

Electric smelting at Rustenburg Platinum Mines Limited of nickel - copper concentrates containing platinum-group metals

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SYNOPSIS

The processes involved in the smelting in an electric furnace of nickel-copper concentrates containing platinum-group metals are described. These are concentrate drying, pelletizing, pellet drying, and flux and pellets proportioning, followed by smelting of the concentrate in a submersible-electrode electric furnace, which is discussed in more detail. The furnace matte is blown in a converter to the white-metal stage and is cast in moulds, whilst the slag is granulated, ground, and floated for recovery of valuable metals and afterwards pumped to the tailing dam.

SINOPSIS

Die prosesse betrokke by die smelting van nikkel-koper konsentrete, wat platina groep metale bevat, in 'n elektriese oond word beskrywe. Hulle omvat konsentraat droging, konsentraat korrelvorming, korrel droging en proporsionering van smeltmiddels en korrel konsentraat wat opgevolg word deur smelting van die konsentraat in 'n elektriese oond met ondergedompelde elektrodes wat meer noukeuriger beskrywe word. Die rusteenkoper-nikkel van die oond word in 'n omsetter tot die witmetaal stadium geblaas en gegiet terwyl die slak gegranuleer, gemaal en geflooteer word vir die herwinning van metale, en daarna na die sliksdamme gepomp word.

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INTRODUCTION

The Rustenburg Platinum Mines, Limited, Rustenburg Section (R.P.M.) is situated near the town of Rustenburg in the Transvaal Province of the Republic of South Africa. The ore-body lies in the Bushveld Igneous Complex, and the major economic metal in the mineral deposit is platinum, though the other platinum-group metals, nickel, copper, gold, and silver are also recovered. Ore is treated by gravity and flotation processes, and the flotation concentrate is smelted to a matte.

When the mine embarked on an expansion programme for increased production of platinum-group metals, it was necessary to decide which type of smelting should replace the blast-furnace smelting practised at that time.

Not only was blast-furnace smelting labour-intensive, but the price of coke had risen steadily and there was no room for further expansion at the site. Moreover, at that time

the South African Government also introduced stringent antipollution laws. The emission of a very large volume of gas containing between 1 and 2 per cent sulphur dioxide presented a pollution problem that would be costly to solve and control.

The Consulting Metallurgists of the Johannesburg Consolidated Investment Company, the parent company of R.P.M., therefore sought alternative smelting methods. Realizing that the R.P.M. concentrate contains a high percentage of magnesium, they immediately focused their attention on electric smelting, where high operating temperatures could be tolerated. A further attraction was that power costs in the Republic of South Africa are fairly low at about R0,05 per kW h and that the cost of power tends to rise more slowly than the cost of most other commodities.

Eventually the Consulting Metallurgists recommended a 19,5 MV A rectangular electric furnace with six in-line electrodes operating in pairs in a three-phase electrical system. They also recommended that pelletization of concentrates, followed by drying of the green pellets, should be adopted to prepare the concentrate charge to the furnace.

This recommendation was approved by the Board early in 1967,

when planning of the new smelter commenced. The plant was finally commissioned in 1969.

TECHNOLOGY OF ELECTRIC SMELTING

Electric smelting of sulphide ores as compared to reverberatory smelting has been described in some detail by Dr Otto Bath¹.

There is a similarity between the two smelting operations in the sense that charges to both types of furnaces are basically the same. The shapes and sizes of the furnaces are also similar in the rectangular form, although there has recently been a move towards circular electric furnaces.

The main differences, however, lie in the energy input and heat transfer, the formation and distribution of magnetite, and the roof construction.

Since the temperature above the charge is relatively low in an electric furnace as compared with a reverberatory furnace, a sprung arch of cheap firebricks as opposed to a suspended arch of expensive bricks is an obvious choice. The two most vulnerable parts in a reverberatory furnace are the roof and the side walls at the slag line. In electric smelting the strains on the roof and furnace side walls have been eliminated by the difference in energy input.

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The heat required to smelt a charge in a reverberatory furnace is mostly generated from the combustion of coal, oil, or natural gas. The combustion flame travels through the furnace with decreasing temperatures towards the slag end and leaves the furnace through the flue system. The charge is heated mainly through convection, but also by direct impingement from the flame or indirect radiation from the roof at the burner end where the temperature is the highest. The transfer of heat in a reverberatory furnace is from the top to the bottom zone, i.e., from the flame to the charge to the slag and matte. The thermal efficiency in the reverberatory furnace is therefore low — about 30 per cent (sum of the heat in the matte and slag, as well as the heat required for endothermic reactions, divided by the heat input) — as compared with that of electric smelting, which is about 60 per cent or higher.

The major heat source in an electric furnace is from electrical energy and is produced in the slag by the electrical current according to the formula I^2R . The heat produced in the

matte can be ignored because of the very high conductivity of the matte. It is thus obvious that the slag must have fairly low conductivity to promote the usage of relatively high voltage to avoid excessive amperage. Heat development efficiencies will be dependent on (a) the intensity of the current, (b) the distance between electrodes and between the electrodes and matte surface respectively, and (c) the specific resistivity of the slag. The specific resistivity of the slag is determined by the composition of the slag, which in turn is dependent on the composition of the raw materials. Slag composition thus offers an operational variable within economic limits. For high electrical efficiency it is advantageous to work at a high power factor, which involves changing of the other two variables, viz, the current and the distance through which the current must flow through the slag. The current normally flows through the liquid slag between a pair of electrodes ($\frac{1}{3}$ to $\frac{1}{4}$ of total energy) and between the tip of the electrode and surface of the matte (the remaining $\frac{2}{3}$ to $\frac{2}{4}$ of total energy). Practical limits in the design of the furnace

more or less determine the distance between electrodes, which leaves only one operational variable, viz, the distance between the electrodes and the surface of the matte. A deeper slag layer, higher voltage, and better power factor are thus the objectives for efficient operation of submerged-electrode slag-resistance electric furnaces.

The heat transfer in an electric furnace is from the slag layer up and down, i.e., from the slag to the charge and slag to matte, which results in electric-furnace matte being hotter than reverberatory-furnace matte. There are no combustion gases in an electric furnace — only gases formed by the reactions in the charge.

Magnetite not reduced and fluxed dissolves in the matte and slag. In a reverberatory furnace, the undissolved magnetite settles on the furnace bottom and forms accretions, mostly because of the lower temperatures and laminar flow of the matte and slag towards the slag-tapping end. In the electric furnace, matte and slag are both in turbulent motion and, since the matte is at a

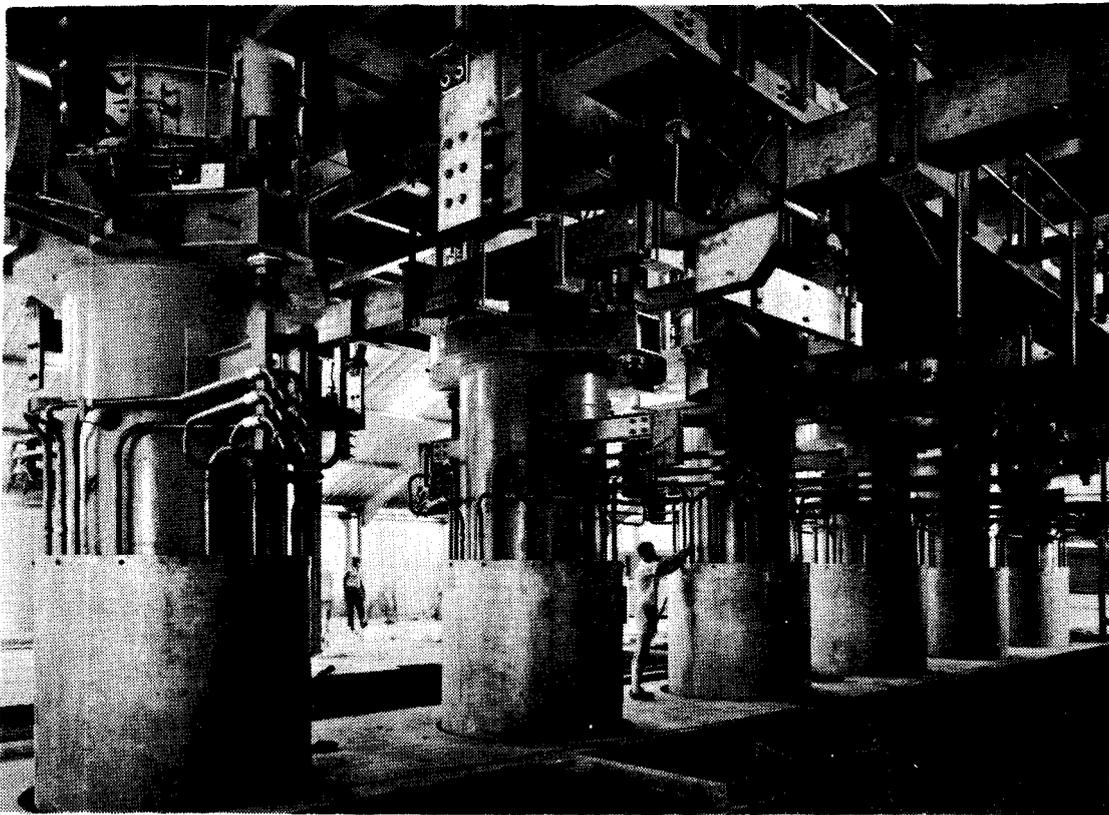


Plate 1—Six in-line Soderberg-type electrodes for the 19,5 MV A electric furnace

higher temperature, more magnetite is dissolved in the matte. The magnetite sometimes forms an intermediate zone of high viscosity between the matte and slag layers. Magnetite layers or accretions can readily be dissolved in an electric furnace by lowering the voltage and bringing the electrodes nearer the matte surface, thus imparting extra heat into the matte at localized spots. Sometimes it is necessary to charge coal and silica to these areas — an operation that is not always successful.

ELECTRIC-FURNACE CONSTRUCTION AND PRINCIPLES OF OPERATION

The electric furnace as supplied to R.P.M. by Elkem, Norway, is rectangular in shape with external dimensions of 27,2 m (89' 8") long X 8,0 m (26' 4") wide X 6,0 m (19' 8") high. The measurements inside the brickwork are as follows: length 26,0 m (84' 11"), width 7,0 m (23'

and height 4,5 m (14' 10") (on centre line between roof and bottom). The furnace is built on 102 concrete pillars positioned in six rows of seventeen each. The floor of the furnace is built on cast-iron base plates cooled by circulating air. The side walls have 2,54 cm steel guard plates cooled by the external application of water. The buckstays and tie rods are the same as in a reverberatory furnace except that the tie rods are manufactured with sections of anti-magnetic stainless steel. The correct tension on the rods is maintained by means of helical springs. The furnace lining is of magnesite, but the underbase of the inverted arch, the upper part of the walls above the slag level, and the roof (sprung arch) are constructed of superduty firebricks.

The charge openings are placed in the roof, four around each electrode for centre charging and one near each side wall opposite the elec-

trodes. Four gas off-take openings and 16 inspection doors are also situated in the roof. There are 7 inspection doors on each water-cooled side wall. There are three slag tap holes at the slag tap end and three matte tap holes at the matte tap end of the furnace (see Fig. 1 and 2).

Energy input to the electric furnace is by means of three pairs of electrodes rated at 6,5 MVA per pair for a total rating of 19,5 MVA. The electrodes are of the Söderberg type and are 1,25 m (49") in diameter, each with a current density of 2,65 A/cm² (17 A/in²) (Plate 1). The distance between the electrodes measured centre to centre is 3,4 m (11' 2") between phase pairs and 3,4 m between phases. There are three transformers rated at 6,5 MV A each. Power is supplied from the South African Electricity Supply Commission grid at 6600 V to the primary windings of the transformer. On

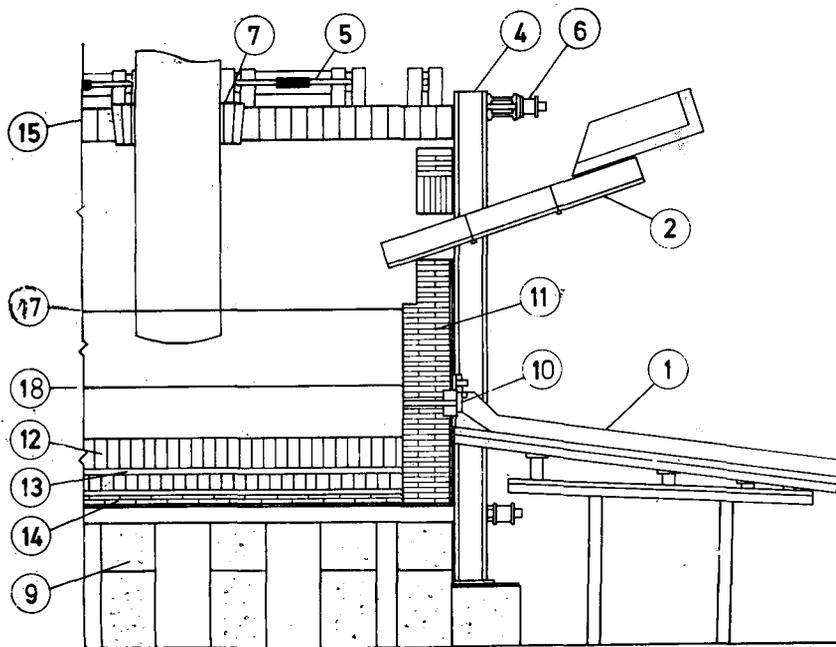


Fig. 1—Longitudinal section of matte tapping end of 19,5 MVA electric furnace

- | | |
|----------------------------------|-------------------------------------|
| 1. Matte launder | 10. Matte tapping blocks |
| 2. Converter slag return launder | 11. Magnesite refractory bricks |
| 3. Drain out tap-hole | 12. Magnesite hearth and sub-hearth |
| 4. Buckstay | 13. Magnesite castable |
| 5. Tie-rod | 14. Fire-clay bricks |
| 6. Tie-rod spring | 15. Firebrick roof |
| 7. Electrode openings | 16. Slag tap-holes |
| 8. Furnace inspection ports | 17. Average slag level |
| 9. Furnace support pillars | 18. Average matte level |

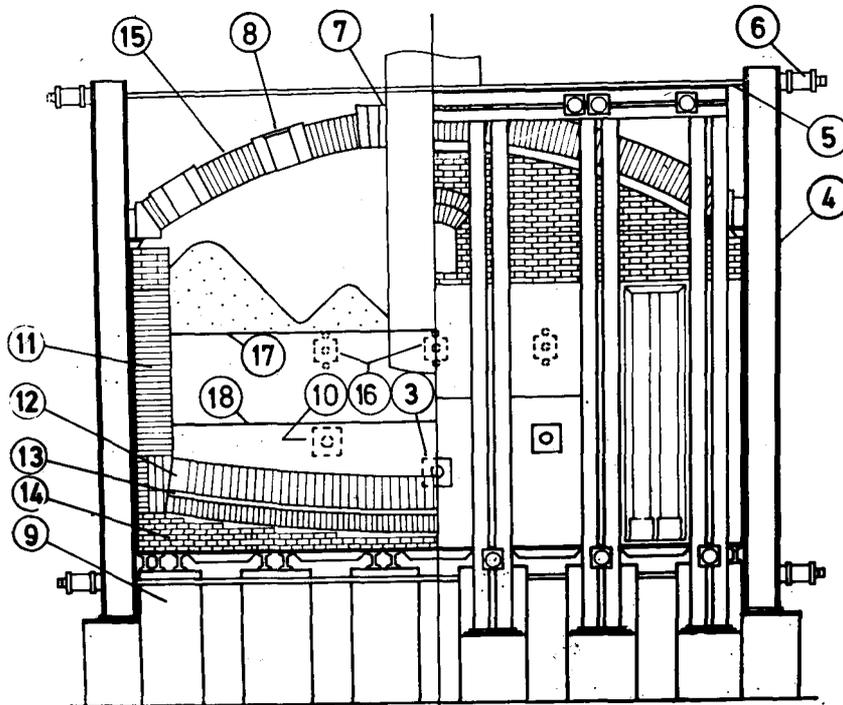


Fig. 2—Cross-section through 19,5 MV A electric furnace

- | | |
|-----------------------------|-------------------------------------|
| 3. Drain out tap-hole | 11. Magnesite refractory bricks |
| 4. Buckstay | 12. Magnesite hearth and sub-hearth |
| 5. Tie-rod | 13. Magnesite castable |
| 6. Tie-rod spring | 14. Fire-clay bricks |
| 7. Electrode openings | 15. Firebrick roof |
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| 9. Furnace support pillars | 17. Average slag level |
| 10. Matte tapping blocks | 18. Average matte level |

delta connection, the secondary voltage of the transformer can be varied from 170 to 350 through seventeen steps by on-load tap changers. The maximum electrode current is 32 400 A at transformer tap 4 or 201 V (6,5 MV A). The current is carried by copper bus bars and flexible cables, which are connected to the six contact pads by means of water-cooled copper pipes. The demineralized water not only serves as a heat-dissipating medium, but also exerts pressure of 70 kPa (40 lb/in²) on the contact pads to secure good electrical contact. From the contact pads the current is carried by baked-carbon paste and flows through the liquid slag between electrodes, the tips of which are submerged in the slag to a depth of about 48 cm (19"), as well as between the electrode tips and matte layer. Owing to the resistance of the slag, heat is generated in the slag and is transferred to the charge by direct contact and convection.

The electrodes are loaded with

carbon-paste cylinders as illustrated in Fig. 3, which ensures that the paste is well baked and all volatiles driven off by the time electrical contact is made. The electrodes are consumed at a rate of about 13 cm (5") per day, and the electrode casing is extended whenever required by welding on 1,75 m (69") sections to the top. Argon welding is used to secure a solid weld free of pinholes. Because the electrodes are continuously burnt away at the bottom, slipping of the electrode casings through the holders by means of a grab-and-release pneumatic-hydraulic combination system is required. The slipping device consists of an upper and a lower ring connected by four hydraulic jacks. The rings are cast steel with rubber diaphragms in the inside. During slipping, the upper ring is deflated and pushed up along the electrode casing by means of the hydraulic jacks. The upper ring is then inflated, the lower ring deflated, and the electrode casing pushed down

through the contact pads, the hydraulic jacks being used to overcome the friction of the contact pads. The lower ring is inflated again and the process repeated automatically at fixed time intervals.

The power input through the three phases (three pairs of electrodes) is kept in balance automatically by regulating the depth of immersion of each individual electrode in the slag, and thus regulating voltage, by means of the two 24 cm (9½") dia. hydraulic hoists per electrode. The hoists have a maximum travel of 1,22 m (4'), and are operated at 5,86 MPa (850 lb/in²) hydraulic pressure. The power input can be set manually and controlled automatically at each desired setting. This is done by means of instruments situated in the control room.

Other instrumentation includes power-consumption meters, temperature recorders, furnace-draft gauges, automatic slipping-device regulators, water-pressure and temperature indicators, etc.

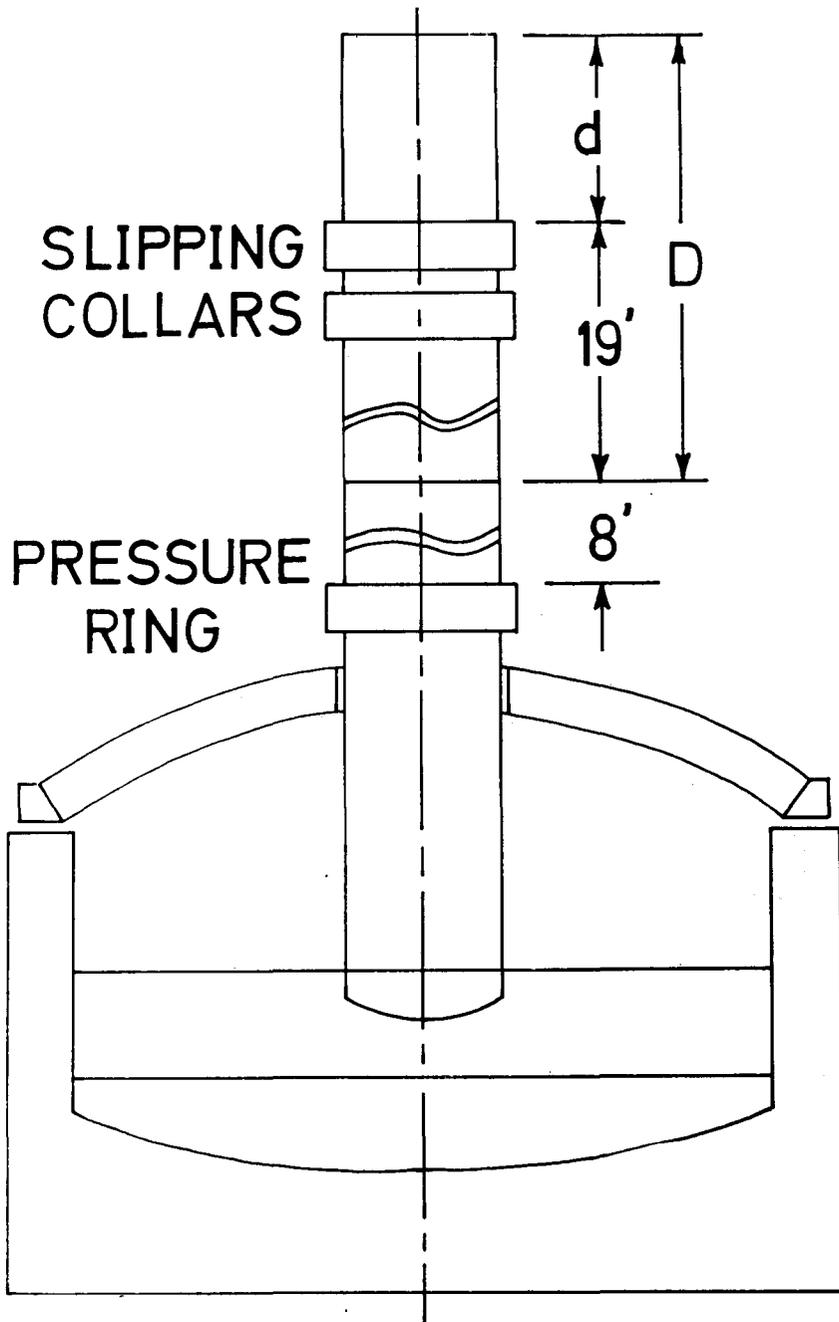


Fig. 3—Paste loading diagram to formula $L = D - d = 5,8 \text{ m (19 ft)}$, where L is the correct length

SMELTING PRACTICE AT WATERVAL ELECTRIC SMELTING PLANT — R.P.M.

A flow diagram of Waterval Smelting Plant (Fig. 4) is included as a basis for the description of electric smelting and its related operations.

Typical analyses of concentrates obtained from ore mined from the Bushveld Igneous Complex are shown in Table I. The major gangue minerals are pyroxene, plagioclase, feldspar, and biotite. The base-metal sulphide

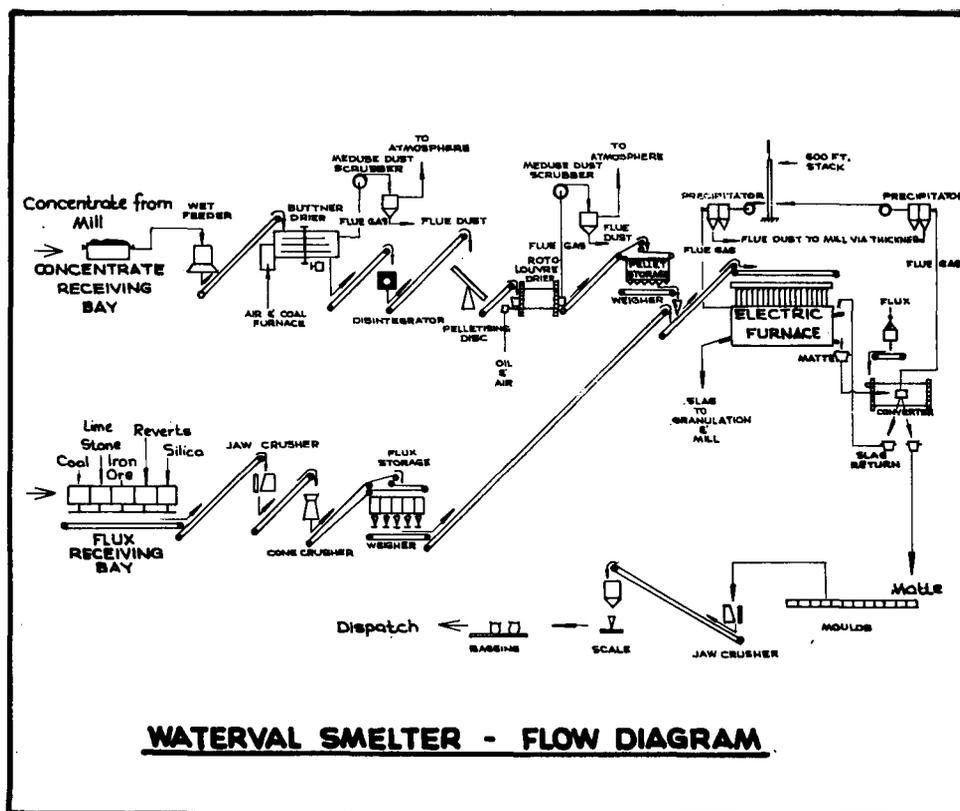
minerals are chalcopyrite, pentlandite, and pyrrhotite. The platinum-group metals are either associated with the base-metal sulphides or occur in the form of minerals such as braggite, cooperite, laurite, or ferroplatinum. It is obvious that R.P.M. aims at the production of platinum-group metals at maximum recovery. The concentrate is thus a low-grade base-metal concentrate by most standards with a high MgO content that requires fairly high smelting

temperatures.

Preparation of the charge

The concentrate is dried in Büttner dryers from 21 per cent to about 7 per cent moisture. The driers have a number of rotating hearths with fixed rabbles. The concentrate is top-fed and passes downwards, while hot air from coal-fired furnaces is distributed uniformly through the dryer by a centrally located turbine.

The temperature and volume of the hot air are automatically controlled,



WATERVAL SMELTER - FLOW DIAGRAM

Fig. 4—Flow diagram of Waterval smelter

TABLE I
TYPICAL ANALYSES OF CONCENTRATE AND FURNACE PRODUCTS

Material	PGM g/t	Ni %	Cu %	Fe %	S %	FeO %	CaO %	MgO %	Al ₂ O ₃ %	SiO ₂ %
Concentrate (pellets)	110-150	3,5-4	2-2,3	15,0	8,5-10	—	3	15	6	39
Furnace matte	500-600	16-18	9-11	38-42	26-28	—	—	—	—	—
Converter matte	1 800-2 000	47-48	27-28	1-2	20-21	—	—	—	—	—
Furnace slag	0,54	0,10	0,06	—	0,27	20	15	15	6	41

and gas is exhausted at low temperature to the wet scrubbers.

The dried concentrate is disintegrated to a fine powder and fed to pelletizing pans of 3 m diameter. (Plate 2). The pellets are about 1,5 cm ($\frac{3}{8}$ " diameter, contain 10 per cent moisture, and have a breaking strength of 5 to 6 kg, although no binding agent is used.

The green pellets are dried to about 2 per cent moisture in oil-fired roto-louvre dryers. The dried pellets have a crushing strength of 9 to 13 kg, and are transferred to storage bins.

Fluxes, mostly limestone (94 per cent CaCO₃) and occasionally iron ore, required to produce a suitable slag, and reverts, are crushed to

minus 1,91 cm ($\frac{3}{4}$ ") and stored in their individual bins. Pellets and fluxes are blended by a weight-proportioning system (Kukla weighers) and transported to the electric furnace by conveyor belt and fed to the charge bins by means of shuttle conveyors. At present the charge consists of pellets 76 per cent, limestone 22 per cent, and reverts 2 per cent.

Smelting

In the original planning, it was intended that wet concentrate should be fed to the furnace. On this basis, the capacity of each 19,5 MVA furnace was estimated at 10 000 tonnes of concentrate per month. Before the plant was erected, however, it was decided that the concentrate would be pelletized. At an operating rate of

18,5 MW, a smelting rate of 12 500 tonnes of concentrate per month can be maintained. The factor that most influenced the 25 per cent increase in smelting rate is undoubtedly the pelletization of concentrates. Because pelletization reduces flue-dust circulating loads, it permits sufficient draught to keep roof temperatures within operational limits, which are in the region of 250°C. Pelletization also provides a means of feeding a well-blended porous charge to the furnace, which allows reactions to proceed rapidly and reaction gases to pass unhindered to the atmosphere above the charge. Since the charge floats on top of the slag, it completely covers the slag to form an insulating medium

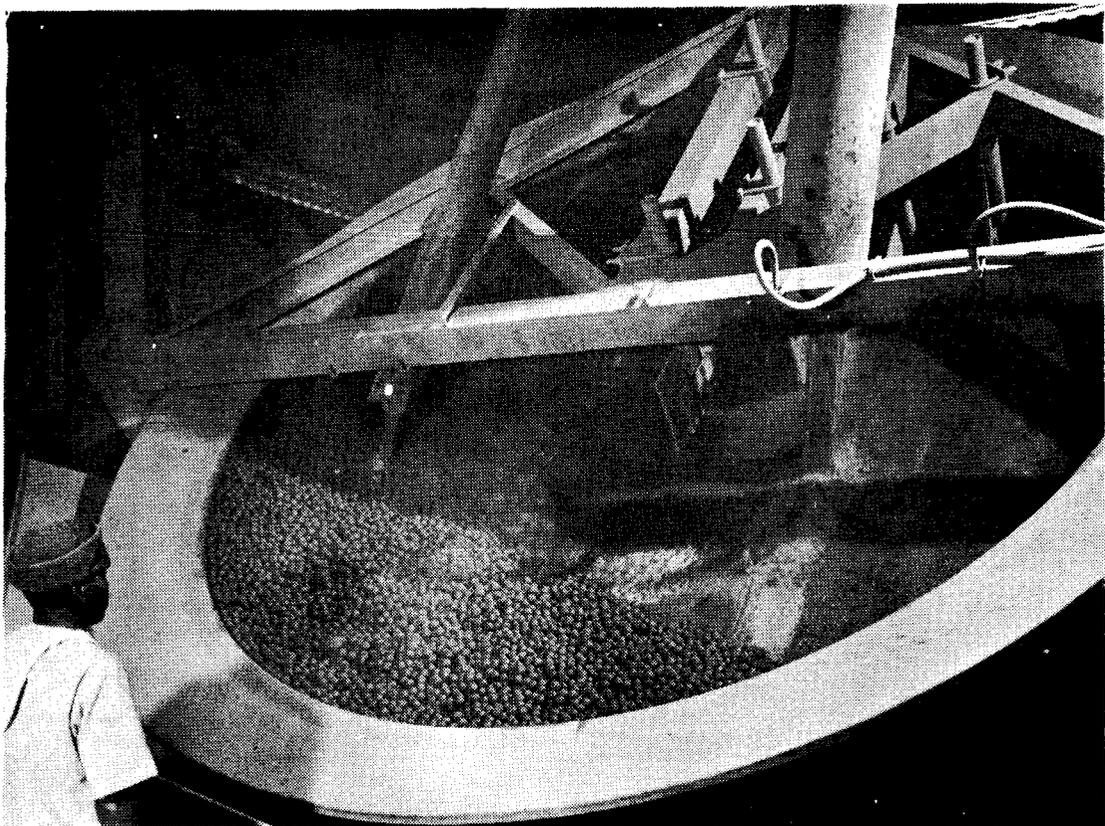


Plate 2—Pelletizing pans (3 m in diameter) forming about 1,5 cm diameter pellets

between the hot slag and the atmosphere above the charge. This not only promotes faster smelting of the charge, but also protects the furnace roof from direct heat and hence prevents excessive heat losses. The mixed charge can also be piled up around the electrodes to stabilize them, thus preventing drifting and swinging. Pelletization at R.P.M. is a relatively cheap operation and offers numerous advantages.

The centre of the furnace is intermittently charged through four ports by means of pneumatically operated chutes, which are manually controlled. The ports are situated around the electrode. Two ports opposite each electrode, one for each side wall, are operated on a choke feed for side-wall protection above the molten-slag line. These side ports are self-feeding because of the nature of the charge, and so require no control.

Molten converter slag is returned by ladle to a cast-steel launder projecting slightly into the furnace through the matte-tapping end wall. The converter slag, consisting mainly of fayalite ($2\text{FeO}\cdot\text{SiO}_2$) with some

dissolved magnetite, has a higher conductivity than furnace slag, which results in a lowering of the voltage between the tips of the first pair of electrodes and the matte layer. Since the power input through the three pairs of electrodes is in balance and automatically controlled, the first pair of electrodes will automatically be raised to increase the voltage between them and the matte layer. This has a cooling effect on the matte. Magnetite tends to settle out from the slag to form a layer between the slag and the matte. This not only results in the formation of accretions on the front wall, but also tends to have a cooling effect on the matte, from which some magnetite settles out on the bottom of the furnace. The net effect is a bank formation against the front wall that interferes with matte tapping procedures.

The problem has been overcome by operating the first pair of electrodes on manual control while converter slag is returned. The voltage of the pair of electrodes is reduced by lowering the transformer taps, and the electrodes are

manually lowered deeper into the slag bath until power input equal to the input to any other pair of electrodes flows through the slag at higher amperage and lower voltage. Extra heat is thus supplied to the matte layer, preventing magnetite from settling out and favouring its reaction with FeS ; it is thus reduced to FeO which is slagged off as $2\text{FeO}\cdot\text{SiO}_2$. Care must be taken not to overheat the matte. Although this procedure has been effective in the control of magnetite build-up, the problem of magnetite build-up against the front wall would be less if the converter slag were returned by a cast-iron launder running on top of the furnace roof and discharging the slag near the first electrode, as at Rönsskar, Sweden. The slag return hole in the roof at Rönsskar is sealed off with a cast-iron ball.

Furnace slag is continuously tapped at $1300\text{--}1380^\circ\text{C}$ from one or two of the three tap-holes, depending on slag production in the furnace. The three tap-holes are located at 1,56, 1,76 and 1,96 m (62", 70" and 78") intervals above the furnace hearth

bottom; one is centrally positioned, and the other two are set at 1,44 m (4' 9") off centre. A slag level of 2,1-2,3 m (82" to 90") is normally maintained in the furnace, i.e., approximately 1,32-1,52 m (52"-60") of slag on top of the matte layer, for optimum smelting conditions. With a slag level of 2,08 m (82") and a matte bath of 0,76 m (30"), the actual depth of the slag layer will be 1,32 m (52"). With an electrode immersion of 0,48 m (19"), the distance between the tip of the electrode and the slag-matte interface will be 0,84 m (33"). The greater the distance between electrode tip and slag-matte interface, the greater the difference in the matte and slag temperatures. This is an important operational factor in dealing with slags of high formation temperatures (1300-1400°C) in comparison with mattes of relatively low formation temperatures ($\pm 1000^\circ\text{C}$), especially when the viscosity of the matte is low at the slag formation temperature. The differences in slag and matte formation temperatures are thus of vital importance in the design of furnace hearths and transformers.

It has also been noted that, in instances where the matte level is high—about 1,0 m (40")—and slag level low—about 1,8 m (70")—with distances between the electrode tip and slag-matte interface of the order of 0,28 m (11"), submerged arcing occurs and results in excessive turbulence around the electrodes, causing electrical instability in the furnace. To rectify these conditions, the power has to be reduced until the slag level is sufficiently raised or the matte level is lowered.

Disposal of slag

Furnace slag is tapped continuously, generally from the 1,76 m (70") level spout. It is carried in a high-pressure—about 550 kPa (about 80 lb/in²)—stream of water (water to slag 15:1 by weight) into a Duplex Rake classifier, where it is further granulated, de-watered, and transferred to a belt conveyor feeding the slag grinding and flotation circuit. Typical slag analyses are given in Table I.

The following factors influence the effective granulation of slag by the described method:

(a) water to slag ratio,

- (b) water pressure,
- (c) water temperature,
- (d) size, shape, and slope of launder,
- (e) smoothness of lining of launder, and
- (f) speed of classifier rakes.

The granulated slag is milled in ball mills in closed circuit with cyclone classifiers to about 60 per cent minus 200 mesh. Overflow from the classifiers passes direct to a conventional flotation circuit, where some of the residual metal is recovered. Flotation tailing is thickened and pumped to the concentrator tailing sump for final disposal in the main tailing dam.

Tapping of matte

A matte level of 59 to 76 cm (23" to 30") (see Fig. 2) is maintained in the furnace, and matte is tapped at regular intervals through one of two tap-holes situated 44 cm (19") above the furnace bottom 1,44 m (4' 9") on either side of the furnace centre line. The third tap-hole is set at the furnace centre line at the bottom of the furnace above the inverted arch to serve as an emergency drain hole. The original tap blocks were chrome refractory bricks of 35 X 35 X 15 cm (14" X 14" X 6") with a 3,8 cm (1½") dia. hole in the centre, held in position by steel wedges attached to the main copper cooling jacket. These have been replaced successfully by 60 per cent alumina bricks, which are less prone to cracking under the impact of the mud gun. The refractories around the tap holes are watercooled by means

of copper jackets. Owing to the high pressure exerted by the matte and slag at the tap-holes, these must be closed by a clay gun. Typical furnace-operation data are presented in Table II.

Converting

The furnace matte (similar in composition to that given in Table I) is tapped into 20-tonne capacity cast-steel ladles refractory lined. The matte is transferred by 40-tonne overhead cranes to 3,05 m X 6,1 m (10' dia. X 20') long Pierce-Smith type converters with 28 tuyeres of 5 cm (2") dia. each. The tuyeres are automatically punched using Ken-necott type 4B tuyere punchers. An air blow of about 30-33 m³/s (10 000-11 000 ft³/min) at 70-80 kPa (10 to 12 lb/in²) is maintained. The converters are lined with chrome magnesite bricks of South African manufacture, which are consumed at a rate of 35 kg per tonne of converter matte produced.

Silica flux is weighed and added by means of a belt conveyor through a side spout to the converter. The free silica in the flux is controlled to 65 per cent by reverts addition, and the moisture content is kept below 4 per cent. The temperature of the refractories is measured by optical pyrometers and kept to below 1300°C at all times. Typical converter-operation data are presented in Table III.

About 14 tonnes of converter matte is cast into cast-iron moulds at the end of the blow. The converter

TABLE II
TYPICAL FURNACE-OPERATION DATA

POWER	
Load	18,5 MW
Power factor	0,98
Consumed: per tonne of concentrate	897 kWh
per tonne of charge	689 kWh
MATERIALS INPUT	
Concentrate (pellets)	18 t/h
Total charge	23 t/h
Converter slag	90 to 110 t/d
MATERIALS OUTPUT	
Furnace matte	4-5 t/h
Furnace slag	21 t/h
TEMPERATURES	
Furnace matte	1 180°C
Furnace slag	1 350°C
RATIOS	
Slag:matte ratio	4,6:1
Slag:concentrate ratio	1,1:1
Slag: granulation water ratio	1:15
ELECTRODE-PASTE CONSUMPTION	
Per tonne of concentrate	3,5 kg
Per tonne of charge	2,7 kg

TABLE III
TYPICAL CONVERTER-OPERATION DATA

Converter blows	2-3 per day
Average cycle time	5,26 h
Average blowing time	3,82 h
Furnace matte, tons per blow	44 t
Converter matte, tonnes per blow	14 t
Converter slag, tonnes per blow	45 t
Air volume	30-33 m ³ /s
Pressure	70-80 kPa

matte (for typical analysis see Table I) is dumped from the moulds by cranes, crushed to minus 1", and bagged for dispatch to the matte refineries. The platinum-group metals follow the nickel and copper into the matte. The total recovery of platinum-group metals from the base-metal sulphides is about 99 per cent or better. The recoveries of copper and nickel from electric-furnace and converting operations are of the order of 97 and 96 per cent respectively.

Disposal of exhaust gases

The exhaust gases, 30 m³/s (60 000 ft³/min) at 250°C and 88 kPa (26" Hg) from the furnace, and 34 m³/s (68 000 ft³/min) at 370°C and 88 kPa (26" Hg) from the converters, pass through the flue system, electrostatic precipitators, and up a 183 m (600') stack

with an acid-resistant brick lining. This stack was recently erected to replace the original two 61 m (200') stacks (Plate 3). Future plans include sulphuric acid production from the SO₂ in the converter gases. Of the sulphur entering the smelter and converter operations, some 60 per cent leaves with the converter gases varying in SO₂ content from 2,5 per cent to 6 per cent (for 70 per cent of blowing time the SO₂ content is higher than 4 per cent), 20 per cent with the furnace gases averaging 0,4 per cent SO₂, 15 per cent with the converter matte, and 5 per cent with the furnace slag.

Recovery of flue dust

The flues are fitted with a series of bottom outlet ports from which dust is removed. This dust, together with

that recovered from the precipitators, is washed down launders to a central pump pit, from where it is pumped to a thickener. The thickened pulp is filtered and added to the flotation concentrate ahead of the Büttner dryers.

Heat balance

Furnace heat input can roughly be divided into electrical energy 87 per cent, converter slag 10 per cent, and combustion heat of carbon electrodes 3 per cent. Furnace heat output can be divided into the following categories:

Furnace slag	49 per cent
Endothermic reactions	18 per cent
Furnace matte	7 per cent
Flue gases	10 per cent
Radiation and electrical losses	16 per cent

ELECTRIC FURNACE — GENERAL NOTES

Furnace start-up

The procedures adopted by R.P.M. for commissioning the electric furnaces were to cover the hearth with a high-conductivity sacrificial lining of calcined dolomite with 40 per cent



Plate 3—Recently erected 183 m (600 ft) stack for combined furnaces and converter gases, replacing the two 61 m (200 ft) stacks

tar binder and to use electrical power for heating the brickwork. The sacrificial lining was laid and tamped onto the hearth as close as possible to start-up date to prevent moisture absorption.

Electrical contact was established by means of mild-steel cylinders filled with crushed coke underneath each electrode. Contact between electrodes was established by crushed furnace matte spread out on the bottom of the hearth on top of the lining. Dry granulated slag was charged to the furnace level to the top of the coke cylinders and banked up along the side and end walls. With the transformers in star/delta connection and tap one (98.1 V), the power was switched on and the electrodes lowered onto the coke bed till electrical contact was made.

During the first week the electrodes were allowed to bake, volatiles were driven off, and heat was generated to heat up the furnace brickwork, especially the unprotected roof. Sufficient furnace draught during this period was thus essential. The coke underneath the electrodes was replenished as required from time to time. The spring tension on the rods was regularly checked and adjusted to allow even expansion of the brickwork.

As soon as the electrodes were baked, the power was increased and

a molten bath of slag started to form. The furnace expanded rapidly during this period. As soon as charging of the furnaces with pellets and fluxes commenced, the resistance of the slag increased and higher voltage was required. This necessitated the switch from star/delta transformer connection to delta/delta connection to enable personnel to operate the furnaces at higher voltage.

Although the start-up was fairly successful, it is generally felt that the sacrificial lining was not absolutely necessary, especially since it lifted in places and restricted the matte flow. Because matte has a substantially lower melting point than slag, it should not have been used as a conductive medium. It would perhaps have been better to establish initial electrical contact between the electrodes by means of steel plates. This would have allowed the molten slag to fill any cracks created by uneven expansion, thus sealing the brickwork before matte was formed.

Furnace repairs

The refractory rings around the electrodes were damaged by electrodes moving out of alignment because of crust formations in the furnace, poor charging operations, and defective valves on one of the two main hoists per electrode. Casting a ring of castable refractory

material instead of rebricking these rings prevented any further damage to the roof.

Apart from minor repairs to refractories around the matte tap-hole and complete rebricking of the matte end wall and partial rebricking of the slag end wall after three years of successful operation, no other serious drawbacks have been suffered in the operation of the electric furnace.

CONCLUSION

Not only has the electric furnace successfully been adapted to the production of a copper-nickel matte containing platinum-group metals, but there has been a significant reduction in operating costs. This, coupled with other advantages, has established electric smelting of platinum-bearing concentrates as an obvious choice for any future expansion programme.

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REPORT ● VERSLAG

No. 1435

THE DETERMINATION OF CHROMIUM, MANGANESE, IRON, COBALT, AND NICKEL BY ATOMIC-ABSORPTION SPECTROPHOTOMETRY USING THE NITROUS OXIDE —ACETYLENE FLAME

16th August, 1972

Investigators: D. C. G. Pearton, R. C. Mallett

SYNOPSIS

The method developed (a detailed laboratory procedure is given as an appendix) is suitable for the determination of the transition elements in ores and silicate materials.

The sample is dissolved in hydrofluoric, nitric, and perchloric acids, and the residue is fused with sodium peroxide. Potassium nitrate is added as a de-ionizing agent, and the atomic absorption of the elements in solution is measured by use of a lean nitrous oxide—acetylene flame. There is no interference from the elements that are likely to occur in ores and silicate materials, and the coefficient of variation of the method is less than 2 per cent.