Platinum Mining at Rustenburg

THE DEVELOPMENT OF OPERATING METHODS

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The deposit worked in the two mines operated by Rustenburg Platinum Mines Limited forms part of the Merensky Reef Horizon of the Bushveld Igneous Complex. This reef, named after the geologist Hans Merensky who was responsible for the prospecting programme that led to its discovery, produces nearly all of the platinum metals mined in the Union of South Africa. It comprises a layered pyroxenite horizon that has been traced continuously, except for a few minor breaks, around both the eastern and western limbs of the Complex. The prospected strike of the eastern limb amounts to 42 miles and that in the western limb to 70 miles. Of this great length of strike only 12 miles of outcrop are being actually mined down dip at the two mines operated by Rustenburg Platinum Mines Limited.

The platinum values are concentrated near the top and bottom contacts of a coarsely crystalline felspathic pyroxenite which has the characteristics of a pegmatite. Chromite concentrations also occur at these contacts and usually form two thin chrome seams.

This pegmatitic pyroxenite of the Merensky Reef is overlain by a pyroxenite band of similar composition but of a less coarsely crystalline structure, which may be referred to as the Merensky pyroxenite. This band grades rapidly at its upper contact into an anorthositic gabbro and this in turn is followed by a zone of mottled anorthosite.

In the Rustenburg mine the Merensky platinum reef averages about a foot in thickness and reaches two feet only in certain limited areas. This narrow width allows the whole pegmatitic pyroxenite together with a portion of both the hanging wall and footwall to be included in the stoping width mined.

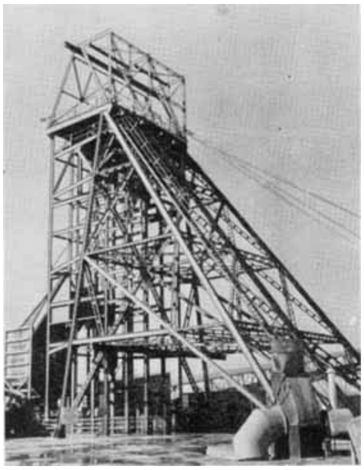
At the Union mine the pegmatitic pyroxenite averages some 15 to 18 feet in thickness. This thickness shows a gradual decrease from over 20 feet in the north-eastern to less than 10 feet in the south-western areas of the property. Platinum values are found near both the upper and lower contacts, but in the present workings the values at the upper contact are appreciably higher than those at the lower contact. As the distance between the two potential economic zones is far too great for them to be stoped together and the average tenor of values on the lower zone normally falls below the level of payability, mining is confined to the zone near the upper contact.

Platinum values show a very low range of distribution, giving a very even tenor of values. The platinum is partly in the form of native metal, invariably alloyed with iron (ferro-platinum) and partly as the sulphide, arsenide and sulph-arsenides, these latter being in intimate association with sulphides of iron, nickel and copper.

At the Rustenburg mine the platinum compounds preponderate, while at the Union mine the ferro-platinum forms the major platinum mineral. Associated with the platinum are smaller proportions of the platinum group metals. Platinum forms the major component of the platinum group metals plus gold. Palladium is next in importance, with ruthenium, rhodium, iridium and osmium in descending order.

The World's Largest Platinum Producer

This article, dealing with the geology of the platinum deposits at Rustenburg, the mining methods in use and the metallurgical treatment of the ore, is abridged from a comprehensive paper presented by the authors to the Seventh Commonwealth Mining and Metallurgical Congress during its technical sessions held in Johannesburg in May. Rustenburg is the world's largest producer of the platinum metals and works in close association with Johnson Matthey, who refine and distribute the metals produced by the mining company.



Sulphides of iron, nickel and copper are always associated with the platinum. Chromite is also present and the highest platinum values occur in association with this mineral. The chromite itself does not contain platinum, which is in the interstitial silicate minerals.

Mining Methods

The reef mined at Rustenburg consists of a thin chromite-rich layer averaging ¾ inch in thickness with, above it, a layer of coarsely crystalline felspathic pyroxenite about 12 inches in thickness. The lower nine inches of the Merensky pyroxenite above the reef and the upper nine inches of the footwall anorthosite immediately below the chrome band contain platinum group metal values of small account.

It may be accepted generally that the platinum-bearing horizon covers a maximum stoping width of 30 inches; the surveyor's measured width in stoping operations has averaged 28½ inches over the past ten years. The strike of the reef is east-west and the dip is at 9° 30′ to the north. The reef strike tends to bend slightly northwards on the western half of the property in conformity with the curved shape of the Mafic zone.

The strike and dip are most regular. This is also true of the mineral contents of the deposit. Over the life of the mine to date it has been found that while appreciable variations in value may occur from one sampling section to the next, yet the average values of ore reserve blocks show little variation. The few dykes encountered have had little or no influence upon the regularity of the deposit, while serious or even minor faulting is unknown.

Conditions of regularity of strike, dip and value have allowed the mine to be laid out in a simple and straightforward manner. The length of strike and the comparative shallowness of the deposit have made possible the rapid development of large areas of ground whenever a demand for platinum arose. Thus the output of the mine has followed world demand very closely.

The upper levels of the mine have been and will be largely worked from an incline and sub-incline haulage system, followed in depth by a vertical shaft. There are three vertical shafts in operation on the property, while a fourth will be sunk shortly. All three shafts are comparatively shallow, their depths being 500 feet, 780 feet and 1,500 feet respectively from surface. The west vertical shaft is a three-compartment timbered shaft with two skip hoisting compartments and a pump/ladderway. It has been in use for the past twelve years but ore in the area is almost exhausted and this shaft will shortly become an upcast ventilation shaft. The Waterval vertical shaft is a 15-foot diameter concrete-lined shaft which serves the Waterval stoping area. Five-ton skips, interchangeable with double-decked cages are installed.

The central deep shaft is a four-compartment shaft of "square-rectangular" section, measuring $24\frac{1}{2}$ feet by $11\frac{1}{4}$ feet inside timbers. Single-decked cages capable of carrying forty persons are installed in the two south compartments. These cages are sufficiently large to accommodate trucks of explosives, mining timber, etc. Skips are of five-ton capacity and are interchangeable with double-decked cages capable of carrying forty persons. The choice of the "square-rectangular" shaft section was made for two reasons, to ensure a large volume of air being delivered to the underground workings, and to facilitate the loading of men and material into large-sized cages.

A headgear of comparatively light construction is erected at all three vertical shafts. The ore or waste rock hoisted is tipped into a small headgear bin from which it is led by conveyor either to an ore bin situated over the mine surface haulage railway line or to a waste bin whence it is taken by self-dumping skip to the waste dump.

A 15° three-compartment sub-incline shaft is now in the course of sinking in the Waterval area. This shaft will be equipped with two 10-ton skips. Ore and waste will be delivered to their respective passes at the top of the incline and trammed to the Waterval vertical shaft tips by overhead electric locomotive.

Generally, except in the eastern extension area, the haulage of ore underground is by overhead electric locomotives pulling trains of 4-ton hoppers through footwall haulages. In the eastern extension area 2-ton diesel locomotives are in use.

Stoping Practice

A great deal of thought and investigation has gone into the laying down of standard working methods to ensure the delivery of a product of the highest value to mill at the lowest cost. The regularity of the deposit has made the laying down of these standardised procedures comparatively simple, and has to a

Drilling a round of holes before blasting in a reef drive at Rustenburg

Ore is tipped from panel cars into the central gully scraper path

large extent eliminated the necessity of having to work round large blocks of unpayable ore. Regularity of dip and strike and freedom from faulting allows stope faces to be carried on a true dip line, which leads to optimum breaking.

The aim in stoping is to advance the face in as straight a line as possible on dip at the lowest possible stoping width compatible with efficient breaking. Shallow gully tracks on strike at 40 foot centres on dip divide the working face into panels and serve as passage ways to and from the face. The raise-winze connection becomes the path of the central gully scraper.

Ore broken on the face is first washed down and examined by the panel-cleaning native. All footwall norite and Merensky pyroxenite of sortable size is picked out and thrown back on to the face scatter-pile. The ore remaining is then shovelled 30 feet down dip and 10 feet up dip to the panel track, where it is loaded into $\frac{1}{3}$-ton cars, trammed along the gully track and tipped into the central gully scraper path. It is then scraped to the stope box connecting the stope to footwall haulage, loaded into hopper trains and delivered to the shaft box or ore pass system.

The Metallurgy of the Platinum Ores

The discovery of platinum in the Bushveld deposits in 1924 posed several problems for the metallurgists, as this was the first time that ores had been mined primarily for the production of platinum. Previously they had been recovered as alluvial platinum from placer deposits, or as a by-product from the mining of nickel-copper ores.

The ore first treated at Rustenburg was from the oxidised zone. It was soft and friable and was mined by pneumatic picks. A recovery plant was built and started operations in 1930. The ore was ground in ball mills, with great care to avoid sliming, the pulp went to the concentration section, consisting of James tables and corduroy tables, by means of which a rich gravity concentrate was made and the pulp was finally sent to a flotation section to recover a concentrate.

The concentrates recovered by the gravity concentrate section were reconcentrated by dressing on James tables. By repeated dressing they were brought up to a suitable grade which was sent overseas directly to the refiners.

As the mine progressed to deeper mining the proportion of oxide ore milled decreased as operations got into the sulphide zone. Apart from other effects on the metallurgical treatment, the sulphide ore reduced the proportion of platinum group metals recovered in the gravity section. The question has often been discussed as to whether it is advantageous to carry on with the gravity recovery, or whether flotation alone should be relied upon. Apart, however, from the safety factor, that by taking out a rich fraction early in the treatment a slightly lower tailing may be obtained, there is a definite economic advantage in making this product. As the product is high in platinum metals it is sent directly to the refiners, thus avoiding the small but inevitable losses incurred in the further treatment of the flotation concentrates.

The gravity concentrates consist of sperrylite (platinum arsenide), cooperite (platinum sulphide), braggite (platinum palladium nickel sulphide), stibio palladinite (palladium antimonide), laurite (ruthenium sulphide), native platinum, native gold, mixed with chalcopyrite, pyrrhotite, pentlandite and other sulphides, with some chromite and graphite.

As a greater proportion of the platinum group metals was then recovered by flotation, the problem of the treatment of the flotation concentrate arose and there was much experimentation (and controversy) to discover the best method.

A number of methods were given intensive investigation, but the one that was most successful, and is now adopted, was to smelt to a nickel-copper matte. Preliminary work was carried out on this method by H. R. Adam at the Government Areas Laboratory, but it was Johnson, Matthey & Co., Limited, who first successfully used the method on a large scale. That this method should be considered was of course obvious, as the concentrate contains nickel, iron and copper sulphides—all the requirements for a matte—but the high recovery of the platinum group metals in the matte was somewhat of a surprise.

The mill at Rustenburg is on the same site as the original 300-ton-per-day mill, and great care and ingenuity have had to be exercised to extend and modernise it to its present capacity.

The ore is first crushed in two jaw crushers, to reduce it to minus six inches, the fines (minus two inches) are screened out and the coarse washed on vibrating screens and then passed over sorting belts. The coarse ore is then crushed in two Symons standard crushers; the crushed product from here goes to the ½-inch screens, with the oversize from the ½-inch screens returning to two Symons

From the central scraper gully, ore is loaded into hopper trains and delivered to the shaft box

Part of the flotation plant at Rustenburg. Eight banks of eighteen cells act as roughers, with a further two banks of eighteen cells as cleaners

shorthead crushers in closed circuit with these screens.

The washings go to four rake classifiers, the rake product being added to the mill feed product, the classifier overflow going direct to the mill circuit.

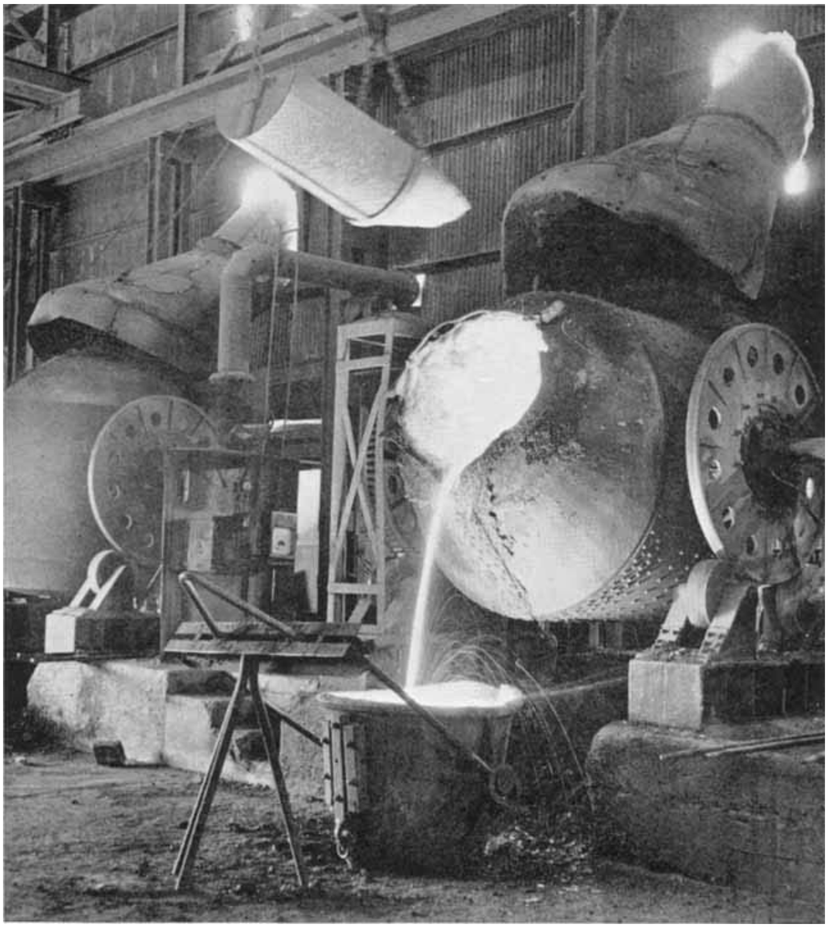
Ball milling is carried out in two stages—primary and secondary. There are a total of 22 mills, including the original four mills installed in 1932. The mills are in closed circuit with hydrocyclone classifiers, which give much sharper separations.

Gravity concentration is by means of corduroy tables, which so far have proved the best practical means of recovering the fine platinum minerals. The corduroy concentrate is dressed on an elaborate system of James tables up to a final concentrate which is sent directly to the refiners.

The tailings from the tables, in a considerably diluted pulp, are returned to the mill circuit, the final pulp from the mill being thickened to flotation density in four 75-foot thickeners fitted with diaphragm pumps. The thickener underflow then goes to the flotation plant. The circuit used is a straightforward rougher-cleaner circuit, with the cleaner tailings returned to the head of the roughers. The thickener underflow is divided into two halves by a splitter in a launder and each half is divided into four streams by revolving distributors. The eight streams are fed to eight banks of eighteen cells acting as roughers. A further two banks of eighteen cells act as cleaners.

The flotation tailings are sampled by an automatic sampler, thickened in a 250-foot traction thickener and discharged to the tailings dam. The concentrates are filtered on three 8-foot by 8-foot drum filters before passing to the smelter. The small tonnage of concentrates to be smelted initially led inevitably to the use of a small blast furnace. As the tonnage to be smelted has increased considerably thought has been given to the possible use of reverberatory furnaces, but it has been decided to retain the blast furnace for smelting. Among other reasons the small blast furnace is very flexible—it can be stopped and started very readily.

There are four blast furnaces, 10 feet long by 36 inches at the tuyeres, the number in operation depending on the output required. The furnaces are water jacketed and run with a trapped spout into forehearths. Matte is tapped periodically into ladles and blown in



Pouring blown matte from one of the Great Falls converters. After cooling and breaking up, the matte is shipped to Johnson Matthey for the extraction of copper, nickel and the six platinum metals

three upright (or Great Falls type) 8-foot converters. The blowing is carried as far as the white metal stage, and the white metal is poured into moulds. When solid this is broken up in a small jaw crusher, bagged and sent to the refiners. It contains about 74 per cent copper plus nickel. The converter slag is cast and returned with other secondary material to the blast furnaces. The blast furnace slag is run off continuously, and quenched in a launder with a strong stream of water. The slag granules are raked on to a conveyor belt which conveys them to a storage bin, from which they are dumped.

The converter matte is treated at the Brimsdown works of Johnson, Matthey & Co., Limited, by the "tops and bottoms" process. The matte is smelted with salt cake and the copper passes into the "tops" and the nickel into the "bottoms". 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 nickel oxide. This is reduced to metallic nickel with coal in a reverberatory furnace and cast into anodes for electrolytic refining. The anode slime from both the electrolytic sections contains the platinum metals and is then sent to the platinum refinery for chemical processing.