

Platinum in South Africa (Metal Review Series no. 3)

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SYNOPSIS

The paper gives a technical review of the history, occurrence, mining, metallurgy, and end-uses of the platinum-group metals in South Africa, with most emphasis on platinum and palladium.

The platinum industry has grown, from the first discovery of the metal in South Africa (in the central Transvaal) in 1923, to the present time, when the country is the largest producer of platinum in the world, supplying almost 70 per cent of the platinum on the world market.

SAMEVATTING

Die referaat gee 'n tegniese oorsig oor die geskiedenis, voorkoms, ontginning, metallurgie en eindgebruike van die platinumgroepmetale in Suid-Afrika, met die klem veral op platinum en palladium.

Die platinumbedryf het sedert die eerste ontdekking van die metaal in Suid-Afrika (in Sentral-Transvaal) in 1923 gegroei tot waar die land vandag die grootste produsent van platinum in die wêreld is en byna 70 persent van die platinum op die wêreldmark voorsien.

HISTORY AND GEOLOGY¹⁻⁵

Discovery of Platinum in South Africa

In 1923 Mr Adolf Erasmus discovered an unusual platinum deposit near Naboomspruit, in central Transvaal. Quartz veins infilling brecciated fault zones within the felsites of the upper phase of the Bushveld Complex yielded significant platinum values, and these were exploited on a limited scale from 1924 to 1926.

While in themselves short-lived as commercial enterprises, these discoveries aroused fresh interest in platinum prospecting in the Transvaal, and in 1924 Mr Andries Lombaard obtained platinum grains in a panned concentrate from a stream bed on the farm Maandagshoek in the Lydenburg district. Mr Lombaard brought this discovery to the attention of Dr Hans Merensky, who devised a prospecting programme to establish the source of the platinum grains. Unfortunately, this proved to be a pair of small pipe-like bodies of limited areal extent, which were relatively rich but insignificant in terms of ore reserves.

Undeterred, Merensky continued the search, and in the latter part of 1924 he discovered a layered gently dipping platiniferous pyroxenite on the farm Maandagshoek. Far from being a small pipe-like body of negligible extent, this appeared to be a continuous horizon, traceable for kilometre after kilometre on strike. Although he did not realize it at the time, Dr Merensky had found the world's largest known repository of the platinum-group elements (PGE).

History of Platinum

It is unlikely that platinum or the PGM were recognized as separate elements by the early civilizations, although

platinum has occasionally been found in artifacts from ancient Egypt. The best known example is a strip of native platinum set in a box that is ornamented with gold on one side and silver on the other. One of the metallic inlays on the side ornamented with silver was established by French scientists in 1900 to be platinum. There was no evidence that the Egyptian craftsmen had treated this piece any differently from the rest, and its occurrence was probably quite fortuitous.

The earliest probable reference to the PGE in European literature was made in 1557 by a gentleman of the unlikely name of Julius Caesar Scaliger. At the time he was engaged in vigorous scientific disagreement with a fellow Italian, who, among other things to which Scaliger took exception, had defined metal as 'a substance that can be melted and which hardens on cooling'. Having set his opponent reeling with mercury, Scaliger delivered the *coup de grace* as follows: 'Metal you say is something that can be melted but when it cools remains hard. . . Moreover, I know that in Honduras, a district between Mexico and Darien, there are mines containing a substance which it has not hitherto been possible to melt by fire or by any of the Spanish arts. Thus we see that the word *melt* cannot be applied to all metals'. This spirited exchange was followed by a whole series of references from South America over many years to a white or grey metal that was difficult to melt. It was referred to variously as *papas de plata*, *smiris Hispanica*, or *platina*.

Finally, in 1879 a German geologist, Theodor Wolf, published a paper describing his findings in the district of Esmeraldas in north-western Ecuador. He had unearthed from local burial mounds a tiny ingot of platinum, which he analysed. It contained 84,95 per cent platinum, 4,6 per cent palladium, rhodium, and iridium, 6,94 per cent iron, and around 3,5 per cent copper. Later work suggested that the inhabitants of the Esmeraldas district had mastered the techniques of working with platinum long before the Spanish invasion.

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It is ironic that one of the first recorded uses of platinum was thoroughly fraudulent. Unscrupulous individuals used to alloy it with gold as witness the following report by Mr William Brownrigg in 1750: 'I am told that one Mr. Ord, formerly a Factor to the South Sea Company, took in payment from some Spaniards Gold to the value of five hundred pounds sterling which being mixed with Platina was so brittle that he could not dispose of it, neither could he get it refined in London, so that it was quite useless to him'.

Rustenburg. Various small companies were formed to exploit the deposit, including Premier Rustenburg Platinum Ltd, Eerstegeluk Platinum Mines Ltd, and the Rustenburg Platinum Company. These were taken over by the Johannesburg Consolidated Investment Co. Ltd (J.C.I.) in 1926, and just three years later a J.C.I.-administered company, Potgietersrust Platinum Ltd, started production from the Merensky Reef.

Production has continued ever since, and today three large-scale mining concerns are engaged in winning platinum from the Bushveld Complex. These are J.C.I.'s Rustenburg Platinum Mines Ltd, Gencor's Impala Platinum Ltd, and Lonrho's Western Platinum Ltd.

By 1929, the Merensky Reef had been traced on the eastern limb of the Bushveld Complex for a distance of 150 km, and in the western limb for about 200 km (Fig. 1), thus making it by far the largest platinum deposit in the world. But yet another surprise was in store.

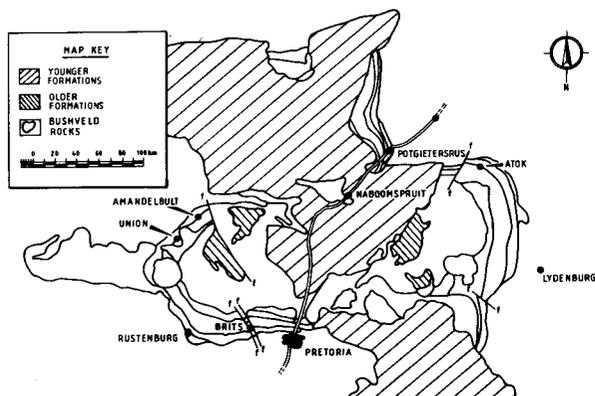


Fig. 1—Geology and mines of the Bushveld Complex

As far back as 1908, Messrs A.L. Hall and W.A. Humphrey had published a report in which they noted that certain chromitite layers within the Bushveld in the

Rustenburg area and in Sekhukhuneland assayed up to 2,6 g of platinum per ton. Chromitite layers continued to yield sporadic platinum values for many years, and it was only upon systematic prospecting within the lower portion of the Bushveld sequence following the discovery of the Merensky Reef that there was located in the Rustenburg and Lydenburg districts a particular chromitite layer that carried persistent PGE values of between 3,5 and 29 g/t. This chromitite layer lay anywhere between 20 and 370 m below the Merensky Reef, and its discovery, obscured the true potential of the area for many years. It is now clear that substantial platinum deposits, which are unusually enriched in copper and nickel, are to be found here.

Geological Setting of the Merensky and U.G.2 Reefs

Both the Merensky Reef and the U.G.2 chromitite layer occur as distinct, persistent layers within the gigantic layered intrusive known as the Bushveld Complex, which underlies an area of about 40 000 km² in the central Transvaal.

Some two thousand million years ago, a major magmatic event occurred, which resulted in vast volumes of molten rock from within the Earth's mantle being injected into the higher-level crustal rocks of the Transvaal Supergroup. This injection probably took the form of a series of pulses that introduced successive units of magma, each of which differed subtly from that preceding it. Some differentiation of individual layers may also have taken place as different minerals crystallized out at different temperatures, and the net result of these processes was that the Bushveld Complex became clearly stratified into distinct compositional units (Fig. 2).

In general, there is a transition from high-temperature rock types that are rich in magnesium in the lower portion of the Bushveld Complex, to low-temperature silica- and iron-rich rocks at the top.

In terms of the PGE, the most important unit is the Critical Zone (Fig. 3), which consists in its lower part of a series of chromitite layers within pyroxenite, and in its upper part of a series of cyclically repetitive triplets of chromitite, pyroxenite, and norite. The Critical Zone is divided rather unimaginatively according to the chromitite layers into the Lower Group, Middle Group, and Upper Group. The basal chromitite of each unit is identified by being numbered according to its group from the bottom upwards, e.g. L.G.1, L.G.2 etc., for Lower Group chromitites, M.G.1, M.G.2 for Middle Group,

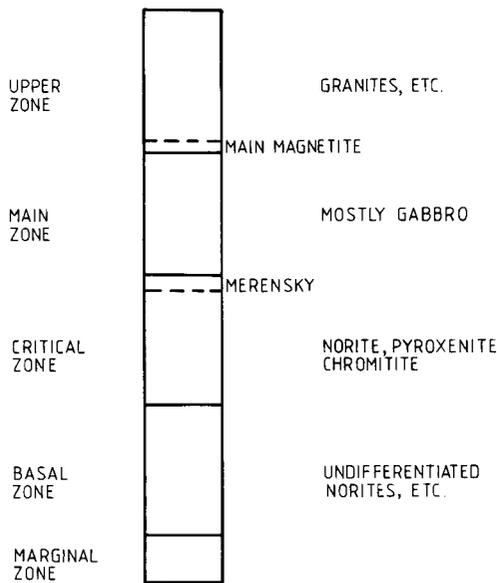


Fig. 2—Generalized stratigraphy of the Bushveld Complex (after Willemse and Von Gruenewalt, 1969)

and so on (Fig. 3). As can be seen in Fig. 3, there are no fewer than thirteen of these chromitite units in the Critical Zone in the Rustenburg area. Of all these chromitites, only the U.G.2 carries significant platinum values throughout the Bushveld Complex.

Near the top of the Critical Zone is an assemblage that could be described as the penultimate happening of the Critical Zone event. It looks like an abortive attempt to form yet another chromitite-pyroxenite-norite triplet, for all the essential components are there. The chromitite, however, is extremely poorly developed. The pyroxenite varies in width from around 60 cm to a maximum of about 3 m. This relatively insignificant assemblage is the Merensky Reef.

A few metres above the Merensky Reef occurs the final chromitite-pyroxenite. This is the Bastard Reef, and its

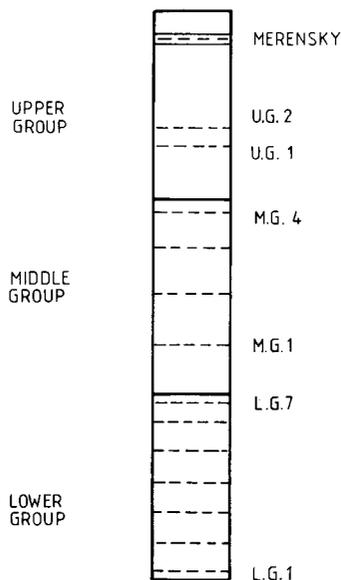


Fig. 3—Stratigraphy of the Critical Zone

chromitite is so poorly developed as to be absent in many places. It is not significantly platiniferous. Above the Bastard Reef, the norites of the Critical Zone give way to the thick, monotonous gabbros of the Main Zone, which appear to represent a new and compositionally different infusion of magma into the Bushveld Complex.

The rocks of the Critical Zone crop out around the margins of the Bushveld Complex (Fig. 1) and dip relatively gently towards its centre. This meant that mining operations could be initiated at small cost from surface, and it is only now after half a century of production that some mines are approaching reasonable depth. This generally shallow dip is important, because the thermal gradient within the Bushveld rocks, which may well be linked to the Earth's mantle through feeder channels, is almost three times as steep as that in the Witwatersrand gold mines.

Merensky Reef

Substantial mining operations on the Merensky Reef began in the Rustenburg area, and for this reason the Rustenburg area has always been considered to be the 'type locality' of the orebody.

The Merensky Reef at Rustenburg normally consists of a coarse-grained 'pegmatoidal' feldspathic pyroxenite containing appreciable phlogopitic mica. It is really very narrow, varying in width from 15 to 40 cm. The top and bottom contacts of the coarse-grained material are usually marked by thin chromitite layers a few millimetres thick. Immediately above the top chromitite layer is a brownish pyroxenite that is around 60 cm thick. This is known as the 'Merensky pyroxenite'. Directly below the bottom chromitite is a light-coloured norite. PGM occur within this assemblage in two different forms.

A relatively small proportion of the PGM forms alloys with iron. These ferroplatinum alloys are very dense, and this is used to advantage in their recovery.

By far the greater part of the PGE associate as complex sulphides within base-metal sulphides. Three principal base-metal sulphides occur in the Merensky Reef. These are, in order of decreasing abundance, pyrrhotite (FeS), pentlandite (Fe,Ni)₃S, and chalcopyrite (CuFeS₂). As the platinum minerals occur within and associated with these sulphides, the Merensky Reef yields substantial copper and nickel as byproducts, together with minor amounts of cobalt and selenium. The lack of these byproducts makes the U.G.2 a somewhat less attractive proposition, the unit value of its ore being usually less than that of the Merensky Reef.

All six of the PGE and gold are found within the Merensky horizon. Their proportional percentages are approximately as follows:

Pt	57%	Ru	8%
Pd	25%	Os	1%
Au	4%	Rh	4%
Ir	1%		

The high ratio of platinum to palladium substantially increases the unit value of Merensky ore.

The base-metal sulphides and their associated PGE are not restricted to the Merensky Reef itself but are disseminated for some way into the hangingwall pyroxenite and the footwall norite, thus facilitating sto-

ping widths of 70 to 90 cm. This is of considerable metallurgical significance in that a concentrator may have to process a mixture of three different rock types, each with its own grindability and recovery characteristics.

It should be remembered, however, that the Merensky horizon varies considerably throughout the Bushveld Complex and, for that matter, within the Rustenburg area itself. In the eastern and southern Bushveld, the Merensky pyroxenite is strongly developed, and much of the mineralization is located within it. The pegmatoidal phase may be absent, or if it is present it may be virtually barren. In the western and south-western Bushveld, the pegmatoidal phase is strongly developed, as are the chromitite contact layers.

The nature of the Merensky Reef has a direct bearing upon the distribution of sulphides, and hence PGE, within the one horizon. Three forms of the Merensky Reef within the Rustenburg area and their PGE distributions are illustrated in Fig. 4. It is clear from the diagrams that the highest PGE values occur near or associated with the chromitite seams, and that the distribution of the PGE determines the composition of the ore that is sent for milling.

Structure of the Merensky Reef

Throughout the Bushveld Complex, the Merensky Reef is affected to a varying degree by the phenomenon known

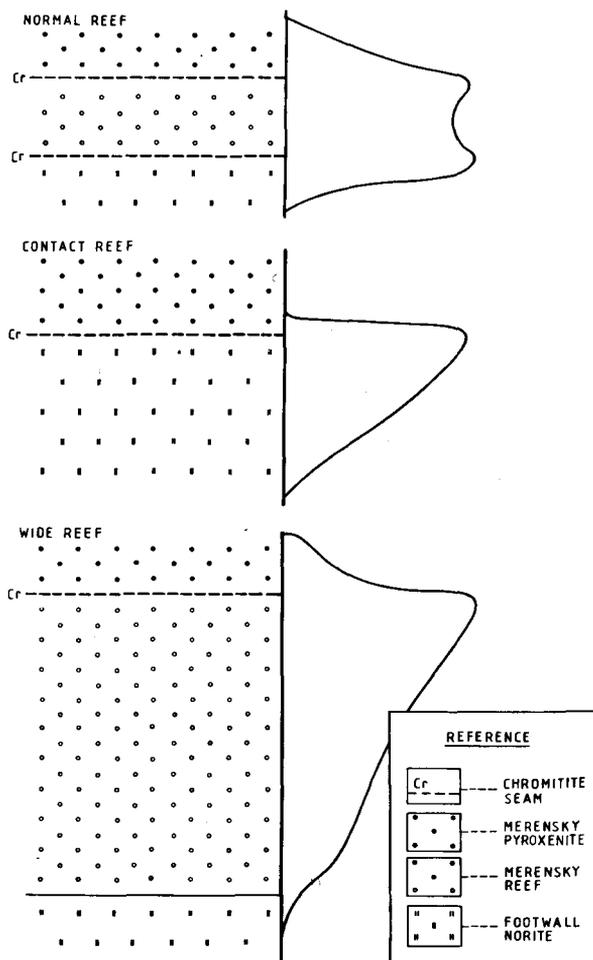


Fig. 4—Distribution of PGE in various types of Merensky Reef

as 'potholing'. Potholing seems to be a characteristic of large layered complexes, and within the Bushveld Complex the Merensky appears to be a particularly vulnerable horizon when compared with other economic layers such as the U.G.2 and the L.G.6.

In brief, potholing occurs when the Merensky Reef (or other horizon) suddenly slumps, cross-cutting its own footwall (Fig. 5). The hangingwall rocks descend concordantly with the potholing stratum. The size and amplitude of these structures vary from a few metres across and a metre or so deep, to severe disturbances that may be 300 m across and up to 90 m deep. They tend to be approximately circular in plan. Their cause is not clear, but they seem to have a strong association with late-stage volatile activity.

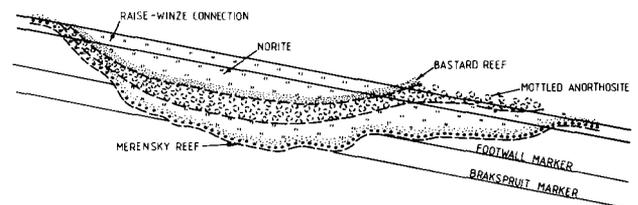


Fig. 5—Section through a typical pothole

In many cases, pothole reef is unmineable since it is often highly disturbed, and sometimes even absent save for a thin, virtually barren pyroxenite. The frequency and severity of these disturbances may govern the choice of mining method in any given area.

Faulting occurs on the reef horizon, but this is generally on a relatively small scale. These faults rarely throw more than a few metres, and their principal effect on mining operations is to weaken the hangingwall substantially. Most are relatively low-angle reverse faults, which may be considerably serpentinized. Particular care must be exercised in support design under these fault planes.

Major dykes are an important structural factor in the western Bushveld. While most of the dykes are doleritic, they are accompanied in the Rustenburg area by large syenite dykes associated with the Pilanesberg intrusive event. The dykes tend to cluster into groups of four or five within a relatively confined zone, and may follow pre-existing lines of weakness. Two major trends are apparent in the Rustenburg area, namely northwest-southeast and east-west.

These dykes have a significant effect on mining in that they induce a horizontal stress component that is not normally present beyond that induced by shallow mining operations. Severe shearing is commonly present on the dyke margins, and water is occasionally intersected in the vicinity of the dykes.

MINING METHODS

In addition to the publications referred to directly in this section, there are several others of interest. These are given as numbers 6 to 10 in the list of references.

Mining methods at present are restricted to underground operations, although the Potgietersrus limb, Platreef, could lend itself to future open-pit mining. Any exploitation system must be aimed essentially at the removal of a narrow sheet of platinum-rich rock contain-

ed within the mass of the Bushveld Complex.

Where the Merensky Reef outcrops on surface, such as at the Amandelbult Section, an inclined-shaft system is ideally suited to the exploitation of the tabular, gently dipping (at approximately 10 degrees) orebody. At intervals of about 1 km along the reef outcrop, inclined shafts are sunk on or below the reef in order to gain access to the first 1200 m of ore. Since inclined shafts become uneconomic to operate beyond a certain depth, vertical shafts are sunk at specific locations to take over from these inclined shafts when the ore so exposed has been mined out (Fig. 6).

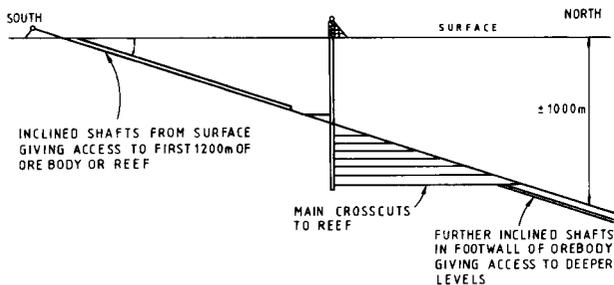


Fig. 6—Schematic section through a typical shaft at Rustenburg Platinum Mines, showing the major development

Access to the reef plane from these vertical shafts is by means of a series of horizontal tunnels or 'main crosscuts', which are the main access for men, materials, services, ventilating air, and the transport of broken rock.

The main crosscuts are driven to a point below the reef plane where the competency of the rock is favourably suited for the siting of footwall haulages, approximately 25 m below the reef horizon, which require long lifespans; for example, in the Rustenburg area the Boulder Bed horizon is the most suitable. Footwall haulages are horizontal tunnels that are parallel to the strike of the reef. Should distances from the shaft to the reef become excessive, only every second or third level has a main crosscut, and a system of interlevels closer to the plane of the reef is used with access to these interlevels via short material inclines.

At intervals of approximately 150 m along the footwall haulages, crosscuts are driven to intersect the reef. From this point, inclined tunnels on the reef plane, known as raises and winzes, are driven on dip. Boxholes, or ore-passes, are mined at an inclination of 60 degrees upwards from the crosscuts to intersect the raise above at intervals determined by the stope layout, viz panel length. From these raises and winzes, the orebody is mined by an operation known as stoping.

Four distinct underground mining methods are in use on South African platinum mines^{1,11,12}. The methods are termed the scattered mining method (mechanized and Bilby methods), and longwall mining method, and the up-dip mining method.

Scattered Mining Methods

Mechanized Method

The scattered mining method (Fig. 7) is used where geological disturbances (e.g. potholes) prohibit the mining of extensive areas. From the original raise, which

becomes the centre gully of the stope, a series of panels is advanced in the direction of strike. Advance strike gullies or A.S.G.s form the top and bottom limit of each panel. A.S.G.s are always kept slightly ahead of the advancing panels to establish a free face for blasting and for purposes of exploration, and to facilitate the removal of ore.

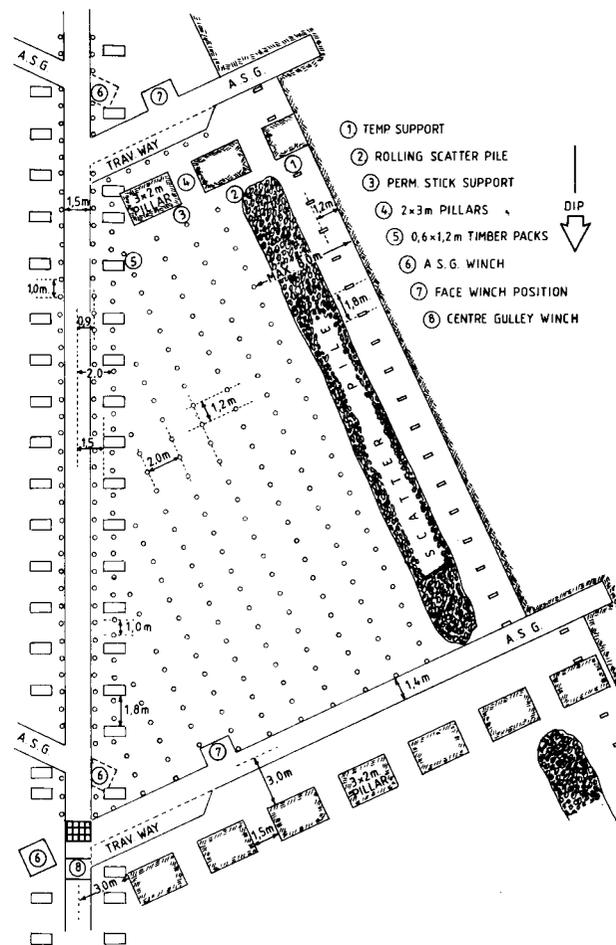


Fig. 7—The scattered mining method

The broken ore is removed from the panel by means of a system of scrapers (hence the term mechanized)¹³. A face scraper is rigged by means of a series of snatch-blocks and steel ropes to run along the face of each panel (Fig. 8). A 22 kW double-drum electric winch sited within 30 m of the panel is used to pull the scraper. The ore is scraped down-dip into the A.S.G. (The night-shift personnel remove ore from the whole length of the face of the panel for 2 m perpendicular to the face, to the start of the scatterpile (Fig. 7). The day-shift personnel then remove ore from behind the scatterpile to the standard thickness required (hence the term *rolling scatterpile*). The purpose of the scatterpile is to eliminate widespread scattering of the ore from the blast and to confine the broken ore within reasonable distance of the face (since the scatterpile extends from footwall to hangingwall). Blasting barricades may similarly be used to confine the scattering of the ore.

The ore pulled into the A.S.G.s is scraped into the centre gully by use of a second (37 kW) winch (Fig. 8). Finally, the 55 kW centre-gully winch scrapes the ore into 'box-

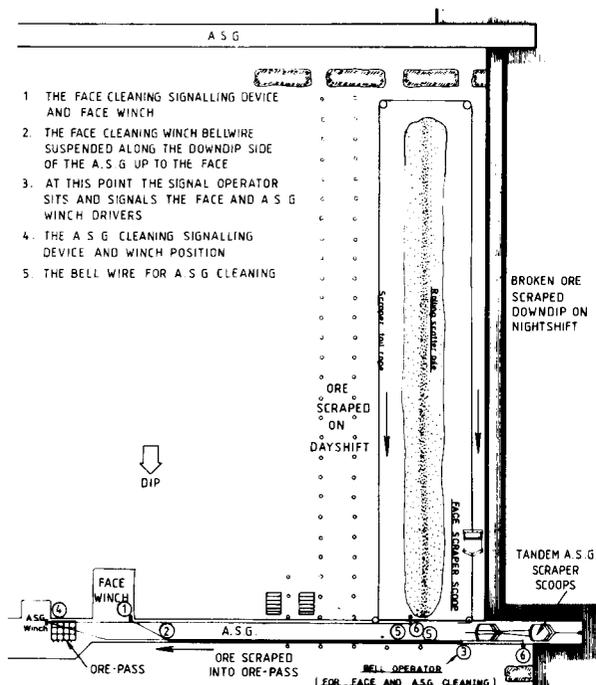


Fig. 8—The arrangement for panel scraping

holes' or ore-passes fitted with a screen or grizzly (the size of which is determined by the size of the desired crusher feed), leading down to the crosscuts below (Fig. 9). The ore is then loaded into hoppers and pulled by small electric or diesel locomotives to conveyor transfer points or to the main shaft itself, where it is hoisted to surface. On surface it is tipped into surge bins, from where it is transported to the mill (the route of the ore from stope to mill being the same for all four methods).

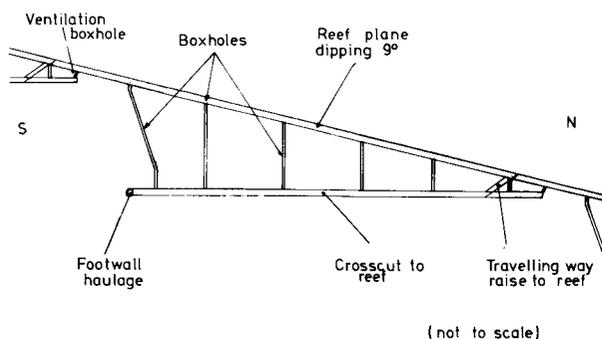


Fig. 9—Section showing the layout of boxholes or ore-passes along a cross-cut

The support system generally used for scattered mining is the stick-and-pillar method¹⁴. A series of small pillars is left *in situ* on the plane of the reef as the panel is blasted (Fig. 8). These pillars are sited at the top of each panel. Support of the worked-out area is by means of wooden timber props 150 mm in diameter. A row of props is installed along dip as the panel is advanced on strike and kept within a maximum standard distance from the panel face, e.g. 8 m (Fig. 10). Face props are installed as temporary support along the face of the panel for the protection of the workers responsible for drilling and blasting. Mat packs are installed along both sides of the

A.S.G.s and centre gully, which are important access routes.

Bilby Mining Method

The Bilby method is similar to the mechanized scattered-mining method but differs mainly in the method of removal of the broken ore. Hand lashing is employed, with two lashing persons per cutting (Fig. 11). The ore is lashed into Bilby cars operating on monorail strike tracks along which the Bilby cars (Fig. 12) are pushed towards the centre gully. At this discharge point, stop blocks allow easy tipping of the car into the centre gully, where a scraper is used to pull the ore into the boxhole. The Bilby method has the inherent flexibility that the A.S.G.s need not be straight, as is necessary for scraper gullies. Hence, in geologically disturbed areas, panels can be maintained on reef (although the panel length changes) by simple swinging of the A.S.G. along a local strike.

Support may take on various patterns, and a popular system is shown in Fig. 11.

A notable feature of the Bilby method is that it is very labour-intensive, utilizing approximately twice the number of persons as in mechanized stoping methods.

Longwall Mining Method

The longwall mining method is well suited to the exploitation of large, geologically undisturbed areas of reef. It requires numerous panels on dip (Fig. 13) advancing in unison along the direction of strike.

Again, the ore is removed by means of face and A.S.G. scrapers. A notable feature is that each A.S.G. has a box-hole, which is developed ahead of the stope along with the other footwall development. De-stressing of the latter development by initial overstoping is not necessary at present mining depths (1000 m, 30 MPa). Normal footwall norites have uniaxial compressive strengths of about 190 MPa. Boxholes are planned 100 m apart in the A.S.G.s, and the winches are moved forward shortly after holings have been effected and grizzlies installed. Face winches are always closer to the panel than A.S.G. winches.

The support system used differs from the scattered mining system in that cement-grout packs are used above and below the A.S.G.s and as dip ribs (Fig. 14). These packs consist of circular steel rings connected with polypropylene netting on the inside, in which is placed a polypropylene knitted bag. This is wedged against the hangingwall by means of four sticks and is then filled with grout that is pumped underground from a cementation plant on surface. Scatter piles may be used to contain the blast and avoid contamination of the worked-out back areas. Recently, however, blasting barricades have been introduced, especially in areas where the hangingwall conditions are bad (Fig. 15).

Up-dip Mining Method

The up-dip mining method is so called because the panels are advanced in an up-dip direction (Fig. 16). Very little cleaning of the blasted areas is needed. The blast of the more advanced panel is directed at the solid side of the dip gully (Fig. 16), and the blast of the lagging panels is directed at the scatterpile, which is left in place, resulting in most of the blasted rock being concentrated

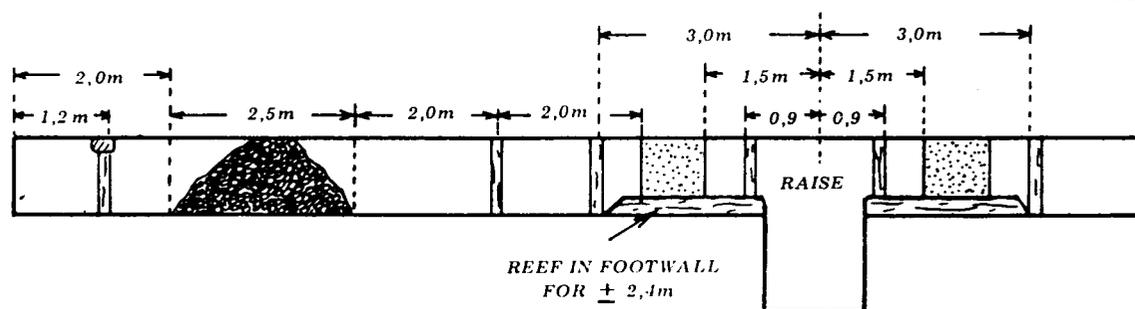


Fig. 10—Strike section showing the positions of the scatterpile and support

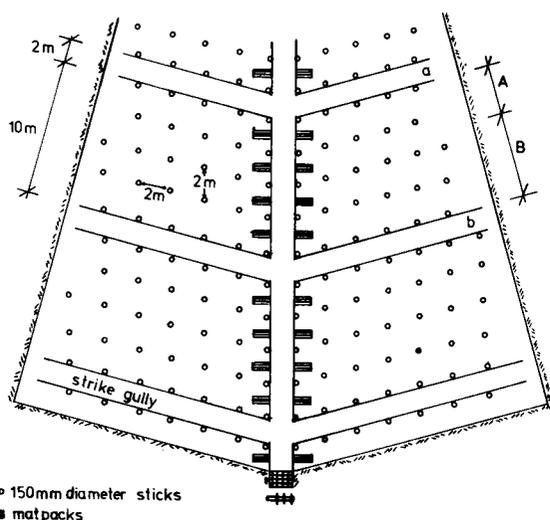


Fig. 11—Layout of a Bilby stope (person A lashes into gully a and person B into gully b)

in the dip gully. A dip-gully scraper then scrapes the ore down to boxholes.

The support system used may vary, with dip rib pillars being left at standard intervals along strike. Stick support is used in the worked-out area. When the panels have advanced to the level above, strike pillars are left for regional support.

Off-reef development is minimal, requiring only footwall drives or haulages, a boxhole for each dip gully, and an access raise. However, extensive on-reef development of reef drives and up-dip raises necessitates careful planning.

Drilling

Drilling is generally conducted with hand-held pneumatic jackhammers, with a crew of two persons per drill in the stopes, and one person per drill in development aided by an airleg.

Drilling patterns in the stopes and development ends are shown in Figs. 17 and 18 respectively. Hole directors are utilized to ensure parallel drilling of holes: the miner marks off the direction of the top and bottom holes (about 70 degrees to the dip) every 3 m; the machine assistant then places the hole director into the previous hole, and the subsequent hole will be drilled parallel to and at the correct burden and spacing to the previous hole. A.S.G.s are drilled to a simple burn-cut pattern. The length of drill steel used is 1,2 m. Stopping widths vary

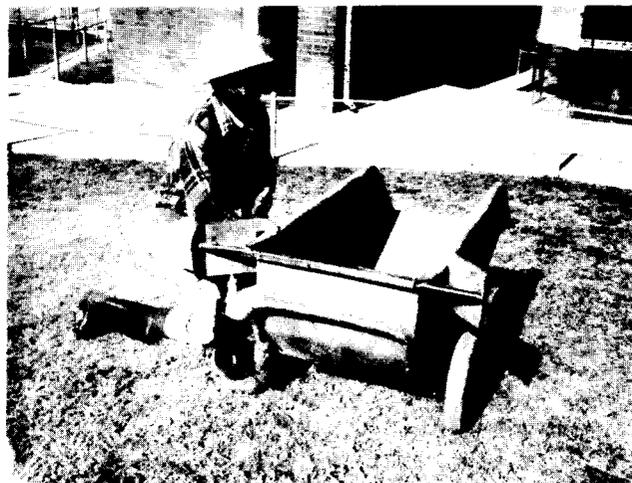


Fig. 12—Mounted underneath the Bilby car at the centre of gravity is a guide wheel, which runs on the monorail in the A.S.G.s

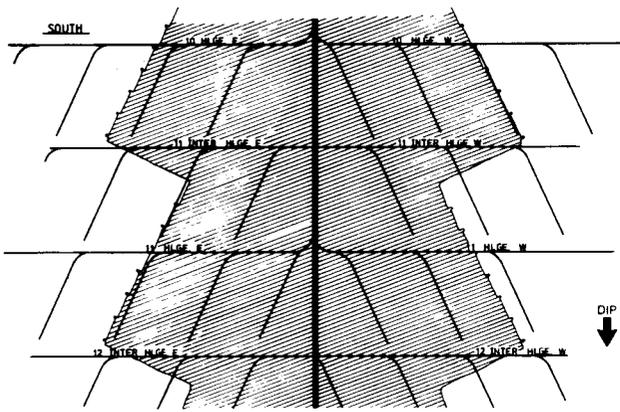


Fig. 13—Diagrammatic plan showing the longwall method of stoping

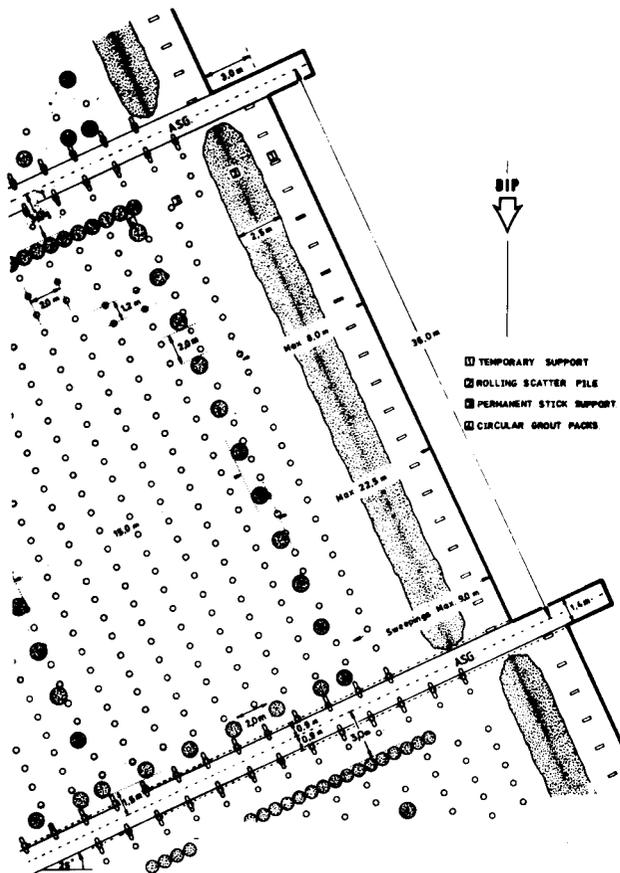


Fig. 14—Longwall support pattern for normal ground conditions

from 70 to 150 cm. Mechanization of drilling operations has been attempted¹⁵ but is not used at present.

Development ends (haulages and cross cuts) are excavated by the use of drag rounds or burn cuts. Advances of 2 to 3 m are gained per blast, depending on the length of drill steel used. Burn cut rounds are drilled for raises, winzes, and ore-passes.

Raise boring is used mainly on shaft sinking and boxholing¹⁶.

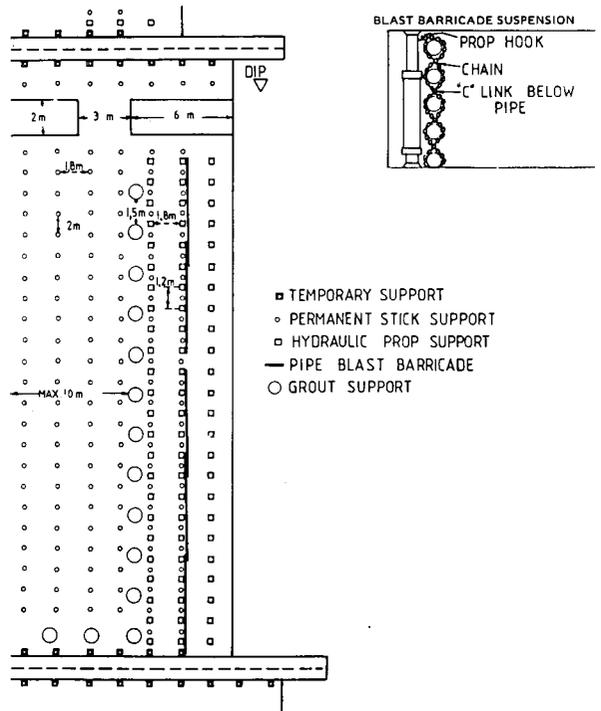


Fig. 15—Hydraulic props and blast barricade for friable ground

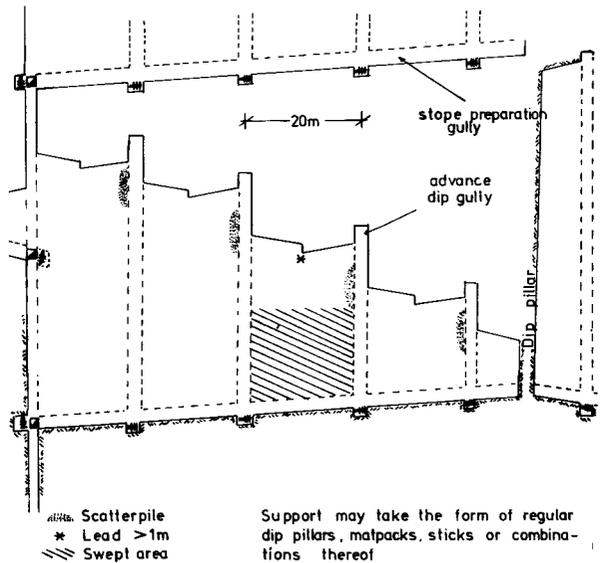


Fig. 16—Layout for up-dip stoping

Blasting

The explosive predominantly used is ammonium nitrate fuel explosive (Anfex). This is loaded into the blastholes by means of loaders operated by compressed air, giving a constant charge per stope blasthole of 0,7 kg and 1,5 kg per 2 m of development blasthole.

When water-resistant explosive is required, Dynagel 60, a stick explosive, is favoured in diameters of 22 mm and 25 mm. The latter is used for lifter and water-filled holes in development ends.

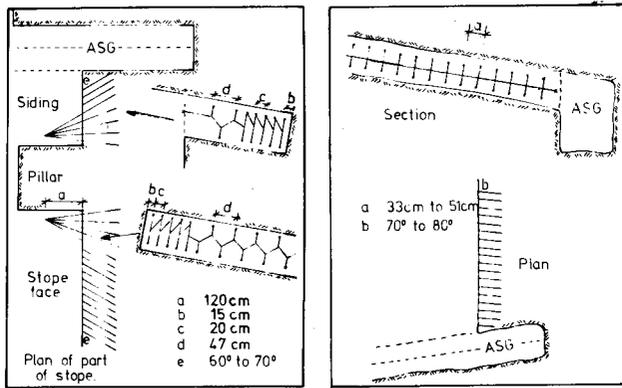


Fig. 17—Examples of the drilling patterns used in stoping

Capped fuses, 0,9 m in length with the MK1 connector 8D, are used to detonate Anflex without the use of primer charges, and MK1 6D fuses 1,8 m long are used for nitroglycerine-based explosives.

Slow (30 to 45 s/m) igniter cord is used for development owing to its high resistance to moisture, and IC 57 (13 to 26 s/m) or S.C.9 (8 to 16 s/m) is used for the sequential firing of stope blastholes.

Other explosives are used, but for more esoteric applications Western Platinum Ltd has manufactured its own Anfo mixture explosives¹⁷.

Cleaning

Stope cleaning has already been described in some detail, viz the use of scrapers. The subsequent cleaning is done by hand lashing, and the worked-out areas are then swept with hand brooms and sweeping tools, or are washed down as required.

Development cleaning is accomplished by means of pneumatic shovel loaders, which lash the broken rock directly into 6 t hoppers (i.e. 8 hoppers per 2 m blast). Tidying up is done by hand lashing.

Raises and winzes are cleaned simply by rigging of a scraper to the blasted end and scraping of the broken rock to a convenient boxhole.

Grade Control

The value distribution within the Merensky Reef is a positively skewed log-normal distribution with semi-variogram ranges of the order of 20 m in all directions. The low sample variances, compared with those for gold and base metals, and almost pure nugget-effect nature of the semi-variograms render kriging procedures inefficient.

Grade can be controlled most effectively by the keeping of ore dilution to a minimum. After the reef has been photographed¹⁸, and the top and bottom chromite seams, the stoping width, and the reef position have been identified along regular points in each panel, the results are averaged by a computer using digitizers. Printouts giving the best cut and stoping width for optimum grade recovery are then supplied to the stopers.

Sampling is carried out periodically, but is more a confirmation method than a grade-control method.

Ventilation

The high geothermal gradient of the Bushveld Com-

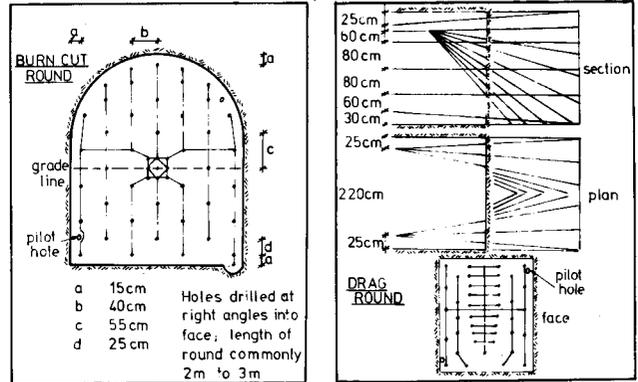


Fig. 18—Examples of the drilling patterns used in development

plex (21,2 °C/km) gives rise to a comparatively high temperature in the working environment. At depths of 1000 m, virgin-rock temperatures of 43°C can be expected, compared with 27°C and 31°C in the gold mines of the Witwatersrand and Orange Free State respectively. However, wet-bulb temperatures are lower owing to limited autocompression, and a wet-bulb temperature of 27,5°C is generally maintained. (The future planned mining depths of 1200 m will call for a consideration of refrigeration plants to reduce the temperatures of the working place from 45 to 26°C.)

A limited number of workers are put through heat-tolerance tests (HTT) at wet-bulb temperatures of 31,3°C for working in areas of higher temperature.

Ventilation requirements are based on a requirement of 3,6 m³/s per kiloton of ore broken per month.

Development ends are ventilated by the use of auxiliary fans, and dust suppression is accomplished with delayed-operation water blasts¹⁹.

Labour

Tens of thousands of men are employed throughout the platinum mines. The administration of this labour is extensive and costly, and involves continuous recruitment together with training, accommodation, catering, recreation, medical services, and an on-going labour-relations programme²⁰.

The South African platinum industry recruits its own labour force through a network of offices and employment representatives throughout South Africa and the independent States, in addition to Lesotho, Mozambique, and Botswana; whereas the gold mines use the recruiting organization of the Chamber of Mines of South Africa. The Black personnel represent over a dozen different tribes and speak a dozen different languages and dialects.

Preferential employment opportunities are afforded to those employees who have had previous service on the platinum mines. Most of the new recruits are unskilled and unsophisticated, with little or no education. Thus, the first education encountered by many is in the industry's employment. The *lingua franca* of the entire South African mining industry, Fanakalo, is included in the training programme, as well as first aid, loss control, safety, and simulated-job instruction.

Computerized labour control is in widespread use on South African platinum mines, because of the large number of personnel employed^{21,22}.

Short periods of employment with regular leave are encouraged with a view to stabilization of the labour force. All employees are covered by insurance in the event of injury or death, with free hospitalization, and compensation for loss of earnings and any permanent disability. Educational assistance for degree, diploma, and certificate purposes is available to all employees, who also are provided with single or married accommodation. A communications network based on tribal customs and traditions is maintained to ensure healthy and progressive labour relations.

Efficiencies

The efficiencies achieved by the platinum mines in South Africa compare very favourably with those on the gold mines and, although many reasons can be put forward, the platinum mines continue to return higher efficiencies in tabular reef mining. Table I gives an overall indication of the efficiencies attained in recent years.

TABLE I
EFFICIENCIES ON SOUTH AFRICAN PLATINUM MINES

Stoping	Long-wall	Scattered	Bilby	Up-dip
Area mined per underground worker, ca	35	35	15	35
Area mined per case of explosive, ca	7,5	7,0	8,0	8,0
Output per underground worker, t	90	90	50	90
Average stoping width, cm	75 for normal reef			
Monthly advance, m	15	15	15	15
Area mined per contractor, ca	2000	2700	2000	3100
<i>Development</i>				
Per underground worker, m	5,5			
Per contractor, m	300			
Per case of explosive, m	1,0			

ca = centare = 1m²

Rock Mechanics

Well-defined structural weaknesses divide the hangingwall into large fragmented blocks. Broken rock can have good structural properties, but these are lost when tensile stresses are present, which leads to bed separation. Thus, a necessary design criterion is that the full thickness of rock up to a plane of weakness within the tensile zone should be supported. As most platinum mines are shallow to moderately deep, the tensile zone extends far into the hangingwall, and this dictates the types of support to be used.

Traditionally, as in all the mines in South Africa, timber support was used in conjunction with waste stowing. This support system proved to be very labour-intensive, and when waste stowing was abandoned, only timber matpacks were used. However, owing to the increase in depth and span of the workings, timber packs were inadequate to cope with the support demands. Scattered mining also contributed to the elimination of matpacks as support, and timber sticks were introduced into the stoping environment. These support units were found to be adequate for face support, but were unable to fulfil the requirements of regional support²³. Small reef pillars were introduced in some mines, while other mines introduced barrier pillars to achieve the designed support.

With the small reef pillars²⁴, inelastic as well as elastic closure is allowed to take place, but detachment is prevented in the hangingwall, which in turn reduces the actual tensile zone. Barrier pillars, on the other hand, limit the span and thus prevent any form of inelastic closure, but often result in large quantities of strain energy being stored in them, which could lead to violent failures of these pillars. In most mines in the Bushveld Complex, the convergence in the mined-out area is generally very limited. Timber matpacks require substantial deformation before any support is generated, and to overcome this grout packs were introduced as internal support and are now used in longwall stopes. Today much work is being done with support systems utilizing rapid-yield hydraulic props, blast barricades, and profiled timber props to enhance productivity without sacrificing safe underground support.

METALLURGY OF PGM

As already mentioned, the occurrence of PGM in the ores of the Transvaal is most complex. Apart from the multitude of platinum-group minerals present, their association is also diverse. The three main base-metal minerals present are chalcopyrite, pentlandite, and pyrrhotite. The platinum-group minerals can occur as associations with any of these minerals, or as discrete particles within the gangue minerals. It has also been established that PGM can occur in solid solution within the lattice of base-metal minerals, e.g. palladium in pentlandite. Recent research work has also indicated that PGM metals can occur within the lattice of the gangue minerals, which, with present known extraction techniques, renders them non-recoverable. In addition to the above, in areas of the orebodies the base-metal minerals have been altered and so made more refractory.

The range of minerals present, their relative densities, shape, discrete particle size, and association, all therefore present a challenge to the metallurgist in optimizing the primary extraction process. This challenge is heightened by the fact that the deposits currently being exploited are not constant in mineralization, either along strike or down dip. Similar problems are encountered in the final separation and refining processes, where, owing to the similar chemical properties of the various PGM, sophisticated techniques have had to be developed to give the desired separation and levels of purity.

Currently, the extraction process can be divided into several different stages:

- (1) comminution of the ore and concentration by physical techniques such as gravity concentration and flotation,
- (2) pyrometallurgical concentration producing copper-nickel sulphide mattes,
- (3) hydrometallurgical separation of the base metals and concentration of the PGM product, and
- (4) final refining to produce the individual PGM.

There are potential processes that claim to eliminate the necessity for one or other of the earlier stages^{25,26}.

Primary Concentration

Comminution

Initially, mining was limited to the upper regions of

the orebody, i.e. the oxidized zone, and, after crushing and milling, the extraction process was limited to gravity-separation methods^{5,27}. It was only as mining progressed deeper into the sulphide zones and a viable treatment route for flotation concentrate was developed that flotation became an economic proposition.

The comminution processes used at present follow established practice. Crushing is normally carried out in two or three stages in either open or closed circuit. The small stoping width in the majority of the mines results in the run-of-mine ore having a much smaller size distribution than that in conventional base-metal mining. It is practice, in Rustenburg Platinum Mines, to remove both the primary floatable-size fines and those generated during the crushing process by wet screening ahead of the second crushing stage. This material is normally treated on an independent flotation circuit.

The grinding circuits vary in configuration. Earlier plant design favoured two-stage grinding, consisting of ball mills in open circuit followed by closed-circuit ball mills. Newer plants tend to be based on single-stage closed-circuit ball mills.

Impala Platinum Limited, owned by Gencor, which started operations in 1968, is the only known platinum venture to eliminate conventional crushing and utilize autogenous milling of run-of-mine ore. However, it is understood that, when market demands require higher throughput rates, the mills will be converted to a semi-autogenous mode by the addition of steel balls.

From the milling point of view, the ore from the Merensky Reef is extremely hard. Bond indices of more

than 20 kW · h/t have been monitored, which can be compared with indices in the region of 16 kW · h/t of normal gold ores.

The extreme range of mineral densities within the ore, from 2,7 for talc to between 16,0 and 17,0 for ferroplatinum, causes problems in the grinding and classification circuits. Good recovery requires that the ore should be ground to a size range adequate to ensure the liberation of the valuable minerals. However, the classification process, now mainly based on cyclones, does not differentiate between small particles of high-density minerals and larger particles of lower-density minerals. It is therefore essential for the metallurgist to control the circuit to ensure adequate liberation for the subsequent flotation stage while avoiding overgrinding of the denser minerals.

To obviate this problem, corduroy tables were initially used to remove the denser floatable-size particles from the mill discharge prior to the classification stage. In some platinum mines, the corduroy tables have been replaced by Davcra cells, which are utilized as unit cells within the grinding circuit. Although their presence has increased the problem of grinding-circuit control, their contribution in preventing overgrinding and improving recovery has been proved conclusively. A typical comminution circuit is illustrated in Fig. 19.

Gravity and Flotation Concentrate

As mining progressed in depth, the proportion of oxide ore decreased and that of sulphide ore increased. The increase in sulphide ore reduced the proportion of PGM that could be recovered by gravity processes and increased

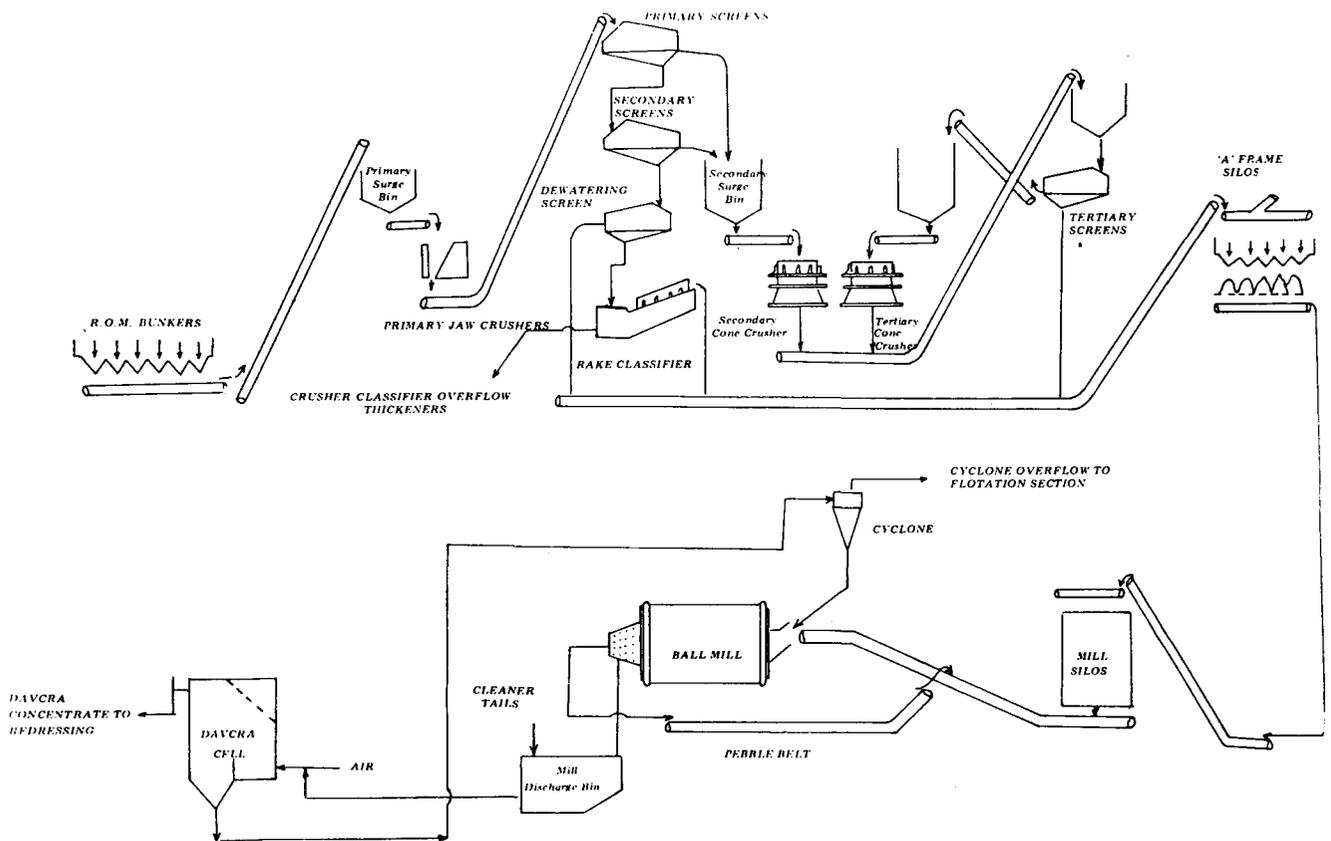


Fig. 19—A typical comminution circuit

that proportion recoverable by flotation. There has been continual controversy as to whether the removal of 'metallics', by gravity concentration is advantageous or whether flotation alone will give the desired recovery.

The term *metallics* is misleading in that the material referred to contains little metallic material. It consists of ferropatinum, braggite (Pt,Pd,Ni)S, cooperite (Pt,Pd)S, sperrylite (PtAs₂), laurite (RuS₂), and other denser minerals, base-metal sulphides, and chromite.

In practice most producers now rely on flotation alone for the primary recovery of the liberated minerals, although gravity separation is still utilized to separate the 'metallics' from the flotation concentrate produced. Corduroy material still remains the best material available for the collection of the fine minerals containing the PGM. Although there has been much experimentation on the development of substitutes that would reduce the labour-intensiveness of the process, none has so far given the collection efficiency or ratio of concentration that can be obtained from well-operated corduroy tables. The 'cords' concentrate is redressed on a series of James tables to produce a high-grade concentrate.

The advantages of the gravity-concentration process are as follows.

- (a) A high-grade concentrate is produced, reducing the cost of subsequent treatment.
- (b) The smelting stage is bypassed, eliminating the losses associated with that process.
- (c) Because the smelting stage is eliminated, the sales value of the metals is realized much earlier and the processing inventory is reduced.

The flotation circuits employed are designed essentially for the flotation of base-metal sulphides; PGM are recovered as a consequence. However, it should be noted that the base-metal head content of the ore is well below the tailings value of many conventional base-metal flotation circuits. It is only the presence of the PGM that makes the process economical.

The normal flotation circuits involve roughing, scavenging, and multistage cleaning to produce a concentrate of the desired grade. The required residence time depends on the mineralization of the ore. The base-metal minerals are essentially chalcopyrite, pentlandite, pyrrhotite, or alteration products of these. Pyrrhotite is notoriously slow-floating, and the residence times in the rougher and scavenger circuits therefore tend to be extended. Studies²⁸ have shown that the floatability of pyrrhotite greatly increases under acidic conditions, and this was substantiated both by tests carried out on ores from the Stillwater Complex²⁹ and by research work at Johnson Matthey on ores from the Merensky Reef. This procedure is not yet practised on an industrial scale, and flotation normally occurs at natural pH.

The flotation reagents normally consist of copper sulphate as an activator, xanthate or a xanthate-dithiophosphate mix as a promoter, a frother, and a gangue depressant. Until recently cresylic acid was the most common frother in use, either alone or with mixes of alcohol-base frothers. The current tendency is away from the use of cresylic acid to frothers that are easier to handle and of more consistent composition. Owing to the talcose nature of the ore, use is made of a gangue depressant, which is normally an organic colloid. Dex-

trin, natural and modified guar, and starch have all been used.

The concentrates produced, after the removal of a gravity concentrate if practised, are subjected to thickening and filtering prior to smelting. At Western Platinum Limited (Lonrho) and Impala Platinum, filtration is omitted because the concentrate is subjected to spray drying prior to smelting.

The flotation tailings are thickened before being impounded within tailings dams. It is common practice, certainly within the Rustenburg group, to cyclone the tailings prior to thickening, the cyclone underflow and the thickener underflow being recombined for disposal. This practice reduces the loading on the thickeners and improves the ease of operation. A typical flotation circuit is shown in Fig. 20.

As noted earlier, fines removed from the crushing circuit by wet screening and classification, where practised, are floated separately. This is done because their viscosity requires an entirely different flotation density for optimum recovery. Similarly, owing to the fineness of the material, the unit surface area and surface characteristics are different from those of normal flotation feed, and different rates of reagent addition are required.

Smelting

The decrease in ores amenable to gravity-concentration methods and the increased reliance on flotation for the recovery of sulphide materials in the early 1930s posed an additional problem to the metallurgist of how to enrich the flotation concentrate to render it suitable for refining.

The first process, the chlorination process, was developed by K.L. Graham and his associates in the Rand Mines Laboratories. It started with roasting of the sulphide concentrate to a dull-red heat to eliminate sulphur and oxidize the base metals. The roasted concentrate was then mixed with salt and placed in a chlorination furnace maintained at a temperature of between 500 and 600°C while chlorine was passed over the surface of the material. The PGM and base-metal sulphides were thus converted to water-soluble chlorides. The material was treated with acidified water and the enriched solution separated from the residue by filtration. The insoluble residue was cyanided for the recovery of gold. Copper was precipitated from the solution as a carbonate. After the removal of the precipitated copper, the PGM were precipitated as a black powder by agitation with zinc dust. The dried precipitate after roasting had a PGM content of 70 per cent, and was suitable for refining. A small plant using this process was constructed, but it proved dangerous to health and was shut down.

A paper¹ published in 1961 states 'other methods of treating the concentrate were under intensive investigation, but one of the most successful and which is now adopted was to smelt to copper nickel matte. Preliminary work was carried out on this method, notably by H.R. Adam at the Government Areas Laboratory, but it was Messrs. Johnson Matthey who first successfully used this method on a large scale'. However, Wagner⁵ in 1929 outlined processes developed by Krupp and Chemical and Metallurgical Corporation Limited, which appear to be very similar to the Johnson Matthey process.

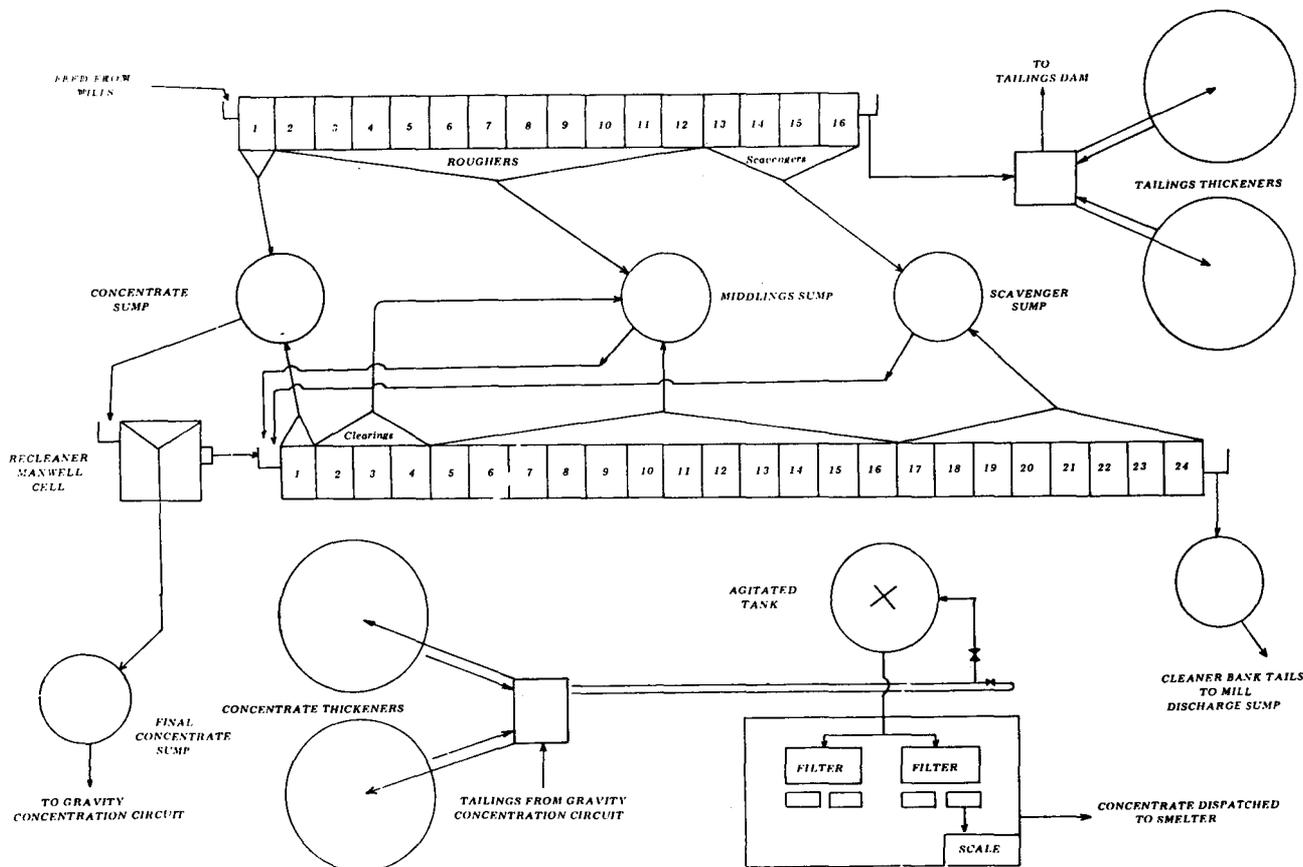


Fig. 20—Typical flotation and filtration circuits

Initially, the concentrates produced by Rustenburg Platinum Mines were smelted by Johnson Matthey at Brimsdown, U.K. In 1937, the first blast furnace and a small (2,4 m in diameter) Great Falls type of converter were commissioned at Klipfontein (Rustenburg). The smelter was expanded to four blast furnaces, 3,0 by 0,36 m at the tuyère line. The furnaces were water-jacketed and run with tapped spouts into a forehearth. Various methods of feed preparation were tried, including sintering on a small Dwight Lloyd machine. The method eventually adopted was to dry the concentrate in a Buttner Turbo dryer, and pelletize the resulting powder in Lurgi pelletizing discs. The green pellets were fed together with coke, limestone, and iron ore into the furnaces. The matte was tapped periodically and blown in one of three upright converters. The white metal was cast, crushed, and bagged for dispatch to the refineries. The converter slag was also cast and returned to the furnace. The blast-furnace slag was continuously tapped and granulated prior to disposal.

An increased market demand for platinum, coupled with lack of expansion room at their existing smelter, resulted in the decision by Rustenburg Platinum Mines in 1967 to construct a new smelter at their Waterval site. The then recently introduced anti-pollution laws, combined with the increasing cost of blast-furnace operations and the low cost of electric power, favoured an electric furnace. The furnace chosen was an Elkem 19,5 MVA rectangular submerged-arc furnace. It is of interest to note that, while Rustenburg Platinum was the first com-

pany in South Africa to decide to use such furnaces, Impala Platinum commissioned their smaller furnace in 1968, when that mine started operations.

The Rustenburg process has been described by Mostert and Roberts³⁰. The furnace has a rating of 19,5 MVA, with energy input by means of three pairs of electrodes rated at 6,5 MVA per pair. The electrodes are of the Söderberg type and are 1,25 m in diameter, each with a current density of 2,65 A/cm². The electrode distance centre to centre is 3,4 m between phase pairs and phases. The secondary voltage of the transformer can be varied between 170 and 350 V, with a maximum electrode current of 32 400 A.

The concentrate is dried in Buttner dryers. The dried concentrate (7 per cent H₂O) is fed to pelletizing pans, which are 3 m in diameter and produce pellets 15 mm in diameter with a moisture content of 10 per cent and a breaking strength of 5 to 6 kg. No binding agent is used. The pellets are dried to 2 per cent moisture in roto-louvre dryers, and the crushing strength increases to between 9 and 13 kg. The pellets and fluxes, mostly limestone and occasionally iron ore, are fed through a proportional weighing system. Molten converter slag is returned to the furnace through a launder at the matte-tapping end wall. Furnace slag is tapped continuously and granulated prior to treatment in a slag-grinding and flotation circuit. Matte is tapped at regular intervals and converted in Peirce-Smith type converters. The converter matte is cast into moulds, crushed to minus 25 mm, and bagged for dispatch to the refiners.

Electric-furnace submerged-arc smelting is also practised by Impala Platinum Limited and by Western Platinum. It is understood that the converter matte from all three smelters is of similar composition, assaying 1500 to 2000 g/t PGM, approximately 50 per cent nickel, 28 per cent copper, 1 to 2 per cent iron, and approximately 21 per cent sulphur. A typical smelter flowsheet is given in Fig. 21.

U.G.2 Reef

The U.G.2 Chromite seam has not been exploited to any great extent, mainly owing to economics and the difficulties in effecting a clean separation between the chromite, the base-metal sulphides, and the platinum-group minerals. Difficulties are also experienced during the smelting stage owing to the tendency of the chromite to 'freeze' in the slag and build up in the furnace.

Most platinum-producing companies have been researching the primary recovery and smelting processes to develop viable processes for the exploitation of this deposit. Understandably, the companies are reticent about publishing their results.

The National Institute for Metallurgy, now the Council for Mineral Technology (Mintek), has also carried out considerable research on these ores. They have shown that good recovery and separation of the PGM and chromite can be obtained by the use of a combination of gravity-separation and flotation methods, and good recovery of the PGM alone by flotation^{31,32}.

J.C.I. has patented a process for the smelting of a concentrate with a high content of chromium³³. This involves the smelting of the concentrate in the presence of a carbonaceous reductant to produce a slag phase, a matte phase, and a ferronickel alloy containing PGM.

Texas Gulf was well advanced on the treatment of U.G.2 concentrates by the use of the expanded precessive plasma process. Again, the current state-of-the-art has not been published.

It is known that Western Platinum are successfully exploiting the U.G.2, utilizing conventional flotation techniques for the recovery of the PGM. The chromite-rich tailings are impounded in separate tailings dams for future treatment, and the flotation concentrate is smelted in a circular submerged-arc electric furnace either alone or with Merensky concentrate.

As a result of the stringent anti-pollution regulations, all current smelting operations have associated sulphuric acid plants for the treatment of converter off-gases. These processes are not described here, and the reader is referred to a paper by Plasket and Ireland that describes the manufacture of sulphuric acid at Impala Platinum Limited³⁴.

Separation of Base Metals

Originally, all the converter matte (white metal) produced by Rustenburg Platinum Mines (the sole producer in South Africa until 1966) was treated by Johnson Matthey at their Brimsdown Works, in the U.K. The process used was the Orford process (or 'tops and bottoms' process), in which the matte is smelted with salt cake and the copper is passed to the 'tops' and the nickel to the 'bottoms'. The two elements were separated physically and recovered by roasting and dissolution as sulphates.

However, a new process was soon introduced in which the copper sulphide 'tops' are blown to blister copper and cast into anodes for electrolytic refining. The nickel sulphide 'bottoms' are ground and roasted to oxide form. The nickel oxides are reduced to metallic nickel with coal in a reverberatory furnace, and are cast into anodes for electrolytic refining. The anode slimes from both electrolytic sections contain the platinum metals and are then treated in a platinum refinery.

In 1954 Matte Smelters (Pty), a company jointly owned by Rustenburg Platinum Mines and Johnson Matthey, started operations in Rustenburg, using the Brimsdown process to treat part of the smelter matte. In the mid 1960s following commercial transactions, Rustenburg Platinum as part of the deal agreed to supply Engelhard Minerals with a quantity of gravity concentrate and matte arising from tributed mining areas. This material was treated in two plants in Newark, New Jersey.

The separation process utilized by Engelhard involved melting of the matte and casting it into anodes. The anodes were subject to electrolysis, the contained copper being deposited at the cathode. The nickel remained in solution in the sulphuric acid electrolyte, while the platinum metals reported to the anode slimes. The copper cathode was refined, and the nickel sulphate solution was purified and crystallized. The anode slimes were treated chemically to remove sulphur and concentrate the PGM, which were then extracted and purified³⁵.

Johnson Matthey also developed a direct electrolysis process for converter matte similar to the Engelhard process, which was used at Brimsdown. J.C.I. Research Laboratories developed a pressure-leaching process in which the nickel and copper were dissolved in successive stages and recovered by electrowinning. The PGM remained in the residues, which were upgraded prior to refining. This process was introduced to Matte Smelters in 1966.

With the start of operations at Impala Platinum Mine in 1968, Union Corporation established a refinery at East Geduld, near Springs. The converter matte is treated by the Sherritt Gordon pressure-leaching process, which involves successive stages of acid-oxidation leaching. Nickel is removed in the first stage, and, after being modified with ammonium sulphate, the solution is reduced with hydrogen under pressure to produce a nickel powder. Copper is removed in the second stage and recovered by electrowinning. The residue from the second stage is leached further under acid conditions, and the final residue, rich in PGM, constitutes the feed to the metals refinery of Impala Platinum, which is situated on the same site. It is understood that the process³⁶ has since been modified, but exactly how is unknown.

In 1974, Rustenburg Platinum Mines and Matthey Rustenburg Refiners commissioned a new plant for the treatment of converter matte by a process that had been developed by Johnson Matthey and piloted at Rustenburg. In the process, the cooling rate of the converter matte is controlled from the commencement of solidification to at least the completion of crystallization, producing at least two phases within the matte: a PGM-free nickel-copper matte, which is non-magnetic, and a PGM-rich phase, which is magnetic. The phases are separated by grinding of the matte and magnetic separation, each

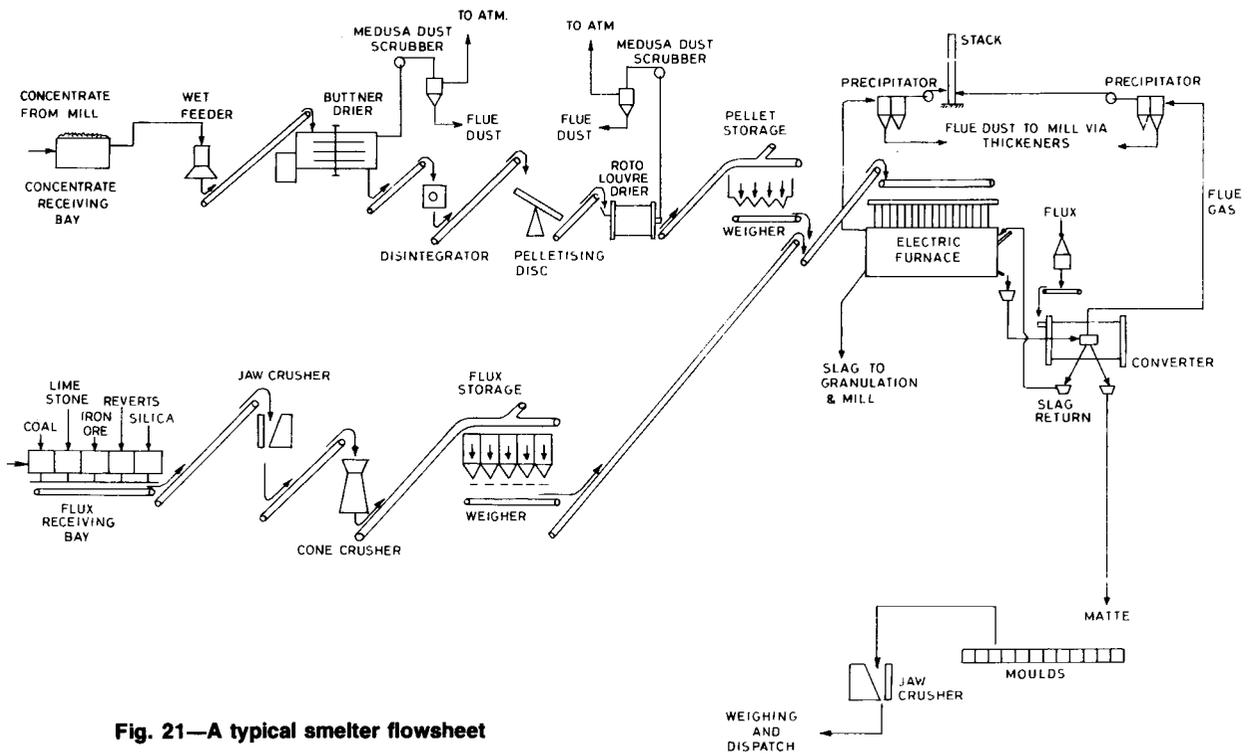


Fig. 21—A typical smelter flowsheet

phase then being treated separately. The nickel-copper matte is subjected to two-stage acid leaching under an overpressure of air. The primary leach is an oxidizing pressure leach with sulphuric acid, the leaching solution being a mixture of tankhouse spent electrolytes. Conditions are controlled to maximize the dissolution of nickel while retarding the dissolution of copper. The resultant solution is then purified for the removal of the copper, iron, lead, and cobalt prior to recovery of the nickel by electrowinning. The cobalt removed from the solution is purified and produced as crystals of cobalt sulphate. The residue from the primary leach passes to the second leach. Spent electrolyte from the copper tankhouse is utilized as the leaching solution. The resultant leach liquor, following purification (specifically the removal of selenium) is subjected to electrowinning for the recovery of copper.

The cleaned magnetic concentrate is also subjected to three stages of leaching, nickel and copper being dissolved in sulphuric acid and the leaching conditions being carefully controlled to minimize the dissolution of PGM. After filtration, the leach residue is suitable for dispatch to the PGM refinery. The solution rich in base metals joins the solutions from the leach of the non-magnetic fraction for the recovery of the base metals.

The flowsheet for the recovery of the base metals is illustrated in Fig. 22. All the converter matte from Rustenburg Platinum Mines is now treated by this process³⁷.

Refining of PGM

In the early stages of the industry, virtually all the refining of the PGM from South African ores was carried out by Johnson Matthey at their Hatton Garden Refinery, and latterly at their Royston Plant. Engelhard were also involved in refining a very small proportion of the PGM.

As the demand for platinum increased, it became necessary to increase the refining capacity, and in 1969 a plant, also totally owned by Johnson Matthey, was erected at Wadeville, near Germiston. In 1968, after the start of Impala's operations, their refinery at Springs was commissioned. In 1971, Western Platinum started operations, the converter matte produced being treated at Falconbridge Refinery, in Norway. The PGM-containing residues were refined by Engelhard Industries until 1974, when Lonrho opened their own refinery at Brakpan.

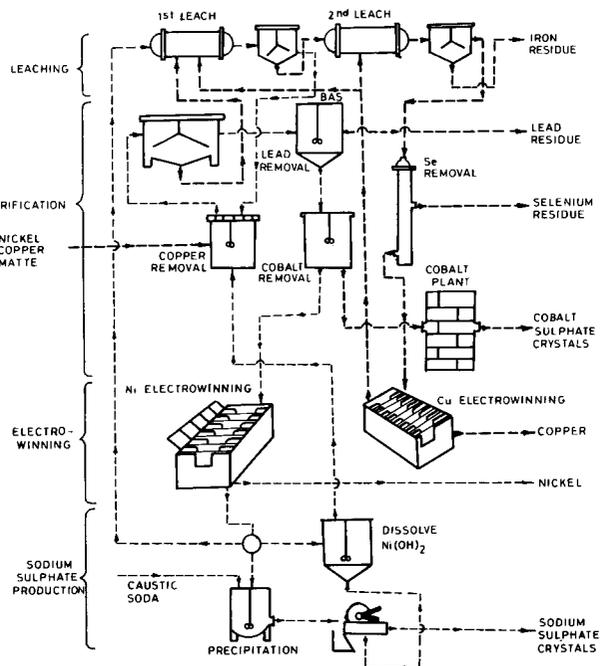


Fig. 22—Flowsheet for the base-metal refinery

The refining of PGM is a complex operation, involving the separation and purification of the six PGM metals (platinum, palladium, osmium, ruthenium, iridium, and rhodium), gold, and silver. The conventional or classical process depends on selective precipitation of the metals from solution, the chemistry of the process being based essentially on the techniques originally used to isolate the elements. The complexity leads to many different variants of processes relying on the same chemistry. The operations of Matthey Rustenburg Refiners at Royston can be regarded as typical modern practice.

The gravity concentrate is roasted for the removal of sulphur, and leached for the removal of soluble base metals. The residue is then dissolved and reduced chemically to precipitate a crude gold. Hydrolysis at about pH 1 precipitates deleterious base metals, which are filtered off. The solution is then oxidized, and ruthenium and osmium tetroxides are distilled off. Further hydrolysis precipitates rhodium and iridium, together with copper. Semi-refined platinum is then precipitated by the addition of ammonium chloride, followed by the precipitation of palladium as a complex amine. A parallel stream is employed for the treatment of the product from the leaching process used on the copper-nickel matte.

The semi-pure platinum and palladium complex products are calcined and redissolved before being subjected to a similar process route that leads to the output of pure products. The other PGM are refined individually through a number of stages utilizing the characteristics of their ammonium complexes for the extraction of the individual metals.

A description of the Engelhard process has been published³⁵, and an outline of a further variation to the conventional process is given by Edwards³⁸. The major objection to these techniques is that the separation is never ideal, necessitating multistage purification and complex recycles.

Considerable research work has been carried out on the development of less complex and more efficient processes. Both Johnson Matthey and Mintek evolved processes utilizing solvent-extraction and ion-exchange techniques, although the two processes differ in many respects. The solvent extraction of gold is well known and has been used commercially for some years. The gold is extracted as AuCl_4^- with reagents such as methyl isobutyl ketone and dibutyl carbitol. The gold is recovered as metal by direct reduction from the organic phase. Platinum can be removed in the absence of palladium, and gold by ion exchange, if iridium is in the Ir(III) oxidation state, using tri-n-octyl amines. The extraction of palladium based on hydroxy-oximes is in use, which, when coupled with accelerating additives, overcomes the slow kinetic reactions.

The extraction of ruthenium from the chloro-species is difficult. However, the removal of these as tetroxides by the distillation of carbon tetrachloride extractant is known. Iridium can be oxidized to the Ir(IV) state and extracted by the use of amine solvents, and rhodium can be recovered by either ion exchange or conventional precipitation methods. The chemistry and kinetics involved in both the Johnson Matthey and Mintek processes are described in various papers³⁹⁻⁴².

The Mintek process employs a novel pre-leaching treat-

ment in which the concentrates are alloyed with aluminium, which, it is claimed, greatly enhances leaching in hydrochloric acid in the presence of chlorine. The alloying procedure is covered by patent⁴³. The Johnson Matthey process is in use commercially at Royston. It is understood that the Mintek process is either partly or totally used by Impala at Springs and by Lonrho at Brakpan⁴⁴⁻⁴⁶. The flowsheets of both processes are given in Fig. 23, which shows the difference in approach.

The advantages claimed for the solvent-extraction processes are as follows:

- (a) increased operational safety by the elimination of the allergenic problems created by some platinum compounds,
- (b) reduced overall processing time,
- (c) improved primary yields of each element,
- (d) total recovery of all the elements, and
- (e) reduced recycling, and thus reduced process inventory.

With such advantages it is likely that the solvent-extraction process will be used in any future installation.

END USES OF PGM

In terms of the South African industry, platinum is of prime concern, with palladium and rhodium attracting more attention than ruthenium, iridium, and osmium. The uses of platinum, and to a certain extent its associated metals, are split into two main disciplines: one depending on their unique physical and catalytic properties, and the other on the more esoteric precious nature of the PGM, i.e. their store of value. The industrial and scientific uses depend on their physical properties, and the use of platinum in jewellery and as an investment medium relies mainly on its rarity.

Jewellery

Platinum in jewellery has roots in history, being a popular metal before the Second World War, when owing to its strategic value in industry it was withdrawn from the jewellery market. Its return has been slow, hindered by a lack of knowledge about the metal in consumer markets. Apart from its rarity, platinum has greater strength than silver and gold, which gives it superior stone-setting qualities and the potential to be used in delicate and unique designs. It is usually alloyed with other PGM and some base metals to improve the metal-working characteristics. Palladium, rhodium, and iridium are among the other PGM used. In Europe and North America, the hallmarking standards specify at least 95 per cent platinum, but in Japan, where the main alloying metal is palladium, specifications range from 80 per cent platinum upwards. The Japanese platinum-jewellery industry is the largest in the world and at its peak consumed almost one-third of the world production. In addition, the Japanese jewellery industry is a significant consumer of palladium. The demand for platinum in jewellery in the U.S.A. is relatively small, but in Europe West Germany is leading a fledgling industry that has doubled its demand for platinum in the past two years.

Investment

The use of platinum in the production of coins and bars

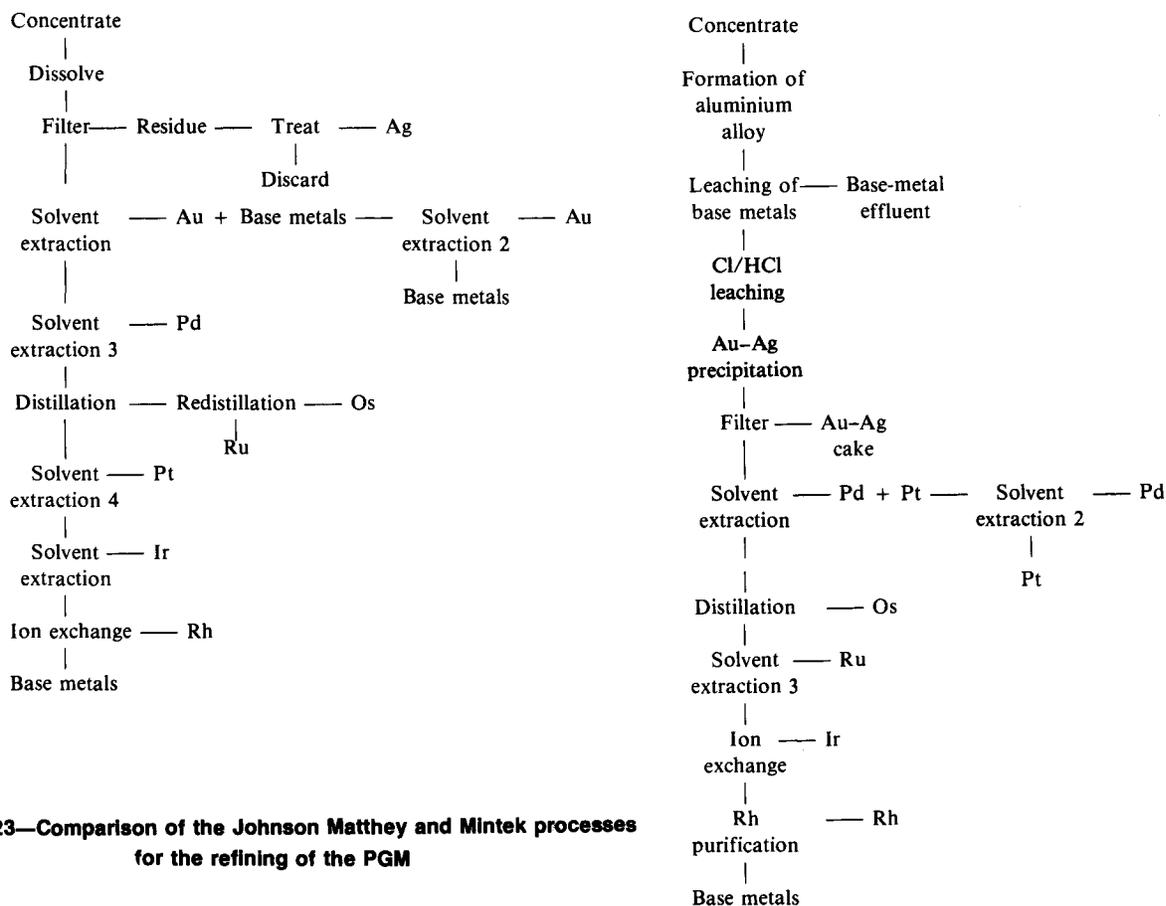


Fig. 23—Comparison of the Johnson Matthey and Mintek processes for the refining of the PGM

(palladium bars are also produced) specifically for the investment market is a recent development. Interest in platinum, and to a lesser extent palladium, has grown with the popularity of gold as an inflation hedge, and within the past three years platinum bars in sizes ranging from 10 tr. oz down to 1 g have been marketed in America, Europe, and Japan. These are designed for hoarders rather than for speculators. The year 1983 saw the launch by the Isle of Man of a 1 oz platinum coin as an investment vehicle. Simultaneously, platinum ingots of various sizes were launched in America, Europe, and Japan.

Industrial Uses

Their physical properties have resulted in a wide variety of industrial uses for the PGM. Among these are high melting points, an ability to resist attack by most mineral acids and chemicals other than halogen elements, and a selectivity in catalytic uses. The major uses are found in the chemical, electrical and electronic, glass, and automotive industries.

Automotive Industry

The use of platinum in the automotive industry is fairly recent, resulting from emission-control legislation in the U.S.A., and rivals jewellery as the largest single use in the world. The exhaust gases are passed over a catalyst that contains platinum, palladium, and rhodium in the ratio 67:26:7, which converts the hydrocarbons, carbon

monoxide, and nitrous oxide to harmless emissions. The ratios of PGM in catalysts vary considerably, depending on the size of the engine and the automobile manufacturer. The approximate quantity of PGM per automobile is 2,4 g.

At present, legislation in the U.S.A. is confined in the main to gasoline-powered automobiles. Standards have been proposed for the control of particulates from diesel-powered engines, which is likely to result in an increased usage of platinum. Japanese automobile emissions are similarly controlled, significant amounts of platinum being required by Japanese manufacturers for vehicular exhaust catalysts.

A major new market for the use of PGM in automobile emission control is expected to arise in Western Europe. Unleaded gasoline is expected to be phased in over the next decade, which will allow the use of PGM catalysts, but some countries are not prepared to wait beyond 1986, which is likely to precipitate the introduction of catalysts. The expected demand arising from Western Europe could eventually match that from the U.S.A.

Also in the automobile application, research and development continue into the use of PGM in a catalytic engine. The main purpose of the use of catalysts is to reduce compression ratios and thereby enhance fuel economy and reduce toxic emissions.

Glass Industry

The glass industry utilizes the high melting points of

the PGM and their resistance to the abrasive nature of molten glass. In the manufacture of high-quality and optical glass, unalloyed platinum, platinum alloyed with rhodium, or platinum cladding of molybdenum and refractories is used in the construction of the equipment. Platinum does not contaminate molten glass. In the manufacture of glass fibre, platinum, often alloyed with rhodium or in a grain-stabilized form, is used in the manufacture of bushings and spinnerets.

Chemical Industry

In the chemical industry, PGM are used extensively as catalysts, as well as in chemical analysis and in laboratory equipment such as crucibles, forceps, combustion vessels, and filters. Protective devices made from platinum-rhodium alloys are used in the manufacture of bursting discs. PGM are also used in anodes in the manufacture of hydrogen peroxide.

The major catalytic use is in the production of nitric acid. Platinum alloyed with rhodium is drawn into fine wire and woven into a gauze. Several of these gauzes are installed together in reactors where the catalytic oxidation of ammonia occurs. Gauzes of platinum alloys, when in use, give up part of their mass to the gas stream that is flowing across them. To reduce this loss, catchment gauzes of palladium alloy are fitted downstream, and these recover 70 per cent of the platinum lost. The main uses of nitric acid are in the manufacture of fertilizers and explosives, the latter being widely used in the mining and demolition industries.

Other main catalytic uses are the reforming of petroleum to higher-octane fuels, the production of aromatic chemicals, and the production of hydrocyanic and sulphuric acids. Palladium and rhodium are used in the catalysis of fat and polymer hydrogenation reactions, rhodium being prominent in hydroformylation reactions. Ruthenium and its complexes are extremely versatile oxidation catalysts. They catalyse the oxidation of virtually any oxidizable organic functional group, and, by choice of the appropriate conditions, the oxidation reactions can be made to proceed in high yield and selectivity. They are used in the reforming of natural gas, the production of diesel fuel, and the synthesis of ammonia.

Electronics Industry

The electrical and electronics industry is one of the growth areas for PGM. Palladium is predominant in the electro-mechanical industry, relying on its resistance to corrosion for its application in connectors, sensors, and relays. Mechanical switching devices in telephone exchanges were a major application, but this is being replaced by electronic switching.

By far the major use of palladium in the electronics field is in 'thick film' hybrid integrated circuits (ICs). A 'thick film' is a metal-glass suspension deposited onto a substrate by silk-screen printing and bonded to the substrate by firing at very high temperatures. Ruthenium is also used in considerable quantities in the 'thick film' application. The palladium in ICs forms conductors and capacitors, and the ruthenium forms resistors. Together, these are the basis for virtually all modern telecommunication systems and digital displays. Platinum is used on similar applications where reliability is more critical. It is technically the best metal for 'thick film' circuits,

its higher melting point allowing for higher firing temperatures, and therefore stronger bonds to the substrate or dielectric.

Other Industrial Uses

Other industrial uses of PGM occur in the manufacture of thermocouples, which are used mainly in the steel industry, in the cathodic protection of steel structures in corrosive environments, and in the manufacture of fume detectors. There is a demand for PGM brazing alloys, and platinum-cobalt alloys are used in the making of powerful magnets.

PGM, palladium in particular, have the ability to absorb up to 800 times their own volume of hydrogen at room temperature. This property is utilized in steady-state diffusion technology to purify hydrogen. Palladium-silver alloys are used in portable hydrogen plants and to clean up waste gases, recovering hydrogen as a useful source of fuel.

Medicine

On the medical side, platinum has found application in the prophylactic and therapeutic aspects of both human and veterinary science. One particular compound, CIS platinum II, is now an established drug for the treatment of genito-urinary types of cancer, and second-generation compounds, having a reduced toxicity, are at an advanced stage of development. Osmium is used in the manufacture of drugs for the treatment of arthritis. One other medical application in which platinum is used is in the manufacture of heart-assist pacemakers. The noble character of the metal makes it ideal for minimizing adverse reaction by the body.

In dentistry, PGM confer hardness, strength, and corrosion resistance on dental alloys. Palladium, as a result of its relatively low price compared with that of gold or platinum, is now the most favoured metal, but its use in alloys is limited to 25 per cent, above which the alloy becomes too hard. Base-metal alloys are cheaper substitutes.

Power

The final area, which is more a use for the future, is in the area of power generation. There are several aspects, but the main one appears to be a catalytic use in phosphoric acid fuel cells. The fuel cell is the basic building block of a power plant that generates electricity by catalytically combining hydrogen and oxygen to form water. Extensive research and development are being conducted in Japan and the U.S.A., where the technology has been demonstrated, but its reliability so far has been uncertain. It is unlikely that considerable commercialization will occur before 1990.

Among other uses, platinum is employed in electrodes for the generation of magnetic hydrodynamic power, and for the impregnation of steel used in the manufacture of gas turbine blades, and could be used in the future in the photo-dissociation of water to produce hydrogen as a fuel using sunlight as a source of energy.

Current Market Situation

The main platinum producers of the world are concentrated in the U.S.S.R. and South Africa, and on a very limited scale in Canada. South African producers have at times been responsible for up to 94 per cent of Western

World production. Accurate information about the Soviet production is difficult to obtain, but it is believed that the U.S.S.R. produces approximately 30 per cent of the world's supply, leaving almost 70 per cent to be attributed to South African companies.

Producers have various options in distributing the PGM in the marketplace. Major individual users, who normally require a stable, predictable, and reliable supply over long terms, have tended to contract direct with producers. The latter used to maintain a stable price independent of the free market, but in recent years they have set their daily prices in line with the free market save for special long-term contracts, which still carry stable prices. As users invariably need the PGM in fabricated form, it is usual for the metals to be delivered to fabricators at some stage for the manufacture of actual product. An alternative, which is generally the preferred case, is the purchase of the product from fabricators who have secured the PGM supply themselves. Most fabricators have long-term contracts with producers for a basic supply, and rely on free-market sources for the additional requirements. The free market has developed considerably in the past decade, and has become an important intermediary between producers, fabricators, and end-users. The main participants in the market are the precious-metal dealers and traders. They operate independently or in conjunction with fabricators and producers, securing deals between themselves and between end-users and fabricators.

Official markets also exist that provide a centralized, organized facility for market participants to trade and lay off price risk. The first market to develop futures trading in platinum was the New York Mercantile Exchange (NYMEX). Trading started in a platinum futures contract in 1956, followed by a palladium futures contract in 1968, but it is only in the past five years that significant trading volumes have been recorded. However, much of the increased volume has been attributed to speculator interest, which has become a large influencing factor on the price. The price established in open trading on NYMEX has become an important reference price around the world. Other references on the world price for platinum are the price fixes established in London twice a day at 10.15 a.m. and 4.15 p.m. by two major dealers, Ayrton Metals and Samuel Montagu. A second futures market for platinum was opened in Tokyo in 1984, which provides another forum for hedging. The latest development is the possible establishment of a platinum options market on NYMEX.

Platinum Statistical Position

The supply and demand in the Western World is shown in Table II, and the demand by application is shown in Table III for Japan and Table IV for North America. In regard to these tables, the following should be noted.

- (1) Differences occur between the totals for each year in Tables III and IV and those in Table II as a result of rounding off of values.
- (2) The supply figures are estimates of sales by the mines of primary platinum. The demand estimates shown in this paper are net figures, the demand in each sector being the total purchases by consumers less any sales back to the market. Thus, the annual totals

represent the amount of newly mined metal that is acquired by consumers in any particular year.

- (3) Movements in stocks in a particular year reflect changes in stocks held by other than primary refiners and final consumers, such as metal in the hands of fabricators, dealers, banks, and individuals. A positive figure indicates an increase in stocks, including some platinum bought for investment; a negative figure indicates a rundown in stocks.
- (4) Hoarding is the long-term holding of metal for investment. In this paper, hoarded metal is defined as investment bars and coins weighing 10 oz or less. The 'other applications' sector in Table II includes some larger bars or ingots acquired for investment purposes, especially in Japan.

Table V gives some indication of the breakdown of platinum and palladium demand into end-uses. For palladium, the electrical and electronics usages are given separately for more detail, but the chemical uses are reported collectively. The use in each category can be calculated in troy ounces from the total palladium consumption, which is approximately 3 million ounces.

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TABLE II
PLATINUM STATISTICAL POSITION (TONS)

	1977	1978	1979	1980	1981	1982	1983	1984
Sub-total	92	93	97	12	10	12	93	93
Sales in China and Eastern Europe	2	4	1	1	1	1	1	1
Movement in stocks (decrease)	(6)	(5)	(3)	15	(4)	5	7	3
Total demand	78	82	87	88	73	78	76	84

Source: Rustenburg Platinum Mines, Johnson Matthey

TABLE III
JAPANESE PLATINUM DEMAND (TONS)

Usage	1977	1978	1979	1980	1981	1982	1983	1984
Jewellery	26,1	26,0	18,4	13,7	19,4	19,3	17,4	19,4
Autocatalyst	2,5	5,6	6,2	6,5	5,9	5,3	5,3	5,3
Chemical	0,3	0,3	0,3	0,3	0,3	0,3	0,3	0,5
Electrical	0,5	0,5	0,4	0,4	0,5	0,6	0,6	0,9
Petroleum	0,6	0,5	0,3	0,5	0,5	0,5	0,5	0,6
Glass	0,9	1,2	1,2	1,2	1,6	1,4	1,9	2,3
Hoarding	—	—	—	—	—	—	0,2	0,5
Others	3,3	2,3	1,7	6,5	7,6	5,3	3,4	5,9
	<u>34,2</u>	<u>36,4</u>	<u>28,5</u>	<u>29,1</u>	<u>35,8</u>	<u>32,7</u>	<u>29,6</u>	<u>35,4</u>

TABLE IV
NORTH AMERICAN PLATINUM DEMAND (TONS)

Usage	1977	1978	1979	1980	1981	1982	1983	1984
Jewellery	0,5	0,6	0,5	0,5	0,5	0,5	0,5	0,5
Autocatalyst	11,0	13,4	20,8	13,7	13,4	14,2	13,1	16,2
Chemical	2,8	4,4	4,0	3,6	1,6	2,5	3,1	4,0
Electrical	3,0	3,1	4,0	4,5	2,2	2,2	2,8	3,0
Petroleum	2,5	3,3	6,1	4,4	1,7	0,6	0,5	0,5
Glass	2,0	3,1	3,1	1,6	0,6	0,3	0,5	0,9
Hoarding	—	—	—	—	—	1,2	1,2	0,9
Others	2,8	2,3	3,0	2,3	1,9	0,6	0,8	1,1
	<u>24,6</u>	<u>30,2</u>	<u>41,5</u>	<u>30,6</u>	<u>21,9</u>	<u>22,1</u>	<u>22,5</u>	<u>27,1</u>

TABLE V
END-USES OF PLATINUM AND PALLADIUM

Commodity	End-use	%
Platinum	Autocatalyst	28
	Jewellery	30
	Electrical/electronics	7
	Petroleum refining	1
	Glass	5
	Hoarding	7
	Other	11
	Chemicals	11
Palladium	Autocatalyst	7
	Jewellery	9
	Electrical/electronics	7/25
	Dental/medical	23
	Petroleum refining	7
	Glass	—
	Hoarding	—
	Other	3
	Chemicals	19

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Platinum-group metals

A seminar on platinum-group metals is being planned by the International Precious Metals Institute (IPMI). The Seminar will be held from 6th to 9th October, 1985, in Washington, D.C.

The programme is being organized by J. Michael Sharrott (Manville International Corporation), David C. Fenton (Johnson Matthey, Inc.), Juergen F. Schroeter (Engelhard Corporation), and Donald A. Corrigan (Handy & Harman).

Sessions are being scheduled to provide almost equal emphasis to each of the six platinum-group metals.

Papers will be presented on mining, supply, and demand, as well as on important or unique applications for the individual metals including aspects of the broadening usage in automobile catalytic converters.

The programme will appeal to industrial users, materials and procurement specialists, research scientists, members of the financial and investment community, and key personnel in the government sector.

Additional information can be obtained from the IPMI, ABE Airport, Government Bldg, Allentown, PA 18103, U.S.A. Telephone: 215/266-1570.

Characterization of rock mass

The South African National Group of the International Society for Rock Mechanics is to hold a Symposium on Rock Mechanics Characterization at Mintek, Randburg, on 8th November, 1985.

In rock engineering it is not generally practical to carry out sufficient numbers of large-scale tests and measurements to define with certainty the overall behaviour of a rock mass. Rock masses are therefore characterized by means of parameters such as geological factors and properties, which can be obtained from simple, inexpensive measurements and index tests. The Symposium is aimed at creating a greater awareness of the benefits of rock-mass characterization for design in rock engineering. Emphasis will be placed on practical aspects.

Provision has been made for standard papers, which will be presented at the Symposium, and for short technical notes, which will not be presented but only published in the proceedings.

The topics are as follows:

- Geological factors (weathering, jointing, etc.)
- Rock-mass classification
- Laboratory and *in situ* testing for characterization
- Computer data analysis.

All enquiries relating to the Symposium should be addressed to:

The Organizing Committee
Rock Mass Characterization
National Mechanical Engineering Research Institute
C S I R
P.O. Box 395
Pretoria 0001
Telephone: (012) 86-9211 ext. 2128 Mr W.L. van Heerden
(011) 726-3020 Miss S.A. Thorpe
Telex: 3-21312 SA
Telegrams: NAVORSMEG.

Flotation reagents

For a number of years, the Council for Mineral Technology (Mintek) has expended a considerable amount of research into the nature of the interactions involved in the selective recovery of minerals by flotation. The combination of reagents used in practice, such as the type of collector, frother, and depressant, the necessity for an activating agent, and the use of a combination of different reagents as the collector, can vary significantly from mine to mine although the same valuable mineral (e.g. pyrite, coal) is recovered at each mine. It is felt that a symposium on this subject will highlight the problems as well as the solutions developed by the various mines and manufacturers of reagents. In addition, the possible theoretical interactions and association of the various reagents will be considered.

Mintek is to hold a Symposium on the Current and Future Usage of Flotation Reagents in the South African Mineral Industry on 25th October, 1985, in Randburg.

Papers on the following topics will be presented:

- Reagent assessment on a laboratory scale
- Collector development
- Adsorption mechanism of depressants
- Improved copper recovery with the correct reagent
- Amine collectors in pyrite flotation
- Advances in reagents for coal flotation
- Modification of depressants
- Use of mixed collectors
- Hard water in the flotation of apatite
- Development of frothers
- Application of polyglycol ethers
- New collectors in the flotation of sulphide ores.

Enquiries should be addressed to The Conference Secretary (C. 30), Mintek, Private Bag X3015, Randburg, 2125 South Africa.

Asian mining

Asian Mining '88, the third conference in the series devoted to topics that cover the minerals industry—geology, mining, mineral processing, and metallurgy—will be organized by The Institution of Mining and Metallurgy, in association with other international bodies, and held at the Putra World Trade Centre, Kuala Lumpur, Malaysia, from 8th to 11th March, 1988.

As in the earlier conferences (held in Singapore in 1981 and Manila, Philippines, in February 1985), specific attention will be paid to topics that are of particular relevance to the Asian region, and technical tours and

an associated international exhibition of equipment, products, services, etc., for the minerals industry will be arranged.

Requests for further information and copies of Conference circulars should be made to

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