this end, although the possibility has not been dismissed. The second method is to remove the copper after it has been precipitated on zinc dust with the gold and silver.

Current practice, as already described in Parts I and II, involves converting most of the copper present in the slimes into the form of a lead-copper matte, which requires a considerable amount of further treatment to recover the gold and silver it contains. Moreover, the bullion obtained from the initial smelt still carries approximately 10% of copper, so that the process cannot be considered ideal.

Aciding of raw zinc slime with dilute sulphuric acid was the earliest treatment tried to overcome the copper removal problem when it first appeared. It was abandoned because of non-success. Granulation and sulphuring of bullion were then adopted from sheer necessity, being followed, as a natural development, by the pyrite matte-forming smelt of current practice.

Recent experimental work has been re-directed towards acid treatment of both raw and roasted slimes. Intensive testing has indicated that when suitable equipment, that will withstand the action of the acids used, can be obtained, the method will be applicable on a plant scale. Two advantages should accrue from its use: A higher grade of final bullion, and no matte formation with its attendant complications.

Some form of acid treatment, then, appears to be the trend for the future, but two other methods have been partially explored.

When copper first became troublesome in the plant, small smelts were run using manganese dioxide in an attempt to oxidise the copper and slag it off. Three of the crucibles used contained graphite in their composition, one was almost free from graphite. The grade of bullion obtained from these trial smelts indicated in no uncertain manner, that to smelt in a furnace lined with a reducing agent would vitiate any attempts to slag off the copper. Efforts have therefore been made over a number of years to obtain a furnace liner containing no reducing agent, that will withstand the action of fluxes and oxides, but unfortunately so far none has been found suitable.

Fusion with salt cake is the other method which has been tested as an alternative to the matte process. Fusion is slow, however; extraction of the soluble sulphates is not as simple as it first appears, and although the bullion grade obtainable is high, the method does not appear to possess enough advantages to merit further work or application.

ACID TREATMENT OF GOLD SLIMES.

Laboratory scale tests were commenced in January, 1944, to try to find an acid or combination of acids with which to treat the zinc slimes, so that on smelting with a normal silica-borax flux, they would yield an acceptable grade of bullion without matte formation. From a practical point of view, whether the slime was to be roasted or not, before or after acid treatment, was not considered a major issue, although it was recognised that elimination of roasting would simplify process detail by reducing the number of stages.

Preliminary Tests.

Preliminary trials were made on raw and roasted zinc slimes from the main Chaffers plant, and on raw slimes from the Chaffers retreatment plant, using nitric and sulphuric acids, alone and in combination.

The procedure in most cases was to add water first, then to make successive small additions of the particular acid to initiate and maintain reaction. When action was complete, the residue was filtered, washed, dried and smelted. The table below summarises results from this series of tests.

Acid	Weight of Slimes gm.	Residual Weight gm.	Smelted Weight gm.	Gold Bullion Fineness
Sulphuric	80	60	20	792
Nitric	100	73.5	19.7	992
Sulphuric and nitric	100	75.4	27.0	792
Sulphuric and nitric	50	15.5	12.0	813

Conclusions drawn from the tests were:—

1. Treatment of the Chaffers plant roasted slimes, with nitric acid alone, showed promise.
2. Treatment of the Chaffers retreatment plant raw slimes with either sulphuric or mixed sulphuric-nitric acids, was a decided advantage.

Confirmatory tests were made on further samples of Chaffers plant slime. Some of the plant raw slime was taken and roasted in the laboratory electric muffle furnace to ensure good roasting.

Roasting Test 1:

Sample weight	150	gram.
Roasted in the muffle furnace at 650°C. for 2½ hours with regular rabbling.		
Calcined weight	100	gram.
Percentage of original weight	66.7	gram.

Roasting Test 2:

Sample weight	150	gram.
Added nitre 20% by weight	30	gram.
Roasted as for Test 1		
Calcined weight	161	gram.
Percentage of original weight	89.5	gram.

Plant Roast:

For the same day, 19/6/1944		
Percentage of original weight	71.0	gram

Analyses:

Element	Raw Slime %	Plant Roasted Slime %
Gold	17.5	18.2
Silver	3.7	3.6
Lead	13.0	13.4
Zinc	35.1	32.9
Copper	10.3	11.7
Iron	0.2	0.2
Calcium	1.4	1.5
Sulphide sulphur	3.8	1.3
Sulphate sulphur	1.0	3.7
Totals	86.0	86.5

Base metals, calculated to oxides in the plant roasted slime, totalled 98.4%.

Treatment of Roasted Slime with Sulphuric and Nitric Acids.

Aciding tests were then made on the laboratory roasted, and plant roasted slimes. Solutions from the acid treatment were analysed to determine the effectiveness of copper removal, and the slags resulting from smelting were assayed for copper, because for plant practice they should be low in copper if they are to be cyanided. Table 9 summarises the tests.

Again, nitric acid appeared to give the best results, but it should be noted that smelting was done in clay crucibles, and the weight after aciding had not been reduced sufficiently to be able to use a clay lined crucible on a plant scale.

A larger scale test was next made with slime roasted under plant conditions. Details are as follows:—

Test (*a*): Sample taken 6/7/1944:

Plant results: 3.1 oz. of bullion per lb. of slime, bullion assayed 815 fine gold, 62 silver, 123 base metal.

Sample weight	1,816 gram.
Added manganese dioxide	363 gram.

Test No.	Sample	Wt.Gms	Acid Treatment	Residue Wt. gms.	Acid Solution Assay			Smelting							Remarks
									Bullion				Slag		
					Gold oz/ton	Silver oz/ton	Copper % Removed	Total X Wt. of Charge	Wt.gms	Gold fine-ness	Silver fine-ness	Base %	Wt.gms	% Copper	
1	Plant Roasted Slime	100	Sulphuric Acid 30 ccs,Filter then Nitric Acid 30 ccs	67 44	0.05 0.02	0.13 0.03	3.1 4.27	83	24.2	732	147	12.1	47	0.3	
2	Lab.Roasted Slime Test 1.	100	as above Sulphuric Nitric	73 42	0.04 0.04	0.06 0.03	4.7 4.4	80	20.6	835	155	1.0	47	0.2	Lab Roasted Wt of 100 gms = 107 gms Plant Roasted Slime
3	Lab Roasted Slime Test 11.	100	as above Sulphuric Nitric	69 39	3.25 4.21	0.03 0.22	3.5 5.51	73	21.6	762	152	8.6	41	0.3	Equivalent to 79 gms of Plant Roasted Slime
4	Plant Roasted Slime	100	Sulphuric Acid 50 ccs	51	-	-	5.23	99	28.3	632	130	23.8	73	2.4	
5	Plant Roasted Slime	100	Slime mixed with 20gm Mang.Dioxide & Acid with Sulphuric 50 ccs	65	-	-	7.6	121	21.5	815	140	4.5	126	1.0	Mang. Dioxide produced more violent action than Sulphuric alone
6	Plant Roasted Slime	50	as above but ½ quantities	31	-	-	7.3	61	10.4	906	90	0.4	57	1.5	
7	Plant Roasted Slime	50	As for Test 6. then Nitric 10 ccs	26	-	- -	8.1	48	10.2	885	108	0.7	47	1.7	
8	Plant Roasted Slime	62	Nitric Acid 40 ccs	36	-	- -	3.1	68	13.1	876	112	1.2	57	5.0	

NOTES. X All Smelt Tests fluxed with
(40% Borax
(20% Silica
(10% Manganese Dioxide
(15% Soda
and smelted in clay crucibles

Tests 1,3 & 4 were all too cool during the whole of the Fusion Period.

All acidings occupied 30 mins and kept to boiling point.

Table 9.

Added boiling water	1 gal.
Added crude sulphuric acid in 50 to 100 ml. lots	483 ml.
Agitated with a wooden stick for ½ hour.	
Diluted with 1 gal. water, stirred, decanted and lightly washed.	
Acid liquor yielded cement copper	105 gram.
Residual weight of slime	1,510 gram.
Test (*b*) Sample taken 17/7/1944:	
Sample weight	300 gram.
Added nitric acid	250 ml.
Residual weight	139 gram.

The residues from both tests were divided, and smelting tests made, using several different lining materials for the crucibles. Results are shown in Table 10.

The tests clearly indicate that these acid treatments are not sufficient without the use of clay lined smelting crucibles.

Treatment of Raw Slime with Hydrochloric and Sulphuric Acids.

Exploratory tests on raw zinc slimes, using hydrochloric and sulphuric acids, with and without manganese dioxide, were made. Details are as follows:—

Test 1: Sample raw slimes taken 17/7/1944:	
Sample weight	200 gram. (moist)
Added commercial hydrochloric acid	175 ml.
Washed, semi-dried and weighed	150 gram.
Mixed with 30 gram. manganese dioxide.	
Added 100 ml. sulphuric acid.	
Washed, dried and weighed	80 gram.
Divided into halves, A and B, and smelted with:	

Borax	16 gram.	(40%)
Silica sand	8 gram.	(20%)
Manganese dioxide	4 gram.	(10%)
Soda ash	6 gram.	(15%)

Results		(A) Clay Crucible	(B) Graphite Crucible
Bullion weight:	gm.	22.5	22.4
Assay: Gold: fine		755	767
Assay: Silver: fine		100	86
Assay: Base metal:		145	147
Slag weight:	gm.	45	40
Copper in Slag:	%	0.67	0.67
Base in Bullion: Lead,	%	6.25	4.7
Base in Bullion: Copper,	%	6.72	7.36
Comments:		Matte on bullion and thick sulphate layer on top of good fluid slag	Matte on bullion. Thick slag.

Test 2: Sample of same slime as Test 1:

Sample weight	110 gram.
Treated as for Test 1 with commercial hydrochloric acid	100 ml.
Washed, semi-dried, mixed with 15 gram. manganese dioxide.	
Added 50 ml. sulphuric acid, washed, dried and weighed	49 gram.
Divided into two parts, A and B, and smelted (A) in a Carbolox lined crucible with:	
Borax glass	20 gram. (50%)
Nitre	10 gram. (25%)
Silica sand	2 gram. (5%)

Analysed (B).

(A)		(B)	
Weight:	40 gm.	Weight:	9 gm.
Bullion weight:	31 gm.	Analysis:	%
Assay: Gold	672 fine	Gold	52.1
Assay: Silver	122 fine	Silver	9.5
Assay: Base metal	206	Lead	5.8
Slag weight:	42 gm.	Zinc	1.0
		Sulphide-Sulphur	3.5
Comments: Very thick slag.		Sulphate-Sulphur	2.7
A little matte on the bullion.		Copper	16.2
			90.8

Test 3: Sample weight	10 gram.
Acided with 4 gram. manganese dioxide and 10 ml. hydrochloric acid, dried and weighed	7.7 gram.
Acided with 6 ml. sulphuric acid, dried and weighed	4.2 gram.
Wrapped in own weight of lead and cupelled. Left impure mass. Added more lead and cupelled to a good bullion button, weighing	1.475 gram.
Test 4: Sample weight	10 gram.
Acided with 10 ml. hydrochloric acid, dried and weighed	3.8 gram.
Acided with 6 ml. sulphuric acid, dried and weighed	3.5 gram.
Wrapped in own weight of lead and cupelled. Gave a button of poor-looking bullion with black copper oxide coating, weighing	1.995 gram.
Tests 5 *and* 6: Sample taken 3/8/1944:	
Preliminary test done first to check Test 4, using 3 gram. hydrochloric acid, followed by 2.6 gram. sulphuric acid: residual weight	2.6 gram.
Test 5: Sample weight	250 gram.
Boiled 30 minutes with 250 ml. commercial hydrochloric acid, decanted and washed by decantation with hot water, added 20 gram. manganese dioxide and 125 ml. commercial sulphuric acid. Boiled 30 minutes, filtered, washed, dried and weighed	77 gram.
Weight of lead chloride recovered from solution	22 gram.
Percentage of original weight	38.5 gram.

Test 6: Sample weight	248	gram.
Procedure as for Test 5, but no manganese dioxide used. No blue colour was obtained in the sulphuric acid treatment: residual weight	75	gram.
Percentage of original weight	30.2	gram.

Residue from Tests 5 and 6 were combined and divided into two halves, (A) and (B), treated as follows:

Sample (*A*): Sample weight	50	gram.
Smelted with:		
Borax glass	25	gram.
Silica	5	gram.
Manganese dioxide	10	gram.
Soda ash	5	gram.
Fluor spar	5	gram.
Bullion weight	33.3	gram.
Slag weight	66	gram.
Bullion Assay:		
Gold	619	
Silver	137	
Base metal	244	

Comments: Good fusion and fluid slag.
Clay crucible used.

Sample (*B*): Sample weight	105	gram.
Roasted in laboratory furnace at 700°C. with frequent rabbling: Residual roasted weight	77	gram.
Smelted with:		
Borax glass	31	gram.
Silica	10	gram.
Manganese dioxide	16	gram.
Soda ash	8	gram.
Bullion weight	39	gram.

Bullion Assay:

Gold	857
Silver	112
Base metal	31

Comments: Good fluid slag.
Clay crucible used.

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Summarising this series of tests, treatment of raw zinc slimes with hydrochloric acid alone, or followed by treatment with sulphuric acid and manganese dioxide, was no more effective in removing base metal than treatment of roasted slime with mixed nitric and sulphuric acids

Treatment of Roasted Slime with Hydrochloric Acid, followed by Mixed Nitric-Sulphuric Acids.

Small scale tests were made on roasted slime, in two stages. Commercial hydrochloric acid was used first, then after filtering and washing, the residue was further treated with mixed diluted nitric and sulphuric acids.

The table below shows the results. The weight of sample taken in each case was 10 gram., equivalent to 14 gram. of raw slime.

Test No	Treatment	Residual Weight Gram.	% of Raw Weight
1	Acid with 10 ml HCl	5.75	41.1
	Acid with 6 ml H_2SO_4	4.50	32.1
2	Acid with 10 ml HCl	5.70	40.7
	Acid with 6 ml H_2SO_4 and 1.0 gm. MnO_2	4.60	32.9
3	Acid with 10 ml HCl	5.40	38.6
	Acid with 5 ml H_2SO_4 and 2 ml HNO_3	4.00	28.6

Test No. 3 was repeated on a larger scale with a sample of plant roasted slime taken on 17/7/1944.

Repeat Test 3: Weight of sample	302	gram.
Added 302 ml. commercial hydrochloric acid; filtered, washed and weighed	183	gram.

Acided with 151 ml. sulphuric acid and 60 ml. nitric acid; filtered, washed and weighed	120	gram.
Percentage of roasted weight	39.8	gram.
Percentage of raw weight	28.4	gram.

Combination: Acid - Roast - Acid.

Aciding of raw slime with hydrochloric acid reduced the weight of slime to 38.5% of the original weight of raw slime taken, (see Test No. 5), by removing zinc and lead as chlorides. It was decided to attempt the further removal of base metals by acid treating slimes, roasted after the hydrochloric acid treatment.

Accordingly the following test was carried out:—

Sample taken 14/8/1944

Slime smelted in the plant on the same day yielded 2.0 oz. of bullion per lb. of raw slime.

The bullion assayed:

Gold	800
Silver	60
Base metal	140

Sample weight	495 gram.	
Acided with 495 ml. hydrochloric acid, semi-dried weight, 300 gram. Rewashed with boiling water, filtered, dried and weighed	120 gram.	(24.3%)
Roasted in the laboratory muffle furnace for 1 hour at 700°C. weight after roasting	108 gram.	(21.8%)
Acided with 100 ml. sulphuric acid and 20 ml. nitric acid. Washed free from copper, filtered, dried and weighed	74 gram.	(14.9%)

Assayed Residue:

Gold	770
Silver	108
Base metal	122

Smelted the residue in three portions, A, B, and C, as below.

Sample		A	B	C
Crucible Used		Graphite	Clay	Clay
Weight taken:	gm.	35.0	35.0	2.7
Flux: Borax glass:	gm.	18.0	18.0	2.0
Flux: Silica sand:	gm.	—	—	—
Flux: Manganese dioxide:	gm.	7.0	4.0	—
Flux: Soda ash:	gm.	5.0	5.0	2.0
Bullion weight:	gm.	33.9	30.6	2.25
Slag weight:	gm.	35.0	35.0	—
Bullion Assay: Gold:		787	867	865
Bullion Assay: Silver:		124	123	123
Bullion Assay: Base metal:		89	10	12
Slag Assay: Copper %		2.05	4.42	2.60

A repeat test was made to confirm the above results, the procedure being the same, except that no nitric acid was used.

Repeat Test: Sample taken 23/8/1944:

Sample weight	860 gram.
Acided with 860 ml. commercial hydrochloric acid, filtered, washed, dried and weighed	216 gram.
Roasted 1 hour at 700°C. and weighed	216 gram. (25.1%)
Acided with 200 ml. sulphuric acid, filtered, washed, dried and weighed	110 gram. (12.8%)

A small sample of residue from the acid-roast-acid treatment was assayed, and the balance smelted:—

Assay:

Metal	%
Gold	79.0
Silver	13.0
Silica	2.65
Lead	1.87
Copper	3.07

Smelt:

Weight taken for each smelt, 25 gram.

Crucible Used		Clay	Graphite	Clay	Graphite
Flux:	Borax glass	12.5	12.5	6.0	6.0
	MnO_2	3.0	4.0	—	—
	Soda ash	4.0	4.0	6.0	6.0
Total weight of charge		44.5	45.5	37.0	26.0
Bullion weight		22.8	23.8	23.0	23.4
Slag weight		23.5	17.5	22.0	7.5
Total		46.3	41.3	45.0	30.9
Bullion Assay:	Gold	847	812	836	810
	Silver	146	154	156	153
	Base	7	34	8	37

Remarks: All were good fusions. There was no matte on any of the buttons.

Conclusions drawn from the whole series of tests, using any or all of the three mineral acids, sulphuric, nitric, and hydrochloric acid, are as follows:—

1. Aciding of raw slime with hydrochloric acid followed by sulphuric acid, and then roasting reduced the weight of slime to 30.8 % of its original weight.
2. Roasted slime treated with hydrochloric acid, followed by sulphuric-nitric acid mixture, reduced the weight to 28% of the original weight of slime.
3. Raw slime when treated with hydrochloric acid, then roasted and treated again with sulphuric acid, was reduced in weight to 12.8% of the original weight.
4. Only the acid-roast-acid method gave a residual material which could be smelted in a silicon-carbide lined furnace and yet yield a bullion of sufficiently high grade to warrant the extra treatment.
5. Advantages to be gained by this last acid treatment are:—
   - (a) The weight of slime remaining to be smelted is small compared with the original weight.
   - (b) Cost of fluxes would be low for this small weight.

(c) Bullion grade is high, even when using a carbon-containing furnace.

(d) There is a better chance of finding a non-carbon lining that might resist slagging, since the treatment removes most of the zinc, the metal which hitherto has rendered high-alumina linings unsuitable.

(e) Lead chloride is easily recoverable from the hydrochloric acid solution, and could be used to replace purchased lead nitrate at present fed to the Merrill precipitator. As lead chloride has a steep solubility-temperature curve, the solution would need to be kept warmed to a minimum of 40°C., but this is neither difficult nor expensive.

(f) Copper is removed by the sulphuric acid wash as copper sulphate, which could be used in the flotation section of the plant.

6. Disadvantages and difficulties of the acid-roast-acid method are:—

(a) Removal of copper is only achieved after the other base metals have been dissolved out.

(b) The precipitate is subjected to three treatments before smelting. As the gold content increases progressively, so do any losses.

(c) Materials of construction to resist the action of hot hydrochloric acid are not readily obtainable.

(d) Washing out of lead chloride must be well done with an adequate supply of boiling water.

Treatment with Ferric Sulphate.

Preliminary laboratory scale tests have been made of the method described in a paper by Hedley and Kress,[2] whereby copper is dissolved directly from raw slime by agitation with a ferric sulphate solution acidified with sulphuric acid. The authors also found that preliminary leaching with sulphuric acid effected economy in the use of ferric sulphate by dissolving first easily soluble metals like zinc.

This method appears to have much in its favour compared with any of the other treatments so far discussed, for several reasons:—

1. Copper can be dissolved without first removing other base metals, as in the acid-roast-acid method.
2. The solutions used are not highly corrosive like hydrochloric acid.
3. Roasting is not necessary from the point of view of copper removal, although it may prove advantageous for the subsequent smelting operation.
4. Copper metal can be readily recovered, together with any gold which dissolves during treatment of the slimes with ferric sulphate.
5. Although lead would not be recovered, the intrinsic value of the lead is small, and the process does not call for an expensive acid to dissolve it.

The procedures adopted in this experimental work followed much the same course as in the original work by Hedley and Kress, viz.: Small samples of zinc slimes were treated at room temperature with the solvents being studied, for comparative periods of time, in open bottles agitated on rollers. The samples were then filtered, and washed, the residues were weighed, and the solutions analysed for copper.

Series 1. — Three tests were made, using 5 gram. (dry weight) of moist, lumpy slimes, which contained 10% of copper. Each sample was mixed with 50 ml. water and 2 ml. of sulphuric acid (sp. gr. 1.8), the required amount of ferric sulphate was added, and the bottles containing the mixture agitated for 1½ hours. The amount of ferric sulphate was the only variable in this set of tests, 15, 10, and 5 gram. being used.

The lumps of slime failed to break up during agitation, so that extraction was not as high as it might have been.

Results are tabulated below with those from the succeeding series.

Series 2. — In this series, three 5 gram. samples of fine dry slime were treated similarly to those of Series 1, except that the agitation time was increased to two hours. Extraction was higher and more consistent.

Series 3. — In the last series, each of three 5 gram. samples, similar to those of Series 2, were first agitated for ½ hour with 4 gram. of sulphuric acid and 50 ml. of water before adding ferric sulphate, 8, 6, and 4 gram. respectively, then agitating for 1½ hours.

Finally, 32.49 gram. of slime were agitated with 320 ml. of water and 24.4 gram. of sulphuric acid for ½ hour, then 32.5 gram. of ferric sulphate were added and the mixture agitated for a further 1½ hours.

Extractions were lower than in Series 2. Comparative results are tabulated below:—

	Test No.	Ferric Sulphate Gram	Copper Extracted %
Series 1	1	15	60.0
	2	10	78.0
	3	5	46.0
Series 2	4	15	75.6
	5	10	74.6
	6	5	73.1
Series 3	7	8	70.0
	8	6	68.0
	9	4	68.7
	10	32.5	68.8

Copper extraction compares with the results of Hedley and Kress, for their low-copper (4.5%) slimes.

Further experimental work is necessary before deciding whether the method can be successfully applied on a plant scale.

SMELTING IN FURNACES LINED WITH CARBON-FREE REFRACTORIES.

When copper first became troublesome about 1936, small scale tests were made in the gold-room on the effect of adding manganese dioxide to the flux, in an effort to oxidise the copper and slag it off. Four charges were run, three in Salamander crucibles, one in a Salamander crucible with a clay liner. All the graphite-containing crucibles yielded low grade bullion, whereas the clay-lined crucible gave a very high grade bullion button. The conclusion was drawn that carbon from the crucible caused reduction of copper from the slag into the bullion, and that manganese dioxide was incapable of preventing this from happening.

Efforts have been made ever since to obtain a commercial product containing no carbon or reducing agent, suitable for lining the plant furnaces, but so far without success.

The first samples tested were obtained from Nonporite Ltd., and Newbold Refractories Ltd., late in 1939. Enough of each was procured to line a small Salamander crucible.

The ground, moistened refractory was rammed into position by hand to form a layer about 1 in. thick around the sides and bottom of the crucible, being then allowed to dry out thoroughly before use by stacking above the brickwork of the muffle furnace in the gold-room.

Crucibles so prepared were annealed carefully by heating slowly in the well furnace to a red heat.

One sample of roasted slime was divided into six equal parts, and smelted with the same flux proportions for each. Results are tabulated below:—

Sample No.	Description	Bullion Assay			Slag Assay Gold Oz./L Tn
		Gold	Silver	Base	
1	Alutite Cement No. 1 "PR"	912.5	58.5	29.0	2.55
2	Mullox PBA Cement, coarse	912.5	58.5	29.0	2.45
3	Diasal Plastic Refractory	726	59.0	215.0	4.50
4	Silfractite Cement, "PR"	923.5	59.5	17.0	2.70
5	Carbolox 90	572.5	38.0	389.5	—
6	Newbold's Fireclay	863.5	86.5	50.0	39.2 and 9.6

The liner corroded very badly with Sample No. 3, and the charge came in contact with the graphite crucible, hence the bullion grade was much lower than with the other samples.

Sample No. 5, Carbolox 90, was the original proprietary silicon-carbide supplied with the Wabi furnaces. Decomposition of the carbide caused copper to be reduced and to enter the bullion.

Slags from the specially lined crucibles were all low in value, although not as fluid as they would have been if no corrosion had taken place.

No tests were made on any of these materials on a plant scale. Confirmation of the reducing effect of carbon-containing furnace lining, and on the other hand, of the advantages that would result from using a carbon-free lining have been obtained on several occasions since these early tests. See Table 10.

Test No.	Crucible Used	Slime Wt. gms.	Fluxes (gms.)				Total Wt. of Change	Bullion				Slag		Remarks
			Borax	Silica	Mang. Dioxide	Soda		Wt. gms	Gold Silver Fine-ness		Base %	Wt. gms	CU. %	
1	Graphite	200	80	40	20	30	370	71.8	74.5	139	11.6	235	1.64	
2	Clay	200	80	40	20	30	370	58.8	893	101	0.6	240	3.60	
3	Graphite	200	80	30	40	20	370	73.5	718	149	13.3	252	-	Matte on Bullion
4	Graphite	200	100	30	60	20	410	71.0	749	134	11.7	288	-	Matte on Bullion
5	Carbolox Mix	200	80	40	40	30	390	71.0	668	115	21.7	301	-	Matte on Bullion
6	Thermo Cast	200	80	40	40	30	390	37.0	894	97	0.9	265	-	Pot absorbed slag possibly some gold

Table 10.

Tests on a plant scale were made in December, 1942, using a high-duty andalusite lining supplied by H. L. Brisbane and Wunderlich Ltd., Perth, W.A. The first liner was made up in special shapes to fit the Wabi furnaces from a plaster cast pattern, and cemented into position with refractory cement.

Three charges in all were run in this furnace before the lining failed by corroding away at the slag line. Results were as follows:—

Melt No.		1	2	3
Roasted slime, lb.		86	84	80
Flux:	Borax glass, lb.	34	33	32
	Silica	17	17	16
	MnO_2	17	13	16
	Soda ash	9	9	Nil
	Fluor spar	4	4	4
Bullion weight, oz.		317.8	329	240
Bullion Assay:	Gold	771.5	716	796
	Silver	154.5	151	133
	Base	74	133	71
Slag Assay, oz./ton	Au	2.8	2.25	1.95
	Ag	125.2	64.4	118.6

Actual smelting time for each charge was 1¼ hours. The whole life of the liner, including preheating time, was 7 hours 10 minutes.

Examination of the lining before Charge No. 3, revealed a large crack across the bottom and a small crack of no consequence at the pouring lip. The lining was cut badly at the slag line to a depth of 1¼ to 1½ in., and fairly evenly over the rest of the working surface to a depth of 1 in. There were no signs of deterioration above the slag line.

During smelting of Charge No. 3, a Brisbane and Wunderlich "Bristile" high-duty andalusite brick, and a Newbold's "Kyanite" brick were thrown into the smelt. They were recovered just before pouring and examined. The Bristile brick was badly pitted and slag had penetrated some distance into it. The kyanite brick was slightly corroded on the surface, but there was no slag penetration.

A second high-duty andalusite lining was tested, with similar results to the first. Both linings were excellent in appearance after one charge had been run, but excessive corrosion started during the second run, indicating that there was an outer layer of skin of good resistance, covering a porous interior which was readily slagged.

Following the failure of these liners, it was decided to suspend further test work, temporarily at least, and proceed with investigations on the recovery of gold and silver from the mattes produced by the semi-pyritic smelting process in use at that time.

FUSION OF GOLD SLIME WITH SALT-CAKE.

Six laboratory tests were made in October, 1944, on the fusion of gold slimes with sodium hydrogen sulphate, the salt-cake obtained as a by-product in the manufacture of nitric acid from saltpetre.

In Series 1, three 50 gram. samples of moist raw slime were mixed with salt-cake in the proportions:—

Fusion 1.—2 parts salt cake to 1 part zinc slime.
Fusion 2.—3 parts salt cake to 1 part zinc slime.
Fusion 3.—4 parts salt cake to 1 part zinc slime

and heated over a kerosene burner, "Primus" type. As there was no sign of action after ½ hour, the samples were melted over a kerosene blow-lamp. The charge in each case swelled while pasty but was quiet when fused.

Fusion 1 fused with difficulty to an earthy brown mass.

Fusions 2 and 3 were blue during the pasty stage, changing to dark green when molten, and blue again on cooling. Small lumps of spongy gold could be seen throughout the mass.

Each melt, after cooling, was dissolved in about 700 ml. of water, boiled for ½ hour, then decanted through filters. With Fusions 2 and 3, the light flocculent precipitate of silver chloride was decanted away from the sponge gold through separate filters, but with Fusion 1, no such separation could be effected.

The precipitates were leached by boiling with twice their weight of sodium hydroxide.

Results are given in Table II, and with those of Series 2, the procedure for which follows:—

In Series 2, the samples were melted under controlled temperature conditions in the laboratory electric muffle furnace.

Fusion 4 was run to determine the temperature required for fusion. The charge was placed in a cold furnace and the

TABLE XI

	Series 1.			Series 2.		
Fusion	1.	2.	3.	4.	5.	6.
Weight of slime	50 gms.	50 gms.	50 gms.	50 gms.	50 gms.	35 gms.
Weight of salt-cake	100 ,,	150 ,,	200 ,,	200 ,,	150 ,,	120 ,,
Time over Blow-lamp	85 mins.	30 mins.	75 mins.	——	——	——
Time Molten in Muffle	——	——	——	15 mins.	20 mins.	45 mins.
Weight of precipitate	17.22 gms.	7.69 gms.	7.25 gms.	8.74 gms.	21.9 gms.	6.8 gms.
Weight of Sponge Gold	——	4.42 gms.	4.15 ,,	4.82 ,,	——	3.55 ,,
Weight of silver chloride ..	0.15 gms.	0.46 ,,	0.75 ,,	1.00 ,,	——	——
Weight of ppt. after NaOH	9.03 ,,	1.86 ,,	1.66 ,,	1.91 ,,	11.06 gms.	1.56 gms.
Assay of sponge gold. Gold	——	841 fine	880 fine	862 fine	——	906 fine
Assay of sponge gold. Silver	——	83 ,,	75 ,,	77 ,,	——	6 ,,
Assay of leached residue. Gold	443 fine	70 ,,	16 ,,	230 ,,	464 fine	142 ,,
Assay of leached residue. Silver	109 ,,	242 ,,	325 ,,	258 ,,	96 ,,	296 ,,
Percentage lead in leached residue	——	——	25.6%	21.8%	——	——

Analyses of Residues after Leaching with Caustic Soda.

Fusion	2 per cent.	3 per. cent.	4 per cent.
Gold	7.0	1.6	23.0
Silver	24.2	32.5	25.8
Lead	27.7	25.6	21.8
Zinc	—	0.63	0.76
Insoluble	2.8	6.2	4.2

temperature gradually raised until the charge melted at 600°C. The mass was similar in appearance to Fusions 2 and 3. Subsequent treatment was as for Series 1. Results were slightly higher, because the slime had dried out somewhat over a 24 hour interval, so that the original 50 gram. actually contained less water and more metal than Series 1 samples.

Fusion 5 was placed in the muffle which had been previously heated to 400°C., and raised quickly to 625°C. It was held at that temperature for 20 minutes, then poured. Part was granulated — not very successfully, as it exploded — by pouring into water. The fused mass was an earthy brown colour and was difficult to dissolve. The sponge gold could not be separated from the flocculent precipitate by decantation.

Fusion 6 was placed in the muffle at 400°C., brought quickly to 625°C., held at that temperature for 45 minutes, then poured. The fused mass, when cold, was a dark olive-green colour. It dissolved easily and a good separation was made by decantation.

A large-scale test, 50 lb. of slime, was next made on 23rd November, 1944, using an oil-fired, cast iron melting pot, and an improvised, rather poor leaching apparatus.

Plant results for the same day were: 764 lb. slime yielded 879 fine oz. of gold and 169 fine oz. silver, including values contained in the matte and slag. Hence 50 lb. taken for the experiment should have contained approximately 57.5 fine oz. gold, 11.1 fine oz. silver.

The 50 lb. sample of slime was mixed with 170 lb. of salt-cake, and fed into the previously heated melting pot at 10.40 a.m. Violent action took place with some spitting. The burner was turned off until 11.05 a.m.

Melting then proceeded unevenly, as one side only of the pot could be kept hot, until 1.45 p.m., when all action had ceased, and the burner was turned off.

The fused mass was ladled out, cooled, and placed in a tub for dissolving. Water was added, and the resulting liquor withdrawn from time to time, being transferred to a drum for settlement. Fresh water was added after each withdrawal of solution, a total of about 40 gal. being used.

Next morning, the liquors were decanted and the undissolved fine sludge recovered.

Two products were obtained for further treatment, gold sponge, and insoluble lead sulphate.

Gold Sponge was boiled with 2 lb. caustic soda to dissolve lead sulphate. Some silver chloride was present, due to sodium chloride in the tap water used.

The extracted gold sponge, after filtering and drying, weighed 69 oz. It was melted with a little borax in a graphite crucible, yielding 56.5 oz. of bullion assaying:—

Gold	989, equivalent to 55.88 fine oz.
Silver	9, equivalent to 0.51 fine oz.
Base metal ...	2

Insoluble Lead Sulphate: The clear liquor was decanted and the solids filtered. The solution contained neither silver nor copper. Owing to an iron drum being used, the solids contained cement copper. The 14½ lb. obtained assayed:—

Gold	268 oz. per ton, equivalent to 1.7 fine oz.
Silver	1,212 oz. per ton, equivalent to 7.8 fine oz.
Lead	22.25%, equivalent to 32.6% $PbSO_4$
Copper	15.0%

On smelting in a clay crucible, this slime yielded 50.5 gram. of bullion assaying:—

Gold	858, equivalent to 1.39 fine oz.
Silver	136, equivalent to 0.22 fine oz.
Base metal	6

Total gold recovered	57.27 fine oz.
Total silver recovered	0.73 fine oz.

A second test, using 20 lb. of slime, was made on 4th December, 1944, after alterations had been made to the furnace brickwork in an effort to make heating of the cast iron melting pot more even.

Plant returns for the same day were: 1,136 lb. of slime yielded 1,228 fine oz. gold and 198 fine oz. silver, including values present in the matte and slag. Hence 20 lb. taken for the experiment should have contained 21.6 fine oz. gold and 3.7 fine oz. silver.

On this occasion the sample was not mixed with salt-cake, but added to the melting pot in layers. Some spitting still took place, however. The weight of salt-cake used was 60 lb. Time of melting was 3 hours 5 minutes.

Treatment was similar to the previous test, except that precipitation of copper was prevented by using an earthenware vessel instead of a steel drum for the decanted solution. Results were as follows:—

Gold sponge:

Final weight after leaching		29 oz.
Yielded bullion		25 oz.
Bullion assay:—		
Gold	924, equivalent to 23.1 fine oz.	
Silver	15, equivalent to 0.35 fine oz.	
Base metal	62	

Solution: No silver in solution, therefore discarded.

Lead Sulphate Residue: Weighed 6½ lb. This was thrown into the plant slag.

Conclusions drawn from the whole set of tests are:—

1. Fusion with salt-cake removes nearly all the base metals, leaving gold sponge, which can be melted and poured directly into bars of high grade bullion, and a lead sulphate precipitated, which can be readily smelted in a clay-lined crucible to yield high grade bullion.
2. The melt can be performed in an iron furnace that requires no high-cost refractory liner.
3. Melting is slow.
4. The molten sulphates are difficult to handle as they cannot be granulated.

CONCLUSION.

While it is realised that the formation of a matte and the subsequent retreatment of the products from such a procedure is not ideal, it is still current practice in the gold-room at the Lake View and Star Ltd.

It can be considered as a compromise method of treatment until the ideal method is found.

The latter part of this paper has dealt with attempts to find this ideal method, and work is still continuing with that object in view.

The method desired should fulfil the following requirements:—

1. All bullion produced should be higher in grade than the minimum of 98% total gold plus silver, set by Government regulations, to avoid Mint refining charges.
2. No by-product to be produced containing quantities of payable precious metals that cannot be treated on the mine.
3. The whole procedure to be quick in the return of bullion, to avoid locking up values in the smelting room.
4. It must be safe from the operatives' point of view, and not unduly arduous or exhausting.
5. It must be sound economically, so that the cost for smelting and refining does not exceed the benefits to be derived from lessened Mint charges.

The matte smelting procedure in use at present, and described fully in this report, when analysed on the above lines is found to fulfil the conditions set out in (2), (3) and (4), in that no by-product has to be shipped for recovery of values, that it is a quick and easy smelting procedure with no difficulty or arduous operation, and that it does not require highly skilled operatives to watch continuously any step in the whole procedure.

It falls far short of satisfying condition (1). It is a cheap process, however, and the additional work done on by-products is more profitably done on the mine than by sending them to a smelter.

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