THE EXPLOITATION OF THE PLATINIFEROUS ORES OF THE BUSHVELD IGNEOUS COMPLEX WITH PARTICULAR REFERENCE TO THE RUSTENBURG PLATINUM MINES

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SYNOPSIS

The Merensky Reef of the Bushveld Igneous Complex, exploited in the two mines operated by Rustenburg Platinum Mines, Limited, produces nearly all of the platinum and platinum group metals mined in the Union of South Africa.

The Reef is a layered band of coarse pegmatitic pyroxenite containing platinum and platinum minerals in association with chrome and the sulphides of iron, nickel and copper. The Reef is a unit in a regular layered succession of basic igneous rocks and has been traced, with only minor breaks, around the outcropping belts of mafic rocks.

In both Sections of the Rustenberg Platinum Mines the Reef is characterized by remarkable regularity of dip ($9\frac{1}{2}°$ N. at the Rustenburg Section and 23° S.E. at the Union Section) and strike, a very even tenor of values and few disturbances, allowing for simplicity in mining layout.

The shallower levels were developed from outcrop by means of incline haulages, but, for the deeper levels, vertical shafts are employed. Ore is conveyed to the shafts along footwall haulages. Owing to the narrow reef widths, stope height is kept at a minimum, averaging $28\frac{1}{2}$ in. Breast stoping, strike tracks and centre gully scraping are used throughout.

Improved methods and better supervision have resulted in marked improvement in rock breaking efficiency.

Much research has been carried out to develop satisfactory methods of extraction. The present system involves concentration of the ground ore on corduroy tables and dressing this concentrate on James tables. This final concentrate goes directly to the refiners.

The final pulp from the mill is treated in flotation cells and produces a concentrate of the sulphide minerals including a proportion of the platinum minerals. This concentrate is pelletized and smelted in blast furnaces and converters to produce a matte. This matte is treated to recover nickel, copper and a concentrate of the platinum metals which goes to the refiners.

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THE EXPLOITATION OF THE PLATINIFEROUS ORES OF THE BUSHVELD IGNEOUS COMPLEX WITH PARTICULAR REFERENCE TO THE RUSTENBURG PLATINUM MINES

1 : INTRODUCTION

IN 1923 the finding of the lode deposits of the Waterberg District gave impetus to the search for platinum in the Transvaal. This was followed in 1924 by the discovery of the Lydenburg deposits and, in 1925, the Potgietersrus and Rustenburg reefs were located in the Mafic Zone of the Bushveld Igneous Complex.

Within a short period a number of mining companies were formed to work the various discoveries and the ill-starred platinum boom of the middle 1920's started. The boom was, however, short-lived and, following upon reductions in the price of platinum, a slump occurred in the late 1920's. Only two mining companies survived. These companies were the Potgietersrust Platinums, Limited and Waterval (Rustenburg) Platinum Mining Company, which, in the interests of economy, were merged in 1932 to form the Rustenburg Platinum Mines, Limited.

In 1947 the Union Platinum Mining Company was formed to work the Merensky Reef Horizon on the farm Swartklip some 60 miles to the north of Rustenburg. This company was amalgamated with the Rustenburg company in 1949 and its property now constitutes the Union Section of the Rustenburg Platinum Mines.

The Rustenburg Platinum Mines, Limited, has within recent years become the world's largest producer of platinum, and is, in fact, the major producer whose primary business is the production of platinum group metals.

The Rustenburg company works in close association with Messrs. Johnson, Matthey & Company of London who refine and sell the metals produced by the mining company.

The deposit worked on both the Rustenburg and Union Sections of the mine forms part of the Merensky Reef Horizon of the Bushveld Igneous Complex.

This paper touches on the geology of the platinum deposits generally, the mining methods in vogue at the Rustenburg Section of the mine and the metallurgical treatment of the ore at that section.

2 : THE GEOLOGY OF PLATINUM DEPOSITS IN THE TRANSVAAL

The invariable association of other metals of the platinum group (palladium, rhodium, ruthenium, osmium and iridium) with platinum necessitates the consideration of platinum group metals deposits as integral with platinum deposits. Although platinum and its associated metals have been recorded from a wide variety of localities, economic deposits have been limited to three different geological environments, namely:

(i) Vein deposits in the Waterberg district.

(ii) Deposits in the Bushveld Igneous Complex.

- (i) Pipe-like bodies of hortonolite-dunite,
- (ii) The deposits of the Potgietersrus area,
- (iii) The layered Merensky Reef.

(iii) The gold-bearing conglomerates of the Witwatersrand system.

1. The vein deposits of the Waterberg

A series of brecciated quartz veins, filling post-Karroo fault zones in Rooiberg felsite, carry sporadic platinum values. One particular vein, occurring on the farms Welgevonden and Rietfontein near the village Naboomspruit about 100 miles north of Pretoria, was the site of the first discovery of platinum in South Africa. This vein was worked for platinum in 1924-1926.

The platinum was erratically distributed, but extremely rich lenses, sometimes containing more than 1000 dwt/ton platinum group metals were found.

The irregular distribution of the platinum metals, the scarcity and size of the rich lenses together with extraction difficulties, caused the mine to close.

2. The Bushveld igneous complex

The Bushveld Igneous Complex is a layered complex, its mafic zone consisting of basic and ultrabasic igneous rocks. These include peridotites, harzburgites, pyroxenites, gabbro and norite, anorthosite, seams of chrome and seams of titaniferous magnetite.

The mafic zone outcrops in two wide curved arcs within a basin of the sedimentary rocks of the Transvaal System, the upper members of which have suffered extremely severe thermal metamorphism.

The western arc extends as a thinner sheet reaching the western boundary of the Transvaal. A northern limb of the mafic rocks extends northwards from Potgietersrus cutting across the entire Transvaal System.

The inward dips of the layered mafic rocks and their position within the basin of the Transvaal System suggested that the Complex was in the form of a gigantic lopolithic sheet.

Recent geophysical and other evidence indicates that the mafic zone of the Bushveld Complex is not a lopolith but that the eastern and western limbs appear to be separate 'T'-shaped curved dykes, extending to abyssal depths and that the mafic rocks do not seem to underlie the central portion of the basin.

In the Complex the layering of the various types of igneous rock is more regular than most sedimentary formations. Individual layers and very similar successions of rocks can be traced from north to south around the eastern and western curved belts.

The genesis of the Bushveld Complex is still speculative. Originally it was believed to be the result of differentiation of an 18,000 ft thick sheet in situ.

Recent geological work has found evidence that the emplacement took place in a number of thinner sheets. These sheets are considered by many geologists as being the result of successive intrusions, each rock type being a separate injection, with differentiation playing only a minor part.

A more recent suggestion visualizes a series of extrusive sheets, each sheet being differentiated. This idea would account for the regular layering and the absence of cross-cutting in the sheets.

A fourth theory, which has received less support, is that the rocks of the Complex are the result of metasomatism (transformation) of the upper sediments of the Transvaal System.

Within the Complex numerous platinum discoveries have been reported, but of these only three types have been found of sufficient economic interest to reach the stage of actual mining up to the present time. (See Fig. 1).

The economic deposits can be classified as follows:

(*a*) *Deposits in hortonolite-dunite*. Pipe-like, dyke-like and sill-like bodies of iron-rich dunite are found at various points around the Complex. They are mostly found in the layered zone from the Merensky Reef to the Lower Chrome seams but are clearly associated with faults, fractures and areas of disturbance. Most of the hortonolite-dunite deposits are barren or only contain low platinum values, but three pipe-like bodies in the eastern limb of the Complex contained economic values. Three mines, the Onverwacht, Mooihoek and Driekop mines, reached the production stage in this area.

These deposits consisted of nearly vertical pipe-shaped bodies some 40 to 75 ft dia, gradually decreas-

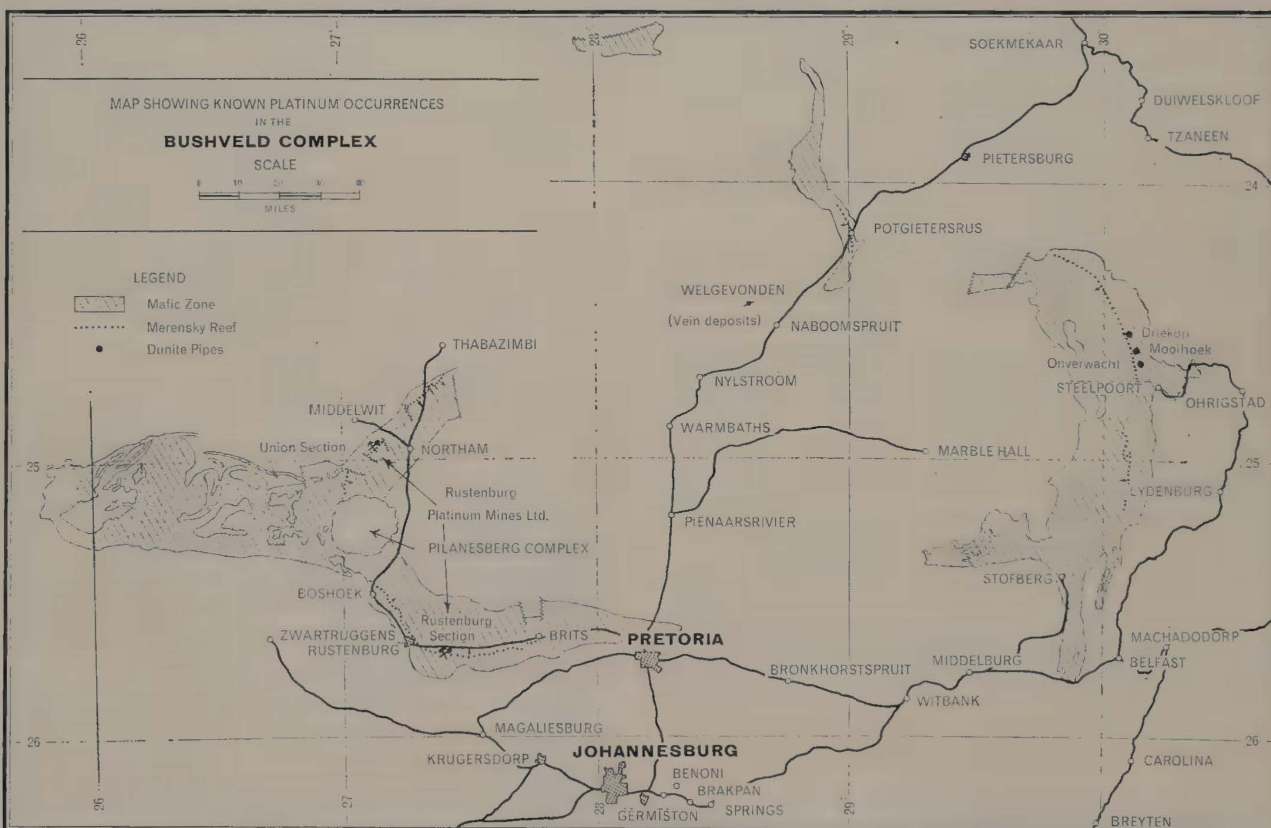


Fig. 1

ing in diameter with depth, which have been traced to a vertical depth of 350 to 800 ft. The platinum values showed a wide distribution. The platinum content in the shallow levels was high but there appears to have been a drop in values in the deeper levels.

The platinum occurred mainly in the metallic state (ferro-platinum) with smaller proportions of sperrylite ($PtAs_2$) and cooperite ($Pt(AsS)_2$).

(*b*) *Deposits of the Potgietersrus area.* In this area the Bushveld mafic zone cuts across the sediments of the Transvaal System. Two types of platiniferous deposits are developed. In the one, magmatic deposits are found in a thick layer of pyroxenite which shows some resemblances to the Merensky Reef of which it is considered to be the possible equivalent.

The other type of deposit is contact metasomatic, platinum being found in veins, fissures and disseminated in the altered dolomite and banded ironstone of the Dolomite series of the Transvaal System.

The platiniferous zones of the pyroxenite layer consist of lenses of pegmatitic pyroxenite of considerable thickness, containing platinum and platinum group metals associated with the sulphides of nickel, copper and iron. The platinum minerals include native platinum (ferro-platinum) together with sperrylite and cooperite. The platinum group metals contain 50-60% palladium.

The metasomatic deposit consists of a wide zone of contact impregnation together with vein and fissure deposits, some within the altered dolomite and some within crush zones in the banded ironstone beds which lie at the top of the dolomite.

The ore is associated with the sulphides of copper, nickel and iron and the platinum group metals consist of platinum arsenides and sulph-arsenides together with palladium in the form of stibiopalladinite (Pd_3Sb).

(*c*) *The Merensky Reef.* The Merensky Reef, named after the geologist Dr. Hans Merensky who was responsible for the prospecting programme which led to its discovery, is a layered pyroxenite horizon which has been traced continuously, except for certain breaks, around both the eastern and western limbs of the Complex. On the eastern limb it has been traced along a strike of 95 miles and on the western limb it has been followed for 125 miles.

The prospected strike of the eastern limb amounts to 42 miles and that in the western limb of 70 miles. Geological breaks due to faulting, disturbed areas and the Pilanesberg alkali complex, are partly responsible for the shortening of the proved strike of the Merensky Reef. Of this great length of strike only twelve miles of outcrop are being actually mined down dip at the two mines operated by Rustenburg Platinum Mines, Limited.

The Merensky Reef lies nearly at the top of what is known as the 'critical' or 'differentiated' zone of the Bushveld Complex, lying some 6,000 ft above the floor of the mafic zone in the Rustenburg area. The floor of the Complex rises from here eastwards and at Brits the Merensky Reef lies only 3,000 ft above the floor. Farther east a rapid rise in floor level cuts out the Merensky Reef. Field evidence indicates that around the Complex the mafic zone was emplaced on a folded and somewhat irregular floor of the Transvaal System.

3. Reefs of the Witwatersrand System

The gold-bearing conglomerates of the Witwatersrand System contain minute proportions of platinum group metals, osmiridium being in the greatest proportion. The platinum group metal content varies from 1/500 to 1/10,000 of the gold content, (2 to 50×10^{-8} of the total rock) according to the locality and reef mined. Owing to the enormous tonnage of ore mined and treated to recover the gold and the practicability of extraction by simple gravity methods, the platinum group metals are recovered on certain mines, yielding a total annual production of some 6,000 ozs.

The following is a typical analysis:

Pt—11%, Ir—30%, Os—34%, Ru—13%, Rh—0.7%, Au—0.6%.

The minute quantities of osmiridium in the Witwatersrand conglomerates are considered to be of detrital origin. Evidence suggests that the conglomerates which carry the coarser gold are generally richer in osmiridium than those in which the gold particles are finer.

Of the reefs in the East Rand area, both the Main Reef Leader and the Kimberley Reefs have been found to carry the greatest quantities. The younger Black Reef, the basal formation of the Transvaal System, lies on the eroded Kimberley conglomerates at the mine of the Government Gold Mining Areas (Modderfontein) Consolidated, Limited, and carries comparatively rich osmiridium values.

The central Witwatersrand reefs carry very small quantities, while those of the West, the Far West Rand, Klerksdorp and the Free State are all much poorer in osmiridium than those of the East Rand.

3 : THE MERENSKY REEF HORIZON AT THE UNION AND RUSTENBURG MINES

Geological sections through the Merensky Reef zone at the Union and Rustenburg mines are shown in Fig. 2. 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 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, which is again overlain by what is known as the 'Bastard' reef, a band of pyroxenite almost identical with that overlying the Merensky Reef. It sometimes carries low platinum values and a thin chromite seam has been known to occur at its lower contact. The hangingwall rocks of this reef show the same sequence of anorthositic gabbro and mottled anorthosite.

The footwall rocks of the Merensky Reef consist of a thin zone of almost pure anorthosite on which the footwall contact of the reef (normally a thin chrome seam

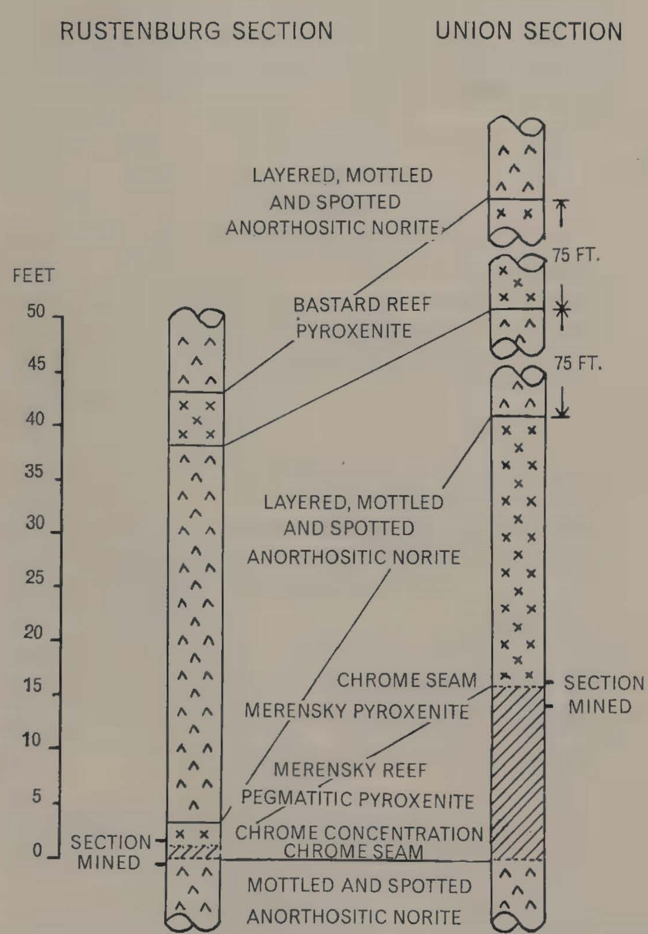


Fig. 2

¼ in to 1 in thick), rests with an extremely sharp contact. This thin band of white anorthosite (usually an inch or less in thickness) is followed by a zone of spotted anorthositic norite containing mottled phases.

Although considerable variations in the thickness of the individual layers within this zone are found around the outcrop of the Merensky Reef, the succession and individuality of each of the bands remain amazingly similar around both the eastern and western limbs of the Complex.

In the Rustenburg mine the Merensky platinum reef averages about 1½ ft in thickness and reaches two ft only in certain limited areas. This narrow width allows the whole pegmatitic pyroxenite together with a portion of both the hangingwall and footwall to be included in the stoping width mined.

At the Union mine the pegmatitic pyroxenite averages some 15 to 18 ft in thickness. This thickness shows a gradual decrease from over 20 ft in the north-eastern to less than 10 ft 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 value 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 value distribution giving a very even tenor of values. The platinum is partly in the form of native platinum, invariably alloyed with iron (ferro-platinum) and partly as the arsenide and sulph-arsenides, these latter being in intimate association with sulphides of iron, nickel and copper.

At the Rustenburg mine the platinum arsenides 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, with ruthenium, rhodium, iridium and osmium in descending order.

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 confined to the interstitial silicate minerals.

The nickel and copper produced bear a constant ratio to the platinum recovered, but this ratio is higher at the

Rustenburg mine than at the Union mine. In both mines the ratio of nickel to copper is 1.8 to 1.

A characteristic of the Merensky Reef is the regularity of the dip in both mines. At the Rustenburg mine the dip averages 9°30′ to the north, while at the Union the dip is 23° to the south east.

A small number of post-Bushveld dykes occur, but generally cause no appreciable disturbance. One larger dyke, a foyaite porphyry, cuts through the eastern portion of the Rustenburg mine.

Faults are also rare and of small displacement.

The only structural disturbances of any importance are local dome-structures known as 'koppies' and local depressions known as 'potholes'.

In a koppie, a roughly circular dome-shaped contact of the footwall anorthosite projects upwards eliminating the hangingwall beds to an extent depending upon the height of the dome. It may thin or cut out the Merensky Reef. In the case of a very large koppie even the Merensky pyroxenite may be cut out. At the contact between the footwall and the hangingwall rocks there is invariably a thin chrome seam. The dome-structure is not due to updoming of the footwall rocks as their layering remains unfolded.

No satisfactory geological explanation for the structure has as yet been put forward.

Over a 'koppie' the platinum values fall commensurate with the thinning of the reef, but values continue to be associated with the contact chrome seam. They are, however, generally unpayable. They are rarely as large as 100 ft dia and are not encountered very often.

'Potholes' consist of roughly circular areas in which the footwall beds appear to have been hollowed out to permit of the hangingwall beds being folded or faulted into a basin-shaped structure. The footwall beds are unfolded.

In the Rustenburg mine the average 'pothole' is some 100 ft dia and its depth a few ft. A few larger 'potholes', up to 500 ft dia and up to 75 ft in depth have been found.

At the Union mine the 'potholes' are larger, from 200 to 700 ft dia with depths correspondingly greater.

The reef at the bottom of most 'potholes' is flat but not quite parallel with the undisturbed reef. A considerable proportion of 'potholes' carry normal payable values; only a few have been recorded as unpayable.

At the rim of most 'potholes' the reef suddenly assumes a steep dip and pinches out, leaving only the Merensky pyroxenite to follow the contact. Very frequently the contact then becomes overturned and only near the floor of the 'pothole' comes rapidly back to a dip nearly parallel to that of the reef outside the 'pothole'. At the floor the Merensky Reef reappears and continues over the complete floor.

A small proportion of 'potholes' have a second, very much smaller, 'pothole' structure extending below the floor of the main 'pothole'.

A satisfactory geological explanation of the cause of a 'pothole' is lacking.

Owing to their small displacement and general payability, the Rustenburg mine successfully mines out the majority of its 'potholes' in the course of normal stoping operations. In the Union mine the much greater depth and consequent additional development has caused the exploitation and mining of 'potholes' to be deferred to a later date.

4: MINING METHODS AT THE RUSTENBURG SECTION OF THE RUSTENBURG PLATINUM MINES, LIMITED

The ground held under mining title by the Rustenburg Platinum Mines, Limited, at the Rustenburg Section consisted originally of portions of the farms Klipfontein No. 538, Kroondal No. 177 and Waterval No. 537. An additional lease area was recently granted to the company by the State over the remaining portions of the Klipfontein and Waterval farms.

The western boundary of the mine adjoins the Rustenburg townlands. The southern boundary runs slightly to the south of the reef outcrop except at the eastern end of the property where a large area of ground is available for reduction works, slimes dams, workshops, offices and European housing. A similar, but smaller area, is available for European housing on the Waterval farm.

The Pretoria—Rustenburg branch line of the South African Railways passes over the property and two sidings are available for use by the company.

The reef mined is the Merensky Reef of the Bushveld Igneous Complex, which, on this mine, consists of a thin chromite-rich layer averaging $\frac{3}{4}$ in in thickness with, above it, a layer of coarsely crystalline felspathic pyroxenite about 12 in in thickness. The lower 9 in of the Merensky pyroxenite above the reef and the upper 9 of the footwall norite 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 in.

The economic metals produced after reduction of the ore and final refining are:

(i) The platinum group metals.

(ii) Gold.

(iii) Nickel.

(iv) Copper.

These are contained in gravity concentrates and converter matte which are the end products of the mine's reduction plant.

The strike of the reef is east-west and the dip is consistently 9°30′ to the north. The reef strike tends to turn slightly northwards on the western half of the property in conformity with the elliptical shape of the Complex.

Over the life of the mine to date it has been found that whilst appreciable variations in value may occur from one sampling section to the next, 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. Serious, or even minor, faulting is unknown and the only general geological disturbances met with are the 'koppies' and 'potholes' described earlier.

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 enlargement of the property with the recent acquisition of the additional mining lease has not upset the balance of the overall layout. The two halves of the old area now constitute roughly the inner quarters of the new area, so that the length of strike can be divided up into four nearly equal lengths for exploitation from four incline haulage and vertical shaft systems.

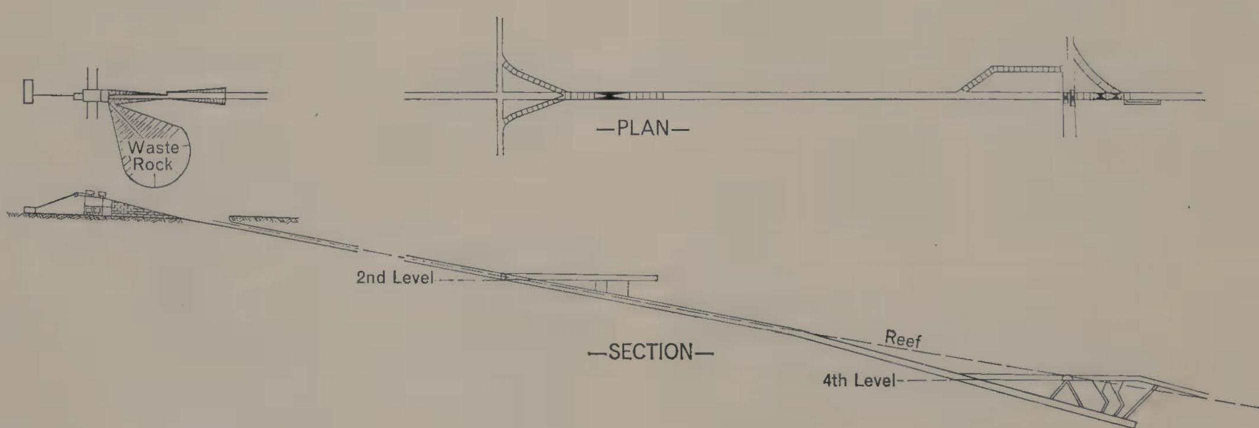
1. Shaft systems

In the foreseeable future, working will extend downwards to the 12th or 13th levels. 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. In the Waterval area a sub-incline shaft will follow the vertical to serve this area down to the 12th level.

In the early days of the mine only the upper 700 ft of reef measured on the incline were worked. This zone, consisted of oxidized ore and a gravity concentrate caught over corduroy cloths and dressed on shaking tables, was extracted. A further small amount was recovered as a flotation concentrate. A large area of the original mine in this zone was worked out by resue stoping methods. Mining operations were carried out from surface winzes spaced approximately 500 ft apart on strike.

(*a*) *Incline haulages.* When a suitable reduction plant had been installed to treat sulphide ore, every fifth winze was equipped as an incline haulage on the plane of the reef and carried down to below the 4th level to an incline depth of 2,300 ft. These haulages were sited at from 2,200–2,500 ft apart and were connected to one another on the 2nd and 4th levels at incline distances of 1,000 and 2,000 ft from surface.

A strike of 18,000 ft in the old mine was served by



RUSTENBURG PLATINUM MINES LTD.
RUSTENBURG SECTION
E.6.E. INCLINE

Fig. 3

these inclines. At a later stage a series of sub-inclines were sunk to serve 15,000 ft of strike down to the 8th level.

Fig. 3 shows the general layout of an incline and sub-incline haulage. The haulage from the surface follows the plane of the reef to a point 375 ft above the 4th level where the dip of the haulage is increased to 14° to obtain box capacity for the 4th level tonnage. The incline is continued to connect up with the ore and waste boxes from the sub-incline. An airway is put up from the bottom of the upper incline to connect with the sub-incline to provide for through ventilation past the loading points.

The 2nd level drives lead into an X-cut over the incline haulage where ore and waste can be tipped into a bin divided into two by a wall.

The sub-incline starts above the reef plane and dips down to it over a suitable distance, after which it follows the reef to a point above the 8th level where it is steepened to 14° to allow for the 8th level reef and waste bins. The ore and waste boxes of the intermediate levels between 4 and 8 are constructed on the same layout as for the 2nd level.

Under normal conditions the stoping area served by the upper incline is more or less exhausted before much tonnage from the sub-incline is handled by the upper incline hoist.

i *Hoisting arrangements.* The type of hoist in use is a 150 h.p. single drum hoist with a rope speed of 400 ft per minute. Two side discharge hoppers are connected to the rope and these between them carry a load of 10-11 tons per trip. Suitable safety devices, approved by the Inspector of Mines, are fitted to the hoppers and rope to prevent a break-away in the event of breakage of links or coupling pins.

The hoist is driven by a Native and hoisting operations are controlled by Natives.

Material in suitable trucks can be handled by means of a long sling attached to the lower hopper. Each reef and waste box on these inclines has its delivery chutes set at the same centres as the hoppers so that these may be filled simultaneously by two operators.

The surface bin is divided into ore and waste compartments. Ore is transferred to the reduction works by locomotive haulage and the waste is hoisted by self-dumping skip to the waste dump.

Brow bins on the surface are provided for sand and crushed stone to facilitate the transport of such items into the mine.

No persons may be transported on these inclines.

ii *Capacity of incline haulages.* This type of incline has worked very satisfactorily and economically. It has been found possible to hoist through an upper and sub-incline system up to 800 tons per 24 hours and to lower material to satisfy the needs of the area from which this tonnage came.

Similar incline and sub-incline haulages have been sunk on the eastern extension area of the mine and to the west of the Waterval Vertical Shaft to serve down to the 7th level.

(*b*) *Vertical shafts.* There are three vertical shafts in operation on the property. A fourth vertical will be sunk shortly. All three shafts are comparatively shallow, their depths being 500 ft, 780 ft and 1,500 ft from surface respectively.

i *West Vertical Shaft.* This 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 serving an area of ground between the 4th and 8th levels over a strike of 8,000 ft. Ore in this area is almost exhausted and this shaft will shortly become an upcast ventilation shaft.

ii *Waterval Vertical Shaft.* This is a 15-ft dia concrete lined shaft which serves the Waterval stoping area. Fig. 4 shows a plan of the arrangements of shaft buntons, etc. Steel brackets of welded construction are bolted to the periphery of the shaft at vertical intervals of 7 ft and support 9 in × 6 in timber buntons. Steel bearer sets are provided at intervals down the shaft and the longer centre bunton has the support of a studdle from bunton to bunton near the shaft centre. Wooden guides are 6 in × 4 in. Cross timbers, as indicated, support air, water and pump columns.

Five-ton skips, interchangeable with double decked cages, are installed. The double-drum hoist is powered by a 750 h.p. motor and the maximum winding speed is 750 ft per minute.

Material for this area is lowered through an old incline haulage from surface to the 4th level and by a newly installed material winze to the lowest level of the vertical shaft.

iii *Central Deep Shaft.* This is a four compartment shaft of 'square-rectangular' section, measuring 24 ft 3 in × 11 ft 3 in inside timbers. Each compartment is 5 ft 6 in × 11 ft 3 in. Wall plates, end plates, dividers and corner posts are 9 in × 9 in. Inner studdles are 10 in × 4 in and guides are 8 in × 4 in. The vertical distance between set centres is 7 ft. (See Fig. 5.)

Single decked cages capable of carrying 40 persons are installed in the two south compartments. These cages are sufficiently large to accommodate trucks of explosives, mining timber etc. Skips are of 5 ton

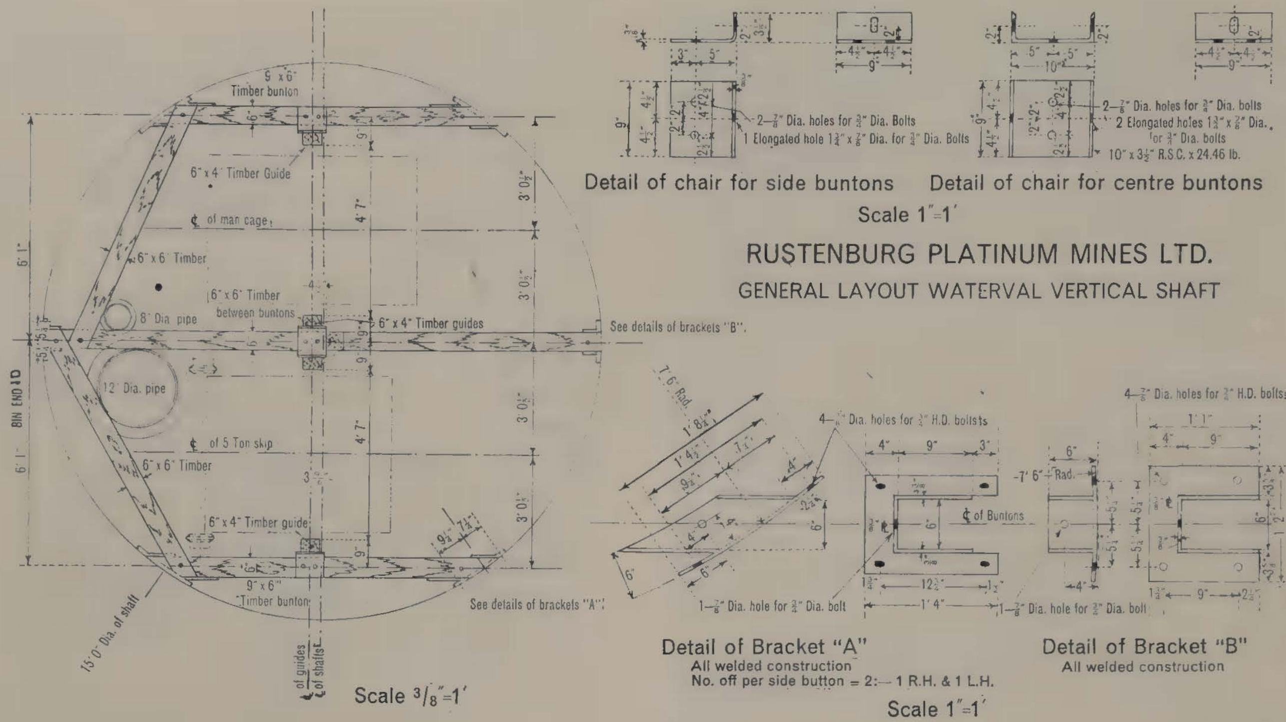


Fig. 4

capacity and are interchangeable with double-decked cages capable of carrying 40 persons.

The ore and waste pass system is shown in Fig. 6. A 'Rayner' dust filter plant is installed at the ore and waste tips on each working level.

In order to bring this shaft into use as soon as possible after sinking, a temporary loading point was constructed on the 10th level station. A conveyor tunnel was put in from the shaft above the 10th level to connect with the ore and waste pass systems. The conveyor delivered ore or waste into a small flask above the measuring chutes situated on the 10th station brow.

The lower shaft stations, ore and waste pass systems and the loading station at the shaft bottom were then completed at leisure.

The choice of the 'square-rectangular' shaft section was made for two reasons, viz., 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.

The man-cage hoist is powered by a 700 h.p. motor and the maximum rope speed is 1,500 ft per minute.

The rock hoist is powered by an 850 h.p. motor and the maximum rope speed is 1,500 ft per minute.

iv *Surface tipping arrangements at vertical shafts.* 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. This arrangement allows of a comparatively small collar set and light headgear, whilst the size of the ore bin over the haulage can be made to any required capacity.

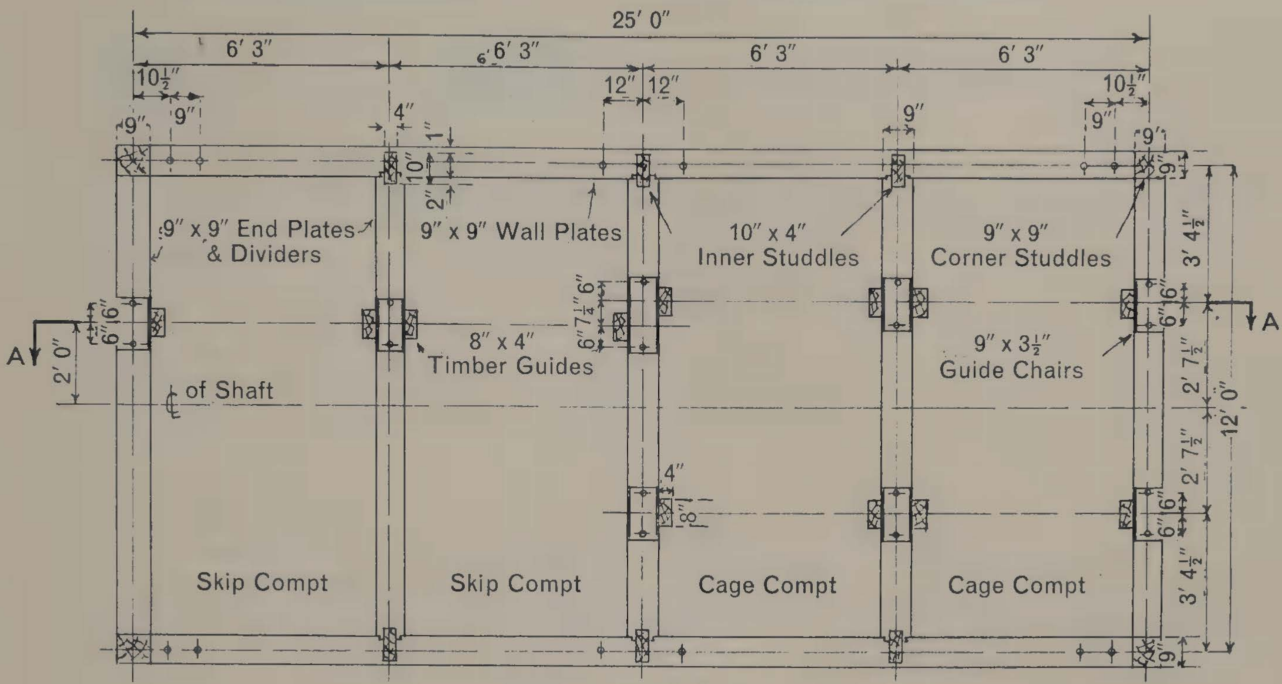
v *Waterval sub-incline shaft.* A 15° three compartment sub-incline shaft is now in the course of sinking from the 7a level to the 13th level in the Waterval area. This shaft will be equipped with two ten ton skips. Ore and waste will be delivered to their respective passes at the top of the incline and trammed to the Waterval vertical shaft tips by overhead electric locomotive.

2. *Underground haulage*

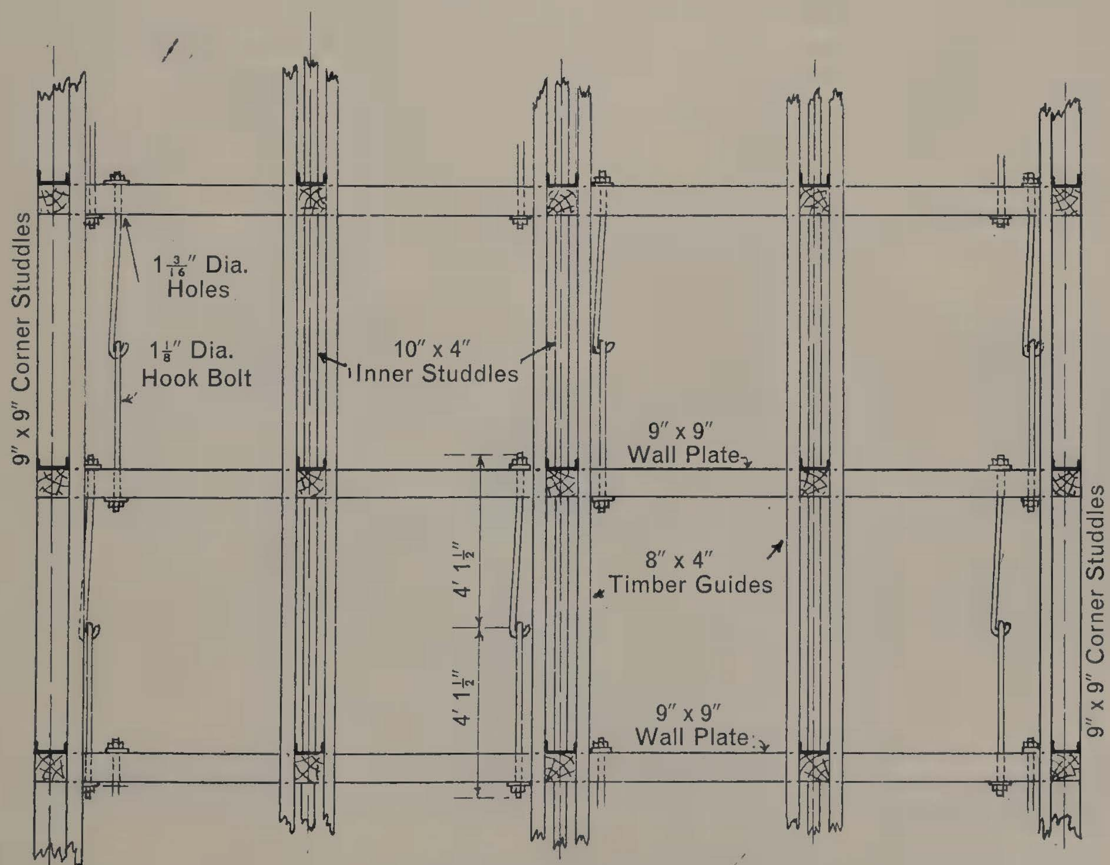
Generally, except in the eastern extension area, the haulage of ore underground is by overhead electric locomotives pulling trains of 4 ton hoppers through footwall haulages. Fig. 7 shows the normal section through a footwall haulage.

In the eastern extension area, 2 ton diesel locomotives are in use. Fig. 8 gives a typical layout of a reef drive, loop and stope scraper loading point in the upper levels of the mine.

All footwall haulages are roof bolted at 4 ft intervals



PLAN ON TIMBER SETT.



SECTION THROUGH A-A

Fig. 5

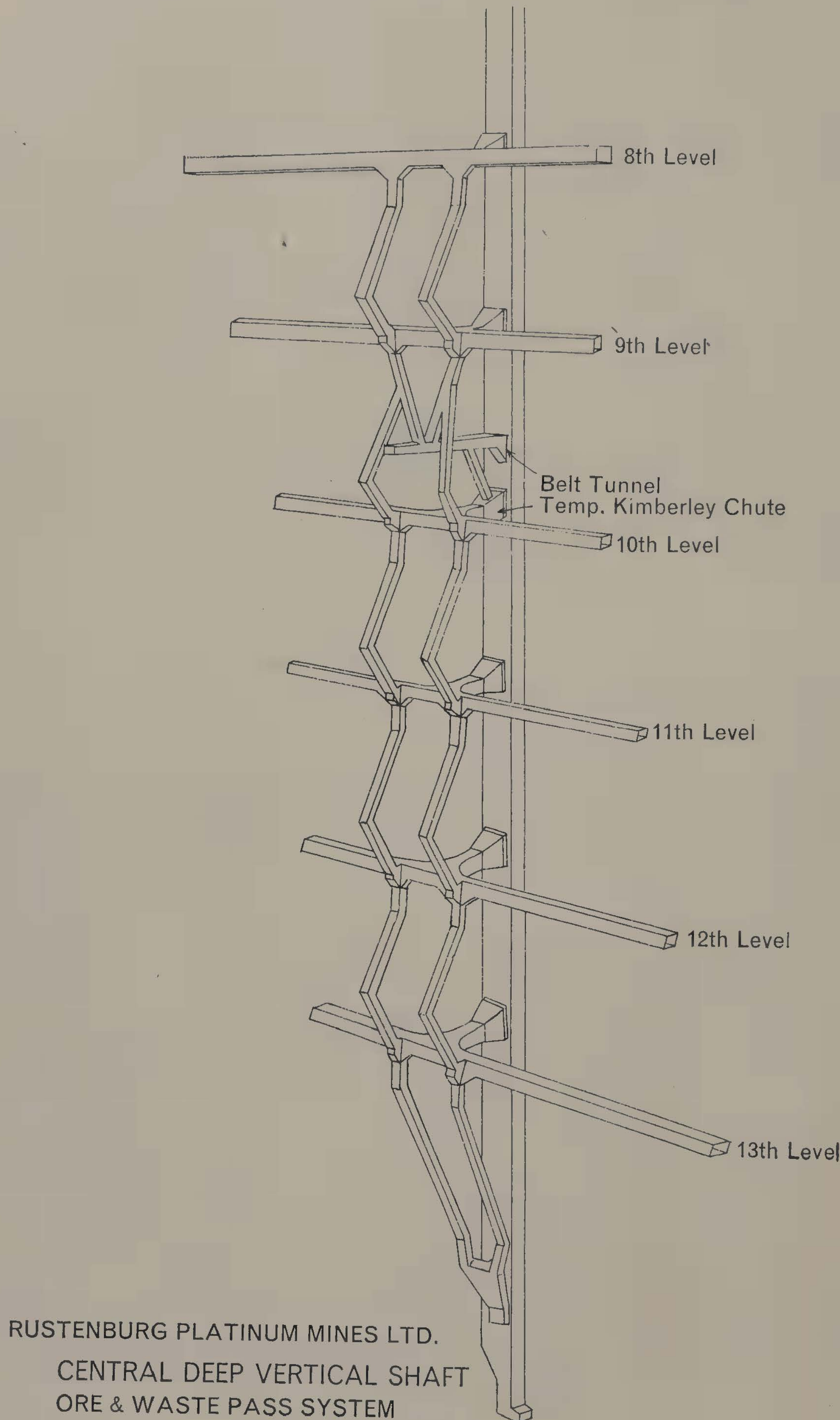

8th Level
9th Level
Belt Tunnel
Temp. Kimberley Chute
10th Level
11th Level
12th Level
13th Level
RUSTENBURG PLATINUM MINES LTD.
CENTRAL DEEP VERTICAL SHAFT
ORE & WASTE PASS SYSTEM

Fig. 6

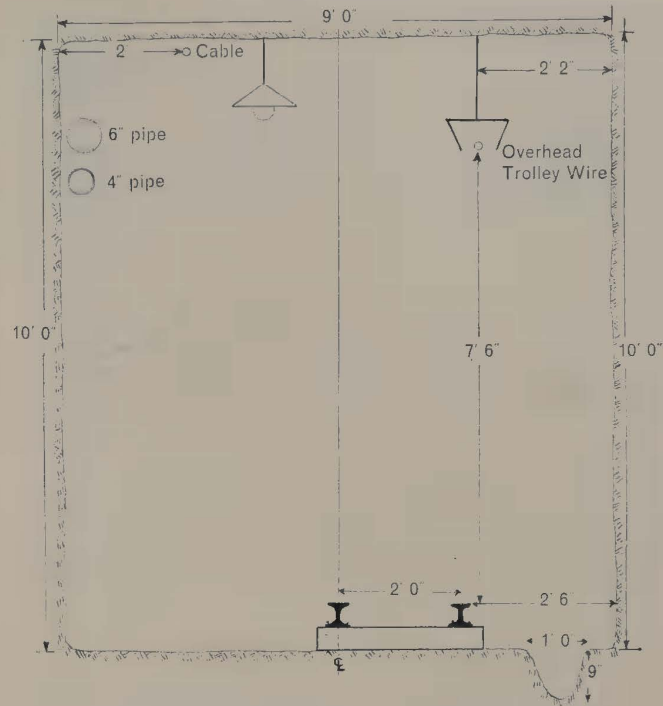
with a pattern of three roof bolts. The use of roof bolting in haulage ways has practically eliminated the use of timber or timber sets and has resulted in clear and safe conditions in all haulages.

In portions of the 2nd and 4th levels in the eastern extension area, drive pillars are being cut and, where necessary, roof bolts put in to maintain conditions in these drives similar to those in footwall haulages. At depths down to 350 ft below surface there is no likelihood of trouble from the failure of these pillars during the working life of the haulage. The pillars will be removed when the drives are no longer required. In future the normal footwall haulage will take the place of reef drives in this area.

Footwall haulages are carried at a distance of 25 ft in the footwall normal to the reef. A distinctive marker exists at this point which simplifies the maintenance of the direction of the haulage in relation to the reef.

X-cuts to raise—winze connections are set off at intervals of 500 ft on strike. An ore box is put up from the footwall haulage to intersect the raise—winze connection. This deals with the ore broken in the stope above the level. A second and shorter boxhole is put up from the X-cut to the reef connection to deal with development rock from the winze below and, at a later stage, ore from the upper portion of the lower stope which can most conveniently be scraped up dip into this box.

An inclined travelling way leads up into the stope connection for the passage of men and material. (See Fig. 9).

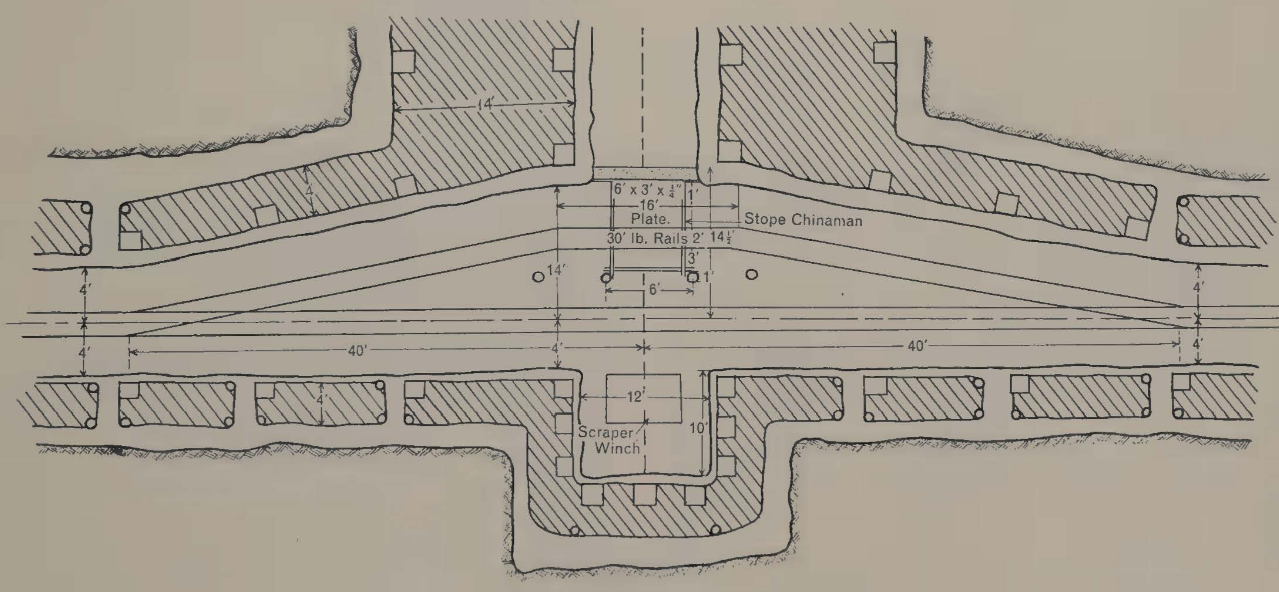


RUSTENBURG PLATINUM MINES LTD.

SECTION THROUGH FOOTWALL HAULAGE.

CENTRAL DEEP EAST-WEST FOOTWALL HAULAGES.

Fig. 7



PLAN OF REEF DRIVE SHOWING SCRAPER & CHINAMAN

Fig. 8

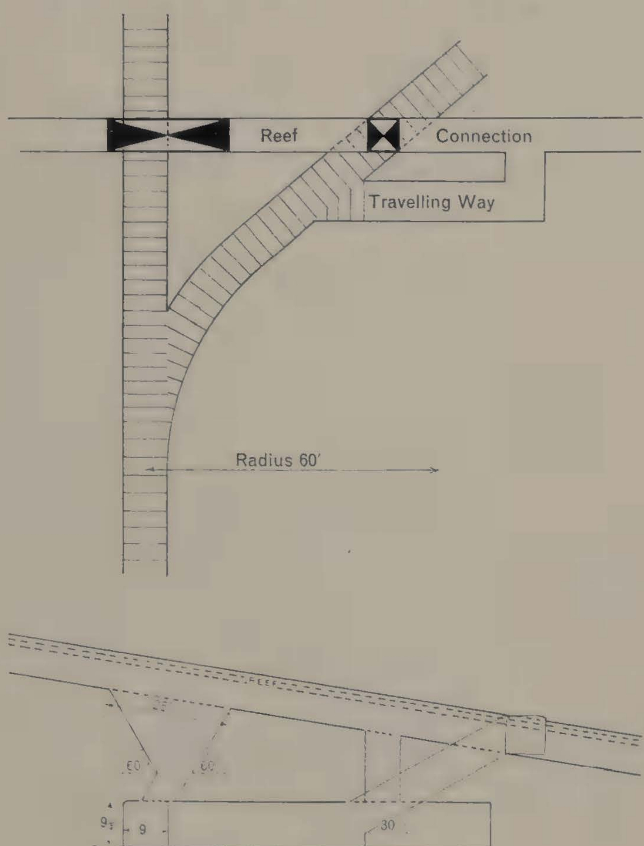
In the upper part of the mine, drives and haulages were set off at varying intervals to suit immediate local requirements. There has thus been a variation in 'back' length. From below the 8th level, 'backs' are being standardized at 800–850 ft, which length has been proved to give the best scraping conditions in the centre scraper gullies of stopes.

3. Stoping

With only a very narrow and not very highly payable reef to mine, 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 standardized procedures comparatively simple.

Regularity of value has to a 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. Even the lack of stratification planes helps to ensure that a hangingwall and footwall can be cut so as to maintain an average mined stoping width of 28.5 in.



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LAYOUT OF F.W. HAULAGE, X-CUT, TRAVELLING WAY & STOPE BOXES ON RAISE-WINZE, CONNECTION

Fig. 9

The only flies in the ointment are the 'koppies' and 'potholes' mentioned earlier.

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 ft 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 the face scatter-pile. The ore remaining is then shovelled 30 ft down dip and 10 ft 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.

The height of the panel gully track should not fall below 42 in or exceed 54 in and all rock broken in the excavation of the gully footwall is blasted and packed separately, including the fines.

All sorted waste is packed finally into stone walls to a definitely laid down pattern. These stone walls are used to provide 'area' support to a hangingwall which can be treacherous if left unsupported as tiny hair-like cracks or calcite-filled seams run in all directions through the reef horizon.

These walls also serve as ventilation walls which force the ventilating current on to the working face and into the central scraper gully where most of the dust generated in mining operations originates. The dust is diluted by the directed ventilating current.

Waste filled timber packs 2ft × 2 ft and, in certain instances, 2 ft × 2 ft mat packs provide the primary support of the stope. A minimum of three props is required per panel between the last line of packs and the face as an added safety precaution. (See Fig. 10)

(*a*) *Opening up of stopes.* At one time the raise-winze connection was opened up for stoping by ledging. During this operation it was found impossible to control the amount of footwall waste which broke off before the ledge was properly established. This led to serious dilution of the ore sent to mill. During the past five years it has become the practice to cut pillars on the connection in opening up work except in the vicinity of the

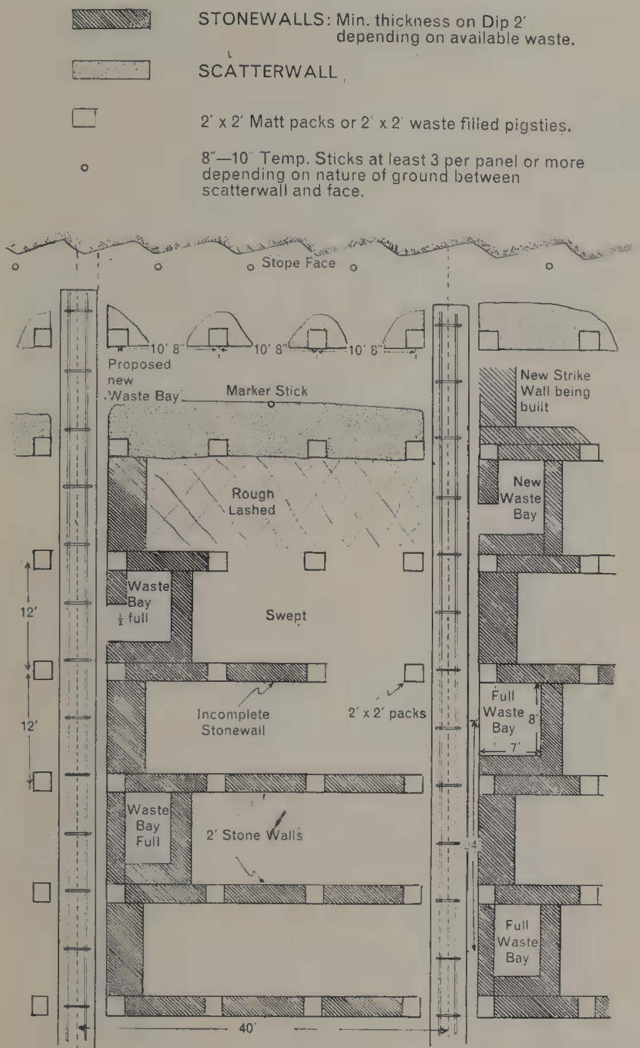


Fig. 10

footwall haulage and X-cut where ledging is still carried out.

It may appear expensive to put in small drives on strike and to parallel the original connection on either side by raising, but it has been found more economical to do so for the following reasons:

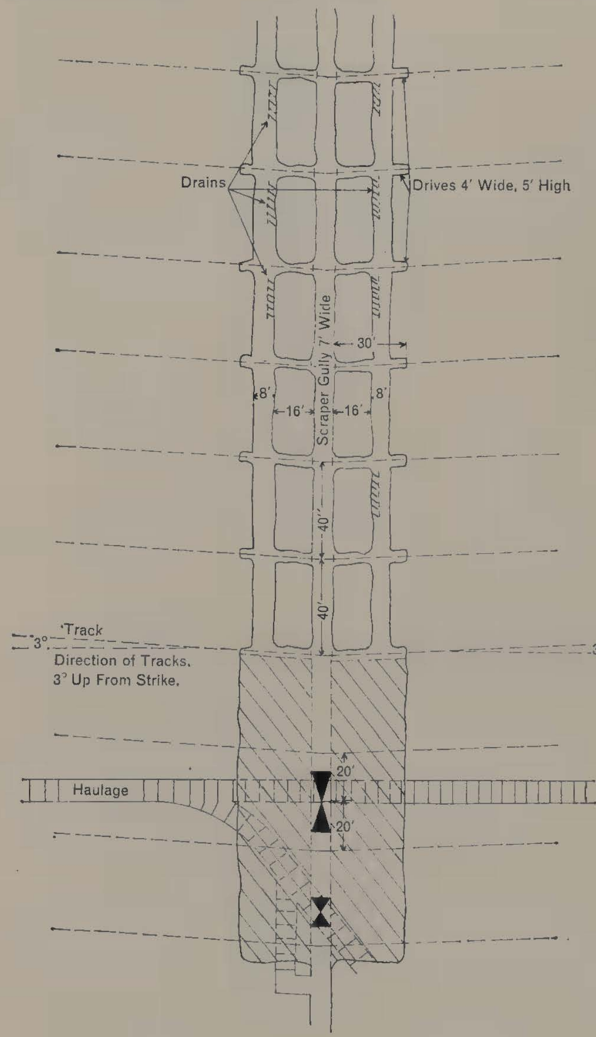
(i) The pillar drive and raise are carried at normal stoping width,

(ii) The footwalling of the drive in waste norite can be done at a definite time and the broken rock treated as waste, and

(iii) most important, the time taken to open up a connection for mining is reduced by half as compared with ledging method. Fig. 11 shows the standard method of pillar cutting.

It is possible for a miner on opening up work to break between 800 ft and 1,000 ft in pillar raising and driving per measurement month. Three drilling crews of two Natives can each drill over three to five ends per shift using 42 in steel.

(*b*) *Drilling in stopes.* Standard $2\frac{5}{8}$ in jackhammers weighing 65 lb are used throughout the mine. Two holes per bench are placed to cut to a hanging and footwall. Four hole and six hole benches have been used in places to straighten out a face when necessary. Each hole is carefully marked by the miner-in-charge, using a hole director. The start of the hole and its 'pitch' are marked in chalk by the stoper according to their positions on the face given by the hole director. The drilling crews use a similar director and carefully follow the chalk marks when starting each hole.



RUSTENBURG PLATINUM MINES LTD.

STANDARD PILLAR CUTTING IN STOPE CONNECTIONS

Fig. 11

The upper hole on each bench is, as a rule, the director hole for burden. The lower hole is aligned with the upper hole and the director used only to give the correct 'pitch'. Stoping holes are marked to start and end at definitely prescribed distances above and below the upper reef contact so that the hole sockets are left at definite distances above and below this contact.

Varying burdens are used depending upon the nature of the ground, but the average burden may be taken as 18 in. The average angle of break is found to be 135°. In order to ensure maximum breaking it is the rule that, when drilling 42 inch holes, the length of the bench should not be less than 5 ft 6 in.

(*c*) *Drill steel.* $\frac{7}{8}$ in hexagonal drill steel with a 30 mm tungsten carbide chisel bit insert is used. The rock is not as abrasive as the Witwatersrand quartzite and a drill life in excess of 800 ft is normal. There is comparatively little loss in gauge during the life of the steel which is one of the reasons which led to the general adoption in August 1959 of the 30 mm bit in place of the 32 mm bit.

The other reason was the greater penetration speed of the smaller gauge bit which resulted in a net increase of 20-25 ft drilled per machine shift.

Failure of the tungsten carbide inserts causes approximately 75% of drill steel losses. Stem failures account for the balance.

All drills are equipped with a rubber collar and a rubber buffer collar. The dimensions of the rubber collar and steel ferrule before crimping are:

Rubber collar—$1\frac{1}{2}$ in outside dia and $1\frac{1}{2}$ in long.

Steel ferrule—$1\frac{3}{4}$ in outside dia, $1\frac{1}{2}$ in long with $\frac{3}{32}$ in walls.

The dimensions after crimping are:

Steel ferrule—$1\frac{5}{8}$ in outside dia, $1\frac{1}{2}$ in long with $\frac{1}{8}$ in walls. (See Fig. 12).

The process of rubber collaring is as follows:

The rubber collar is slipped over the drill to a position $6\frac{1}{4}$ in plus the length of the rubber buffer collar from the end of the stem. The steel ferrule is placed over the rubber collar. The ferrule is then crimped down to a smaller dia by means of an Ingersoll-Rand '34' collaring machine, clamping the rubber very tightly to the drill. A certain amount of rubber squeezes out at either end of the ferrule. The excess rubber on the shank end is cut off and the rubber buffer collar forced up against the ferrule collar. The purpose of the rubber buffer collar is to prevent metal to metal contact between the ferrule and the machine chuck which would tend to damage the chuck and displace the collar.

The approximate time taken to put on a collar is three minutes. With a daily circulation at present of over 700 drills some five collars are replaced per shift due to slipping of the collar and fifty put on to new replacement steel. This work is largely done by Natives under the supervision of Europeans.

(*d*) *Footage drilled.* The footage drilled per machine shift has increased at a steady rate over recent years from about 160 ft in 1950 to an average of 220 ft per shift during 1958. During the last six months of 1959 the average footage drilled amounted to 270 ft per shift.

The attainment of this steady increase has been due largely to increasingly good supervision on the part of shift bosses and miners and the sudden jump in 1959 is attributed to the following two changes in conditions which took place during that year:

(i) the use of a delay fuse to ignite the igniter cord, and

(ii) the introduction of 30 mm gauge drill steel in place of 32 mm gauge.

The use of the delay igniter fuse to fire the igniter cord allowed a common lighting-up time to be introduced throughout the mine and increased the average

STANDARD RUBBER COLLAR FOR DRILLS

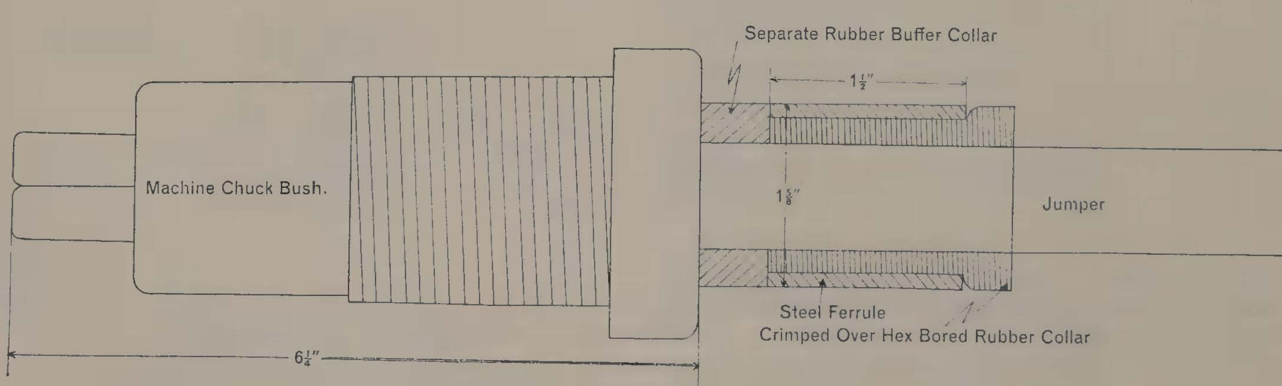


Fig. 12

length of the drilling shift by 15-20 minutes. This resulted in an additional 16 ft being drilled on average by every stoping machine.

The greater penetration speed of the 30 mm gauge steel led to a further increase of over 25 ft per shift.

In a number of stopes an average of over 300 ft has been maintained by the drillers over several months. In such cases the gangers-in-charge appear to be better-than-average organizers.

This high drilling rate demands a long length of face per machine, and 300-350 ft of face must be allocated to each machine if the full benefit of high footage drilling is to be attained. At least two face panels must be left unblasted per machine each day. The face on these panels should be clean, holes marked and ready to drill after inspection and watering down takes place at the start of the next day's shift.

(*e*) *Breaking efficiency*. Table I gives an indication of the increase in breaking efficiency which has been attained over the past six years. In 1951 full standard procedures were laid down for every underground operation. In 1954 a method of supervision by shift bosses was introduced which, it is felt, has been of inestimable value in maintaining these standard procedures. The value of this form of supervision has been gradually proved as more and better holes were drilled in stopes. The increase in the square fathoms broken per machine shift from 2.85 in 1953 to 4.17 in 1958 and 4.77 in 1959 is important from the cost point of view.

TABLE I

Year	1950	1951	1952	1953	1954	1955	1956	1957	1958	1959
Square fathoms broken per machine shift	2.66	2.71	2.76	2.85	3.07	3.03	3.13	3.51	4.17	4.77

If the 1953 figure of 2.85 fms/machine shift had been the 1959 average it would have required 68 more machines per 100 actually working to produce the fathomage broken in that year. With two stoping machines in operation per average stoping contract 84 European miners would have been required for every 50 presently employed.

The normal stoping gang consists of roughly 35 Natives, thus 34 more stoping gangs would have to be manned by 1,190 more Natives as against the 1,750 now employed per 100 machines.

The additional faces worked would have increased the official supervisory complement and additional survey and sampling staff would also have been required. The extra jackhammers in operation would have required the provision of additional compressor capacity.

It may be added that the original target aimed at in 1953-1954 was a breaking rate of 3.25 square fathoms per machine shift.

(*f*) *Blasting in stopes*. Blasting in stopes is carried out by the ignition of 24 in and 30 in capped fuses by igniter cord. The igniter cord is ignited by means of a 30-minute delay fuse, which, in effect, gives a common lighting-up time throughout the various sections of the mine.

The shorter length of capped fuse is placed in the leading hole, usually the top hole, on each bench. Fast igniter cord, burning at 1 ft per second is used. As the stopes are generally dry the igniter cord is knotted to the fuses.

The cost of the delay fuse is 1/6d and the cost of igniter cord is just under 1d per ft. In order to ensure against failure of the charges to ignite only forty holes approximately are connected to one delay fuse. Where gaps between holes in excess of 20 ft occur in the face another delay fuse is used.

Comparatively few ignition failures are recorded.

4. Development

The following dimensions of development ends have been adopted as suitable to our requirements:

Incline haulages	—	9 ft high × 10 ft wide.
Main X-cuts	—	10 ft high × 12 ft wide.
Footwall haulages	—	10 ft high × 9 ft wide.
Reef drives	—	9 ft high × 8 ft wide.
Raises and winzes	—	8 ft high × 7 ft wide.
X-cuts to stopes	—	9 ft high × 8 ft wide.

Standard drag rounds are drilled applicable to each size of end. The length of hole drilled, except in the shorter cut holes, is 7 ft. The drag cut is drilled using two hole directors. Director 'A' gives the directions of the top pair of cut holes. Director 'B' is a 7 ft length of drill steel with a rubber collar and washer fixed 4 ft from one end. (See Fig. 13).

The starting points of the drag cut holes are marked out on the face in the form of a V at intervals of 10 in apart vertically and at varying distances on either side of the centre line depending upon the size of the end. After the top pair of cut holes has been drilled according to director 'A', director 'B' is placed in one of these holes and the hole below it drilled so that the jumper is held in a line between the collar of the lower hole and the end of director 'B' on the inner side.

The following hole of the drag cut is directed by placing director 'B' in the second hole and starting the third hole from the collar mark using the end of director 'B' as a guide. This results in a double set of holes drilled fan-shaped in the form of a V, all correctly burdened in relation to one another.

The grid for the remaining holes in the round is marked out on the hanging wall of the end in crayon and the holes drilled according to the various lines marked. All holes drilled are related to the centre and grade lines of the end. In a development end on dip of reef the round is marked out relative to a direction line and the normal upper reef contact except where 'potholes' and 'koppies' are encountered when a grade line at normal reef dip is followed. (See Fig. 14).

Jackhammers are usually hand held, the drillers using an easily rigged platform of pipes and planks from which to drill the majority of the holes in the round.

Until recently a machine crew of handle-and spanner-boy was used per end. It has been found that better drilling results can be obtained by using this crew of two, each drilling a machine. The two-man crew collars all the holes in the end with one machine working together. Thereafter, they complete the drilling of half the round, each using a machine.

Each member of the crew uses much less energy in drilling half the round separately than both consumed in drilling the whole round together. As a result it is possible for the two-man crew to drill further holes in an additional working place in the same shift with a consequent gain in the footage broken per shift.

In normal development work on the flat or on the plane of the reef the use of rigs or airlegs has no attraction for the drillers. Their use does, in fact, add to the actual time taken to drill over the round. In steeply inclined work airlegs are used.

CUT DIRECTORS TO BE USED IN DRILLING OF
STANDARD DRAG-ROUND CUT

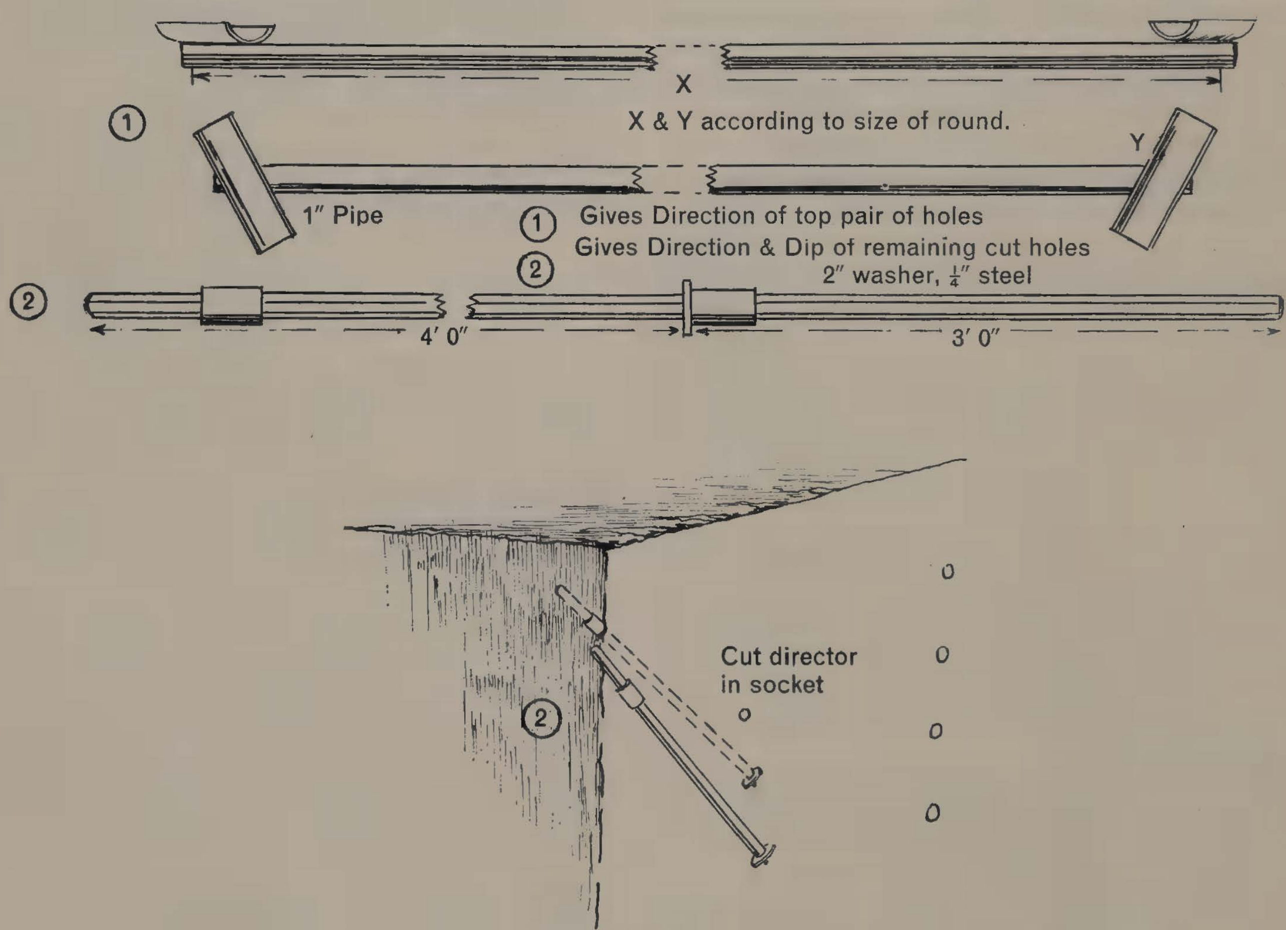


Fig. 13

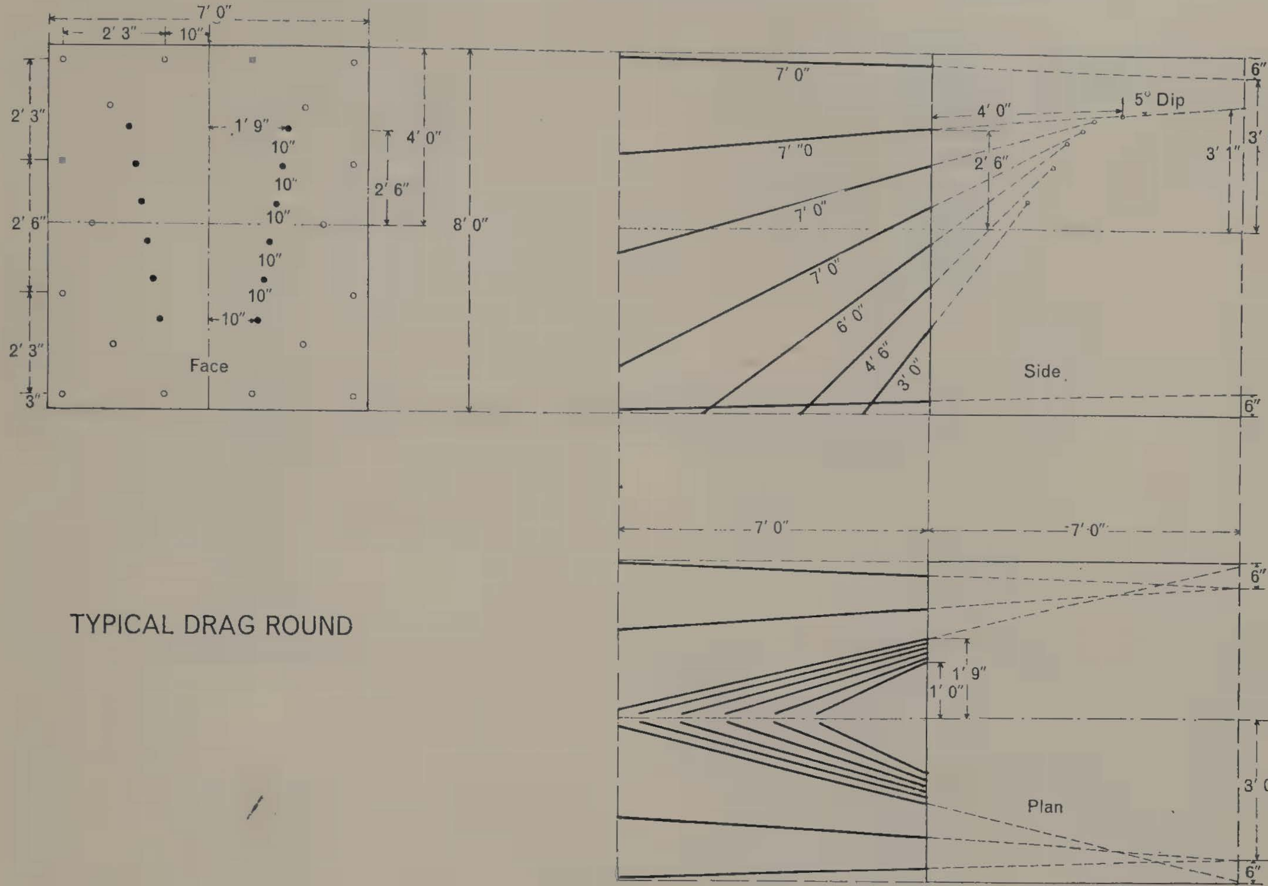


Fig. 14

(*a*) *Drill steel.* The drill steel used is $\frac{7}{8}$ in 'hexagon' with tungsten carbide chisel bit inserts. Three drills are used per 7 ft hole—a starter, 36 in in overall length and of 36 in mm gauge, a second, 60 in in overall length and of 34 mm gauge and a third, 96 in in overall length and of 32 mm gauge. All drills are rubber collared. On long development steel a second and shorter rubber collar and ferrule is added to the single rubber collar.

Footages drilled by the two-man drilling crew are normally in excess of 230 ft per shift. An average advance of 6 ft per day is obtained per machine crew.

A development contract under a European miner usually consists of three machine crews with three or four ends available for drilling daily. Experienced developers break between 450 and 525 ft per measurement month.

(*b*) *Development blasting.* In blasting in development work, suitably timed 6 ft fuses are ignited by means of igniter cord. The igniter cord is ignited by a 30-minute delay fuse. In order to synchronise the action of the water blast with the ignition of the round, a 30-minute delay fuse and igniter cord is used to operate a special form of waterblast. (See Fig. 15).

(*c*) *Development cleaning.* Development ends are cleaned on night shift. As a rule cleaning is done by hand lashing into 1 ton cars or 2 ton hoppers except in the case of raises which are equipped with scrapers. Hand cleaning limits the length of round drilled to 7 ft.

(*d*) *Ventilation of development ends.* Normally a development end is ventilated by means of an exhaust fan of suitable size with 16 or $22\frac{1}{2}$ in galvanized ventilation piping. The air intake end of this column must at all times be within 50 ft of the face. A venturi blower with a length of 14 in canvas tubing delivers fresh air to within 2 ft of the face. A minimum overlap of 18 ft between the end of the exhaust column and the intake to the venturi must be maintained.

5. *General mine ventilation*

The long length of strike worked and the comparatively shallow depth of the workings necessitate the provision of a number of main ventilating fans to provide adequate air in the mine. At present eight 54 in and two 40 in propeller-type fans, situated at various points on the surface, exhaust a total of about 600,000 cu ft per

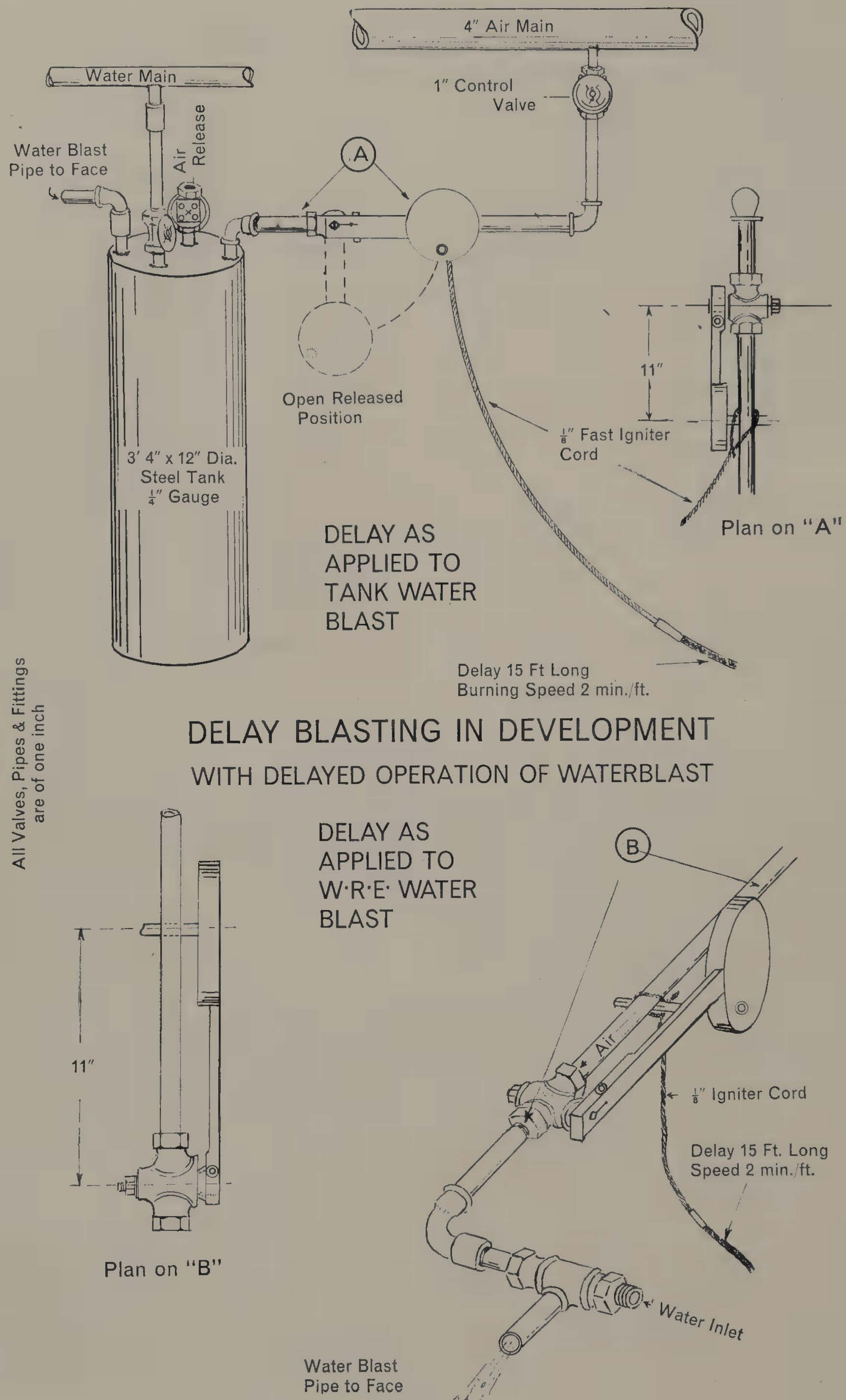

4" Air Main
Water Main
1" Control Valve
Water Blast Pipe to Face
Air Release
A
Open Released Position
3' 4" x 12" Dia. Steel Tank ¼" Gauge
⅛" Fast Igniter Cord
11"
Plan on "A"
DELAY AS APPLIED TO TANK WATER BLAST
Delay 15 Ft Long Burning Speed 2 min./ft.
All Valves, Pipes & Fittings are of one inch
DELAY BLASTING IN DEVELOPMENT
WITH DELAYED OPERATION OF WATERBLAST
DELAY AS APPLIED TO W·R·E· WATER BLAST
B
11"
Air
⅛" Igniter Cord
Delay 15 Ft. Long Speed 2 min./ft.
Plan on "B"
Water Inlet
Water Blast Pipe to Face

Fig. 15

minute from the mine. The normal duty of the 54 in fans is about 68,000 c.f.m. at 10 in water-guage.

The three vertical shafts and some of the incline and sub-incline haulages act as down-casts and it is possible to arrange suitable splits to provide a good ventilating current through each working place. In the worked-out areas of the mine the hanging wall subsidence is a few inches only. The waste filled timber packs and stone-walls are capable of maintaining clear and adequate return airways through the old workings.

At some later date the West Vertical Shaft will become an upcast ventilating shaft, serving a length of strike of about 10,000 ft. Upcast air to this shaft will travel through sub-incline haulages to footwall haulages on the 4th level. This will ensure the passage of air through well protected airways. Similar arrangements will be possible in other areas of the mine as their workings become deeper.

The total ventilating current as determined by the ventilation officer in the December 1959 quarterly survey amounted to 570,000 c.f.m., suitably split to provide a quantity of 123 c.f.m. to every person employed underground.

Air temperatures in the mine workings range from 70.8°F wet bulb/71.8°F dry bulb in summer to 69.1°F wet bulb/70.0°F dry bulb in winter.

The use of strike stonewalls on each stope track tends to concentrate the ventilating currents on the face and in the central scraper gully so that the dust content of the air is generally low. Konimeter dust counts are on average 78 particles per cc. at the face and 144 particles per cc. in scraper gullies.

6. *Power supply*

The Rustenburg mine is connected to the Electricity Supply Commission's mains. Two sub-stations receive power at 88,000 volts which is stepped down to 6,600 volts to the company's mains. With the exception of the feed to the Klipfontein mill and the compressor station, distribution is by overhead line.

At the mill and shaft sub-stations the mine uses 1,500 kVA transformers to step the voltage down from 6,600 to 2,200 or 525 volts. The feeders to the underground workings are mostly at 2,200 volts but 6,600 volts feeders are taken down the Central Deep Shaft where the supply is stepped down to 2,200 to 525 volts underground.

A diesel driven generator station, comprising four 1,000 kW units, is available for use in the event of a breakdown of the Escom supply. This can be connected to the Klipfontein mill and smelter feeders.

7. *Compressed air supply*

There are two compressor stations on the property, one, at the Klipfontein reduction works, is used to supply low pressure air for metallurgical purposes and the other, at the central compressor station, where air is compressed for mining and power purposes.

In the latter station the following five compressors are available for use:

Make	Type	Capacity in drills	H.P.	Revs/ min.
Bellis & Morcom	Reciprocating	65	1,000	163
Bellis & Morcom	Reciprocating	65	1,000	163
Polorney–Wittekind	Rotary	120	2,810	3,000
Bellis & Morcom	Reciprocating	35	750	245
British General Electric	Rotary	250	5,339	4,520.5

Note: The unit 'Drill' mentioned above is nominally 100 cu ft of free air per minute.

The first four compressors are direct-coupled to their motors. The B.G.E. speed is stepped up through a gear box from 1,500 to 4,520 5 r.p.m.

The setting of all compressors is such that the maximum pressure of the supply is 80 lb sq in.

The number and varying capacities of the machines available permits the use of various units or combinations of units throughout the 24 hours depending upon the duty required of the station.

8. *Water supply*

The Rustenburg mine draws water from two sources, one from the Municipal supply and the other from a series of boreholes situated on farming ground between the property and the town of Rustenburg.

A minimum amount of 350,000,000 gallons per annum may be taken from the municipal source. Normally, except in times of severe drought, this amount of water is sufficient to maintain present operations.

The borehole supply of over 300,000,000 gallons per annum is held in reserve.

9. *Surface ore haulage*

The haulage of ore from the various points of supply to the Klipfontein mill is by locomotive of 2 ft gauge drawing trucks of approximately 15 tons capacity. The types of locomotive in use are varied. Two 45 ton diesel locomotives are in operation at present assisted by two 18 ton steam driven locos. A large Garrett type steam locomotive is also available. Haulage of ore is carried on throughout the 24 hours.

The new mill in the Waterval area can be served by means of a 3 ft 6 in gauge steam locomotive and 45 ton hoppers from a stockpile-type bin adjoining the Waterval Vertical shaft.

Future haulage of ore to the Klipfontein mill will continue to be by 2 ft-gauge track, but extensions to the west of the Waterval mill will be 3 ft 6 in-gauge.

10. Surface material haulage

A large tonnage of bulk materials has to be handled. This includes timber for use underground, coke, ironstone and limestone for use in the mine smelter. A large siding area has been provided in the centre of the property including an extension into a central explosives magazine area.

Here the various bulk stores are unloaded from South African Railways' trucks and reloaded either on to 2 ft gauge trucks or lorries as required. Coke, ironstone, limestone are delivered by rail to bunkers at the smelter. Timber supplies to the various shafts are handled by motor lorry.

Smaller and less bulky goods are delivered to a second siding on the Klipfontein farm and transported by lorry to the main store.

11. Labour

European and Native labour complements vary with the output of the mine.

European labour is drawn partly from the farming area around Rustenburg and partly from other mining areas. Local young men are recruited as learner miners, plant operators, mechanical trades apprentices and junior clerical staff, whilst trained miners and artisans are drawn from other areas. The number of trained local men is increasing steadily which tends towards a more settled complement.

The company has provided a number of houses on the mine property and in the neighbouring town for a fair proportion of its employees. Many of the remainder own their own houses in the town or own small farms in the neighbourhood.

The Native workers are largely of the migratory type who come from as far afield as Nyasaland, Northern and Southern Rhodesia, Bechuanaland, Basutoland and the Transkei.

Good housing, good feeding and good underground conditions make the mine popular with its Native personnel. Many of these workers have records of service with the company ranging from 10-25 years.

A modern 135 bed hospital is maintained by the company for the benefit of its Native staff.

12. Administration and organization

In common with most of the mines of the Transvaal and Orange Free State, the Rustenburg Platinum Mines are controlled by one of the South African mining houses, in this case it is the Johannesburg Consolidated Investment Company, Limited, and the two mines, therefore, have the benefit of technical and administrative services of a high order.

On the Rustenburg section, control works downwards from the mine manager through the various departmental heads.

All senior members of the staff have had experience in other mining, engineering or metallurgical fields, thus a fund of knowledge is available of which use has been made in laying down standard procedures in all departments.

New ideas and suggestions are constantly being put forward which are tested and, if found of value, are incorporated in the laid down standards of the department concerned.

A form of stores budgeting is in operation. At the beginning of every month each departmental head prepares a budget showing the amount and value of the stores which he proposes to use during that month. The detailed information is contributed mainly by underground shift bosses and surface foremen which is then collated by the head of the department and submitted at a meeting of all senior officials and the manager. It is felt that this arrangement has resulted in a more realistic approach to production and work planning and to the development of 'cost consciousness' and some sense of responsibility for money at comparatively low levels of management.

A study department is in existence which has various laid down functions. Its work covers the following matters:

(*a*) *Personnel.*

(i) Aptitude testing.
(ii) Boss boy training.
(iii) New boy instruction.
(iv) Underground European engagements.

(*b*) *Organization and methods.*

(i) Shift boss supervision—training and checking.
(ii) Shift boss bonus—planning and calculation.
(iii) Native labour control—allocation, incentive payments, efficiencies and statistics.
(iv) Stores control—planning in conjunction with shift bosses, small stores issue, second-hand stores, statistics.
(v) Underground salvage.
(vi) Drill steel control—issue and statistics.
(vii) Lamp rooms—control and statistics.

(viii) Official learners—control and training.

(ix) Learner miners—lectures.

(x) Full scale testing of new suggestions.

This department functions in a purely advisory capacity and its services may be called upon by anyone who may require them.

5: THE METALLURGY OF THE PLATINUM ORES

The discovery of platinum in the Transvaal — in the Waterberg in 1923, and the Bushveld deposits in 1924, posed several problems for the metallurgists.

Firstly, this was the first time that ores had been mined primarily for the production of platinum. Previously the platinum metals had been recovered as alluvial platinum from placer deposits or had been obtained as a by-product from the mining of nickel—copper ores.

Secondly, there were three different types of ores found in the Transvaal—each requiring somewhat different treatment.

In the dunite deposits of the Lydenburg district[1], the bulk of the platinum was present in the metallic state in well formed crystals, nuggets and irregular grains. At Onverwacht[2], the largest nugget was 0.56 x 0.67 x 0.20 in, but most of the platinum was much finer. At the Onverwacht mine[2], the ore was crushed in a stamp mill followed by a ball mill and the metallics collected on Wilfley and James tables followed by corduroy tables.

The concentrates were redressed on James tables, and the final concentrate amalgamated in a revolving barrel, by an interesting process, using zinc amalgam and copper sulphate[3], whereby a thin copper film was deposited on the platinum which then amalgamated with the zinc amalgam.

The Maandagshoek plant[1], where the platinum was mainly metallic and in a comparatively finely divided condition, crushed the ore in ball mills and concentrated on Fraser and Chalmers tables followed by corduroy tables.

The tailing from these tables was treated in an 8 cell M.S. flotation machine, using cresylic acid and potassium xanthate. The overall extraction was 82%, of which approximately 74% was obtained by the gravity section.

The lode deposits of the Waterberg district[1], contained platinum grains in quartz stringers. A plant was operated in 1926, for a few months, but the ore proved too erratic to sustain payable operations.

The third type was proved to be the most important. This is the Merensky horizon in the Rustenburg and Potgietersrus districts, and it is this horizon which is being worked at Rustenburg Platinum Mines, Limited.

As an example of the extraordinary range of methods tried on these platinum ores, Cooper and Watson (in their paper on the chlorine process of extraction) report that the following unsuccessful methods of treatment were tried on the Waterberg ores. Retarded settlement of the gangue in viscous liquids and in liquids of high specific gravity; accelerated settlement of platinum in centrifuges; elutriation; flotation; magnetic separation; hysteresis; dieletric attraction.

Numerous solvents were tried, for example, cyanide solution, dilute hydrochloric acid, ferric chloride, brine saturated with chlorine, zinc chloride, copper chlorides, sodium thiosulphate, cyanogen chloride, ethyl cyanide, hydrocyanic acid, bromine, iodine, acid hydrogen peroxide, alkaline peroxides, permanganates etc.

Many investigations were carried out on the concentration of platinum ores by flotation, and some of the early investigations are reported by T. K. Prentice[5].

F. Wartenweiler and A. King in their paper[6] presented to the Third Empire Mining and Metallurgical Congress 1930, discuss the methods in use, at that time, for the treatment of the Transvaal platinum ores.

1. The Rustenburg deposits

The ore, which was first treated at Rustenburg, was from the oxidized zone. The ore 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, the grade of the concentrate was 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 the mining 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, although a greater proportion was recovered in the flotation 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 are locally called 'metallics' which is a misnomer as there is little 'metallic' material in the concentrates which 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.

One process[4] which was put into operation for some time was the 'Chlorination Process' developed at the Rand Mines Laboratory by Messrs. Graham, Cooper and Watson.

The flotation concentrate was dried and roasted for about six hours at a dull red heat to eliminate sulphur and oxidize the base metals. The roasted concentrate was then mixed with salt and placed in a muffle furnace where it was maintained at a temperature of between 500-600°C for five hours. During this period chlorine is passed over the surface of the material. The material is then withdrawn from the furnace and dropped into acidified water, and the solution containing the platinum metals, copper and nickel as chlorides is filtered from the residue. The copper in the solution, was precipitated as carbonate by powdered limestone and filtered off. The platinum metals were then precipitated from the resulting solution by means of zinc.

A small plant treating Rustenburg concentrates was operated for a short time at the property of the Government Gold Mining Areas Limited.

Other methods of treating the concentrate were also under intensive investigation, but the one that was most successful, and which is now adopted, was to smelt to a nickel-copper matte. Preliminary work was carried out on this method notably by H. R. Adam at the Government Areas laboratory, but it was Messrs. Johnson and Matthey 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.

2. Present operations at Rustenburg

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

The ore is first crushed in two jaw crushers to reduce it to −6 in, the fines (−2 in) are screened out and the coarse washed on vibrating screens and then passed over sorting belts.

The coarse ore is then crushed in two $4\frac{1}{4}$ ft Symons standard crushers, the crushed product is screened on $\frac{1}{2}$ in screens, the undersize going to the mill bins, the oversize going to 4 ft short head Symons crushers. The crushed product from the short head crushers returns to the $\frac{1}{2}$ in screens, so that the short head crushers are 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 which include the original four mills installed in 1932. $3\frac{1}{2}$ in balls are used in the primary mills and 3 in balls in the secondary mills.

The mills are in closed circuit with hydrocyclone classifiers. The use of these classifiers was a great advance at Rustenburg. The ore contains heavy minerals—chromite, ferro-magnesian silicates, etc.,—and if the pulp is diluted the heavy minerals tend to settle out, whereas if the pulp is thick, classification is bad. The result with the old, somewhat flimsy rake classifiers was either a classifier choked with sand or bad classification. The use of the hydrocyclone has given much sharper separations.

Gravity concentration is by means of corduroy tables. So far, the corduroy blanket has proved the best practical means of recovering the fine platinum minerals. They require considerable labour however, and experiments continue to try and find a substitute. In the past James tables, travelling belts and Buckman tables have been tried but none gave the high ratio of concentration which is obtained on a corduroy blanket.

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 is thickened to flotation density in four 75 ft thickeners, fitted with diaphragm pumps.

The thickener underflow then goes to the flotation plant.

In the 25 years this plant has been running almost all types of cells have been tried—air lift cells as well as different types of mechanical cells.

The present cells which have been found most suitable for this specialized type of ore is a mechanical cell with a fairly high power consumption.

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.

Flotation reagents consist of copper sulphate as an activator, xanthate for the promoter, a frother and a gangue depressant. The frother is cresylic acid, although various combinations with M.I.B.C. Frother 70 and so on have been tried.

The gangue depressant is an organic colloid, and here starch, gum arabic and dextrin have been used at various times.

The flotation tailings are sampled by an automatic sampler and then thickened in a 250 ft Traction thickener. The pulp is pumped first to two 30 in cyclones, which by-pass the coarser fraction out of the thickener feed while the finer fraction goes to the thickener feed well. Cyclone underflow and thickener underflow join and are pumped to the tailings dam. This use of hydrocyclones as a protective measure before thickeners, has saved a considerable amount of thickener trouble.

The concentrate after thickening is filtered on three 8 ft × 8 ft drum filters, after which it goes to the smelter.

The small tonnage of concentrates to be smelted, at the start, inevitably led 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.

The chief disadvantage of the blast furnace is, of course, the preparation of the feed from the fine concentrate. Among other methods which have been tried are sintering on a small Dwight Lloyd machine, preparation in a plunger press (similar to a brick press) and so on. The method now used is to dry the concentrate in a Buttner Turbo dryer sufficiently to form a free flowing powder and then pelletize the powder in Lurgi pelletizing pans. The green pellets are then fed, along with coke, iron ore and limestone, into the blast furnaces.

There are four blast furnaces, 10 ft long by 36 in at the tuyeres and the number in operation depends 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 three upright (or Great Falls type) 8 ft converters. The blowing is carried as far as the white metal stage, and the white metal is poured into moulds. When solid, the white metal is broken up in a small jaw crusher, bagged and sent to the refiners. It contains about 74% 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 granulated slag and water runs into a rake classifier (one for each furnace) and the slag granules are raked on to a conveyor belt which conveys them to a storage bin, from which the slag is carried to the dump. This granulation method works admirably with the relatively small quantity of slag being handled and avoids the excessive hand labour involved in casting into small pots.

The converter matte, or 'white metal', is treated at the Brimsdown Works[7] of Messrs. Johnson, Matthey & Company 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.

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DISCUSSION

H. J. R. WAY (*South Africa*):

I would like to ask Mr. Cousins if he has formulated any ideas yet on the surface which occurs at the bottom of the Merensky Reef? From what I have been able to see from the few visits that I have made to the mine, it is ordinarily of an extremely undulating nature, and occasionally these undulations are not ordinary in any way, but extremely irregular. I would, therefore, like to ask him if he has as yet formulated any ideas on the subject and, if so, whether he can let us have them?

There is also the point of the so-called potholes and koppies which in a way interfere with this surface. It seems to me that a rather important point is the fact that the bottom of the Merensky Reef in the potholes in contra-distinction to the above, is extremely regular and that this difference may possibly help in the elucidation.

C. A. COUSINS in reply:

Dr. Way has asked whether we have formed any explanation for this wavy lower contact of the Merensky Reef. It was not mentioned in the paper but, to those of you who have visited Rustenburg Platinum Mines in the last two days, it was one of the features which attracted quite a lot of attention. I may say that our underground geologists are at present doing an intensive study of these features, namely the potholes and koppies. Included in this study is the wavy lower contact of the Merensky Reef. Their investigation is, so far, only in the preliminary stages.

We were not encouraged previously in the carrying out of geological work by the extreme regularity of the deposit. This made the mining engineers reluctant to incur the extra expense of a mine geologist, but we have managed to persuade them to let us do this investigation and, as a consequence, we are busy on it. I may say that I have a feeling that this wavy contact at the base of the Merensky Reef may have some connection with the mode in which it was emplaced. It might be some type of flow structure, resembling the flow structures found in lavas. This is purely a tentative suggestion and we must await further research underground before the problem can be solved.

R. J. WESTWOOD in reply:

In connection with the metallurgical side, there are one or two points about the blast furnace which may be of interest. Many people nowadays consider small water-jacketed blast furnaces as obsolete, but we find they suit our conditions very well. We have cheap metallurgical coke and we find them very flexible.

As a matter of interest we are replacing a portion of the coke by anthracite following some work carried out by the U.S. Bureau of Mines on the use of anthracite in iron blast furnaces.

We do know that the anthracite is not equal to coke but we can get by with a small proportion, which helps to keep the costs down.

The other point, with regard to the feed to the furnace is that, as you probably know, the reverberatory furnaces were developed essentially to take care of the fine product from the flotation cells. If one has a blast furnace one must agglomerate the charge and that has proved to be one of the most difficult parts of this operation.

We have tried sintering, but sintering is not suitable as the sulphur content is very low. We tried briquetting presses, we tried the kind of press used in brickmaking and, finally, finished up with pelletizing which works very well mechanically.

Again, I do not think we have got quite the ideal pellet, but it goes into the furnace and if you can get it into the furnace undamaged it will descend in the charge, be dried, baked and melted.

We have removed the jackets of the furnace and the pellets half-way down or so are very nicely baked. You must handle the pellets very carefully, not drop them and treat them with great care. I think that is the major feature of the furnace plant.

J. K. E. DOUGLAS (*South Africa*):

I would like to ask Mr. Westwood with regard to the problem he has with pelletization whether he has considered the use of an electric furnace and how the economics of that would compare with the blast furnace.

R. J. Westwood in reply:

Mr. Douglas has raised a very wide question. We have considered the use of electric furnaces but not in very great detail. Firstly, when Rustenburg started there was no electric power, it generated its own power from a set of gas engines. With the availability of large amounts of power now at a later stage in the mine's life, we would have to go into that in great detail before changing over.

The other point is, I think, that even in an electric furnace I should rather hesitate to put in raw wet concentrate. You might even have to pelletize the concentrate feed to an electric furnace.

G. F. Mead (*Australia*):

This is not a question. I should just like to make the observation that at the Mount Lyle Mine, Tasmania, Australia, they smelt about 10,000 tons of copper per year by putting wet flotation concentrates straight into a blast furnace.

To some degree that is also done at Port Kimbler where they put a mixed charge of sun-dried concentrates, then raw ore, into a blast furnace. It seems to work fairly well, but Mount Lyle is the principal one where wet flotation concentrates are fed straight into a blast furnace.

R. J. Westwood in reply:

I should like to thank the last speaker for his very interesting remarks. We, in our briquetting days, almost got to that stage because the briquettes were very wet. We finished up with something like 25 to 30% of our concentrates circulating as flue dust.

I wonder whether similar conditions exist at Mount Lyle or whether, perhaps, their concentrate agglomerates better in the furnace. Do they get a very high flue dust production?

G. F. Mead (*Australia*):

Yes, that is correct Mr. Westwood. They do have a very high circulating flue dust but it does not seem to worry them. It is the way the ore is smelted.

W. H. Wainwright (*Australia*):

I would like to ask Mr. Westwood if they have considered the heat hardening of pellets before introducing them into the blast furnace.

R. J. Westwood in reply:

In the preliminary work on pelletizing, considerable pilot plant work was done in Germany on the making and drying of pellets. There is no doubt that if you can dry the pellets by means of hot air they are harder in the sense that it takes more pressure to break them, but they are also more brittle. We have a very rough air dryer at Rustenburg but it did not work too well. We have found from experience that it was better to leave the pellets wet and plastic and then feed them into the furnace and allow the furnace to do the drying. If you were feeding the pellets into some other type of furnace where you would not be getting that slow drying condition, I think it would be very much better to dry them first. We cannot burn them because we would lose sulphur.