

An overview of Southern African PGM smelting

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ABSTRACT

The largest known platinum group metal (PGM) deposit in the world is the Bushveld Complex in South Africa, with the Great Dyke in Zimbabwe also being one of the biggest. It is therefore not surprising that the majority (just over 5 million troy ounces or 156 metric tons per annum) of the world's platinum is produced in Southern Africa. Primary smelting of ore concentrates is carried out in that region by five companies, namely Anglo Platinum, Impala Platinum, Lonmin Platinum, Northam Platinum, all of South Africa, and Makwiro Platinum in Zimbabwe. The only other primary smelter of platinum group metals is Stillwater Mining of Montana, USA, although very significant quantities of PGMs are produced as co-products by Norilsk Nickel of Russia. Smaller (but still significant) quantities of PGMs are produced by Falconbridge and Inco of Canada, also as co-products from nickel sulphide smelting. There are many similarities between PGM smelting and nickel sulphide smelting, and the range of technologies in use includes six-in-line rectangular electric furnaces, three-electrode circular AC furnaces, Peirce-Smith converters, and Anglo's ACP (based on Ausmelt technology). PGMs are also recovered from residue materials using DC arc furnace technology.

INTRODUCTION

The platinum group metals (PGMs) are a family of six greyish to silver-white metals with close chemical and physical affinities. The three heavier metals platinum (Pt), iridium (Ir), and osmium (Os), have densities of about 22 g/cm³; and the three somewhat lighter metals palladium (Pd), rhodium (Rh), and ruthenium (Ru), have densities of about 12 g/cm³. The PGMs belong to the transition metals of Group VIII in the Periodic Table, as do iron (Fe), nickel (Ni), and cobalt (Co). These metals have similar geochemical behaviour and tend to be concentrated together geologically. The PGMs, along with gold (Au) and silver (Ag), are classified as noble metals because of their high resistance to oxidation and corrosion.

Their great scarcity classifies them as precious metals. Only about one thirteenth as much platinum is produced as gold, itself a very rare metal. (By contrast, about 5 million times as much iron as platinum is produced in the world.) As precious and noble metals, PGMs are chemically more versatile than gold, and have found numerous industrial applications. They are also the only competitors for gold as investment metals and for jewellery purposes.

Platinum was originally called “platina” or “little silver” in Spanish, as it was considered a poor-quality by-product of silver mining operations 400 years ago in Colombia. Platinum was formally discovered only in 1751, although it (possibly mistaken for silver at that time) was used as far back as the 7th century BC in Egypt, when the ‘Thebes casket’ was produced. (This was made for Shepenupet, daughter of the King of Thebes, and has gold hieroglyphics on one side and platinum on the other.) The catalytic properties of PGMs were described in the period 1823 – 1838. Interestingly, jewellery and catalysis remain the most important applications of these metals today.

PGMs have extraordinary physical and chemical properties that have made them indispensable to the modern industrial world. The PGMs have very high melting points, and are chemically inert to a wide variety of substances (even at very high temperatures), and thus resist corrosion. They also have excellent catalytic properties, and are widely used in the chemical industry and in automobile catalytic converters. Commercial substitution by cheaper metals has rarely been successful, although an individual platinum group metal may readily be replaced by another.

The most economically important of the PGMs are platinum, palladium, and rhodium, with ruthenium, iridium, and osmium being less prevalent and less in demand. Gold, though it is a precious metal, is not one of the PGMs although it is often lumped together with the PGM content when talking about the valuable products from PGM smelting. The base metals nickel, copper, and cobalt commonly occur together with the PGMs and are produced as co-products in the smelters and refineries. The PGM market is fundamentally strong, particularly in platinum, where recent years have shown good

growth in jewellery demand, and in autocatalysts, especially for the diesel vehicles that now make up 50 per cent of new car sales in Europe.

Platinum group elements are generally associated with nickel-copper sulphides in magmatic rocks. Depending on the relative concentrations (and market prices) of the precious and base metals, the PGMs are produced either as the primary products, or as by-products of the nickel and copper. The primary PGM-rich deposits include the Bushveld Complex in South Africa (the largest known layered igneous complex of its type in the world, extending some 350 km from west to east, and some 250 km from north to south, containing more than two thirds of the world's reserves of PGMs), the Great Dyke in Zimbabwe (the second largest known deposit of platinum in the world), the Stillwater deposit of the USA, and the Lac des Isles deposit of Canada. PGMs are produced in significant quantities as by-products from the Norilsk-Talnakh area of Russia and the Sudbury deposit of Canada. Other deposits occur in Finland, the Jinchuan deposits of northwest China, the Duluth complex of the USA, and in numerous smaller deposits. PGMs are also produced in small quantities as by-products from the nickel-copper industry in Australia and Japan (1).

Table I shows the geographical distribution of the most important PGMs, and Table II shows the demand by market sector for Pt, Pd, and Rh, courtesy of Johnson Matthey (2).

Table I - PGM production in 2004, in millions of troy ounces per annum
(1 troy ounce = 31.1035 g; 1 million troy ounces = 31.1035 metric tons)

	Pt supply		Pd supply		Rh supply	
	Moz	tons	Moz	tons	Moz	tons
South Africa	5.030	156.5	2.500	77.8	0.587	18.3
Russia	0.850	26.4	3.800	118.2	0.105	3.3
North America	0.385	12.0	1.055	32.8	0.018	0.6
Other	0.235	7.3	0.265	8.2	0.016	0.5
Total	6.500	202	7.620	237	0.726	22.6
Plus Recycled material	0.700	21.8	0.530	16.5	0.141	4.4

Table II - Demand by market sector for 2004

	Pt demand		Pd demand		Rh demand
Automobile	43%	Automobile	50%	Automobile	85%
Jewellery	33%	Electronics	14%	Glass	6%
Industrial	23%	Jewellery	14%	Chemical	6%
Investment	0.6%	Dental	13%	Other	3%
		Other	9%		

* The figures for Automobile use reflect the demand for new material only, i.e. recycled materials are excluded here.

PGM PRODUCTION IN SOUTHERN AFRICA

South Africa has more than three quarters of the world's platinum reserves, and is the world's largest producer of platinum group metals (PGMs). These vast resources occur together with the world's largest reserves of chromium and vanadium ore in the unique Bushveld Complex geological formation. South Africa's PGM output is derived almost exclusively from the Bushveld Complex, with only about 0.1 per cent coming from the gold deposits of the Witwatersrand and Free State, and the Phalaborwa copper deposit.

Platinum group metals now constitute South Africa's largest mining export earner, accounting for a third of all sales last year. Revenue from PGMs surpassed those from gold for the first time in 2001, although gold dominated briefly again in 2002.

Since the identification of economic deposits of platinum in South Africa in 1924 by Hans Merensky, a number of platinum mines have come and gone, and some have merely changed identity (3-5). South Africa currently has four integrated primary platinum producers, namely Anglo American Platinum Corporation Ltd (6) (formerly Rustenburg Platinum Holdings Ltd), Impala Platinum (7), Lonmin Platinum (8) (which includes Western Platinum), and Northam Platinum (9). Their range of operations includes open-cast and underground mining, milling, flotation, drying, smelting, converting, refining, and marketing. Anglo Platinum, Impala Platinum, and Lonmin Platinum are the three largest producers of platinum in the world. Since 1971, these operations have established South Africa as the world's largest producer of PGMs. The precious metals are the most valuable products in South African platinum ores, unlike the situation in many other countries where smaller quantities of platinum are produced as by-products or co-products of base-metal production, particularly of nickel. Apart from South Africa's platinum mines, only Makwiro Platinum (10) (formerly Hartley Platinum) in Zimbabwe, and Stillwater Mining (11) in Montana, USA are major primary producers of PGMs. For comparative purposes, mention should also be made of Norilsk Nickel (12), which, although strictly not a primary PGM smelter, is the world's largest producer of nickel and palladium, and is also one of the leading producers of copper, platinum and gold. Other significant producers of PGMs include Falconbridge and Inco of Canada.

Table III shows the production by individual companies of the economically most important PGMs, namely platinum, palladium, and rhodium. Most company production figures are taken from their most recent annual reports (for 2003 in most cases) that are mostly available from their company websites. Note that Russian law was amended only this year to allow the release of official production figures for PGMs, and it is still too early for this information to be available. However, estimated production figures produced by Johnson Matthey (13) for the Polar Division (which produces the bulk of Norilsk's PGMs) are believed to be fairly reliable.

Table III - Company production figures for 2003, millions of troy ounces per annum

	Pt	Pd	Rh	Pt + Pd + Rh
Primary producers of PGMs				
Anglo Platinum, South Africa	2.31	1.19	0.23	3.73
Impala Platinum, South Africa	1.96	1.05	0.25	3.26
Lonmin Platinum, South Africa	0.93	0.42	0.14	1.49
Stillwater Mining, USA	0.13	0.45	0.005	0.58
Northam Platinum, South Africa	0.21	0.10	0.02	0.33
Zimplats, Zimbabwe	0.09	0.07	0.01	0.17
Producers of PGMs as by-products				
Norilsk Nickel, Russia	0.65	2.70	0.06	3.41
Falconbridge, Canada	0.11	0.33	0.002	0.44
Inco, Canada	0.09	0.11	0.017	0.21

SOUTH AFRICAN PLATINUM-GROUP METAL ORES

South Africa has the greatest concentration of primary PGM producing companies (14,15), each of which has their own approach to smelting, governed to a large extent by the type of ore that they process.

The currently exploitable South African reserves of platinum-group metals are concentrated in narrow but extensive strata known as the Merensky Reef, the Platreef, and the UG2 chromitite layer. These three layers in the Bushveld Complex each have their own distinctive associated mineralogy, and have been well described mineralogically (16-18). The Merensky Reef has the PGMs occurring in conjunction with base metal sulphides; the Platreef has an even greater quantity of base metal sulphides present; and the UG2 chromitite layer has a high chromite content together with relatively low quantities of base metal sulphides. Platreef is currently smelted only by Anglo Platinum, but Merensky and UG2 ores are processed by all the producers. These ores are quite different from each other, and require different approaches to metallurgical processing. For example, UG2 ore has a much lower content of nickel and copper sulphides, and contains much more chromite than Merensky ore. The Platreef can be considered as metallurgically similar to Merensky ore, although somewhat enriched in palladium.

The extent to which UG2 ore is processed has a major influence on the smelting behaviour, as traditional six-in-line furnaces are susceptible to build-ups of high-melting chromite spinels if the Cr₂O₃ content of the feed is too high.

With the change in political dispensation in South Africa, and the change in mineral rights legislation (bringing South Africa closer in line to common international practice), there has been a great deal of new mine development, especially on the Eastern Limb of the Bushveld Complex, with much of the new capacity centred around UG2

ores. Most of these new mines have concentrate offtake agreements with existing smelters and refiners.

Ore from the Merensky Reef contains up to 3% base-metal sulphide minerals, distributed as follows: pyrrhotite (45%), pentlandite (32%), chalcopyrite (16%), and pyrite (2 to 4%). The majority of the PGMs in the Merensky ore are associated with pentlandite, occurring either in pentlandite grains or at the pentlandite-gangue grain boundaries. To a lesser extent, the PGMs are associated with other base-metal sulphides or occur in the form of minerals such as braggite, cooperite, laurite, or ferroplatinum. The major gangue minerals are pyroxene, plagioclase feldspar, and biotite.

The principal constituents of UG2 ore are chromitite (60-90%), orthopyroxene, and plagioclase, together with minor amounts of talc, chlorite, and phlogopite, as well as smaller amounts of base-metal and other sulphides and platinum-group minerals. The base-metal sulphides are predominantly pentlandite, chalcopyrite, pyrrhotite, pyrite, and to a lesser extent millerite. The sulphide grains of UG2 ore are generally finer than those of the Merensky Reef.

Merensky ore contains much more sulphide than does the UG2 ore, and the minerals are found in a silicate substrate, while UG2 ore has a chromite matrix. The Cr_2O_3 content of the UG2 ore presents major challenges in processing. In Merensky ores, the ratio of nickel to copper is fairly constant at about 1.7, but the PGM to base metals ratio is not constant (19).

The Merensky and UG2 reefs are situated in close proximity to each other. The UG2 reef lies anywhere between 20 and 330 metres below the Merensky horizon, and varies in thickness between 0.15 to 2.5 metres. Reserves of PGMs plus gold are estimated (20) at 547 million ounces in the Merensky Reef, and more than 1000 million ounces in the UG2 reef. Another estimate (21) says the UG2 reef contains about 800 million ounces of PGMs.

The PGM content of the UG2 reef is comparable with, and sometimes higher than, that of the Merensky Reef. The PGM content in the Merensky Reef ranges between about 4 and 10 g/t, whereas the UG2 reef contains between 4.4 and 10.6 g/t. UG2 ore is by far the richest source of rhodium, which is currently the highest-priced PGM and an important constituent of the catalysts used in motor car exhaust systems. The copper and nickel contents of UG2 ore are generally less than a tenth of those found in the Merensky Reef. The Cr_2O_3 content of UG2 ore is about 30%, as opposed to about 0.1% for Merensky ore. The low-grade chromite produced as a by-product during the treatment of UG2 ore is also sold, and there is no reason why it could not be used for the production of ferrochromium (22,23). The high demand for palladium also makes the processing of UG2 concentrates very attractive.

Average grades and current values of the individual precious metals in Merensky, UG2, and Platreef ores are shown in Table IV. Further detail regarding the distributions

of the individual PGMs in various reefs and sectors of the Bushveld Complex is available elsewhere (17). The content and value of base metals in the three ores are shown in Table IV.

Table IV - Typical content of precious and base metals in Merensky, UG2, and Platreef ores (16), and their current potential value. Market prices (24) are as of the week preceding 16 May 2005.

	Price	Merensky ore			UG2 ore			Platreef ore		
	\$/oz	g/t	\$/t ore	mass %	g/t	\$/t ore	mass %	g/t	\$/t ore	mass %
Pt	875	3.25	91.4	59	2.46	69.2	41	1.26	35.4	42
Pd	193	1.38	8.6	25	2.04	12.7	34	1.38	8.6	46
Rh	1550	0.17	8.5	3	0.54	26.9	9	0.09	4.5	3
Ru	67	0.44	0.9	8	0.72	1.6	12	0.12	0.3	4
Ir	155	0.06	0.3	1	0.11	0.5	1.9	0.02	0.1	0.8
Os	400	0.04	0.5	0.8	0.10	1.3	1.7	0.02	0.3	0.6
Au	425	0.18	2.5	3.2	0.02	0.3	0.4	0.10	1.4	3.4
Total PGM+Au		5.5	112.7	100	6.0	112.4	100	3.0	50.5	100
	\$/t	% in ore	\$/t ore	mass %	% in ore	\$/t ore	mass %	% in ore	\$/t ore	mass %
Ni	17403	0.13	22.6	62	0.07	12.2	80	0.36	62.7	67
Cu	3300	0.08	2.6	38	0.018	0.6	20	0.18	5.9	33
Total Ni+Cu		0.21	25.3	100	0.09	12.8	100	0.54	68.6	100

It is evident from the data above that Pt, Pd, and Rh make up a remarkably constant 96% of the value of all the precious metals, for all three ore types. In the case of Merensky ore, these three dominant PGMs make up about 80% of the value of all the metals produced (*i.e.* PGMs plus base metals). For UG2 ore, the fraction is closer to 90%. The situation is quite different for Platreef ore, where the contained base metals may be worth even more than the precious metals (if base metals prices are high, as they are at present).

The average grain size of the PGM minerals is about 45 μm in Merensky ore, and 15 μm in UG2. In order to liberate the PGM minerals, UG2 concentrate is more finely milled (about 80% less than 75 μm) than Merensky concentrate (about 55% less than 75 μm) (25). During concentration, the recovery of PGM+Au is around 80 to 87 per cent (25). Typical analyses of the Merensky and UG2 concentrates at Lonmin's Western Platinum Mines have previously been published elsewhere (25-27).

From a given quantity of ore, the mass of UG2 concentrate is generally lower (around 1.3% of the feed ore) than that of Merensky concentrate (around 2.5% of the feed ore). Hence, the grades of UG2 concentrates are commonly higher, and the amount of concentrate to be smelted is smaller.

BENEFICIATION PROCESSES

Each processing step is designed to increase the grade (concentration) of the valuable components of the original ore, by reducing the bulk of the products. The mined ore undergoes comminution, and a gravity concentrate is extracted. The sulphides are concentrated by flotation. The flotation concentrates undergo smelting and converting, to produce a PGM-containing nickel-copper matte. The matte is treated hydrometallurgically to separate the base metals from the precious metals. Finally, the PGM concentrate is refined to separate the individual precious metals into their pure forms. As a rough guide (25), the PGM contents during the various stages are as follows.

Ore	0.0005% (5 g/t)
Flotation concentrate	0.0150% (100 – 400 g/t)
Converter matte	0.20%
PGM concentrate	30 – 65%
Refined metals	99.90% for Rh, Ru, Os 99.95% for Pt, Pd, Au

During each separation stage of the process, there is an increase in the concentration of PGMs – about 30:1 in the concentrator, about 10:1 in the furnace, about 3:1 in the converter, and about 200:1 in the base-metals refinery.

For South African producers, the approximate distribution of the operating costs for each stage is as follows (28):

Mining: 72% Concentrating: 10% Smelting: 9% Refining: 9%

PGM recovery is typically about 85% in the concentration stage, 95 to 98% in smelting, and 99% in refining. By far the greatest loss of PGMs occurs during crushing, milling, and flotation, and research into these operations could prove very rewarding, as could the development of new processes that remove some of the constraints on the various concentration stages.

DESCRIPTION OF THE CONVENTIONAL MATTE-SMELTING PROCESS

Please note that analyses of concentrate, furnace slag, furnace matte, converter slag, and converter matte have been presented elsewhere (14).

Concentrate Drying

The concentrate is dried in a spray drier or flash drier. This reduces the energy requirement for smelting, as well as decreasing the occurrence of ‘blowbacks’ or explosions in the furnace. The dry concentrate is transferred pneumatically from the drier into the furnace.

Smelting

Smelting is intended to separate the gangue (oxide and silicate) minerals from the sulphide minerals associated with the noble metals. The sulphide minerals form a matte that is treated further; the gangue is discarded as slag. As the concentrate melts, two liquid phases form: a lighter silicate- and iron-rich slag with a relative density around 2.7 to 3.3, and a denser molten matte (rich in nickel and copper sulphides, and other base and precious metals) with a relative density of about 4.8 to 5.3. Prills of molten matte grow in size by coalescing with other prills, then settle out from the slag under the influence of gravity, at a rate which depends on the viscosity of the slag. A flux, often limestone, may be added to reduce the viscosity and liquidus temperature of the slag.

PGM smelting in Southern Africa takes place exclusively in electric furnaces at present. Rectangular six-in-line submerged-arc electric furnaces are the most widely used, although there are also some circular three-electrode furnaces in operation. Smelting typically takes place at temperatures around 1350°C, although smelting of UG2 concentrates can require temperatures in the region of 1600°C or higher.

Because of the low concentration of valuable minerals in the concentrate, the furnace is operated at a high slag:matte ratio (between about 4 and 9). These two phases are tapped separately from the furnace (from opposite ends, in the case of a rectangular furnace). In the case of Merensky ore being smelted in a six-in-line furnace, the slag is tapped at temperatures around 1350°C, and the matte is somewhat cooler, around 1200°C. The unwanted slag constituents are discarded (usually after being subjected to granulation using a high-flow water stream, milling, and flotation to re-capture any entrained droplets of matte). The furnace matte contains nickel, copper, cobalt, iron, sulphur, and the PGMs. The furnace matte is tapped into ladles and transferred by crane to a converter vessel.

The furnaces are normally operated with a ‘black top’, *i.e.* with a layer of unsmelted concentrate on top of the molten bath. This limits the amount of radiation from the surface of the bath to the walls and roof of the furnace. In one documented case (19), a 15 cm layer of concentrate covers a 100 cm layer of slag, which in turn covers a 58 cm layer of matte.

The electrical power consumption in the furnace is approximately 600 to 1100 kWh per ton of concentrate, but depends on the nature and grade of the material

being treated, as well as the operating conditions in the furnace. Electrical power typically accounts for approximately 40 per cent of the direct smelting costs (19).

During smelting, some magnetite (and other spinels such as chromite) that is not reduced and fluxed, dissolves in the matte and slag. Magnetite sometimes forms an intermediate viscous zone between the matte and slag layers, causing an increase in entrainment. A build-up of magnetite or other spinels causes a reduction in operational furnace volume. Near the slag-matte interface, the concentration of matte particles in slag is at its highest, as is the concentration of chromium oxide in the slag.

Converting

During the converting process, air is blown into the molten matte, over a period of a few hours, in order to remove much of the iron and sulphur by oxidation (primarily of FeS). The converters most commonly employed are of the Peirce-Smith type; these are of horizontal cylindrical shape with an opening at the top for charging and discharging; tuyeres for blowing are arranged in horizontal rows along the lower back of the vessel; a tilting mechanism allows pouring to take place. Silica sand is added to the converter to flux the iron oxide that is formed by the oxidation of the iron, and to form an iron silicate slag having the approximate composition of fayalite ($2\text{FeO}\cdot\text{SiO}_2$), with some dissolved magnetite. Some of the sulphur leaves the system in the gas phase as sulphur dioxide (SO_2). The oxidation reaction is sufficiently exothermic to maintain a temperature around 1250°C in the converter. The temperature is controlled by adding cold feed or revert materials (spillages, *etc.*) to the converter if it becomes too hot. The converter slag is periodically skimmed off, but the matte is poured out only once it has attained the desired iron content. The required degree of iron and sulphur removal during converting is dictated by the choice of the subsequent refining process. The converter matte is either cast into cast-iron moulds or refractory-lined pits, and crushed, or it can be granulated by pouring it into a very fast-flowing stream of water.

The converter slag requires further treatment, as the vigorously turbulent conditions cause the entrainment of prills of valuable converter matte, and the oxidizing conditions cause some of the valuable base metals (especially cobalt and nickel) to dissolve in the slag in oxide form. Until a few years ago, it was common for the molten converter slag to be returned intermittently to the primary smelting furnace (by ladle to a cast-steel launder projecting slightly into the furnace through the matte-tapping end wall). In other cases the slag is granulated, and subjected to milling and flotation; it is also possible to introduce the slag into a slag-cleaning furnace. Breaking the recycle loop, by not returning the converter slag to the furnace, is rather attractive, as the quantity of PGMs locked up in this loop can represent a large financial investment. It is not uncommon for up to a third of the matte produced in the converters to be returned to the furnace.

A further advantage of breaking the recycle loop is that this reduces the load of chromium to the primary smelting furnace. Some magnetite and chromite spinels form

in the oxidizing conditions of the converting process. If the converter slag is returned to the furnace, these can settle out and precipitate on the furnace hearth, thus considerably reducing the volume of the furnace over time.

The converter matte (also known as white metal) has a relative density of about 6, and consists primarily of Ni_3S_2 , Cu_2S , and FeS , along with small amounts of cobalt and precious metals. The matte also contains small amounts of impurities such as selenium, tellurium, arsenic, lead, tin, antimony, and bismuth.

Off-gas handling

It used to be common practice for furnace exhaust gases to pass through an electrostatic precipitator and then to be discharged to the atmosphere through a tall stack. The SO_2 in the gas can be used in the production of sulphuric acid, but the low concentration produced from the furnaces, and the intermittent production from the converters makes this challenging.

Of the sulphur entering the smelter, 60 per cent leaves in the converter gases, 20 per cent in the furnace gases, 15 per cent in the converter matte, and 5 per cent in the furnace slag (29). The furnace gases have an SO_2 content of around 0.4 per cent, which is generally considered too low for efficient recovery. The converter gases, for 70 per cent of the blowing time, have an SO_2 content of more than 4 per cent; the overall variation is typically from 2.5 to 6 per cent. The difficulty in dealing with a gas of such a variable composition is one of the driving forces behind the development of continuous converting processes.

Refining

The converter matte is usually milled prior to treatment in the base-metal refinery, where the copper and nickel are extracted by a sulphuric-acid leaching route. In most plants, the leach residue makes up the high-grade PGM concentrate that is provided to the precious metals refinery for final separation of the pure precious metals.

LIMITATIONS OF THE CONVENTIONAL MATTE-SMELTING PROCESS

Environmental concerns have focused on the problem of SO_2 emissions, especially the stray emissions around the mouth of the converter. Even with a large fume hood above the mouth of the converter, fugitive emissions remain a problem. A sulphuric acid plant is probably the most effective means of capturing the sulphur. However, the intermittent nature of converting operations makes this rather challenging.

As increasing amounts of UG2 concentrate are processed (to utilize deposits accessible from existing mines, and to maximize production of rhodium, as well as

platinum), so the quantity of base metal sulphides decreases. The conventional process requires sufficient matte (at least 10% of the mass of the slag) to be present to allow for effective coalescence of droplets and collection of the valuable metals. This causes limits to be placed on the mining of ore such that only material containing more than a specified amount of nickel and copper is acceptable to the process. This limitation can be lifted only if additional collector material is available.

The UG2 concentrates contain significant quantities of chromite, which easily results in the build-up of (highly refractory) chromite spinel layers in the furnace. This affects furnace operation, and the accumulation reduces the working volume of the furnace over a period of time. This can be mitigated to some extent by the addition of some carbon to the furnace, as more reducing conditions allow for greater solubility of chromium oxide in the slag.

The intermittent batch mode of converting is not conducive to good plant operation, and there is a significant move towards the development of continuous converting processes.

Although most current smelters and refineries have PGM recoveries in the region of 95 to 99% each, the recovery from concentrators is only around 85%, and that from mining itself is also relatively low. Clearly, any new processes being developed should be sufficiently flexible to allow greater recoveries in these areas, preferably by removing some of the constraints imposed by present practices.

The long processing times in the refining of PGMs result in a large lockup of precious metals. Sometimes, the value of the PGMs permanently locked up inside process units exceeds the capital cost of the units themselves. The composition of the metal produced in the smelter can make a difference in reducing the length of the processing pipeline in the refinery. This should be taken into account when investigating new processes.

HISTORICAL SMELTING DEVELOPMENTS

The early days

Platinum mining on a large scale began around 1926, and, before the 1920s were over, no less than seven mining operations had started in South Africa. The platinum ores were mostly processed by traditional milling and gravity-table concentration. Flotation was used for the first time to produce a sulphide platinum concentrate in 1926, at Potgietersrus (21).

The weakening of the platinum price in the early 1930s led to widespread closures and amalgamations, resulting in the formation of a single dominant company, Rustenburg Platinum Mines, in 1931. By 1936, throughput had expanded to 18 000 tons

of ore per month, and the oxidized ore was nearly exhausted. It became necessary to commission a flotation plant and to install a small blast furnace and converter unit for the production of platinum-rich copper-nickel matte, which would cost less than bulk concentrate to transport to the UK refinery. A second blast furnace was commissioned in 1953. Blast-furnace smelting was labour intensive, and utilised expensive coke. Furthermore, a very large volume of gas was emitted containing between 1 and 2 per cent sulphur dioxide, posing a serious pollution problem.

Interestingly, reverberatory smelting (where the energy is supplied by the flame generated by the combustion of coal, oil, or natural gas, as well as indirect radiation) was never applied to platinum production in South Africa, despite this technology being used quite extensively for copper production. The probable reason for this is the difficulty in achieving the somewhat higher temperatures required for platinum smelting. The slags produced in platinum smelting have liquidus temperatures one or two hundred degrees Celsius higher than those produced in copper smelting.

Electric smelting

Electric smelting was used for the first time in the primary production of platinum, with the commissioning of a 19.5 MVA Elkem rectangular electric furnace (with six in-line submerged electrodes operating in pairs in a three-phase electrical system) at the Rustenburg Section of Rustenburg Platinum Mines in 1969. The furnace was 27.2 m long, 8.0 m wide, and 6.0 m high. The sidewalls of the furnace were externally water cooled. The furnace was lined with magnesite, and utilised firebrick for the roof. Concentrates were pelletized and dried prior to being fed to the furnace.

Electric smelting remains the most prevalent today, with all PGM smelting done this way, with the exception of that carried out at Norilsk Nickel. The furnaces in use today are described further below.

Drying

There has been a significant move away from pelletization, and towards the pneumatic feeding of dried concentrates. The lowering of the amount of moisture introduced into the furnace has lowered the energy requirement of smelting, and has drastically reduced the occurrence of 'blowbacks' or furnace eruptions. This has reduced the quantity of dust emitted from the furnaces, and has significantly improved the safety and cleanliness of the smelting operation.

Sidewall cooling

There has been an increasing move in recent years to higher-intensity furnaces fitted with copper cooling, both for rectangular furnaces (at Anglo Platinum's Polokwane smelter) and circular furnaces (such as those at Lonmin Platinum and Makwiro

Platinum). Containment issues are important, especially at the higher operating temperatures used to cope with increasing chromite contents in the furnace feed.

Hatch copper-finger coolers have been installed in the sidewalls of some furnaces, and this has improved the integrity of the furnace linings. More recently, waffle coolers have been used on the newer higher-intensity furnaces. Although copper cooling has allowed the use of higher power fluxes in furnaces, it has brought with it new challenges of its own in terms of localised copper wear resulting from gaseous attack and possible electrochemical effects.

UG2

Although the UG2 chromitite horizon was identified as containing PGMs in the 1920s, it took many years for this reef to be exploited. Traditionally, the grades were regarded as lower than those of the Merensky Reef, but more recent developments have shown that in many areas the PGM values are higher than in the Merensky Reef.

A blend of Merensky and UG2 concentrates has been processed since the late 1970s. During the 1980s, Mintek developed a process for the treatment of UG2 concentrates without the requirement for blending. Testwork (30) showed that a UG2 concentrate could be produced having a PGM grade around 430 g/t, at a recovery of 87 per cent. This was achieved with a mass pull (*i.e.* concentrate to ore ratio) of about 1 per cent, and a Cr₂O₃ content of 2.9 per cent. Even higher grades (more than 1000 g/t) could be achieved at even higher recoveries (more than 90 per cent), if the constraint on the Cr₂O₃ content was relaxed (to between 4 and 10 per cent).

The higher concentrations of MgO, SiO₂, and Al₂O₃ in the UG2 concentrate require a higher smelting temperature. The Cr₂O₃ content of UG2 concentrate is typically seven to ten times that of Merensky concentrate, and if allowed to deposit in the furnace hearth, would rapidly build up and reduce the volume of the furnace. Depending on the individual process, UG2 smelting may have a higher energy requirement per ton of concentrate. For example, pilot tests (26,27) demonstrated the smelting of Merensky concentrate at 1350°C and 896 kWh/t, and UG2 concentrate at 1470°C and 1088 kWh/t. However, because UG2 concentrates have a higher concentration of PGMs, as a result of the small quantities of sulphide minerals in the ore, they actually require significantly less energy per mass of PGMs produced. (In addition, the chromite content of UG2 ore is potentially saleable, after recovery of the PGMs, and the removal of gangue.)

Pilot-scale tests (26,27) have shown that adequate coalescence of matte prills can be obtained by the use of a higher smelting temperature, and higher power flux (kW/m² of furnace hearth area) to increase mixing in the bath. The pilot tests led to the adoption of circular electric furnaces with three graphite electrodes for UG2 smelting, as this configuration can better withstand the higher temperature and power flux required. The slag from this operation has a PGM content too high (2.5 to 3.5 g/t) to be discarded, so it is granulated and returned to the flotation circuit for the recovery of the PGMs. Lime or

limestone is sometimes used as a flux, to improve the compatibility of the slag with the basic refractory lining.

Recycling or cleaning of converter slag

The chromite problem is seen as increasingly important as the amount of UG₂ ore being mined continues to grow faster than the amount of Merensky ore. A recent trend is the breaking of the traditional recycle of converter slag to the primary smelting furnace. The converter slag can be treated either in a slag-cleaning furnace (as practised by Anglo Platinum) or by subsequent milling and flotation.

Continuous converting

Continuous converting has been investigated by a number of platinum producers. This is seen as a way to improve environmental issues, and to de-bottleneck those plants where the converters are the limitation to increased production. The steady stream of SO₂ generated during continuous converting is suitable for sulphuric acid production.

SO₂ emissions

Various techniques have been used in the attempt to reduce SO₂ emissions from smelters, with different degrees of success. These are discussed further in the sections detailing the individual smelters.

NEW PROCESSES UNDER DEVELOPMENT

ConRoast

Matte smelting has been in use for the production of PGMs for many decades, but new PGM smelting processes continue to emerge. An alternative PGM smelting process has recently been developed using alloy smelting in a DC arc furnace (31,32). This process has been implemented on a large-scale continuous demonstration basis (32). The ConRoast process eliminates many of the traditional constraints on feed composition and also has environmental advantages in the control of SO₂ emissions.

The ConRoast process, jointly developed between Mintek and Falconbridge, is a significant departure from sulphide matte smelting, in that it collects PGMs in an iron-rich alloy in a DC arc furnace. Sulphide ore concentrates are dead-roasted in a fluidized bed prior to the smelting step. This enables a continuous high-strength SO₂-containing gas to be captured from a single enclosed vessel, and avoids the presence of sulphur from that point onward in the process.

The DC arc furnace attains the correct balance of high temperature and reducing conditions to achieve very high recoveries of PGMs without suffering a spinel build-up

in the furnace. The similarity of the alloy and slag liquidus temperatures greatly simplifies the problem of containment of the high-temperature molten phases in the furnace.

A long-term large-scale demonstration of the DC arc furnace for PGM smelting has been underway at Mintek since 2004, and the recovery of PGMs has been demonstrated very successfully on over six thousand tons of low-sulphur high-chromium revert tailings to date, at a throughput of approximately one thousand tons per month (32).

SOUTHERN AFRICAN PLATINUM SMELTER DATA

Production data and equipment and processing details for the Southern African platinum-group-metal smelters (and some others for comparison) are given in Tables V to X.

Table V – Smelter location, year of first production, and annual production

Smelter	Location	Year	PGMs, Moz/a	Ni, kt/a	Cu, kt/a
Anglo – Waterval	Rustenburg, South Africa	1926	4.162	22.1	12.9
Anglo – Mortimer	Swartklip, South Africa	1973	(0.754)	(2.6)	(1.5)
Anglo – Polokwane	Polokwane, South Africa	2003			
Impala Platinum	Rustenburg, South Africa	1969	3.725	16.4	8.7
Lonmin Platinum	Marikana, South Africa	1971	1.758	3.18	2.12
Northam Platinum	Northam, South Africa	1992	0.385	1.51	0.86
Makwiro Platinum	Selous, Zimbabwe	1997	0.187	1.63	1.0
Stillwater Mining	Columbus, Montana, USA	1990	0.599	0.75	0.5
Norilsk Nickel – Nickel Smelter	Taimyr Peninsula, Siberia, Russia	Early 1940s	3.410	239	451
Norilsk Nickel – Nadezhda Smelter	Taimyr Peninsula, Siberia, Russia	Early 1980s	-	-	-
Norilsk Nickel – Copper Smelter	Taimyr Peninsula, Siberia, Russia	-	-	-	-

Table VI – PGM Furnace information

Smelter	Year	Furnace type	Power	Power flux, kW/m ²	Dimensions, m	Electrode diameter, mm	Cooling system	Concentrate smelting rate, t/h
Anglo – Waterval	1990s	Six-in-line rectangular (Hatch)	34 MW (30 MVA)	165	25.8 x 8.0	1100	Copper finger / plate coolers	71 (total for both furnaces)
1990s	Six-in-line rectangular (Hatch)	34 MW (30 MVA)	165	25.8 x 8.0	1100	Copper finger / plate coolers	(see line above)	
2003	Three-electrode AC circular slag cleaning	28 MVA	-	-	1200	Copper coolers	-	
Anglo – Mortimer	1973	Six-in-line rectangular	19.5 MVA	110	25.3 x 7.0	1250	Forced air-cooled	20
Anglo – Pookwane	2003	Six-in-line rectangular	68 MW (168 MVA)	250	28.7 x 9.6	1600	Copper waffle coolers	87 capacity
Impala Platinum	2001 (F3)	Six-in-line rectangular	38 MW	180	25.9 x 8.2	1140	Copper plate coolers	92 (total for both furnaces)
1992 (F5)	Six-in-line rectangular	35 MW	180	25.9 x 8.2	1140	Copper plate coolers	(see line above)	
1974 (04)	Six-in-line rectangular dormant	15 MW	-	-	-	-	-	
1972 (02)	Six-in-line rectangular modified	7.5 MW	-	-	-	-	-	
Lommin Platinum	2002	Three-electrode AC circular (Hatch)	28 MW (60 MVA)	320	11 (diameter)	1400	Copper waffle coolers	30
1991	Three-electrode AC circular (Pyromet x 3)	5 MVA (three)	235	5.2 (I.D.)	500	Falling film	-	
1991	Six-in-line rectangular (Barnes-Birlec, Merensky)	12.5 MVA	120	18.2 x 5.3	900	-	-	
Northam Platinum	1992	Six-in-line rectangular (Davy)	15 MW (16.5 MVA)	90	25.9 x 8.7 x 5.6	1000	None	10
1997 (2002)	Three-electrode AC circular (Elkems/Hatch)	12.5 MW (9-10 MW) (13.5 MVA)	90	12 (O.D.)	1200	Copper plate coolers	10	
Makwiro Platinum	1999	Three-in-line rectangular (Hatch)	5.0 MW (5.3 MVA)	150	9 x 5	305	Copper plate coolers	5.0
Stillwater Mining	1990	Three-in-line rectangular (Lecromselt)	1.5 MW (1.68 MVA)	150	7.5 x 2.6	305	Copper finger coolers	1.3
Norilsk – Nickel	Early 1940s	5 x six-in-line furnaces (3 working, 1 standby, 1 repair)	75 MW (normally 45 MW)	-	-	-	-	-
-	-	2 x slag-cleaning electric furnaces (for converter slag) (1 working, 1 standby)	20 MW	-	-	-	-	-
-	-	6 x electric anode furnaces (3 graphite electrodes) reduction by coal	-	-	20 t capacity	1500	-	-
Norilsk – Nadezhda	Early 1980s	2 x Onokampu flash	-	-	245 m ² hearth	-	-	135-180 each
-	-	1 x Vanyukov	-	-	-	-	-	-
-	-	4 x 3-electrode slag-cleaning furnaces	18 MW each (normally 8-11 MW)	-	-	-	-	-
-	-	1 x slag-cleaning rotating vessel	-	-	-	-	-	-
-	-	4 x rotating anode furnaces	-	-	-	-	-	-
Norilsk – Copper	-	2 x Vanyukov	-	-	15.6 x 2.3 hearth	-	-	-
-	-	1 x Vanyukov	-	-	20 m ² hearth	-	-	-
-	-	Reverb	-	-	250 m ² hearth	-	-	-
-	-	Reverb	-	-	92 m ² hearth	-	-	-
-	-	4 x Reverb anode furnaces (green poles for reduction)	-	-	-	-	-	-

Table VII – PGM Furnace operation information

Furnace	PGM Conc. grade, g/t	Energy consumption, kWh/t of concentrate	Slag Tapping Temperature, °C	Matte Tapping Temperature, °C	Matte Fall *, %
Anglo Waterval #1	150	700	1550	-	22
Anglo Waterval #2	150	700	1550	-	22
Anglo Waterval Slag cleaning	-	-	-	-	-
Anglo Mortimer	145	820-850	1650	1550	15
Anglo Polokwane	150-200	800-850	1600	1550	15
Impala #3	130	680	1460	1300	12
Impala #5	130	680	1460	1300	12
Lonmin #1	300-350	850	1600-1650	1500-1550	14
Lonmin – Pyromet (3)	-	880	-	-	-
Lonmin – Barnes-Birlec	-	1270	-	-	-
Northam	130	1044	1485	1385	18
Makwiro	75	750-850	1580	1420	12
Stillwater - 5 MW	600-1200	850-950	1500-1550	1150-1250	14-25
Stillwater - 1.5 MW	-	900-1000	-	-	-

* Matte fall is the mass of matte produced relative to the mass of the original concentrate feed.

Table VIII – PGM Converter information

Smelter	Year of First Production	Type of Converter	No. of Converters	Converter Dimensions, m	No. of Tuyeres per Converter	Converter Slag Recycle to Furnace
Anglo Waterval - ACP	2002	Ausmelt (ACP)	1	-	1	No
Anglo Waterval - PS	-	Peirce-Smith	6	3.0 x 7.6	-	No
Anglo Mortimer	-	None	0	-	-	No
Anglo Polokwane	-	None	0	-	-	No
Impala	-	Peirce-Smith	2	3.6 x 7.3	32	No
Impala	-	Peirce-Smith	4	3.0 x 4.6	16	No
Lonmin	1971,2002/3	Peirce-Smith	3	3.0 x 4.6	18	No
Northam	1992	Peirce-Smith	2	3.0 x 6.1	22	Yes
Makwiro	1998	Peirce-Smith	2	3.0 x 4.6	16	Yes
Stillwater	1999	Kaldo (TBRC)	2	2.0 x 3.32	-	Yes (a)
Norilsk – Nickel	Early 1940s	Russian	6	100 t capacity	-	Yes (b)
Norilsk – Nadezhda	-	Peirce-Smith	6	-	-	-
Norilsk – Copper	-	Peirce-Smith	9	4 x 9	-	Yes (c)

(a) Recycled in granulated form

(b) From copper smelter

(c) To reverb furnace

Table IX – Driers, Acid plants, Gas cleaning, and Stack height

Smelter	Drier Type	Acid Plant	Gas Cleaning Equipment	Stack Height, m
Anglo Waterval	Flash (4)	Yes	Ceramic filters	183
Anglo Mortimer	Flash	No	ESP	100
Anglo Polokwane	Flash (2)	No	Baghouse	165
Impala	Spray (4)	Yes	ESP; Sulfacid plant	77 (91)
Lonmin	Flash; Spray (2 old)	No	Baghouse for drier; ESP and dual-alkali wet scrubber	120
Northam	Flash	No	ESP and wet scrubber	200
Makwiro	Flash (1)	No	ESP	105
Stillwater	Fluid bed	No	Baghouse and SO ₂ wet scrubber	26
Norilsk – Nickel	Sinter plant	-	-	-
Norilsk – Nadezhda	Niro spray (6)	No	ESP (6); Elemental S production	-
Norilsk – Copper	Rotary (3)	Yes	Wet gas cleaning; Elemental Sulphur production	180

Table X – Granulation

Smelter	Furnace		Converter	
	Slag	Matte	Slag	Matte
Anglo Waterval	Yes	Yes	Yes	No *
Anglo Mortimer	Yes	No **	-	-
Anglo Polokwane	Yes	No **	-	-
Impala	Yes	No	Yes	Yes
Lonmin	Yes	No	Yes	Yes
Northam	Yes	No	No	Yes
Makwiro	Yes	No	No	Yes
Stillwater	No	Yes	Yes	Yes

* Waterval converter matte is slow-cooled

** Mortimer and Polokwane furnace matte is cast and crushed

ANGLO PLATINUM

Anglo Platinum has three smelter plants – Waterval, Mortimer, and Polokwane. Anglo Platinum have chosen to have one central facility where all their converting and subsequent treatment by slow cooling and magnetic separation prior to refining can take place. Furnace matte from all three of their smelters is treated in the Anglo Platinum Converting Process (ACP), at the Waterval Smelter near Rustenburg. This strategy of decoupling their furnaces and converting operations provides some operational flexibility as well as offering a range of siting possibilities for future expansions.

The year 2003 saw significant changes in Anglo Platinum's smelting capacity, with the commissioning of the Polokwane Smelter, the Anglo Platinum Converting Process (ACP) plant, and the Waterval Slag-cleaning Furnace.

Waterval Smelter

The largest of Anglo Platinum's smelting plants is the Waterval Smelter at their Rustenburg Section. All converting of furnace matte from Anglo's three PGM smelters takes place here.

The first electric furnace installation for platinum matte smelting was commissioned in 1969, and has been described by Mostert (29) and others (33). The 19.5 MVA six-in-line submerged arc furnace used electrodes 1.25 m in diameter, spaced 3.4 m apart. The maximum electrode current was 32.4 kA at 201 V. Based on the cross-sectional area of the electrodes, this results in 2.65 A/cm^2 . The electrodes were normally submerged about 48 cm into the slag layer, which varied in thickness between 1.3 and 1.5 m. The thickness of the matte layer was around 76 cm. A second furnace was installed in the early 1970s. The mean residence time in the furnace was around 20 hours. A 25 per cent addition of limestone was added to the concentrate as a flux. The resulting slag had a liquidus temperature of 1300°C , an electrical resistivity of $4.7 \Omega\text{cm}$ at 1400°C , and a viscosity of 3.7 poise at 1400°C (33). A temperature gradient of 0.75°C per cm was measured in the slag (33).

During the 1990s, most of the smelter was upgraded. The rotary multi-hearth driers and pelletizing plants which produced semi-dry pellets (about 10 per cent moisture) were replaced by flash driers in 1992, thereby eliminating the labour-intensive process of pellet production, as well as lowering the cost of smelting. Flash drying technology was seen to have lower capital, maintenance, and labour costs, higher efficiency, and more effective dewatering capacity than conventional spray/rotary drying or pressure filtration technologies. The two 18 MW Elkem furnaces were replaced by the current Hatch furnaces. The Peirce-Smith converters were lengthened from their original 6.1 m (20 ft) to 7.6 m length, and their number was increased from four to six.

Waterval Smelter treats concentrates from a number of different concentrators. There are three flash driers; one smaller and two identical larger units. The dried concentrate is blended with lime, and is pneumatically transferred to the furnaces. The two furnaces are of a Hatch design, accommodating six Söderberg electrodes 1.25 m in diameter. Both of the furnaces at the Waterval Smelter are rated at 39 MVA (34 MW). The two furnaces measure 25.8 m x 8.0 m inside, and have a combination of chrome-magnesite and magnesite refractories. Based on these figures, they have a power flux of 165 kW/m^2 . The maximum voltage supplied by the transformer is 350 V, and the maximum current is 27 kA per phase. The electrode consumption is around 3.5 kg of electrode paste per MWh. Limestone is added to the furnace as a flux, to the extent of about 10 per cent of the mass of the concentrate. The furnace off-gases pass through ceramic filters, resulting in significantly reduced dust losses, and are then expelled to the atmosphere via the main stack.

Anglo Platinum uses a matte slow-cooling process (34) for the recovery of PGMs. In this process, the converter matte (consisting predominantly of nickel, copper, and

sulphur, together with minor amounts of iron, and trace quantities of PGMs) is chill-cast into 30-ton ingots in refractory-lined moulds in the ground, covered with a lid for about a day, and cooled for approximately five days. During slow cooling, an iron-nickel phase and a copper-nickel alloy phase separate. Around 95% or more of the platinum group elements concentrate in a relatively small volume of magnetic Ni-Cu-Fe alloy. This alloy can be magnetically separated after crushing and milling, with the intention of shortening the overall processing time for the recovery of the noble metals. (A quicker process reduces the hold-up of precious metals in the extraction and refining circuits.) This enables the PGMs to be processed directly in a precious metals refinery, without the need to first pass through a base metals refinery. In this way, a clear separation is made between the business of processing the base metals and the precious metals business.

Anglo Platinum Converting Process (ACP)

Anglo Platinum announced in 1998 that they were busy with the development of a new continuous converting process, now known as the Anglo Platinum Converting Process (ACP). The prime objective of the ACP project was to reduce emissions of sulphur dioxide at the Waterval smelter to the Department of Environmental Affairs and Tourism's permitted levels of under 20 tons per day by the end of 2004. Prior to the ACP process implementation, the Waterval Smelter was able to capture only 50% to 55% of the sulphur dioxide emissions, and needed to increase the sulphur fixation to 95% in order to come within acceptable world-class environmental standards. (For interest, South African guidelines allow 100 parts per billion of sulphur dioxide.) During May 2003, the Waterval Smelter came under heavy criticism in the media regarding its SO₂ emissions, and it was acknowledged that they had emitted 68 700 tons of SO₂ during the previous year (*i.e.* an average of 188 t/d), with figures as high as 248 t/d at the beginning of 2003. By September 2003, emissions had already been reduced to 130 t/d.

The Anglo Platinum Converting Process (ACP) was also intended to increase the available converter capacity for the group's expansion programme. The plant, developed in conjunction with Ausmelt and others, uses Sirosmelt submerged lance technology from Ausmelt, adapted to the requirements of nickel-copper matte smelting, with copper waffle coolers from Hatch.

The ACP project was started in 1995 with feasibility studies. Final approval of the project was concluded in July 2000, and site construction began in August 2000. At the peak of construction, more than 3 000 workers were employed on-site. Commissioning began in March 2002, and the first slag was tapped on 13 June 2002. The ACP converter plant was brought on stream in 2003, and continues to increase throughput, with the intention that it would be fully operational by the second quarter of 2004. The ACP unit underwent a hearth rebuild in January 2004. The cost of the first stage of the ACP project (Stage A) was about R1.4 billion, and the cost of a second unit (Stage B) is projected to be an additional R0.5 billion. Construction of the second converter was planned to begin in early 2003. The second ACP unit (Stage B) is scheduled for completion by 2006. The phasing out of the existing Peirce-Smith

converters and the old single-contact-absorption sulphuric acid plant is planned once the second standby ACP reactor is commissioned. (This acid plant treated the 4-6% SO₂ off-gas from the Peirce-Smith converters.) This is anticipated to occur during the fourth quarter of 2005, with the first slag tap scheduled for the first quarter of 2006.

While Stage A included the complete infrastructure to support the new Ausmelt converting process, which included raw-material handling, slag and matte granulation, hot-gas cleaning, as well as the acid plant for sulphur fixation, Stage B will include only the converter and gas-cooling component, which ties in with the existing support infrastructure. The second converter will serve to ensure continued operation during periodic converter overhauls or in the event of a major converter failure of the operating unit. This project has a building footprint that measures 24 m x 28 m and is about 54 m high. The height of the building is necessary to accommodate the relatively slender converter and water-cooled uptake as well the lance-handling equipment. It adjoins the existing converter building on its eastern side.

The ACP technology is the first application of Ausmelt converting to the PGM industry. Extensive testwork was carried out on a pilot scale prior to embarking on the original project. Two modes of operation were tested – continuous-processing and batch-processing. Batch-processing was deemed at the time to be the most appropriate. However, during early ramp-up, a decision was made to change to continuous-converting in order to achieve better end-point control of the converting process and, in this way, render the process more stable. This change has not warranted any significant design change in the Stage B converter. It was also found during early ramp-up that accretion build-up within the converter uptake was resulting in unacceptable furnace availability. In addition to measures implemented on the operating unit – which have brought the accretion build-up under control – the design of the Stage B freeboard has been changed to further reduce the possibility of accretion formation.

Air Products has built a R120 million over-the-fence air-separation unit that will supply about 237 t/d of oxygen, 2.1 t/d of nitrogen as well as up to 45 000 normal m³/h of plant and instrumentation air to the smelter.

The converter has the capacity to treat 72 000 t of converter matte per year. This means that the plant will be able to treat material containing 33 000 t of nickel a year, corresponding to about 4 million ounces of platinum per year.

The converter is designed to accept all furnace matte from the three Anglo Platinum smelters (Polokwane, Mortimer, and Waterval), as well as Waterval slag-cleaning furnace matte. The furnace matte is transported in solid form from the smelters to the ACP, where it is fed into the converter together with silica and coal as feedstock.

The furnace technology was a consolidation of Hatch (engineering, procurement, and construction management contractor for the furnace, and copper cooler design), Ausmelt (Sirosmelt lance smelting), and Foster Wheeler technologies. The furnace, with

an uptake of high-pressure water-cooled jacketed membrane and water running at temperatures of up to 220°C and pressures of up to 70 bar was designed by Foster Wheeler. Other technology suppliers were Paul Wurth, as well as Germany-based Lockwood Greene Petersen (acid plant). The ACP uses the most advanced technology, and is the only converter in the world with a combination of waffle coolers in the crucible and high-pressure water membrane boiler freeboard and uptake.

Mortimer Smelter

The Mortimer Smelter at the Union Section has one six-in-line furnace, used primarily for smelting UG2 concentrates. The Union furnace matte is converted at the Waterval Smelter.

Polokwane Smelter

The Polokwane Smelter was built to process concentrates from Anglo Platinum's operations (fully or jointly owned) on the Eastern Limb of the Bushveld Complex. The 68 MW six-in-line furnace is the highest-power furnace in the PGM industry, and is designed to run at a higher power flux than the other six-in-line furnaces. The furnace was designed to achieve the high operating temperatures necessary for smelting concentrates with high chromite contents. The smelter was successfully commissioned in March 2003, ahead of schedule and within budget. Having a design capacity of 650 000 tons per year will allow the Polokwane Smelter to process concentrate from existing and new mines on the Eastern Limb, as well as to provide some backup capacity for the Waterval and Mortimer smelters. The plant is expected to eventually account for 40% of Anglo Platinum's production. The furnace was running at about 50% capacity around the middle of 2004.

Some repairs have been required on the furnace cooling system, and Anglo Platinum announced a six-week shutdown of the furnace from 10 September 2004, to enable essential repairs to be carried out on the smelter's cooling system (copper waffle coolers).

IMPALA PLATINUM

At Impala Platinum, flotation concentrate is electrically smelted in six-in-line electric furnaces, and the resulting furnace matte treated in Peirce-Smith converters to remove the iron. The iron-free (less than 1% Fe) converter matte is granulated and sent to the base metals and precious metals refineries in Springs for further processing.

Impala has four rectangular six-in-line submerged-arc furnaces, of which only the two largest (38 MW and 35 MW) are in use. The furnaces are served by four Niro spray driers. Six Peirce-Smith converters are available, although not all are in operation at any

given time. In addition to the treatment of their own concentrate, Impala also have a significant business in the toll-treatment of a variety of materials.

Impala's first six-in-line furnace (7.5 MVA) was installed in 1969, and a number of different furnaces and driers have been installed (and some de-commissioned) since then. A detailed chronology of the installations has been provided elsewhere (35). The largest furnace (39 MVA) is 25.9 m long and 8.2 m wide, and has electrodes of 1.14 m in diameter, spaced 3.32 m apart (centre to centre). The phase current is maintained at 24 to 26 kA, and the phase voltage is between 500 and 550 V. The furnace is normally operated at 38 MW.

The fine particle size of the concentrate presented serious problems in the initially installed drying process (filters). Filters became blocked; the concentrate became too dry; and dust losses increased. Furnace blow-backs (sometimes explosive in nature) resulted from steam generation inside the furnace, and had a detrimental effect on atmospheric pollution (as well as on the loss of feed material). Niro spray drying, to a moisture content of less than 0.5%, reduced the above problems. 'Dry feeding' of the furnaces was instituted in 1986. Once some materials of construction problems were solved, typical running times on the driers exceeded 95%. Dry feeding of concentrate virtually eliminated 'blow-backs', and made it possible to reduce the number of feed pipes from 28 to between 4 and 6, as the distribution of feed in the furnace was improved. Dry feeding also increased furnace efficiency by 12 to 15%. Using burnt lime in place of limestone increased smelting capacity by about 5%.

Because of the high value of the PGMs, the grade-recovery relationship is heavily skewed towards maximum recovery. This has a major impact on smelter capacity, as has Cr_2O_3 control in the concentrator. The low grade of concentrate smelted, in order to maximize PGM recovery over the concentrator, has a high SiO_2 and MgO content, and a low FeO content. This requires a high operating temperature, namely 1460°C for slag, and 1260°C for matte. High power fluxes help to prevent spinel build-ups, and the Cr_2O_3 contents of the concentrates are carefully controlled. The Merensky concentrate has a Cr_2O_3 content of less than 0.5 per cent, and the UG2 concentrate runs at about 1.6 to 1.7 per cent Cr_2O_3 . This results in an overall Cr_2O_3 content of less than 1 per cent in the blended concentrate. Impala was the first producer to experiment with UG2 on a plant scale as early as 1971 (35).

High-quality magnesite refractory bricks are used to line the hearth and lower side walls, while firebricks are used for the upper walls and roof. Copper cooling of side- and end-walls was provided by Hatch Associates.

A very low tonnage of converter matte (also known as white metal) is the final product from the smelter. This is granulated and supplied as the feedstock for the Impala Refineries. Impala's Base Metals Refinery uses Sherritt Gordon ammonia leach technology.

All waste gases, from driers, furnaces, and converters, are treated prior to disposal. During smelting and converting, two distinct gas streams are produced. The converter off-gas contains 4-9% SO₂ which is treated in a conventional sulphuric acid plant to produce 94% H₂SO₄. (The single-contact Lurgi-designed sulphuric acid plant is one of the few in the world running on converter gas alone.) The furnace off-gas contains 0.6-0.9% SO₂ which was (until 2002) vented through a 91 m concrete stack to the atmosphere. In October 2002, a Sulfacid plant was installed to recover SO₂ from the furnace off-gas into a weak (up to 13%) sulphuric acid solution. Prior to the installation of the Sulfacid plant, Impala's smelter emitted 34 tons of SO₂ per day to the atmosphere from its off-gas stacks (28 t/d from the furnace off-gas stack, 4.5 t/d from the acid plant stack, and 1.5 t/d from the drier stacks). The Sulfacid plant installation was very successful, and achieves an 85% recovery of SO₂ from the furnace off-gas (36). The Sulfacid plant reduced SO₂ stack emissions by 40% to less than 20 tons per day (37).

In the 1990s, the Impala operation was predominantly characterised by smelting Merensky ore concentrates in low power density furnaces, with converter slag being recycled to the furnaces. Now, in 2005, the plant operates fewer (two) higher intensity copper-cooled furnaces treating a much greater proportion of UG2 ore concentrates and other toll-smelted materials. Converter slag is no longer recycled to the furnaces. The chromium content permitted in the furnace feed has risen from about 0.8% to about 1.8% Cr₂O₃. SO₂ emissions have decreased significantly. Over the time that platinum production has doubled, the number of smelter employees decreased from about 500 to about 350. Impala's smelting strategy seems to have been one of using only tried and trusted conventional furnace and converter technologies and spending a lot of effort in getting them to work reliably and optimally.

LONMIN PLATINUM

Operations at the Western Platinum Smelter commenced in December 1971 with the commissioning of a 7.5 MVA Merensky six-in-line furnace. In November 1982, the smelter was expanded with the commissioning of two 2.3 MVA Infurnco circular furnaces to smelt UG2 concentrate. The UG2 smelting facilities were expanded in March 1991 with the commissioning of three 5 MVA Pyromet circular furnaces. Western Platinum was the first mine to commission separate facilities for treating UG2 ore for the recovery of PGMs and associated base metals (38). The Merensky six-in-line furnace was subsequently upgraded to 10 MVA.

For a long time, the Merensky and UG2 concentrates were smelted separately. The Merensky concentrate (received as a slurry) was filtered in a rotary drum filter and partially dried through a rotary kiln (to a moisture content of about 8 per cent) before being fed into the six-in-line furnace. The green charge and limestone flux were manually rabbled inside the furnace. The furnace matte was tapped periodically, while slag was tapped almost continually and granulated in a high-flow water stream.

Converter slag was returned to the Merensky furnace to recover entrained matte. The UG2 concentrates, containing relatively high concentrations of chromite, were dried in spray driers. The bone-dry concentrate was then pneumatically conveyed to one of several circular three-electrode submerged-arc AC electric furnaces. Burnt lime was used as a flux. Smelting of UG2 concentrate in a circular three-electrode furnace (39) requires a higher power flux than the smelting of Merensky concentrate. A higher than usual smelting temperature is used, and the smelting zone is more concentrated, so that the slag is more agitated. The agitation of the slag is necessary to promote coalescence of the small quantity of matte that has to be separated from the slag. The agitation also causes the accretion of chromite on the hearth to be minimized. Around 80 to 90 per cent of the chromite present is discarded in the furnace slag. Furnace matte with a chromium content of 2 per cent could be blown to converter matte containing less than 40 ppm of chromium, which is acceptable to the base-metal refinery. The Merensky furnace matte and UG2 furnace matte were combined and converted together.

All the converter matte is processed at the base-metal refinery (BMR), using Sherritt technology from Canada, to produce nickel sulphate crystals, pure copper cathodes, and a high-grade PGM concentrate. The capacity of the BMR was expanded in 1991 to be able to treat 54 tons of converter matte per day.

Water granulation of the converter matte was introduced to prevent the formation of magnetite and trevorite (which previously formed by oxidation during cooling in moulds before crushing). These materials did not leach significantly in the BMR and reported to the PGM concentrate. The PGM concentrate currently has a grade of about 60 per cent (38).

Western Platinum was the first company to exploit the UG2 on a large scale for its PGM content (18). Metallurgical investigations were undertaken in conjunction with Mintek during 1980. Mining of the UG2 at Western Platinum Mine commenced in 1982, and the UG2 concentrator started up in March 1983. The UG2 ore is generally milled separately from the Merensky ore. More than 80 per cent of Lonmin Platinum's current annual production is sourced from the UG2. Lonmin UG2 ore is commonly mined at depths ranging from 30 m to 700 m below the surface.

In 2002, Lonmin started operating a new 28 MW circular furnace that was able to accept a combined blend of UG2 and Merensky concentrates that were dried in a flash drier. The furnace performed well and was used as the sole production unit. On 26 December 2002, a serious explosion occurred on the south matte tap hole of the No.1 Furnace some nine months after commissioning (40). This resulted in a lengthy shut down of the smelter. Contingency plans were put in place soon after the explosion and included the re-commissioning of the Merensky and Pyromet furnaces, and the signing of an agreement with Impala Platinum to toll treat a portion of the concentrates. The additional costs incurred amounted to some \$34 million (\$8 million for the re-commissioning, and \$26 million for toll smelting). A team consisting of personnel from Lonmin's metallurgical services and Hatch (the consultants used during the design and

construction of the No.1 Furnace) launched an extensive investigation into the cause of the failure. The conclusion reached was that hot matte had come into contact with the deep matte waffle cooler resulting in a release of water into the furnace and a subsequent explosion, which destroyed the deep matte waffle. A review of the original design indicated that improvements could be made to the monitoring and process control systems, feed blending systems, the refractory lining, the copper waffle cooling system, and the tap blocks. These design improvements, together with the re-building should significantly reduce the risk of a similar accident. The repaired furnace was re-started in December 2003, with the first matte tap taking place on 25 January 2004.

However, on 18 November 2004, a second (unrelated) serious incident took place when cooling water leaked into the furnace, causing a steam explosion, resulting in structural damage. Although the smelter was out of action for approximately two months until 20 January 2005, Lonmin was able to treat much of its concentrate output via its old Pyromet furnaces, while sending the remaining material to Impala for toll-refining. The outage consequently had little impact on refined PGM production, although the cost implications were about \$14 million.

A dual alkali sulphur fixation plant, producing a calcium sulphite product, was commissioned in July 2003. This has brought the level of SO₂ discharged to the atmosphere via the SO₂ plant stack to less than 3.5 tons per day (representing about 96% SO₂ capture).

NORTHAM PLATINUM

Northam operates the world's deepest platinum mine, with a lowest depth of 2000 m. The ore grade is 10 g/t *in situ*, and 5.5 g/t mined. The first smelting was carried out in August 1992, with first production in 1993.

Northam uses a very conventional matte-smelting process, using a single six-in-line furnace. In 1999, Northam announced plans to increase their production of UG2 concentrate above the roughly 10 per cent (relative to the combined feed of Merensky and UG2 concentrates) that was in use at the time. Concentrates are dried in a flash drier, and the dry feed is pneumatically fed to the furnace. Burnt limestone is used as a flux. The six-in-line furnace, supplied by Davy, is rated at 16.5 MVA (15 MW), with a normal operating range between 11 and 12 MW. The smelter produces about 360 tons per month of converter matte.

In the first leaching stage of Northam's base metals refinery, nickel is removed as a sulphate. The PGM concentrate is removed as the residue from a pressure leaching stage. Finally, copper is removed by electrowinning. The PGM concentrate is refined by Heraeus in Germany.

In January 2003, the smelter was shut down for a complete furnace rebuild after operating continuously for the past eleven years. The plant was re-commissioned during March 2003.

MAKWIRO PLATINUM (ZIMPLATS)

The Great Dyke in Zimbabwe is the world's second largest platinum deposit, but it took until the 1990s for it to be exploited. PGM concentrates were produced commercially at Mimosa from 1995, but no smelting was done locally. However, BHP and Delta Gold of Australia entered into a joint venture to develop what was then known as Hartley Platinum. The mine and plant was designed for a capacity for treating 180 000 tons of ore per month, with full-scale production of 0.15 Moz/a Pt, 0.11 Moz/a Pd, 0.012 Moz/a Rh (*i.e.* Pt+Pd+Rh = 0.27 Moz/a).

A circular furnace was chosen for the smelter. While evaluating options for the plant, both Ausmelt and conventional electric smelting options were considered, but no significant difference in capital or operating costs was projected. Therefore, conventional technology was implemented. The furnace was lined in mid 1996. Smelting began in early 1997, and had not yet reached full capacity when operations were suspended in 1999. (The sidewalls needed to be rebuilt in August 1998.)

Under the auspices of the new owners Zimplats, the renamed Makwiro Platinum started up again a few years later. Ore is processed at the Selous Metallurgical Complex at the Hartley mine site. The first concentrate was produced in November 2001, the first furnace matte in January 2002, and the first converter matte in April 2002. A major break-out at the smelter occurred in July 2003, resulting in the Elkem furnace being out of operation for seven weeks. The breakout occurred under a slag taphole following magnesite hydration, which took place when cooling water from a slag tapping block entered the furnace. A rebuild of approximately 25 per cent of the lower sidewall, in the region of the plate coolers, was required. The furnace was restarted 6 August 2003, following these improvements. Converter matte is transported to Impala Refining Services in South Africa for refining and marketing.

CONCLUSIONS

South Africa dominates world production of platinum. Because of the high value of the PGM products, a risk-averse, conservative approach is often adopted towards the introduction of changes in processing technology, and PGM matte smelting remains very closely based on traditional nickel-copper matte smelting. However, platinum smelting has undergone many changes during the past three-quarters of a century, and will continue to develop further, in particular to address environmental concerns, and to maximize recovery from all available ore-bodies. The metallurgically challenging

extraction of PGMs will continue to require an advanced understanding of many aspects of PGM smelting, in both chemical and electrical fundamentals (41), as well as an improved understanding of the physical aspects of furnace design. Clearly, large-scale pilot testing will be required for new processes that are currently under development, with a view to addressing the limitations of the conventional matte-smelting processing route.

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