

Special issue on
Platinum in South Africa
to commemorate the
75th anniversary of the
discovery of the
Merensky Reef

Platinum smelting in South Africa

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South Africa has more than 80 % of the world's platinum reserves, and is the 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 Bushveld Complex geological formation. South Africa's PGM output is derived almost exclusively from the Bushveld Complex, with only about 0.1 % coming from the gold deposits of the Witwatersrand and Free State, and the Phalaborwa copper deposit.

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.¹⁻³ South Africa currently has four integrated primary platinum producers, namely Amplats^{4,5} (Anglo American Platinum Corporation Ltd, formerly Rustenburg Platinum Holdings Ltd), Impala Platinum,⁶ Lonmin Platinum⁷ (which includes Western Platinum), and Northam Platinum.⁸ Their range of

operations includes open-cast and underground mining, milling, flotation, drying, smelting, converting, refining, and marketing. Amplats and Impala Platinum are the two 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 Stillwater Mine in Montana, and Hartley Platinum in Zimbabwe are major primary producers of PGMs. This factor is of great importance in the supply of these metals, as South African producers are able to effect a marked change in the output of platinum within a relatively short time to conform to market requirements.⁹

Table 1 shows world PGM supply and demand by country, as well as PGM reserves. It should be noted that the supply from Russia is believed to significantly exceed its actual production capacity, as a large part of its sales are from stockpiles. As Russia does not supply official figures for PGM production, there remains some doubt about the accuracy of these numbers. For example, other estimates show Russia producing only 2.98 million ounces per annum of Pt+Pd.

Table 2 shows the production by individual companies of the economically most important PGMs, namely platinum, palladium and rhodium. Most production figures are from the 1998 annual reports of the companies,^{4,8} although the estimates for production at Noril'sk are based on news articles. The most recent estimate of platinum production at Noril'sk¹⁰ is significantly lower than previous estimates (around 0.4–0.45 million ounces per annum). Other 1997 figures have shown Russia producing 0.23–0.64 million ounces of platinum annually.

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Table 1. Supply and demand figures¹¹ for 1997 are given in millions of ounces (Moz) per annum. PGM reserves¹² are also shown in millions of ounces. (1 million ounces = 31.1 metric tons.)

	Pt	Pd	Rh	Ru	Ir	Os	Total Pt, Pd, Rh	PGM reserves (Moz)
SA supply	3.70	1.81	0.377	0.49*	0.80*	0.016*	5.9	2030
Russia supply	0.90	4.80	0.240				5.9	199
Canada supply	0.16	0.28	0.012				0.5	10
USA supply	0.08	0.27					0.4	23
Others supply	0.13	0.10	0.003				0.2	23
Total world supply	4.97	7.25	0.632				12.9	2280
Total world demand	5.20	7.46	0.460	0.357	0.127	0.005 [#]	13.1	
SA as % of world	74	25	60				46	89

*The estimated figures for South African production of Ru, Ir and Os are based on doubling 1984 production figures,¹³ as has happened with platinum production over the period.

[#]The figure for world demand of osmium is based on a 1993 estimate.¹⁴

Table 2. PGM production figures for individual producers in 1998 (Moz/yr).

	Pt	Pd	Rh	Pt+Pd+Rh
Primary producers of PGMs				
Amplats	1.86	0.93	0.177	2.97
Impala Platinum	1.05*	0.56	0.112	1.72
Lonmin Platinum	0.63	0.29	0.088	1.01
Stillwater, USA	0.11	0.35	0.004 [†]	0.46
Northam Platinum	0.18	0.08	0.015	0.28
Hartley Platinum, Zimbabwe	0.07	0.05	0.004	0.13 [‡]
Producers of PGMs as by-products				
Noril'sk, Russia	0.35	0.76	0.034 [†]	1.14
Inco, Canada	0.14	0.17	0.011	0.33
Falconbridge, Canada	0.05	0.09	0.003	0.14

*The figures for Impala exclude the additional approximately 30 % production from toll treatment.

[†]The figures given for rhodium production at Stillwater and Noril'sk are the author's estimates.

[‡]Hartley started smelting in early 1997, and has not yet reached full annual capacity of 0.15 Moz Pt, 0.11 Moz Pd, 0.012 Moz Rh; i.e. Pt+Pd+Rh = 0.27 Moz. Operations were suspended in 1999, but may yet resume in the future. The production figures quoted here are the author's estimates.

According to Johnson Matthey,¹¹ the total world demand for platinum in 1997 was distributed as follows: jewellery (42 %), autocatalysts (28 %), industrial (25 %), investment (5.6 %).

The total world demand for palladium was distributed as follows: autocatalysts (40 %), electrical (34 %), dental (18 %), other (8.6 %).

Platinum-group metal ores

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.¹⁵⁻¹⁷ The Platreef is mined only at Potgietersrus Platinum (Amplats), but Merensky and UG2 ores are mined 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.

There are currently twelve active or planned platinum mines in the Bushveld Complex, eleven exploiting the Merensky Reef and UG2 chromitite layer, and one, Potgietersrus (an open-cast mine), mining the Platreef of the northern limb of the Bushveld Complex. There is only one active mine on the eastern limb,

namely the Atok Mine of Lebowa Platinum (belonging to Amplats). The other mines are all on the western limb. Amplats has the Rustenburg, Union, and Amandelbult Sections of Rustenburg Platinum, as well as the Bafokeng-Rasimone Mine, which is soon to open. Impala Platinum is supplied by its own Impala Mine, as well as by Kroondal Mine (owned by Aquarius Platinum of Australia), among others. Lonmin has Western Platinum, Eastern Platinum and Karee Mine. Northam Platinum has Zondereinde.

Ore from the Merensky Reef contains up to 3 % base-metal sulphide minerals, distributed as follows: pyrrhotite (45 %), pentlandite (32 %), chalcopyrite (16 %), and pyrite (2-4 %). 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

Table 3. Average grades of the individual precious metals in Merensky, UG2 and Platreef ores,¹⁵ and their current potential value. Market prices¹⁰ are for the last week in February 1999.

	\$/oz	Merensky ore			UG2 ore			Platreef ore		
		g/t	\$/t	mass %	g/t	\$/t	mass %	g/t	\$/t	mass %
Pt	379	3.25	39.54	59	2.46	29.98	41	1.26	15.35	42
Pd	350	1.38	15.47	25	2.04	22.96	34	1.38	15.53	46
Rh	860	0.17	4.56	3	0.54	14.93	9	0.09	2.49	3
Ru	37	0.44	0.52	8	0.72	0.86	12	0.12	0.14	4
Ir	395	0.06	0.70	1	0.11	1.45	1.9	0.02	0.30	0.8
Os	400	0.04	0.57	0.8	0.10	1.31	1.7	0.02	0.23	0.6
Au	287	0.18	1.62	3.2	0.02	0.22	0.4	0.10	0.94	3.4
Total PGM+Au		5.5	62.99	100	6.0	71.70	100	3.0	34.99	100

Table 4. Typical content of base metals in Merensky, UG2 and Platreef ores,¹⁵ and their current potential value. Market prices¹⁰ are for the last week in February 1999.

	\$/lb	Merensky ore			UG2 ore			Platreef ore		
		% in ore	\$/t	mass %	% in ore	\$/t	mass %	% in ore	\$/t	mass %
Ni	2.25	0.13	6.44	62	0.07	3.47	80	0.36	17.84	67
Cu	0.66	0.08	1.16	38	0.018	0.25	20	0.18	2.62	33
Total PGM+Au	0.21	7.61	100	0.09	3.72	100	0.54	20.46	100	

ore, and the minerals are found in a silicate substrate, while UG2 ore has a chromite matrix. The Cr₂O₃ 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 ratio of PGM to base metals is variable.⁹

The Merensky and UG2 reefs are situated in close proximity to each other. The UG2 reef lies anywhere between 20 and 330 m below the Merensky horizon, and varies in thickness between 0.15 and 2.5 m. Reserves of PGMs plus gold are estimated¹⁴ at 547 million ounces in the Merensky Reef, and more than 1000 million ounces in the UG2 reef. Another estimate² has it that 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 Cr₂O₃ content of UG2 ore is about 30 %, compared 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.¹⁸ 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 3. Further detail regarding the distributions of the individual PGMs in various reefs and sectors of the Bushveld Complex is available elsewhere.¹⁶ The content and value of base metals in the three ores are shown in Table 4.

It is evident from the data above (as well as from actual revenues recorded by the producers) that Pt, Pd and Rh make up a remarkably constant 95 % 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 80–85 % of the value of all the metals produced (i.e. PGMs plus base metals). For UG2 ore, the

fraction is 90 %.

The average grain size of the PGM minerals is about 45 μm in Merensky ore and 15 μm in UG2. 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).¹⁶ During concentration, the recovery of PGM+Au is around 80–87 %.¹⁶ Typical analyses of the Merensky and UG2 concentrates at Lonmin's Western Platinum Mines have been published elsewhere.^{16,19,20}

From a given quantity of ore, the mass of UG2 concentrate is 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 higher, and the amount of concentrate to be smelted is smaller. The total cost of treatment of UG2 ore is claimed^{21,22} to be considerably lower than for Merensky ore, for the following reasons:

- Mining costs are lower, mainly because of the higher relative density of the UG2 reef. The relative density of Merensky ore is 3.2 and that of UG2 ore is 4.3.¹⁵
- Crushing costs are lower because UG2 ore is more friable. Milling costs are also lower.
- Flotation reagent costs are much lower, because Merensky ore requires the use of a talc depressant.
- Smelting costs are lower because much smaller quantities of concentrates have to be smelted (per quantity of platinum produced).

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 are smelted and converted, 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,¹⁶ the PGM contents during the various stages are as follows:

Table 5. Concentrate analyses.*

	Al ₂ O ₃	CaO	Co	Cr ₂ O ₃	Cu	FeO	MgO	Ni	S	SiO ₂	PGM	Total
Amplats: Waterval	3.2	4.7	0.08	0.80	2.1	20	15	3.6	9	34	143	92
Amplats: Union	3.8	2.5	0.04	2.59	1.1	15	20	2.2	5	38	142	90
Impala	4.1	2.9	0.06	1.1	1.3	18	18	2.1	5.6	42	138	95
Lonmin: Merensky	1.8	2.8	0.08	0.4	2.0	23	18	3.0	9	41	130	101
Lonmin: UG2 blend	3.6	2.7	0.06	2.8	1.2	15	21	2.1	4.1	47	340	100
Northam	2.6	3.0	0.05	0.87	1.3	17	18	2.5	5.4	47	132	97

*All as percentages except PGM (g/t).

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:¹⁴ mining (72 %), concentrating (10 %), smelting (9 %), refining (9 %).

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

Simple description of conventional matte-smelting process

A simple representation of the most common process is shown in Fig. 1.

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.

Table 5 shows the analyses of the various concentrates. Typical PGM grades are over 100 g/t for Merensky concentrates and around 400 g/t for UG2 concentrates. Some blending takes place.

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 2.7–3.3, and a denser molten matte (rich in nickel and copper sulphides, and other base and precious metals) with a relative density of about 4.8–5.3. 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 that 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 South 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). 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. It is tapped into ladles and transferred by crane to a converter vessel.

The furnaces are normally operated with a 'black top', that is, 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,⁹ a 15 cm layer of concentrate covered a 100 cm layer of slag, which in turn covered a 58 cm layer of matte.

The electrical power consumption in the furnace is approximately 600–1100 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 accounts for approximately 40 % of the direct smelting costs.⁹

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

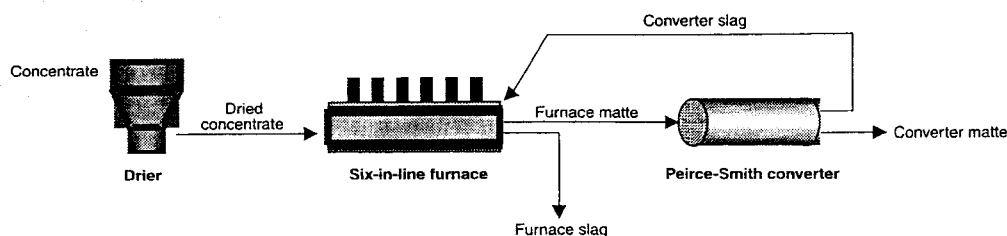


Fig. 1. Schematic representation of a typical platinum smelting process.

Table 6. Furnace matte analyses.*

	Co	Cr	Cu	Fe	Ni	S	PGM	Total
Amplats: Waterval	0.5	0.5	9	41	17	27	640	95
Amplats: Union	0.3	1.9	7	37	12	25	830	83
Impala	0.4		16	34	20	28	1050	99
Lonmin: Merensky	0.5	0.23	9.7	37	17	28	1000	92
Lonmin: UG2	0.5	0.29	9.8	35	17	28	2500	91
Northam	0.4		7.9	41	16	27	724	93

*All as percentages except PGM (g/t).

Table 7. Furnace slag analyses.*

	Al ₂ O ₃	CaO	Co	Cr ₂ O ₃	Cu	FeO	MgO	Ni	S	SiO ₂	Total
Amplats: Waterval	3.3	6.4	0.05	0.8	0.11	31	15	0.19	0.50	46	103
Amplats: Union	3.0	5.8		2.8	0.10	20	13	0.16	0.33	41	86
Impala	6	8	0.03	1.2	0.11	21	18	0.11	0.25	47	101
Lonmin: Merensky	2.0	9.8	0.05	1.2	0.09	28	19	0.15		44	104
Lonmin: UG2	3.9	13	0.02	2.4	0.13	9.2	22	0.11		47	98
Northam	1.5	10	0.03	0.8	0.10	21	20	0.2		44	98

*All as percentages.

entrainment of matte in slag. A build-up of magnetite or other spinels reduces 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.

Table 6 shows the analyses of furnace matte produced in various smelters. Note that the balance of analyses not totalling 100 % is assumed to be entrained slag. The production rate for the largest producer (Amplats Waterval smelter) is approximately 20 t/h of matte.

Table 7 shows the analyses of furnace slag produced in various smelters. As a good general first approximation, the tonnage of furnace slag produced is approximately equal to the tonnage of concentrate processed.

Converting

During the converting process, air is blown into the molten matte, over a period of a few hours, to remove much of the iron and sulphur by oxidation (primarily of FeS). The converters in operation at present are of the Peirce-Smith type; these are of horizontal cylindrical shape with an opening at the top for charging and discharging; tuyères 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 with the approximate composition of fayalite (2FeO·SiO₂), with some dissolved magnetite. Some of the sulphur leaves the system in the gas phase as sulphur dioxide (SO₂). 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 (such as spillages) 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 poured into cast-iron moulds or refractory-lined pits, and crushed, or it can be granulated by tipping it into a fast-flowing stream of water.

The converter slag requires further treatment, as the vigor-

ously 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. In many instances, the molten converter slag is 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 advantageous, 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.

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.

Both Amplats and Lonmin smelt UG2 and Merensky concentrates separately in the main, but the matte from both types of furnace is converted together.

The converter matte (also known as white metal) has a relative density of about 6, and consists primarily of Ni₃S₂, Cu₂S, 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.

Analyses of converter matte and slag are shown in Tables 8 and 9.

Off-gas handling

It is 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 SO₂ 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 % leaves in the converter gases, 20 % in the furnace gases, 15 % in the converter matte, and 5 % in the furnace slag.²³ The furnace gases have an

Table 8. Converter matte analyses.*

	Co	Cu	Fe	Ni	S	PGM	Total
Amplats: Waterval	0.5	26	2.9	47	21	2100	97
Impala	0.4	31	0.5	47	21	3430	100
Lonmin	0.6	29	1.4	48	20	6000	99
Northam	0.5	27	1.0	51	19	2570	98

*All as percentages except PGM (g/t).

Table 9. Converter slag analyses.*

	Al ₂ O ₃	CaO	Co	Cr ₂ O ₃	Cu	FeO	MgO	Ni	S	SiO ₂	Total
Amplats: Waterval	0.7	0.4	0.45	0.4	1.17	63	1.1	2.25	2.4	27	99
Impala	1.8	0.3	0.43	1.4	1.06	64	0.71	1.90	1.0	27	100
Lonmin	0.7	0.5	0.39	1.4	0.94	65	0.78	1.43	1.7	28	100
Northam	1.3	0.7	0.4	0.36	1.37	64	0.82	2.18		27	98

*All as percentages.

SO₂ content of around 0.4 %, which is generally considered too low for efficient recovery. The converter gases, for 70 % of the blowing time, have an SO₂ content of more than 4 %; the overall variation is typically from 2.5 to 6 %.

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.

Historical smelting developments

The early days

Platinum mining on a large scale began around 1926, and before the 1920s were over, no fewer than seven mining operations had started in South Africa. The platinum ores were processed mostly by traditional milling and gravity-table concentration. Flotation was used for the first time to produce a sulphide platinum concentrate in 1926, at Potgietersrus.²

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 from near surface was nearly exhausted. It became necessary to commission a flotation plant to separate the sulphide-rich unoxidised ore from over 30 m depth, 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 U.K. refinery. A second blast furnace was commissioned in 1953. Blast-furnace smelting was labour intensive, and used expensive coke. Furthermore, a very large volume of gas was emitted containing between 1 and 2 % 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 100–200 °C higher than those generated 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 used firebrick for the roof. Concentrates were pelletized and dried prior to being fed to the furnace.

Drying

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

Sidewall cooling

Hatch copper-finger coolers have been installed in the sidewalls of some furnaces, and this has improved the integrity of the furnace linings.

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 have been 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 Merensky ore.

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. Testwork²² showed that a UG2 concentrate could be produced having a PGM grade around 430 g/t, at a recovery of 87%. This was achieved with a mass pull (concentrate to ore ratio) of about 1%, and a Cr₂O₃ content of 2.9%. Even higher grades (more than 1000 g/t) could be achieved at even higher recoveries (more than 90%), if the constraint on the Cr₂O₃ content was relaxed (to between 4 and 10%).

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^{19,20} 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. As shown previously, in Table 5, UG2 concentrate may have more than twice the PGM concentration than Merensky concentrate. (In addition, the chromite content of UG2 ore is potentially saleable, after recovery of the PGMs, and the removal of gangue.)

Pilot-scale tests^{19,20} 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 (2.5–3.5 g/t) to be discarded, so it is granulated and returned to the flotation circuit for the recovery of the metals. Lime or limestone is sometimes used as a flux, to improve the compatibility of the slag with the basic refractory lining.

Continuous converting

Continuous converting is under investigation by a number of platinum producers. This is seen as a way to improve environmental issues, and to avoid the bottleneck of those plants where the converters limit increased production. The steady stream of SO₂ generated during continuous converting is suitable for sulphuric acid production.

South African platinum producers

Production data and processing details for the four South African platinum-group metal producers are given in Table 10.

Amplats

Amplats has two smelter plants. The largest is the Waterval Smelter at the Rustenburg Section, having furnaces and converters. The other is the Mortimer Smelter at the Union Section, which has one furnace (used primarily for smelting UG2 concentrates) but no converters. The Union furnace matte is converted at the Waterval Smelter. The Waterval Smelter is described in more detail below.

The first electric furnace installation for platinum matte smelting was commissioned in 1969, and has been described by Mostert²³ and others.²⁴ 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/cm². 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% addition of limestone was made to the concentrate as a flux. The resulting slag had a liquidus temperature of 1300 °C, an electrical resistivity of 4.7 Ωcm at 1400 °C, and a viscosity of 3.7 poise at 1400 °C.²⁴ A temperature gradient of 0.75 °C per cm was measured in the slag.²⁴

During the 1990s, most of the smelter was upgraded. The rotary multi-hearth driers and pelletizing plants, which produced semi-dry pellets (about 10% 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 has 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 converters were lengthened from their original 6.1 m (20 ft) to the current 7.6 m, and their number was increased from four to six.

Concentrate is received from five concentrators, namely Waterval, Frank, Klipfontein, Amandelbult, and Potgietersrus. 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 furnaces at the Waterval Smelter are rated at 39 MVA (34 MW). They measure 25.8 m × 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/m². 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% of the mass of the concentrate. The furnace off-gases pass through recently installed ceramic filters, resulting in significantly reduced dust losses, and are then expelled to the atmosphere via the main stack.²⁵ The gas produced in the Peirce-Smith converters during blowing is rich in SO₂ (4–6%) and is routed to the single-contact-absorption sulphuric acid plant.

Amplats have announced⁵ that they are currently developing a new, continuous converting process.

Amplats uses a matte slow-cooling process²⁶ 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, to shorten 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 dedicated refinery, without the need to pass first through a base metals refinery. In this way, a clear separation is made between processing the base metals and precious metals.

Table 10. Comparison of South African platinum producers.

	Amplats	Impala	Lonmin	Northam
Year of first production	1926	1969	1971	1992
Annual production:*				
Platinum (Moz)	1.861	1.052	0.628	0.177
Palladium (Moz)	0.931	0.557	0.291	0.083
Rhodium (Moz)	0.177	0.112	0.088	0.015
Gold (Moz)	0.007			
Nickel (kt)	20.6	7.7	2.88 (sold as NiSO ₄)	1.86 (sold as NiSO ₄)
Copper (kt)	10.9	4.5	1.74	0.98
Cobalt (t)	250 (sold as CoSO ₄)	46	0	0
Ore grades in proven reserves (g/t)				
Merensky	5.1	8.3	5.6	9.5
UG2	4.7	9.1	6.1	6.5
Ore milled (Mt/a)	22.01	14.51	9.19	1.80
% UG2 ore mined	19	46	77	~0**
Average head grade (g/t)	5.52	5.17	5.16	6.25
PGM recovery in concentrator (%)	Merensky: 88; UG2: 81; Platreef: 82	Merensky: 90; UG2: 79	84	90
Concentrate smelted, tons per hour (dry basis)	74	67	Mer: 4.8; UG2: 11.0	10.6
Driers	Flash driers	Spray driers	Spray driers	Flash drier
Furnaces	<i>Waterval:</i> 1. 39 MVA (hatch six-in-line, 25.8 m long, 8.0 m wide) 2. 39 MVA (Hatch six-in-line, 25.8 m long, 8.0 m wide) <i>Mortimer:</i> 3. 19.5MVA (Six-in-line)	1. — (decommissioned) 2. 7.5 MVA (Six-in-line) 3. 7.5 MVA (Six-in-line) 4. 15 MVA (Six-in-line) 5. 39 MVA (Six-in-line, 25.9 m long, 8.2 m wide)	1. 10.5MVA (Barnes-Birlec, six-in-line, Merensky) 2. 5 MVA (Pyromet, 3-electrode, ID 5.2 m) 3. 5 MVA (Pyromet, 3-electrode) 4. 5 MVA (Pyromet, 3-electrode) 5. 2.3 MVA (Infurnco, 3-electrode) 6. 2.3 MVA (Infurnco, 3-electrode)	1. 16.5 MVA (Davy, six-in-line, 25.9 m long, 8.7 m wide, 5.6 m high)
Power flux (kW/m²)	165	180	Mer: 120; UG2: 235	90
Slag to matte production ratio	4.5	5.9	Mer: 3.5; UG2: 6.3	8.5
Energy requirement (kWh/t of concentrate)	785	720	Mer: 1270; UG2: 880	1044
Converters	Six Diameter 3.0 m, length 7.6m	Four Diameter 3.0 m, length 4.5 m	Two Diameter 3.0 m, length 4.6 m	Two Diameter 3.0 m, length 6.1 m
Granulation	Furnace slag	Furnace slag Converter slag Converter matte	Furnace slag Converter matte	Furnace slag Converter matte
Stack height (m)	183	92	120	200
Sulphuric acid plant	Yes	Yes	No	No
Smelting	Rustenburg & Union	Rustenburg	Marikana	Northam
Base-metal refining	Rustenburg	Springs	Marikana	Northam
Precious-metal refining	Rustenburg	Springs	Brakpan	Heraeus (Germany)

Note: Amplats quote PGMs as 4E (Pt, Pd, Rh, Au); Impala and Lonmin use 5PGE+Au (excluding Os); PGE, platinum-group elements.

*Annual production figures reported for Impala Platinum exclude toll treatment.

**Northam has announced plans to increase UG2 production in 1999.

Impala Platinum

Impala has four rectangular six-in-line submerged-arc furnaces, of which only the two largest are in use. The furnaces are served by three Niro spray driers. Four Peirce-Smith converters are available, of which only two operate at any given time.

A chronology of the furnace and drier installations is provided below:²⁷

• 1969: the 5 MW turbo-tray drier and the first 7.5 MVA furnace

were commissioned.

- 1972: the first 7 MW spray drier and the second 7.5 MVA furnace were commissioned.
- 1973: the second 7 MW spray drier and the third 7.5 MVA furnace were commissioned.
- 1974: the first 14 MW spray drier and the 15 MVA furnace were commissioned.
- 1986: 'dry feeding' of the furnaces began.

- 1988: the second 14 MW spray drier was commissioned and the 5 MW turbo-tray drier was de-commissioned.
- 1991: the 28 MW spray drier and 39 MVA furnace were commissioned.
- 1996: both 7 MW spray driers and the Number 1 7.5 MVA furnace were de-commissioned.

The fine particle size of the concentrate presented serious problems in the drying process. 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 these problems. Eventually, 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 12–15 %. Using burnt lime in place of limestone increased smelting capacity by about 5 %.

Details on of the furnaces have been provided elsewhere,²⁷ and only those pertaining to the largest will be provided here. The 39 MVA furnace is 25.9 m long and 8.2 m wide, and has electrodes 1.14 m in diameter, spaced 3.32 m apart (centre to centre). The phase current is 21 kA, and the phase voltage is 500 V.

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 Cr₂O₃ control in the concentrator. The low grade of concentrate smelted, to maximize PGM recovery over the concentrator, has a high SiO₂ 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-up, and the Cr₂O₃ content of the concentrates is carefully controlled. The Merensky concentrate has a Cr₂O₃ content of less than 0.5 %, and the UG2 concentrate runs at about 1.6–1.7 % Cr₂O₃. This results in an overall Cr₂O₃ content of less than 1 % in the blended concentrate.²⁸ Impala was the first producer to experiment with UG2 on a plant scale as early as 1971.²⁷

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 feedstock for Impala Refineries. Impala's Base Metals Refinery uses Sherritt Gordon ammonia leach technology.

All waste gases, from driers, furnaces and converters, are treated in a Lurgi radial gas scrubber (installed in February 1999) prior to disposal. The single-contact Lurgi-designed sulphuric acid plant is one of the few in the world running on converter gas alone. Control is such that some furnace gas (at about 1 % SO₂) can also be treated.

Almost the entire plant throughput of concentrate is processed through the 39 MVA furnace. The 15 MVA furnace is used primarily for the toll-treatment of a variety of materials.

Lonmin Platinum

Operations at the Western Platinum Smelter began 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.²⁹ Western Platinum was the first mine to commission separate facilities for treating UG2 ore for the recovery of PGMs and associated base metals.³⁰ The Merensky six-in-line furnace has subsequently been upgraded to 10 MVA.³¹

The Merensky concentrate (received as a slurry) is filtered in a rotary drum filter and partially dried through a rotary kiln (to a moisture content of about 8 %) before being fed into the six-in-line furnace. The green charge and limestone flux are manually rabbled inside the furnace. The furnace matte is tapped periodically, while slag is tapped almost continually and granulated in a high-flow water stream. Converter slag is returned to the Merensky furnace to recover entrained matte.

The UG2 concentrates, containing relatively high concentrations of chromite, are dried in spray driers. The bone-dry concentrate is then pneumatically conveyed to one of several circular three-electrode submerged-arc a.c. electric furnaces. Burnt lime is used as a flux.

Separate smelting plants were erected for treating UG2 concentrate, but UG2 furnace matte is combined with Merensky furnace matte for converter operation.

Western Platinum was the first company to exploit the UG2 on a large scale for its PGM content.¹⁷ 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 75 % of Lonmin Platinum's current annual production is sourced from the UG2. Depth of UG2 mining at Lonmin ranges from 30 m to 700 m below the surface.

UG2 concentrate is smelted in a circular three-electrode furnace²⁰ with a higher power flux than is used in the Merensky furnace. A higher than usual smelting temperature is used, and the smelting zone is more concentrated, so that the slag becomes agitated. The agitation 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 80–90 % of the chromite present is discarded in the furnace slag. Furnace matte with a chromium content of 2 % could be blown to converter matte containing less than 40 ppm of chromium, which is acceptable to the base-metal refinery.

All the converter matte is processed at the refinery, using Sherritt technology from Canada, to produce nickel sulphate crystals, pure copper cathodes, and a high-grade PGM concentrate. The capacity of the refinery was expanded in 1991 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 base-metal refinery and reported to the PGM concentrate. This concentrate currently has a grade of about 60 %.³⁰

Northam

Northam operates the world's deepest platinum mine, at a depth of 1750 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 metal production in 1993.

Northam uses a conventional smelting process. Merensky concentrate (together with up to 10 % UG2 concentrate) is 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 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.³²

In the first leaching stage, nickel is removed as 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.

Limitations of the conventional smelting process

1. Environmental concerns have focused on the problem of SO₂ emissions, especially the stray emissions around the mouth of the converter. Even with a large fume hood above the converter, fugitive emissions remain a problem. A sulphuric acid plant is probably the most effective means of capturing the sulphur. However, the intermittent, rather than continuous nature of converting operations is a handicap.
2. As increasing amounts of UG2 concentrate are processed (to exploit deposits accessible from existing mines, and to maximize production of palladium and 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 valuable metals. This limits 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.
3. The UG2 concentrates contain significant quantities of chromite, which easily results in the buildup of (highly refractory) chromite spinel layers in the furnace. This affects furnace operation, and the accumulation reduces the working volume of the furnace. 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.
4. The intermittent batch mode of converting is not conducive to good plant operation, and moves are being made to develop a continuous converting process.
5. Although most current smelters and refineries have PGM recoveries in the region of 95–99 % 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.
6. The long processing times in the refining of PGMs result in a large retention 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 reduce the length of the processing pipeline in the refinery. This should be taken into account when investigating new processes.

Conclusions

South Africa dominates world production of platinum. Because of the high value of PGM products, a very risk-averse, conservative approach has been adopted to the introduction of changes in processing technology, and PGM matte smelting remains closely based on traditional nickel-copper matte conversion. 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 available

ore-bodies. Clearly, large-scale pilot testing will be required for new processes that are currently under development, with a view to addressing the limitations of conventional processing.

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