Gold in South Africa

by P.R. JANISCH*

SYNOPSIS

This paper reviews the sources, production, and technology of gold in South Africa. First of all it details the geographical distribution of gold-bearing materials, together with the mineralogy and distribution of gold within those materials. It goes on to the geological origins, and to the discovery and early exploitation, of the gold deposits. The laws relating to the mining and disposal of the metal are explained, after which an outline is given of gold exploration, processing, and refining. The economics of gold production are discussed briefly, and it is concluded that, although present expertise in the financial and technological aspects of a gold-mining venture have largely reduced the risks involved, one severe risk remains: that the rate of cost inflation will not be matched by the rate of escalation in the gold price.

SAMEVATTING

Hierdie referaat gee 'n oorsig oor die bronne, produksie en tegnologie van goud in Suid-Afrika. Eerstens verstrekk dit besonderheid van die geografiese verspreiding van goudhoudende materiale evenwgt met die mineralogie en verspreiding van goud in daardie materiale. Daarna behandel dit die geologiese oorsonge, en vervolgens die ont- deking en vroeë ontginning van goudsaamstelings. Die wette in verband met die ontginning van en beskikking oor die metaal word verduidelik, waarna goudprosessing, -verwerking en -raffinering in hooftrekke bespreek word. Die ekonomie van goudproduksie word kortlik bespreek en die gevolgen daarvan word geysa gemaak dat hoewel die huidige kundigheid wat betref die finansiële en tegnologiese aspekte van 'n goudmynderyneming die risiko's daaraan verbonde baie vermind hou, maar dat die eskalasiekoers van die goudprys nie met die inflasiekoers van die koste sal tred hou nie.

Introduction

Sources, Production, and Technology

During the calendar year 1985, the recorded gold production from South African sources passed the 40 kt mark. Official records date from 1884, although there is extensive documentation on activity prior to 1884. Gold production south of the Limpopo river is unlikely to have exceeded some tens of metric tons by that date, and the official statistics can be taken as correct for all practical purposes.

The recorded amount of gold-bearing ore or material treated is 4.3 billion tons, with an average yield of 9.29 g of gold per ton\(^2\). It is possible to relate mill tonnage and gold production directly only from 1910. In that year, the yield was 11.88 g/t for a production of 234.3 t. By 1940, production had increased to 436.9 t, but the yield had declined to 7.12 g/t. Both production and yield fell during and after the Second World War, the pre-war position being restored only in the mid-1950s. Growth was steady, and maximum production was attained in 1970 at just over 1 kt. Under the combined pressure of increasing costs, a static gold price of $35 per ounce, and stable exchange rates, the gold yield per ton milled had peaked two years earlier at 13.11 g/t. The year 1968 marked the start of relief on the price front, with the establishment of two-tier marketing of gold. New economic circumstances in the late 1970s and the 1980s have been such that, despite double-digit cost-inflation rates, the price received made it possible to reduce the yield to 6.16 g/t in 1984 from 110.7 Mt of ore treated for a production of 681.3 t.

As a proportion of world production (excluding that of the U.S.S.R.), South Africa's production peaked in 1971 at 79.1 percent. It has since fallen consistently, to a 1985 level of 55.8 percent, under the combined influence of declining total South African production and increasing output from elsewhere, particularly North America, Australia, and Brazil. As a percentage of new world supply, which includes imports from the Communist sector, South Africa's contribution decreased from 78.7 per cent in 1970 to 47.3 per cent in 1985. Nevertheless, South Africa is still pre-eminent in world supply; the next-largest suppliers in 1985 were the Communist sector (14.8 per cent) and Canada (6.0 per cent).

Within South Africa, the mines of the Witwatersrand Basin dominate gold production. Deliveries of bullion to the Rand Refinery during its 1985 financial year were as shown in Table 1.

### TABLE I

<table>
<thead>
<tr>
<th>Source</th>
<th>Fine gold, kg</th>
<th>%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Underground mines on the Witwatersrand</td>
<td>664 483</td>
<td>97,0</td>
</tr>
<tr>
<td>Dump-treatment plants on the Witwaters-</td>
<td>10 712</td>
<td>1,6</td>
</tr>
<tr>
<td>rand</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Gold mines outside the Witwatersrand</td>
<td>5 318</td>
<td>0,8</td>
</tr>
<tr>
<td>Mines that produce gold as a byproduct</td>
<td>3 837</td>
<td>0,6</td>
</tr>
<tr>
<td>Total</td>
<td>666 350</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Gold mines outside the Witwatersrand include those at Barberton. The largest byproduct mines are the
platinum producers of the Bushveld Complex and Consolidated Murchison, a primary producer of antimony. Palabora Mining Company derives revenue from the gold contained in its copper anode slimes. Other smaller base-metal producers such as O'okiep Copper Mining Company receive a credit for the gold in the metal sold.

Table I suggests that any study of gold technology in South Africa should devote itself mainly to the Witwatersrand Basin. That region, indeed, features largely in the present paper, but due recognition has been given to other sources of South African gold, some of which are growing in importance. For 1980 and 1985, the Rand Refinery report the figures shown in Table II for specific categories of producer, none of which is a member of the Chamber of Mines of South Africa.

### TABLE II

**PRODUCTION OF GOLD BY NON-CHAMBER PRODUCERS**

<table>
<thead>
<tr>
<th>Producer</th>
<th>Fine gold, kg</th>
</tr>
</thead>
<tbody>
<tr>
<td>Witwatersrand mines*</td>
<td>1750 6896</td>
</tr>
<tr>
<td>Non-Witwatersrand mines</td>
<td>3759 3754</td>
</tr>
<tr>
<td>Banks</td>
<td>258 753</td>
</tr>
<tr>
<td>Dump re-processors</td>
<td>731 1106</td>
</tr>
<tr>
<td>Byproduct producers</td>
<td>3866 3964</td>
</tr>
</tbody>
</table>

* Companies such as the Boshoff Group, Golden Dumps, Consolidated Modder, and others, which have been able to re-open and reclaim old mines that had ceased operations in less profitable times.

To some extent the increase in production shown in Table II reflects improved geological knowledge of the reefs that were originally mined and of other reefs that earlier miners had ignored. Similar comments apply to the dump reprocessors such as Ergo, which are profitably applying new metallurgical technology to increasing tonnages of surface-dump material of decreasing grade. The category 'Banks' is of interest in that it reflects production by small-scale, possibly one-man, operators working claims in the traditional manner. However, these figures also include some melted-down jewellery.

Rocks of the Central Rand Group of the Witwatersrand Supergroup began yielding their gold in 1886. In the hundred years that have elapsed since then, mining technology on the Witwatersrand has evolved from something of an art to a precise engineering discipline. There have been several notable technological turning points along the way: the MacArthur-Forrest cyanidation process, which increased gold recoveries from around 70 per cent to 98 per cent or more, despite lower head grades and increasing refractoriness of ores; the introduction of compressed-air jack hammers for the drilling of blast-holes; tube milling; longwall mining at depth; the development of rock mechanics in the 1960s; and the introduction of refrigeration to counteract the heat in deep mines.

One aspect of mining has so far resisted all efforts at the introduction of new technology. That is rockbreaking, which is still fully dependent on explosives, as it has been since 1886. Despite this, Witwatersrand mining has developed with the times. In an introduction to his seminal 1963 paper on rock mechanics, Salamon wrote:

The attitude of mining engineers was and still is to a large extent that the art of mining cannot be acquired by studying the science of mining and then becoming a master by practical experience, but that the secrets of the trade have to be mastered by prolonged practical training, which eventually develops an intuitive sense towards the problems. The outlook is slowly changing since some of the mining problems appear to be in surmountable by this traditional approach (for example the rockburst problem in deep, hard rock mines), and the rapid development of the technologies of the younger industries presents itself as a challenge to the mining industry.

The events of the past few decades demonstrate that mining problems yield to the enquiring mind of the engineer, and the foundations of this new science are already being laid down.

The status of gold mining and extraction on the Witwatersrand in 1986 upholds the validity of Salamon's remarks.

### Note on Units of Measurement

The gold-mining industry metricated its measuring units in 1970, adopting as far as possible the standards of the Système Internationale (SI). Some minor conventions were introduced to accommodate mining conditions. The following units are common:

- **Tonnage of ore** 1000 kilograms, tons, t
- **Gold production** Kilograms, kg
- **Gold value** Grams per ton, g/t
- **Gold content** Parts per million, ppm (equivalent to g/t) or parts per billion, ppb (equivalent to 10^-9 g/t)
- **Particle size** Micrometre, 10^-6 m, μm
- **Mining-lease area** Hectares, ha
- **Area of stoping** Square metres, m²
- **Stoping width** Centimetres, cm
- **Gold accumulation** Centimetre-grams per ton, cm².g/t

The factor for the conversion of kilograms of gold to the more common market unit of troy ounces (oz tr.) is 32,150.74.

### The Literature

Every aspect of gold mining in South Africa has been covered by a vast literature. There are two particularly strong reasons for this. Firstly, it is a romantic story, intimately bound up with South Africa's history and affecting the lives of millions of her people. Secondly, it is a traditional South African requirement that all the gold produced must be sold immediately to the Reserve Bank, which is the only buyer and which then controls the further disposal of the metal. The price received by the producers is determined by the markets, particularly in London, Zurich, New York, and Hong Kong. Therefore, the mines do not compete for markets or market share. The result has been a degree of cooperation in the development and a sharing of technology that is arguably unequalled in any industry anywhere in the Free World.

This has manifested itself in the literature produced as cooperative efforts through the Chamber of Mines of South Africa; through professional bodies such as The South African Institute of Mining and Metallurgy (and its predecessor, The Chemical, Metallurgical and Mining Institute of South Africa), the Association of Mine Managers, and similar associations; and through trade
publications of varying quality. The most notable of the cooperative publications are various textbooks, which include volumes on gold metallurgy, assay practice, environmental engineering, rock mechanics, and mine valuation. These have been continuously updated, and the body of literature currently available can be regarded as definitive. One important exception is the lack of a consolidated, up-to-date volume on Witwatersrand mining practice. The most recent was published in 1946 and, while subsequent South African references abound, the need for a new textbook is apparent.

No study of gold production in South Africa can possibly cover it all. In June 1936, Owen Letcher produced a 580-page volume entitled ‘The Gold Mines of Southern Africa,’ being the History, Technology, and Statistics of the Gold Industry, and Published to Commemorate the Fiftieth Anniversary of the Proclamation of the Witwatersrand and the Jubilee of the City of Johannesburg.’ A review paper that must also deal with an additional fifty years has necessarily to be both selective and subjective. If there are any glaring omissions, they should not be taken as deliberate.

Occurrence

Background Gold

The average gold content of the continental crust is about 3.5 parts per 10^9 (ppb). Sedimentary rocks are relatively enriched at 5.1 ppb, with certain sandstones showing in excess of 50 ppb. Limestone and dolomite contain 7 ppb and chert 17 ppb. Metamorphic sediments, notably quartzite, schist, and marble, carry around 20 ppb. The gold value of ultramafic rocks does not differ significantly from the average crustal values, but some South African basalts are an order of magnitude richer at 23 ppb. Bushveld granites carry up to 9 ppb. By contrast, the Johannesburg Archaean granite dome is poor at 1.2 ppb.

In geochemical exploration, values in excess of 25 ppb are seen as anomalous and potentially indicative of the nearby presence of exploitable gold deposits. Such deposits would have to assay more than 1000 ppb (i.e. 1 g/t) to generate economic interest in the gold alone, but lower values found in association with other minerals may be profitably worked.

Geographical Distribution

Known occurrences in the Cape and Natal are of curiosity value only. At Humansdorp some highly folded, sheared argillaceous rocks carry steeply dipping quartz veins containing sulphides and native gold. Alluvial gold has been found in the Karoo near Prince Albert. Near Knysna, gold-bearing quartzites of the Cape Supergroup have been exploited, and in northern Natal limited quantities of gold have been mined from rocks associated with the gold province of the eastern Transvaal. Byproduct gold is obtained from the platinum mines of the Bushveld Complex, in which the pegmatitic pyroxenite of the Merensky Reef carries gold values around 0.5 g/t. It occurs in minute quantities in the copper deposits at Phalaborwa, O’okiep, and Prieska, and as a co-product with antimony in the Murchison Range.

The major primary South African deposits are located in the Transvaal and Orange Free State (Table III and Fig. 1), and are conveniently described under three headings: Archaean; quartz veins in sedimentary rocks of the Transvaal Sequence; and the quartz–pebble conglomerates of the Witwatersrand Supergroup. Small deposits of disseminated gold have been exploited in the eastern Transvaal, but occurrences on the scale of the Hemlo (volcanic-hosted) or Carlin (sediment-hosted) types have not yet been discovered.

| TABLE III SUMMARY OF THE MAIN GEOLOGICAL UNITS AND ASSOCIATED MINERAL DEPOSITS OF SOUTHERN AFRICA
<table>
<thead>
<tr>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Age (Ma)</td>
</tr>
<tr>
<td>0 Tertiary to recent</td>
</tr>
<tr>
<td>Kimberlite Pipes (100)</td>
</tr>
<tr>
<td>Cretaceous</td>
</tr>
<tr>
<td>Karoo Sequence (200–300)</td>
</tr>
<tr>
<td>Cape Supergroup</td>
</tr>
<tr>
<td>Salem Granite (500)</td>
</tr>
<tr>
<td>Metamorphism of the Damara Belt</td>
</tr>
<tr>
<td>Nama Group (600)</td>
</tr>
<tr>
<td>Damara Sequence (800)</td>
</tr>
<tr>
<td>1000 Metamorphism of Namaqua-Natal Belt (1100)</td>
</tr>
<tr>
<td>Kheis Sequence</td>
</tr>
<tr>
<td>Metamorphism of Kalahari and Rehoboth Belts (1800)</td>
</tr>
<tr>
<td>Waterberg Group and Olifantshoek Sequence</td>
</tr>
<tr>
<td>Phalaborwa Complex (1900)</td>
</tr>
<tr>
<td>Transvaal and Griqualand West Sequences</td>
</tr>
<tr>
<td>Great Dyke (2550)</td>
</tr>
<tr>
<td>Metamorphism of Limpopo Belt (2600)</td>
</tr>
<tr>
<td>Venterdorp Supergroup (2600)</td>
</tr>
<tr>
<td>Witwatersrand Supergroup</td>
</tr>
<tr>
<td>Dominion Group</td>
</tr>
<tr>
<td>3000 Pongola Sequence (3000)</td>
</tr>
<tr>
<td>Granite-gneiss (3000–3400)</td>
</tr>
<tr>
<td>Barberton Sequence and Sequences of other Greenstone Belts (3200–3500)</td>
</tr>
<tr>
<td>Metamorphism of Sand River Gneiss (3800)</td>
</tr>
<tr>
<td>4000</td>
</tr>
</tbody>
</table>

Archaean Deposits

Significant greenstone-hosted gold deposits are found in the eastern and northeastern Transvaal at Barberton and Klein Letaba, and in the Murchison Range. In the Barberton Mountainland, the gold mineralization is struc-
Gold occurrences in South Africa (after van Biljon 16)

Quartz Veins in Sedimentary Rocks, Transvaal Sequence

The Black Reef conglomerate, lying at the base of the Transvaal Sequence, is sporadically mineralized. It is the source of some gold in the Kaapsche Hoop–Sabie–Pilgrims Rest field, and has been mined on the East Rand, West Rand, and northwest of Potchefstroom. Black Reef deposits at Kromdraai, north of Krugersdorp, were found and mined before the major Witwatersrand discoveries of 1886.

Gold orebodies of particular economic and historical importance occur in the Dolomite and Pretoria Series at Pilgrims Rest and Sabie. They appear in several, generally pyrite-rich forms. Flat lodes parallel to bedding planes, in which the mineralization is confined to shoots aligned to dominant structures, carry higher values near dyke contacts and small faults or at changes in dip or strike, and at pinches and swells. The vertical lodes are not particularly well mineralized, but gold shows up well in leaders a few centimetres thick and in ‘blows’ attached to the flat lodes. These primary deposits have also given rise to occurrences of eluvial gold.

The Malmani quartz lodes of the Marico district are hosted by dolomite and are not extensive. Mining was severely hampered by water encountered at shallow depth.

Anomalous gold values have been reported in Polo-ground quartzites, in Daspoort shales and quartzites, and in Timeball Hill quartzites of the Transvaal Sequence at localities in the Pretoria–Witwatersrand region. Prior to 1886, mining at Blaauwbank and Broederstroom was con-
ducted in these formations.

**Quartz-pebble Conglomerates, Witwatersrand Supergroup**

The Witwatersrand Basin, which is roughly oval in shape, covers an area of the Highveld some 350 km long, southwest to northeast, and 150 km wide. A generalized stratigraphic column through the Basin including the underlying and overlying formations would start in the basement granites of Archaean age, 3100 million years (Ma), and progress through five identifiable stratigraphic supergroups or sequences, ending in the Karoo Sequence, which covers its southern and eastern portions (Table IV).

**TABLE IV**

<table>
<thead>
<tr>
<th>Geological entity</th>
<th>Age (Ma)</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>Karoo</td>
<td>154 to 190</td>
<td>Shale, sandstone, tillite, coal</td>
</tr>
<tr>
<td>Transvaal</td>
<td>c. 2000</td>
<td>Shale, quartzite, diabase, dolomite, conglomerate</td>
</tr>
<tr>
<td>Ventersdorp</td>
<td>c. 2300</td>
<td>Lava, quartzite, conglomerate, minor sediments</td>
</tr>
<tr>
<td>Witwatersrand</td>
<td>c. 2500</td>
<td>Quartzite, conglomerate, shale, lava</td>
</tr>
<tr>
<td>Dominion</td>
<td>c. 2800</td>
<td>Lava, quartzite, conglomerate</td>
</tr>
<tr>
<td>Archaean</td>
<td>c. 3100</td>
<td>Granite, gneiss, metamorphosed sediments, etc.</td>
</tr>
</tbody>
</table>

Gold-bearing conglomerates occur in the Dominion Reef Group and the Witwatersrand and Ventersdorp Supergroups. Hamilton and Cooke, who recognized the close association between these entities, proposed the term "Witwatersrand Triad" to encompass all three. There are several outliers of the main Basin, notably the Evander goldfield; at Rietfontein, east of Johannesburg; and the Dominion Reef basin west of Klerksdorp.

Respectable quantities of gold have been derived from the conglomerates of the Dominion Reef and the generally argillaceous lower beds of the Witwatersrand Supergroup (the West Rand Group). However, the bulk of the 40 kt of gold estimated to have come from the Basin has been drawn from the upper, more arenaceous Central Rand Group and from the Contact Reef at the base of the Ventersdorp Reef. Today, an exploration borehole that has passed through beds of the Central Rand Group or has entered the lower beds will usually be abandoned as being of little potential further interest.

The conglomerate reefs of the Central Rand Group have been mined along the eastern, northern, and western rims of the Basin in nine distinctive goldfields, at depths ranging from surface outcrop to 3500 m (Fig. 2). They have also been penetrated by several thousand boreholes. While a very clear picture of the succession has emerged, there is still a measure of disagreement about the correlation of reefs between the different goldfields, particularly in view of the current theories of origin and deposition. One result of this disagreement has been a proliferation of local names for the reefs where correlation is in dispute.

By definition, the Central Rand Group starts at the top of the Jeppestown shales, and ends either at the Ventersdorp lava contact or, if Ventersdorp rocks are absent, at the Black Reef base of the Transvaal Sequence. Its stratigraphic thickness varies from less than 1000 m to 3000 m. Four sets of reef bands are identified. The first is the Main Conglomerate formation, containing the Carbon Leader, North Reef, Main Reef, Main Reef Leader, and South Reef. Where they occur, they occupy a thickness of about 200 m together with the intervening quartzites. The next stage carries the Livingstone and Bird Reefs, which include two important gold producers: the Vaal Reef of the Klerksdorp field, and the Basil and Leader Reefs of the Orange Free State.

A shale horizon, up to 350 m thick, separates the two lower reef sets (together forming the Johannesburg Subgroup) from the Turffontein Subgroup. This contains the Kimberley Reefs, which comprise several bands of economic value (particularly on the East Rand) and the Elsburg Reefs, which include a number of relatively thick bands of special importance on the West Rand (Fig. 3 and Table V).

**TABLE V**

<table>
<thead>
<tr>
<th>Goldfield</th>
<th>Central Rand Group formation</th>
<th>Major producing reefs</th>
</tr>
</thead>
<tbody>
<tr>
<td>Orange Free State</td>
<td>Bird</td>
<td>Kimberley</td>
</tr>
<tr>
<td>Klerksdorp</td>
<td>Bird</td>
<td>Kimberley</td>
</tr>
<tr>
<td>West Wits Line</td>
<td>Main</td>
<td>Kimberley</td>
</tr>
<tr>
<td>West Rand</td>
<td>Main</td>
<td>Kimberley</td>
</tr>
<tr>
<td>Central Rand</td>
<td>Main</td>
<td>Kimberley</td>
</tr>
<tr>
<td>East Rand and Heidelberg</td>
<td>Main</td>
<td>Kimberley</td>
</tr>
<tr>
<td>South Rand</td>
<td>Kimberley</td>
<td>Kimberley Reef</td>
</tr>
<tr>
<td>Evander</td>
<td>Kimberley</td>
<td>Kimberley Reef</td>
</tr>
</tbody>
</table>

The Ventersdorp Contact Reef, strictly a member of the Ventersdorp Supergroup, is significantly enriched in gold, particularly where it appears to truncate tilted older Witwatersrand reefs, as happens on the West Rand and Far West Rand and in the Klerksdorp field.
Fig. 2—Mining areas of the Witwatersrand Basin

Fig. 3—Generalized section through the Central Rand Group (not to scale)
The general strike direction of the reefs is parallel to the rim of the Basin, but there are variations. The dips range from a few degrees to vertical, but average about 23 degrees. Major faulting is known to have eliminated the reef at places such as the Witpoortjie Gap and the Bank Break of the West Wits Line. The structure in other major gap areas around Potchefstroom and Bothaville is still not fully understood. Lesser faulting is pervasive in the Evander, Orange Free State, and Klerksdorp fields, where it gives rise to severe mining problems. Dyke intrusives of Venterdorp and younger age, mainly diabase, dolerite, and syenite, are common throughout the Basin.

Quartzite is the prevailing host rock in the Upper Witwatersrand. A troublesome khaki shale appears in the hangingwall of the Basal Reef in the Orange Free State, and lavas, sometimes tuffaceous, occur in the hangingwall of the Venterdorp Contact Reef.

The southeastern rim of the Basin has not been penetrated and is not exposed anywhere, its surface trace being conjectural. Updoming has occurred in the centre of the Basin around Vredefort, and low-grade reef bands are known to exist there.

Mineralogy of the Witwatersrand Conglomerates

The bands of conglomerate reef (Fig. 4) are marked by the presence of well-rounded ovoid pebbles, which make up some 70 per cent of the volume and are set in a mineralogically complex matrix. The pebbles are mainly vein quartz, varying in colour from white and grey, through blue, to black, and in size from 1 cm to more than 50 cm. Chert, jasper, quartzite, shales, and schist contribute smaller proportions of the pebble mass. Some of the pebbles themselves do not contain recoverable ore minerals.

The compact matrix that cements the pebbles consists essentially of finer-grained clastic and secondary quartz, and fine-grained phyllosilicates (mainly mixtures of sericite and lesser chlorite, with minor amounts of muscovite, pyrophyllite and chloritoid). This matrix is also host to heavy, allogenic minerals consisting largely of pyrite with lesser amounts of zircon, rutile, chromite, uraninite, 'flyspeck kerogen', arsenopyrite, cobaltite, and rare platinum-group metals. Authigenic minerals within the matrix include pyrite, pyrhotite, chalcocyste, uraniferous leucocoxen, brannerite, rutile, galena, sphalerite, and gersdorffite.

In 1956, Liebenberg gave the following approximate composition for reef conglomerate:

- 70–90% quartz
- 10–30% sericite (+ chlorite, pyrophyllite, and chloritoid)
- 3–4% pyrite
- 1–2% other sulphides
- 1–2% heavy detrital minerals (such as zircon, uraninite, rutile, chromite)
- 1% uraniferous kerogen.

Pyrite occurs in several genetically different forms, of which rounded grains, probably of detrital origin and usually known as 'buckshot' are the most common. Certain reefs, such as the Carbon Leader, and the Vaal and Basal Reefs, are characterized by the presence of thin, discontinuous layers of kerogen (previously referred to as thucholite or carbon) at their base. Hallbauer has ascribed a biogenic origin to this kerogen; Zumberge et al. claimed that it was formed from ancient algae or bacterial colonies that had undergone thermal degradation. Kerogen also occurs in other reefs as 'flyspeck carbon', which is believed to have formed in situ when organisms grew on grains of uraninite, or when layers were fragmented and reworked during sedimentation. The kerogen is very significant in that it is always uraniferous and is often associated with gold, which is found on its surface, as well as filling cleats and open spaces between filaments (Fig. 5).

![Fig. 4—A conglomerate reef face (Main Reef, West Drielontein)](image)
There are two main varieties of gold. One is (possibly) primary gold, occurring as rare inclusions in detrital grains of massive pyrite, or as detrital grains and nugget-like particles in the matrix. The second is a younger generation, possibly the result of metamorphism and recrystallization virtually in situ, which seems to have replaced fine-grained matrix material. Fine, flaky, irregular or jagged particles predominate. This form constitutes the bulk of all the gold in the reef, and virtually the entire gold content of the Venterdorp Contact Reef.

Gold is distributed throughout the thickness of the reef, but tends to concentrate where other heavy minerals are found. Particles may form clusters. Hallbauer and Joughin identified five different patterns, all five sometimes occurring within a few tens of metres of stope face. These are as follows:

1. an even distribution of particles a few millimetres apart;
2. single particles up to 100 mm apart;
3. concentrations of particles in a few cubic centimetres of matrix;
4. concentrations in thin patches at hangingwall or footwall contact; and
5. concentrations in very rich isolated volumes of a few cubic millimetres spaced 50 cm or more apart.

There appears to be a direct proportionality between gold content and pebble size, and a less convincing one between uranium and gold values. Von Rahden has found a remarkably positive correlation between gold and pyrite–pyrrhotite. Although gold and carbon generally occur together, investigations of the Vaal Reef have not disclosed a strong statistical correlation.

Electron-microprobe analyses of individual gold grains have shown a variable chemistry between reefs and within reefs. The copper content is consistently 0.1 per cent. Viljoen and Hiemstra give gold and silver contents lying within the ranges shown in Table VI. Von Gehlen studied the mercury content of gold grains, which are given in Table VII.

<table>
<thead>
<tr>
<th>Reef</th>
<th>Gold</th>
<th>Silver</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dominion</td>
<td>80-87</td>
<td>1-15</td>
</tr>
<tr>
<td>Government (Babbrasco)</td>
<td>77</td>
<td>16</td>
</tr>
<tr>
<td>Main (City Deep)</td>
<td>82-94</td>
<td>4-12</td>
</tr>
<tr>
<td>Main Reef Leader (Geduld)</td>
<td>66-70</td>
<td>27-32</td>
</tr>
<tr>
<td>Carbon Leader (Blyvooruitzicht)</td>
<td>83-93</td>
<td>8-10</td>
</tr>
<tr>
<td>Vaal (Hartebeestfontein)</td>
<td>78-86</td>
<td>9-11</td>
</tr>
<tr>
<td>Basel (Loraine)</td>
<td>78-90</td>
<td>9-12</td>
</tr>
<tr>
<td>Monarch (Randfontein)</td>
<td>91-99</td>
<td>1-12</td>
</tr>
<tr>
<td>Elshburg (Loraine)</td>
<td>72-97</td>
<td>9-11</td>
</tr>
<tr>
<td>VCR (Vaal Reefs)</td>
<td>67-93</td>
<td>7-12</td>
</tr>
<tr>
<td>Black Reef (Geduld)</td>
<td>81</td>
<td>20</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Reef</th>
<th>Mercury</th>
</tr>
</thead>
<tbody>
<tr>
<td>Barberton</td>
<td>11-15</td>
</tr>
<tr>
<td>'B' (Loraine)</td>
<td>9</td>
</tr>
<tr>
<td>Steyn (St Helena)</td>
<td>11.5</td>
</tr>
<tr>
<td>VCR (Kloof)</td>
<td>11.6-15.8</td>
</tr>
<tr>
<td>VCR (Western Deep)</td>
<td>8.6-9.5</td>
</tr>
<tr>
<td>Carbon Leader (Blyvooruitzicht)</td>
<td>8.7-8.9</td>
</tr>
</tbody>
</table>

The grain size distributions also appear to be reef-dependent, as Fig. 6 shows. The presence of large grains in milled ore may result in dissolution losses in the plants. Losses also result from coatings of various minerals such as limonite and compounds of iron, which shield the gold grains from the chemical action of cyanide.

Fig. 5—Photomicrograph of Carbon Leader Reef from West Driefontein. Gold grains (arrowed) average 33 μm in size. Uraniferous kerogen shows as black specks. A large detrital uraninite grain with kerogen infilling measures 190 μm along its major diameter.
Fig. 6—Grain size of gold, expressed as cumulative mass percentage less than a given diameter (De Waal).  

Fig. 7—Lognormal distribution of gold values in a reef

\[ \psi(x) = [\alpha \sqrt{2\pi}]^{-1} \exp \left(-\frac{1}{2\sigma^2} (x - \xi)^2\right), \]

where \( x = \ln(z + a) \)

First parameter: \( \xi = \text{mean of } \ln(z + a) \)
Second parameter: \( \sigma^2 = \text{variance of } \ln(z + a) \)
Third parameter: \( a = \text{additive constant} \)

If a large enough sample of values is available from a block of geologically homogeneous reef (a term to...
which we shall return), the parameters of its distribution can be established. If only a small number of samples is available, as with borehole intersections, the distribution can be estimated by use of the Sichel $t$ distribution, for which tables are readily available\(^2\). Table VIII shows distribution parameters for three typical reefs on the Witwatersrand.

### TABLE VIII
LOGNORMAL DISTRIBUTION PARAMETERS FOR THREE TYPICAL WITWATERSRAND REEFS

<table>
<thead>
<tr>
<th>Reef</th>
<th>$\xi$</th>
<th>$\sigma$</th>
<th>cm$^2$/g/t</th>
</tr>
</thead>
<tbody>
<tr>
<td>X</td>
<td>6.7</td>
<td>0.5</td>
<td>206</td>
</tr>
<tr>
<td>Y</td>
<td>6.3</td>
<td>1.4</td>
<td>69</td>
</tr>
<tr>
<td>Z</td>
<td>7.7</td>
<td>1.1</td>
<td>281</td>
</tr>
</tbody>
</table>

**Spatial Distribution within a Reef**

Lognormal distributions make no allowance for the possibility that values in close spatial proximity to one another may be related in some measurable way. This may take the form of a concentration of values in a particular area, or a value trend in a particular direction. The latter form is termed **anisotropy**. Extreme cases of this condition are known, e.g. on the Main Reef of the East Rand, where pronounced payshoots trending south–east have long been recognized.

The **semivariogram**\(^3\) is a tool for studying the continuity of value within a reef. Let $z(x)$ be a value at a point $x$ along a particular direction in a reef, and $z(x + h)$ a value a distance $h$ away. If $z(x)$ is taken as an estimate of $z(x + h)$, the estimation error arising will be $z(x) - z(x + h)$. For a large number of pairs of samples, distance $h$ apart along the given direction, an error variance can be calculated as

$$\gamma(h) = \frac{1}{2n} \sum [z(x_i) - z(x_i + h)]^2.$$ 

The semivariogram is $\gamma(h)$, which depends on the choice of the interval $h$ and, if the reef is anisotropic, on the direction chosen.

Experimental semivariograms (Fig. 8) can be plotted against the interval $h$, and various model types of curve fitted to describe this relationship. If the semivariogram levels off at some point $h_1$, that level is called a **sill**, and it is related to the variance of the lognormal distribution. The distance beyond which samples can be assumed to cease any form of interrelationship, $h_2$, is called the **range**. If the curve cuts the $\gamma(h)$ axis at some point other than the origin, that point describes the **nugget effect**, which is the variance of random fluctuations in value between two points that are very close together.

Fig. 9 illustrates anisotropy in a Witwatersrand reef. The semivariograms are typical for gold accumulation values based on the averages for blocks of 20 by 20 m. In this particular case, the direction of best continuity is in an east–west direction, where the range of influence is 180 m. The perpendicular, north–south direction is that of least continuity, with a range of 80 m. Semivariograms in other directions would have a range somewhere between 80 and 180 m. It is usually found that the direction of best continuity corresponds to the main direction of the palaeochannel. (See next section.)

Both experimental semivariograms can be well fitted by spherical models. The nugget effect and sill are the same, and only the range changes in different directions. This is known as a geometric anisotropy and is characteristic of gold reefs.

Semivariograms are a reflection of the underlying geological structure, taking account of the variability between points or block averages in terms of the distance between them and also their orientation in space. They
have several important uses:

- in the determination of the weights to be assigned to samples when the grade of a block is being estimated (kriging); samples that are aligned in the direction of best continuity will carry more weight than samples at the same distance but aligned in different directions;
- as an aid in the setting of optimal sample spacing;
- in assisting in the determination of the effect on payability when blocks of different sizes or longwalls are to be mined; and
- in highlighting the direction of pay shoots and so assisting in the formulation of a mining strategy.

Surface Accumulations

Not all the gold in the ore mined from underground sources can be recovered by metallurgical extraction procedures. Treated sand and slime accumulations on the Witwatersrand (Fig. 10) contain varying grades of gold, which had escaped recovery for various reasons: the grains were encased in gangue material or pyrite, or they were coated with layers of cyanide-resistant minerals, or they were too large to become fully dissolved in the time allowed. Otherwise, the plants may not have been optimal for the particular mineralogy of the ore they were treating, or deliberate economic decisions had been taken to lose certain quantities of gold to residue dams or
dumps rather than incur the incremental cost of maximum recovery.

The total ore recorded as having been treated in gold plants from 1894 to the end of 1986 is 4.3 billion tons\(^2\). There is no complete record of the average grade of all this tonnage. Company annual reports and other sources suggest that a figure of 0.3 ppm would represent a reasonable low estimate. Potentially, therefore, there are at least 1300 t of gold in milled surface accumulations. The actual grades\(^3\) reported for certain slimes dams on the East Rand range from 0.26 to 0.84 ppm. Sand dumps on the Central and West Rand carry about 0.7 ppm but, in the newer goldfields on the West Wits Line, Klerksdorp, the Orange Free State, and Evander, the gold content of the slimes dams is unlikely to exceed 0.4 ppm.

An unknown tonnage of rock dumps, made up of waste from excavations in country rock, or sorted manually or electronically from ore delivered to plants, also contains quantities of gold arising from ore spillage or inefficient sorting. Certain Witwatersrand dumps contain 1 ppm or more of gold, much of which can be and is currently being recovered.

**Origins and Discovery**

**Pre-Cambrian Origins**

Rocks older than 2500 Ma are found almost exclusively in areas that have been geologically stable, except for faulting, uplift, and erosion, for at least 1000 Ma\(^2\). Such areas, composed largely of granite-greenstone, form the nuclei of continental crustal blocks, and are called cratons. The Kaapvaal craton, which covers the northern, central, and northeastern parts of South Africa, hosts almost all the significant Archaean gold mineralization in its greenstone belts. Its probable source and mode of concentration are described by R.P. Viljoen et al.\(^3\).  

1. Significant amounts of gold were derived from the upper mantle in disseminated form in 'primitive' mafic and ultramafic lavas, which make up the initial volcanic phase of every greenstone belt in South Africa. The presence of anomalously high gold values in these lavas, now universally recognized as komatiites, was described recently by M.I. Viljoen\(^3\).
2. Gold in low-grade concentrations was initially accumulated in particular horizons by magmatic, sedimentary, and possibly biogenic processes.
3. This was followed by deformation and metamorphism caused by the intrusion of low-volatile granites. Structural traps were generated for the replacement of gold-bearing quartz veins from gold-bearing hydrothermal solutions that were derived from the volcanic rocks.
4. Younger high-volatite granites were intruded adjacent to the greenstone belts; the initial low-grade gold occurrences and second-generation concentrations were reconstituted, with formation of higher-grade deposits in suitable structures.

The Witwatersrand Basin carries some of the first sediments to have been deposited after the formation of the Archaean basement. It is contained entirely within the Kaapvaal craton, and few clues have emerged as to how it was formed. Starting off 3100 Ma ago as a shallow-water inland lake, it became filled over a period of 1200 million years with some 14 000 m of sediments derived from its mountainous rim and from areas further afield to the north, northwest, and west. Volcanics also made their entry from time to time. The nature of the sediments was determined by vertical tectonics in the source area: a sequence of differential uplift and downsag. R.P. Viljoen et al.\(^3\) have drawn attention to the presence in the Witwatersrand hinterland of a series of unique domes, often containing eroded remnants of greenstone belts. In a recent study, Robb and Meyer\(^3\) discuss the results obtained on samples from 162 borehole cores in selected granite-greenstone areas adjacent to the Basin. They conclude that the granite basement is pervaded by high-level hydrothermally altered granites, significantly enriched in both gold and uranium relative to other surface granites in the hinterland, probably representing the principal source rock for the Witwatersrand palaeoplacer deposits.

Although the differential geochemistry and morphology of gold grains suggest that Barberton is an unlikely source of Witwatersrand gold, the Barberton Mountainland can be used as a model for the Witwatersrand source area. Viljoen et al.\(^3\) divided the greenstone source into four erosional units: Unit I, at the base, contains komatiite-dominated mafic and ultramafic lava intruded by granites; Unit II consists of mafic rocks with pyroclasts and chemical precipitates, and is surrounded by granites; Unit III contains shales and greywackes with minor banded ironstones, also flanked by granites; and Unit IV, at the top, consists mainly of quartzites and conglomerates at the same level as a homogeneous high-level granite. These units are related to the formation of Witwatersrand conglomerates: Dominion Reef sediments were derived largely from IV; the West Rand Group from III and the bottom of IV; Main-Bird of the Central Rand Group from II and the bottom of III; and Kimberley-Elsburg from the top of I and the bottom of II.

In a series of definitive papers consolidating the views of many workers, Pretorius\(^3\) has developed a model for the depositary in the Witwatersrand Basin (Figs. 11 and 12). He writes (somewhat paraphrased):

A gold field is a fluvial fan or fan-delta that formed where a river system debouched into the lake via a canyon cut through the granite-greenstone high ground . . . after emerging from the canyons the rivers flowed short distances over a piedmont plain and then dispersed through a braided stream pattern into the basin. Six major fans have been discovered to date, and parts of each support a major goldfield. Gold and uranium were transported as detrital particles and in solution. Concentration took place physically through gravity settling and subsequent winnowing by wave and current action, and biochemically through interaction with algal and lichen colonies developed at the mouths of major rivers and in quiet water conditions down the slopes of the major fluvial fans.

The apex of a fluvial fan was located along the tectonically unstable basin edge, where repeated uplift of the source area took place along longitudinal faults. The fanheads of earlier fans were then uplifted and reworked into later fans, while midfan and fanbase sections were structurally depressed and thereby preserved. This caused transgression of the lake waters, producing winnowing of the fines and lag concentrations of the heavier minerals. Longshore currents moved the finer sediments farther away from the entry points.

A fluvial fan was built up in a series of pulses of sedimentation, together constituting a cycle. A new cycle was initiated through tectonic adjustment producing a steepening of the paleoslope. The first pulse laid down a gravel and the next the sand matrix which also brought in the heavy minerals. As the slope became progressively less steep, the energy level dropped...
and the resulting transgression was accompanied by the deposition of finer-grained material, until equilibrium was reached. Further tectonic activity then caused the tilting of the erosion surface, producing unconformities between successive cycles.

Pretorius admits that this model is likely to undergo further modification as new sedimentological, geochemical, mineralogical and, particularly, structural clues are discovered and evaluated.

A particular air of controversy surrounds the formation of the highly important Ventsdorp Contact Reef. The most credible theory (credible because it has led to new discoveries along the West Wits Line) is that it was formed by the reworking of tilted, older, partly gold-bearing sediments under erosional processes prior to being covered by the Ventsdorp lavas. Subsequent increases in pressure and temperature brought about a
further concentration of gold in situ.

Pre-European Workings

While there is strong evidence of indigenous exploitation of copper, tin, and iron deposits in South Africa, the evidence of gold working is less convincing. Friede lists seven possible sites in the northern and eastern Transvaal. He also mentions a report by Bronkhorst, a Voortrekker, who described having been shown in 1836 a mine in the Zoutpansberg from which gold was being extracted and rings being made. Gold artifacts were excavated on Mapungubwe, close to the Limpopo River, in the 1930s (Fig. 13). It is possible that these artifacts were sourced in Zimbabwe, but Fouche in his report on the excavations has suggested that the Venda had developed techniques for mining and extraction prior to the arrival of Europeans. Gold certainly found its way to trading posts at Delagoa Bay in the 18th century. Cartwright records that Portuguese traders did not themselves succeed in finding payable gold mines; the gold they shipped to Lisbon was obtained entirely from Africans.

In 1976 Anhaeusser estimated that there are some 4000 separate primitive diggings in the Southern African sub-continent, and Summers has speculated that the amount of gold produced in pre-European times could have been as high as 700 t.

Early European Discoveries

Letcher, Rosenthal, and Cartwright, inter alia, have recorded the sequence of gold discoveries north of the Vaal River that culminated in the recognition of the great importance of the Witwatersrand in 1886. Three circumstances were of particular significance colon:

(1) Pre-existing indigenous diggings had attracted the attention of explorers and adventurers like Hartly, Mauch, and Thomas Baines.
(2) Diggers had become disenchanted with other earlier discoveries (in California in 1848, and at Bendigo and Ballarat in Australia in 1851) and the subsequent gold rushes. News of the discoveries in South Africa attracted large numbers of experienced prospectors and diggers to the Transvaal, especially to the Tati area of Botswana in 1868. When the gold ran out at Tati, they dispersed over the Transvaal, and further discoveries became inevitable.

(3) Many surficial quartz and banket deposits were of high grade, and readily produced 'tails' of visible gold when crushed and panned. Although high values did not necessarily persist, they continued to draw prospectors to places such as Lydenburg, Barberton, Pilgrims Rest, Malmani, and ultimately the Witwatersrand.

In the years between 1870 and 1886, rocks of the Transvaal Sequence (which had been recognized in the Pilgrims Rest area) and of the West Rand Group (Lower Witwatersrand) were found to be gold-bearing on the farms Blaauwbank, Kromdraai, and Wilgespruit, north of the Witwatersrand. They were mined with moderate success (Fig. 14).

For at least thirty years prior to 1886, outcropping conglomerates (banket) of the Central Rand Group had been known to contain gold. Rosenthal relates that in 1856 Lieutenant Lys, of the Royal Navy, found a tailing in crushed 'puddingstone' at Knights, near Germiston. This was confirmed by Mauch in 1868. At the time, interest in the potential of this area was small by comparison with that in other Transvaal fields. The 1886 discovery of the Main Reef–Main Reef Leader outcrop at Langlaagte must have been triggered by knowledge of the gold-bearing potential of the banket. What made this discovery spectacular was the gold content, which was measured in ounces per ton, and which persisted on both strike and dip beyond all experience to that time.

The competing claims of Harrison, Walker, and others such as the Strubens as to who first recognized the significance of the Witwatersrand have been painstakingly
analysed by James and Ethel Gray, but are still the subject of controversy. Authoritative researches by Pretorius have suggested that most of the credit belongs to the Strubens.

**Gold and the Law**

Referring to the immigrant miners at Tati, Eric Rosenthal writes:

Tati brought the first practical application of the gold mining law, derived by a strange set of circumstances from medieval Spain. Because California until 1848 had belonged to Mexico, Uncle Sam inherited there the ancient usage under which he, as successor to the former Spanish kings, laid claim to all precious metals in the soil. In addition, the diggers, who arrived from 1849 onward, evolved a system by which every qualified man paying his tax and holding the prescribed licence was entitled to peg a claim of specified dimensions. California diggers emigrating to Ballarat and Bendigo carried this system with them to Australia, whence in due course some of them in their turn brought it to Tati. In this way the California claim system, unknown in Europe, travelled around the world to South Africa with only minor changes to suit local conditions.

Rosenthal does not give a primary source for his contention, but there is a reference to the California system in a 1976 paper by Ely and Pietrowski.

The law relating to gold in South Africa was embodied in 47 different statutes of pre-1910 Transvaal, Natal, and Cape of Good Hope, and the Union of South Africa. These were consolidated into the Mining Rights Act, No. 20 of 1967, which is now the principal act governing prospecting for minerals, and the ownership, mining, and disposal of minerals, in South Africa. Where this or other laws do not cover a specific case, common (Roman-Dutch) law prevails.

Two principles underlie gold-mining law: firstly, the owner of land is the *dominus* of the whole land, including the air space above and everything below the land; and, secondly, the right of mining for and disposing of precious metals is vested exclusively in the State. Precious metals are defined in the Act as gold, silver, platinum, iridium, and any other metals of the platinum group, and their ores, plus any other metals so declared by the State President. No other items have to date been added to this list.

The most generally useful reference on South African mining law is Franklin and Kaplan, from which most of what follows has been taken.

**Mineral Rights**

The right to minerals below any land can be separated from full *dominium* in four different ways.

(a) The owner disposes of the land, but reserves to himself the right to minerals. A Certificate of Mineral Rights is taken out by the former owner when transfer is registered.
(b) The owner disposes of his mineral rights to another party and cedes them by a notarial deed of cession registered in a deeds registry.

(c) The land is partitioned among joint owners, for example by legacy. The will may stipulate that mineral rights are dealt with separately. The new owners will then retain those rights over the whole original property in undivided shares. Subsequent generations may continue this partition process. It is not unknown, for example, for the mineral rights of a particular farm to be held by 50 or more persons in varying proportions, none of whom has any relationship to the owner of the surface land. In order to limit excessive splintering of rights, the law now requires that further division of mineral rights in undivided shares needs ministerial approval if such division has the effect of separating mineral rights from ownership of the land.

(d) The owner may himself separate the right to minerals from his dominium and register his title to them separately.

The transfer of title to mineral rights does not transfer ownership of the minerals. This remains vested in the owner of the surface until the minerals are physically separated from the land, after which the holder of the mineral rights becomes the owner.

One consequence of the transferability of mineral rights is the intricate patchwork distribution of holdings over some geographical units, where assembly of a package under one holder for the purpose of establishing a mine would require patient negotiation with large numbers of people over an extended period of time. With the competition for mineral rights that now characterizes the remaining portions of the Witwatersrand Basin, this has become a daunting task (Fig. 15).

**Prospecting**

Armed with a prospecting permit issued by a mining commissioner, the holder of mineral rights is entitled to move upon the relevant property to search for precious metals and, after appropriate arrangements with the State, to carry them away. In cases of conflict with the landowner, the holder of the mineral rights enjoys precedence provided that his rights are exercised in a reasonable way.

It may be that a prospective holder of mineral rights wishes to determine the probable extent of the mineralization before purchasing the rights. He and the owner would then enter into a prospecting contract setting out the right to prospect and the option to purchase within a specified period at a specified price, which is usually expressed per hectare of surface area. Option monies payable annually to the owner during the currency of the contract may also be specified. Such contracts are registrable against the owner's title deed.

Prospecting is defined by the Act as the employment of means that disturb the surface of the earth. Accordingly, geophysical prospecting by such techniques as aeromagnetics and gravity does not require the ownership of mineral rights, nor options to acquire such rights, or even a prospecting permit.

**Proclaimed Land**

Any land, whether privately held or over which the State holds certain rights, may be proclaimed a public digging for precious metals. *Open proclaimed land* is proclaimed land not held under *mining title*, which in turn means any right to mine granted or acquired under Act

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*Fig. 15—Ownership of mineral rights on a farm in the Potchefstroom gap, Witwatersrand Basin. Different shadings on the sub-divisions represent different owners or groups of owners*
Proclamation affects both the right to mine on and the right to use the surface of land. Disposal and surface use of proclaimed land are reserved to the State, and are administered by a mining commissioner. The Minister may deal with proclaimed land not yet held under mining title in two ways: by declaring such land open to the public for the pegging of claims, or granting an exclusive mining lease over the proclaimed land. A licence for a precious-metals claim may be issued by a mining commissioner, authorizing the holder to peg up to 50 claims of given shape and size and to mine them. This licence constitutes mining title, and it can be confirmed absolutely by issue to the claimholder of a certificate of bezitrecht. In the absence of bezitrecht, licenses may expire but can be renewed. They can also be abandoned deliberately. These provisions really apply only to land that had already been proclaimed by 1967. No fresh land has been specifically proclaimed in South Africa for several decades.

Dumps

Mined dumps are capable of being classified either as movable property, or as being attached to land and forming part of the property of the registered surface owner. If it is movable, then whoever acquires the dump is also the owner of the minerals in the dump. The case law has not yet clarified under what conditions a dump can be classified as movable.

A person who has abandoned a mining title or allowed it to lapse may obtain a permit to retain possession of and treat any surface accumulations on proclaimed land. If no such permit has been applied for after three months, the dumps are deemed to be abandoned, and the mining commissioner may issue a permit to any person on application.

Mining Leases

The State's right to mine for precious metals is never alienated, but it may be leased to the holder of the precious-metal rights over unproclaimed land. The Minister may grant such a lease if he is satisfied that there are grounds for believing that precious metals exist in reasonable quantities; that the applicant is in fact the owner of the mineral rights (in the various forms that such ownership may take); that the proposed scheme for carrying on mining operations is acceptable; and that the applicant's financial resources are adequate. The land over which the lease is granted becomes proclaimed when the lease is registered.

The lease document covers the provisions that apply to the lease. These include the manner of working the lease area such that the resource is exploited optimally, the share of profits to be paid to the State for granting the lease, and conditions for its cancellation or abandonment. The first of these provisions is usually expressed by requiring the leaseholder to mine at the average grade of the in situ ore reserve. An Inspector of Mining Leases sees that the lease provisions are adhered to in respect of all the leases granted in terms of the Act.

Surface

Use of the surface of any land that has been proclaimed, or over which mining title has been granted, is served to the State and administered by the Mining Commissioner. The Act provides that surface shall not be used for purposes other than mining without written permission, which is given in the form of a surface-right permit. Such a permit may, with certain qualifications, be granted to any person. It is a requirement if the mining company itself wishes to use land, say for roads, housing, or dumping, even if the company owns the land.

Provision is made for surface owners to reserve to themselves certain land for bona fide agricultural use before the land is proclaimed. Generally, no business, industrial, or trading activity is permitted on proclaimed land or the surface of a mining lease without special authority.

Disposal of Precious Metals

The right of disposal of precious metals vests in the State. The Act prescribes precisely through whose hands precious metals may pass after they have been mined and are still in an unwrought state. Any such person requires a licence or permit issued by the Gold Branch of the South African Police. Registers of dealings in unwrought gold must be kept and are subject to monthly inspection by the police, who supervise all activities closely.

A strict interpretation of this aspect of the law is that it applies even to samples of gold-bearing ore. Mines are therefore not permitted to give such samples away without special authority.

Exploration for Gold

The Legal Environment

Exploration for gold in South Africa is intimately tied up with and regulated by the legal environment, which, while designed both to protect rights and stimulate activity, sometimes acts to retard prospecting severely.

Certain types of surface reconnaissance, as well as aerial and remote-sensing surveys, can be conducted without prospecting permits, licences, or contracts—and remain undetected by competitors. This is not possible when prospecting disturbs the surface of the earth. If reconnaissance and theoretical target-generation work designate an area suitable for closer examination, the essential first step is to secure prospecting rights by option or direct purchase of precious-metal rights. Registration of these contracts in a public registry, as well as loose talk among negotiating participants, may alert others, and in a competitive situation it may not be possible to secure enough consolidated ground to justify an exploration programme.

Opportunities in areas of known gold potential are today becoming limited. Little ground remains available in the Witwatersrand Basin. In the most geologically promising gold regions outside the Witwatersrand, considerable tracts of land have been taken up by exploration companies. The resulting competition for precious-metal rights has had its effect on the prices paid for mineral rights, which are now very expensive in comparison with the prices that were paid as recently as the 1970s. A further consequence is that ownership within potential mining units has become so dispersed that meaningful prospecting by one company cannot be entertained.

Since most large companies are reluctant to sell their
holdings of mineral rights, it becomes necessary to exchange ground—always a difficult negotiating task—or to enter joint ventures with other parties. Conventional practice in South Africa is that the party which owns most of the ground in an area designated for a mining lease becomes the mine operator, and other parties must be content with minority equity holdings in the mining company. Therefore, the bargaining process that precedes the assembly of a prospecting or mining area may be a lengthy one. Moreover, there is no legal machinery available to compel a 'reluctant' owner of a strategic holding to make it available to the owners of adjoining or surrounding ground for prospecting purposes.

While these legal problems have inhibited, and continue to inhibit, both prospecting and mine development, they have not totally prevented either. It is usually possible to reach a conclusion ultimately, although this may not be entirely satisfactory to all parties.

Once the ground has been secured, exploration activity becomes governed by the constraints of time and expenditure. Enough geological knowledge must be gained for decisions to be made about the purchase of mineral rights before the options or joint ventures expire, and this must be done within the budgets allowed. Given the cost and duration of deep drilling programmes, exploration must be conducted systematically and under close supervision.

**Exploration Outside the Witwatersrand**

Archaean or Transvaal-hosted gold deposits are generally close to the surface, and are characterized by some form of surface expression that is detectable from a package of rocks of certain type or age, from a particular structural configuration, or from a geochemical signature.

Systematic exploration starts with an examination of known gold occurrences in the region and the construction of a suitable geological model to fit them. Alternatively, models for gold mineralization in other, similar geological environments may be selected. For example, South African explorationists have been engaged in a study of disseminated gold deposits such as occur at Hemlo and Carlin in North America, with a view to applying those models here.

The selected target area is mapped geologically on a large scale, say 1:50,000, and is sampled on a wide grid. The samples are studied mineralogically, and are assayed for gold as well as for associated elements or minerals expected by the model: possibly antimony and arsenopyrite. Concentrations greater than 'background' are designated anomalous and potentially significant, and their sources are marked for closer examination.

Sampling of stream sediments to bedrock and subsequent geochemical analysis may point to gold-enriched sources further upstream. This is the successor to traditional panning methods, which led to the original (19th century) gold discoveries in North America, Australia, and Southern Africa.

Localized targets are then selected for small-scale mapping and sampling on a grid interval measured in tens of meters. If the model expects rock types that carry particular magnetic signatures, or that are significantly different in density from the surrounding strata, surface geophysics would be employed at this stage. Areas for trenching and drilling are identified from the resulting geological, geochemical, and geophysical maps, studied individually or in superposition.

Even the most scientific of prospectors will admit that intuition and luck play major roles in gold discoveries. Old and even ancient workings may be pointers to lower-grade but 'payable' deposits that were missed or abandoned by early miners. A systematic geological study, for which there is no substitute, must still be accompanied by a keen observational eye and an ability to think laterally. While companies are understandably reluctant to publicize their activities, there is sufficient visual and other evidence to suggest that considerable expenditure is still being incurred in South Africa in the search for this type of gold occurrence.

**Exploration of the Witwatersrand Basin**

Gold-bearing conglomerates of the Central Rand Group outcrop for no more than a fifth of the strike length along the rim of the Witwatersrand Basin. Extensions east and west of the original Langlaagte discovery were quickly traced over a distance of 80 km. Further discoveries of outcrops were made at Klerksdorp, and a barren conglomerate of the Venterdorp System was later found near Odendaalsrus. Outcrop mining established that the reefs of the Central Rand dip southwards, initially at a steep 70 degrees but soon decreasing to about 30 degrees. The continuity of the reefs down dip was confirmed by drilling; *inter alia*, the Rand Victoria borehole intersected the Main Reef Leader at a depth of 729 m in 1892. Exploration based on diamond drilling subsequently continued to disclose gold-rich conglomerates on various horizons of the Central Rand Group, both down-dip to depths of 2000 m or more, and along strike, particularly eastward, where outcrops vanished beneath the cover of younger Transvaal and Karoo rocks.

As the geology of the Witwatersrand Basin, its stratigraphic successions, and the physical properties of its contained rocks unfolded, it became apparent that the 'gap' area between Randfontein and Klerksdorp and extensions south of Klerksdorp and east of Springs might well contain gold-bearing beds of the Central Rand Group underneath younger rocks. The geophysical discoveries, during the period 1930 to 1960, were based, firstly, on the magnetic properties of shales in the West Rand Group (Lower Witwatersrand) and, secondly, on the low density of the arenaceous rocks of the Central Rand Group. The magnetic method of exploration (Fig. 16) identifies the probable sub-outcrop position of known shale bands. A knowledge of the dip of the strata indicates the areas where the Central Rand Group might be profitably drilled. Similarly, low densities would suggest the existence of strata of the Central Rand Group below the cover and would demarcate targets for drilling. The spectacular discoveries of the West Wits Line (1932), the Orange Free State field (1939), Stilfontein (1947), and the Evander field (1951) were the results of a combination of these methods and have been well-described by Roux. Typical density values are shown in Table IX.

From 1960 onwards, the use of geophysics has been supplemented by that of sedimentology and tectonics in the identification of targets for drilling. The process
of understanding the manner of deposition of the reefs, together with the preceding and subsequent tectonic events has given rise to major extensions to existing fields—e.g. East Driefontein, Unisel, Deelkraal/Elandsrand, Southvaal, Beatrix—and to existing mines. Structure is assuming increasing importance, and will probably be the key to further discoveries. A further geophysical technique was introduced in the Witwatersrand in 1983: the high-resolution seismic reflection method developed

from oil exploration". It is currently proving to be of great assistance, both in defining structural features such as faults and changes of attitude of the strata, and in indicating particular Witwatersrand marker horizons in depth and position.

The ultimate proof of the existence of gold-bearing reefs must come from deep core-drilling, which constitutes the major portion of the cost of Witwatersrand exploration. A single hole drilled to a depth of 4000 m would cost around a million rand, and would take up to 3 years to complete, depending on the number of reefs to be probed and the number of deflections drilled on each reef. The normal practice is to drill an initial hole to the lowest reef and then to perform (say) three short deflections within a radius of a metre of the original intersection by installing low-angle wedges in the hole at appropriate distances above the reef. Longer deflections, some tens of metres distant from the original, may then be drilled, each with a number of short deflections. This procedure would then be repeated on successive reefs. The number of deflections is a function of geological judgement, being a compromise between the potential in-

### Table IX

**TYPICAL DENSITY VALUES**

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Density t/m³</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean, West Rand Group</td>
<td>2.75 – 2.83</td>
</tr>
<tr>
<td>Mean, Central Rand Group</td>
<td>2.63 – 2.66</td>
</tr>
<tr>
<td>Venterdorp</td>
<td>2.80 – 2.90</td>
</tr>
<tr>
<td>Transvaal dolomite</td>
<td>2.85 – 2.90</td>
</tr>
<tr>
<td>Karoo</td>
<td>2.67</td>
</tr>
<tr>
<td>Karoo (with 32% dolerite)</td>
<td>2.69</td>
</tr>
</tbody>
</table>
creased knowledge of gold mineralization in the reef and the cost of each deflection. One study of this problem has suggested an optimum of two short deflections of the original hole and two single long deflections.

Deep boreholes tend to deviate naturally to a direction normal to the strata bedding. The point of reef intersection could be a kilometre or more laterally removed from the collar position. This major natural deviation, together with deliberate deflections, can be surveyed accurately by the use of gyroscopic and other devices, making it possible to determine the precise position and depth of each intersection.

Cores can be recovered fully or partially. In ‘softer’ reefs, such as the Carbon Leader of the West Wits Line, valuable minerals may be lost in grinding of the core, leading to undervaluation. The interpretation of gold-assay results is today intimately associated with sedimentological and mineralogical studies of the core—taking in such items as pebble size and distribution, pyrite content, and type of gold occurrence—as well as a knowledge of the behaviour of mineralization on the same reef in nearby mines. In consequence, most exploration companies have ceased publishing gold-assay values as soon as they are received, a practice that was common until the mid 1970s. Nowadays, values are made available to the public only when a new mining area is defined, and the ore resource described and quantified in a geological report.

A typical exploration property of 3000 to 4000 ha in extent will probably have been probed on a grid pattern of holes about 2 km apart. The drilling programme will provide:

- structural information on the position, altitude, depth, and displacement of the reefs, and probable losses or gains resulting from faulting;
- identification, from sedimentological and other studies, of ‘geologically homogeneous’ areas that might be studied as geostatistical units; and
- assay values of gold accumulation (cm·g/t) and reef thickness.

South African contributions to the geostatistical process for the estimation of tonnage and gold accumulation in a mining area have been considerable. In particular, these include identification of the three-parameter lognormal distribution, Sichel’s method of estimating means and confidence limits from small samples, and the generation of kriged contour plans of block values in areas for which there are dense data. Variograms, which describe the gold distribution in such areas, can be tested for their applicability in exploration areas by the use of borehole data, and this enhances the accuracy of an assessment of a new ore resource. The knowledge that has been gained of gold distributions on the Witwatersrand has added significantly to the confidence attached to a published ore reserve.

Mining on the Witwatersrand

Characteristics of the Reefs

The methods used in the mining of reef deposits on the Witwatersrand are determined largely by reef geometry, depths, rock hardness and abrasiveness, and steep vertical temperature gradients. The design of layouts has also to contend with displacement of the reef through major and minor faulting, notably in the Evander and Orange Free State fields.

The continuity of the narrow, tabular reefs both along and transverse to the rim of the basin is the feature that, more than anything else, has ensured continuation of gold production into its second century. On a smaller scale, this continuity has made it possible to plan individual mines and mining operations in an optimal and rational way.

Mining is taking place today at depths varying from the surface (outcrop workings) to more than 3000 m below the surface. Several gold-mine lease areas include reefs that are known to extend down to 4000 m; in at least one case (Western Deep Levels), plans have been announced and capital expenditure started for the extraction of these deposits. Exploration boreholes have encountered gold-bearing formations at even greater depths, and exploration companies are acquiring mineral rights over such ground.

The uniaxial compressive strength of the conglomerate reef and its quartzite host rock is 200 to 300 MPa. The relative density is 2.7. Hardness and abrasiveness (Fig. 17) have combined to severely limit the economic use of mechanical or non-explosive methods of rockbreaking, and place a very finite life on rockhandling and transportation equipment.

Virgin rock temperatures increase with depth as a result of heat flow from the earth’s interior. The gradient is linear, to a first approximation. Where strata of low thermal conductivity, such as those of the Karoo Sequence, overlie Witwatersrand rocks, the gradient is generally steeper (Fig. 18).

Mine Design

The base design parameter for a mine or mining unit is the rate at which ore is to be mined and delivered to a plant. For no apparent reason other than the expected useful life of major buildings and equipment, the mining rate is selected on the assumption of an operating life of about 40 years. The size of the capital investment required also plays a role in the selection process. An upper limit of around 250 kt of ore milled per month (Fig. 19) is set...
by experience of 'manageable' units, and appropriate managerial structures need to be devised for larger mining complexes.

Given an extraction rate and a geological report that defines the specific reef characteristics of the proposed mine, a mine planner is able to proceed with a design that allows for the optimal use of labour, materials, services, and time.

The current underground labour requirements are one employee for approximately 25 t mined per month. A typical underground complement is 8000 men. It is fairly standard practice for some 70 per cent of the work force to be used on day shift, the balance being employed on night-shift cleaning, rock transportation, or maintenance activities. Therefore, the mine must be designed to enable perhaps 5500 men to reach their working places, perform their assigned tasks, and return to the surface within the space of an eight-hour day shift. During that shift, but also during night shift, material must be moved into the mine and transported to the working places. The monthly requirements for a 200 kt per month mine would include inter alia 6 kt of timber, 200 t of explosive, and 1850 m of rail track. Time should also be set aside for the lowering of major items of underground equipment such as locomotives, electricity sub-station assemblies, and pumps as and when they are required.

Water, compressed air, and electrical power are reticulated down the mine and to the working places. Pumps and pump columns remove the water from the mine. Broken rock, ore, and waste are transported to the shafts and hoisted. All this activity takes place through the shafts, which serve also as channels for the entry and egress of ventilation air.

Today's Witwatersrand shafts are overwhelmingly vertical and circular in cross section. The maximum depth is defined by engineering considerations: the proposed monthly rock-hoisting capacity, the length and weight of a single rope, and the power and winder size required. A practical limit at the present stage of development is about 2400 m below the collar on the surface.

The interval between working levels, and therefore stations in the shaft, is determined by the dip of the reef that the shaft will serve. It is selected so as to avoid excessive down-dip movement of broken rock at the working place before being transported out on a working level to the rock passes. The rock then gravitates to loading boxes at the bottom of the shaft. Spillage-free loading of rock at mid-shaft positions has not yet been perfected for normal production, but has been applied with some success in the development phase of a gold mine.

Government regulation requires an independent second outlet from the mine. This is sometimes incorporated in a bratticed upcast-air ventilation shaft. Larger, newer mines have both upcast and downcast air in each of two shafts, the two air compartments (Fig. 20) of a shaft being separated along the length of the shaft by a concrete or steel brattice. Access to workings deeper than those served by the surface shaft is gained by secondary vertical or inclined shafts (called sub-shafts), which are operated by winders installed in excavated underground hoist chambers (Fig. 21). In some very deep mines, tertiary shafts may also be necessary. Main-secondary-tertiary shaft systems are becoming common, especially on the West Wits Line.

The infrastructure for a shaft station provides for the circulation of ore trains, tips for access to rock passes, pump stations, repair bays, and electrical substations. The layouts permit efficient loading and unloading of men and material into and from shaft conveyances.

The overall cost of a single-stage shaft for the handling of men and materials, and rock-hoisting of, say, 150 kt per month was about 160 million rands in 1986. The cost of a main and sub-shaft system with a second outlet represents more than half the total outlay of a new mine. Therefore, the positioning of a shaft system relative to the reef area it is to serve is critical. One decision to be taken is the length of strike to be mined from the shaft. Fig. 22 shows the distribution of strike lengths currently being served by shafts on three fields: Klerksdorp, Orange Free State, and West Wits Line.

Primary underground development is aimed at the opening up of stope face and the provision of access for men, materials, and services, and the egress of rock to
Fig. 20—Cross-section of a deep-level shaft system on the West Wits Line

Fig. 21—Compartment design for a typical bratticed shaft

Fig. 22—Distribution of strike distances served by shafts in the Klerksdorp, Orange Free State, and West Wits Line goldfields
Stoping

A 2 by 2 m raise connection is developed in the reef plane, on major dip, between points where the cross-cuts intersect the reef on successive levels. Reef is exposed in the sidewall of the raise, usually being carried in the upper portion. The lower portion becomes, in due course, a ‘slusher’ for the removal of broken rock.

Ledging is the process of initial widening of the raise, taking a minimum thickness of barren waste-rock above and below the reef band, and establishing timber or other support. Strike rock- and material-transport ways on the reef horizon, known as gullies, may also be started at intervals along the raise, typically 35 or 40 m. These intervals define stope panels (Fig. 23). Once the ledging phase has been completed, full-scale stoping—the removal of the layer of reef, together with minimum footwall and hangingwall waste—can proceed. A stoping cycle comprises the drilling of blast-holes into the exposed face, charging with explosives, blasting, and cleaning out of the broken rock. The panel is the stoping unit, and the system is designed to achieve a maximum number of panel blasts every month. Each panel occupies about 15 men, made up of a certificated miner (who may supervise several panels), drilling-machine operators, winch drivers, support labour, ‘lashers’ (shovellers), team leaders, and other specialists. The panel advances in the general strike direction at about 1 m per blast and about 8 blasts per month. The crew of a 35 m panel would therefore deliver 280 m² of broken reef to the ore passes, which is equivalent to some 750 t at a stoping width of 100 cm.

Blast-holes are drilled by hand-held rock drills driven by compressed air. The hole diameters range between 28 and 42 mm, and the lengths are from 0.9 to 1.2 m. Two, sometimes three, rows of holes are drilled at about 70 degrees to the face and 0.6 m apart (Fig. 24). The blasting medium is nitro-glycerine or is based on ammonium nitrate. The sequential firing of 100 or more blast-holes is essential for efficient rock breaking. This is achieved by the use of a suitable combination of safety fuses and igniter cords. Blasted rock is confined to the immediate face area by previously erected rubber or timber barricades, or a ‘scatter pile’ of freshly broken rock.

A standard re-entry time of 4 hours is observed to enable fumes and dust to dissipate. The broken rock may be cleaned by a reduced crew on night shift. Ore is moved in the plane of the reef by scraper scoops drawn by winches in two or three steps: down the face to the strike gully, along the gully to a box-hole or to the centre gully slusher, and down the slusher to a box-hole. Box-holes emerge either in the cross-cut or in a footwall drive, where a chute controls the loading of ore into trains for onward transportation to ore passes in the shafts. Fine blasted material, usually rich in gold particles, is swept from the panel manually or by a high-pressure water jet.

The use of explosives as the primary means of breaking rock, combined with the regulatory re-entry period, imposes a rigid cycle on stoping operations. One panel cycle can at best be completed during every two day-shifts. Recent attempts to achieve a cycle per panel per day by modifying the stoping system have met with only partial success.

Several stoping systems are currently in use. At shallow depths, less than about 2000 m below the surface, raises are spaced perhaps 150 m apart on strike and may be mined selectively according to their grade. This method is flexible in that, at any working level, more points of attack than are needed at any given time are made available, and stoping activity is ‘scattered’ about the mine.

Steeply dipping reefs may be stoped by advancing the face up-dip or by shrinkage methods, but the more common practice is to advance the face in the direction of strike using a breast or underhand layout.

The main disadvantage of scattered mining is that highly stressed remnant areas are formed between the raise connections. At depth, these remnants give rise to ground-control problems and are the major cause of rockbursts. To overcome these difficulties, a longwall mining system (Fig. 25) is used at greater depths. This system of mining eliminates remnants but suffers from a lack of flexibility, since all the faces have to be advanced at the same rate to ensure a straight face line and so avoid highly stressed face irregularities. The longwall system does not allow for grade selectivity, except over the whole longwall face of 500 to 1000 m. Longwall mining was first described in 1924, but became common only after the Second World War.

The Papers and Discussions of the Association of Mine Managers of South Africa contain many appropriate references to stoping systems.

Grade Control

Gold-bearing ore is discernable at the stope face by its pebbly nature. The first objective of stoping is to break...
and transport only that portion of the ore that carries value. In practice, the entire reef band is mined, together with varying amounts of waste. It is not generally possible to mine reef separately because of the (relatively) uncontrolled nature of the blasting process. Where reefs are very narrow, there is also a need to provide adequate working room for personnel and equipment. One answer to this problem, resue mining, is not widely practised because of its heavy labour requirements and gold losses, and mines prefer simply to accept value dilution in the transported ore.

Grade control on a gold mine has several functions. It attempts to ensure that the gold content of the ore transported from a face is sufficient to pay for the costs of mining, and that gold loss and ore dilution are minimal. On the mine scale, control must also be exercised on the overall value of the ore mined, to ensure that it meets the lease condition requiring that the average grade of broken ore does not differ significantly from the average grade of the in situ ore reserves.

The first two functions of grade control require that:
- the average accumulation value (cm·g/t) for a stope face exceeds a calculated cut-off value;
- the stoping width is kept to a minimum without the leaving of valuable reef in the roof or floor;
- all the broken ore in every size category (from dust to boulders) is transported from the stope; and
- other sources of barren rock that contribute to the ore tonnage but not to the gold value are kept to a minimum.

Grade-control departments on gold mines have developed measures and procedures that meet their particular circumstances most effectively. Two typical control parameters are detailed opposite:

![Diagram](image)

**Fig. 24—Longwall mining system, with stabilizing pillars, West Wits Line**

![Diagram](image)

**Fig. 25—Longwall mining system at depth, showing stabilizing pillars (Kloof Gold Mining Company Limited). Scale in metres**
Milling width (a) Milling width factor = 
\[ \frac{\text{Average channel width of reef}}{\text{Gold in ore to plant}} \]

(b) Mine call factor (MCF) = 
\[ \frac{\text{Gold in ore from mine}}{\text{Gold in ore to plant}} \]

* Milling width = \[ \frac{\text{Tons milled}}{\text{m² mined} \cdot \rho} \]
† Determined from metallurgical balance
‡ Determined from sampling returns

The milling width factor is a measure of ore dilution that is characteristic of a mine and dependent on the average channel width of the reef and whether surface sorting is practised. The mine call factor may be affected either way by pronounced sampling bias, but the main cause of a mine call factor that is less than 100 per cent is gold left underground. Factors such as these two are monitored continuously for changes, and the reasons investigated.

Rock Mechanics and Stope Support

The great depth of mining and the tabular nature of the reef bodies result in very unfavourable rock-pressure conditions in most Witwatersrand gold mines. Rock mechanics principles are used extensively in the design and support of workings to minimize these problems. Work in the early 1960s established that the overall response of the rock mass to mining is linearly elastic if the narrow zones of fractured rock around mining excavations are ignored. This finding has given rise to design tools and concepts, the most important of which are the electrical resistance analogue computer and the digital mining-simulation computer program code-named MINSIM.

Design of off-reef excavations such as tunnels and underground chambers is based on critical stress quantities that take into account the strength of the rock strata and the type of support employed. In very deep mines, excavations are situated in de-stressed (stoped-out) ground. The elastic volumetric closure that takes place in the stoped-out areas is the most important parameter since it governs the energy changes that occur as a consequence of mining. The spatial rate of energy release, ERR, which is measured in megajoules per square metre mined, is a widely used design quantity (Fig. 26). It describes the extent of fracturing that takes place at the stope face, and hence the probability of rockbursts.

Stope support in deep gold mines (Fig. 27) must have good early load-bearing characteristics and good yield properties to ensure stable conditions throughout the life of the excavation. Grouted rock binders supplemented by rope lacing and wire mesh have been found to be most effective as tunnel support under extreme stress conditions. In stopes, rapid-yielding hydraulic props are widely accepted as face support. The backfilling of stopes using deslimed plant tailings or crushed waste rock is gaining in popularity, since it provides not only effective local hangingwall support but also regional support, and has the potential of significantly reducing the rockburst hazard in deep mines.

Another important development in deep-level stoping is the use of a system of regularly spaced stabilizing pillars. They are usually oriented along strike and have a width of 30 to 70 m depending on the depth and pillar spacing. Pillar layouts are designed to minimize volumetric closure in mined-out areas, effectively reducing the energy release rate. They offer the further advantage of clamping fault planes and minimizing slip movement along faults. However, local hangingwall conditions tend to deteriorate in the tight corners formed between stope faces and pillars.

The loose rock in the fractured zone around stoping excavations has to be contained, and various timber and timber-concrete pack designs are available for this purpose. A recent innovation is the 'pipestick', a circular mine pole confined for most of its length in a metal pipe to prevent buckling.

Environmental Engineering

The discipline of environmental engineering in South African gold mines has developed to keep pace with the increasing problems encountered. Initially, it was required to ensure adequate ventilation of working areas and to keep dust and noxious fumes below acceptable levels. Increasing depths brought increasing wet-bulb temperatures and the need to introduce both cooling of the environment and heat acclimatization of the workers. Today, mine environmental engineers are required also to monitor the quality of the underground water, radioactivity, illumination, noise, and fire-control measures. These subjects have been extensively covered in a 987-page volume prepared by the Mine Ventilation Society of South Africa with the support of the Chamber of Mines of South Africa.

In 1982, with a rock-breaking rate of 8 million tons per month at a mean depth of 1650 metres, some 600 megawatts of heat was being liberated in underground
workings. An acceptable working face temperature is 29°C wet-bulb. To achieve this level, 34500 kilograms per second of air was circulated through the mines, supplemented by 400 megawatts of refrigeration. These figures have not changed significantly. "Coolth" is delivered to working places by chilling service water (used in drilling and dust suppression) to below 10°C. The problem of rejecting heat from underground has been resolved to some extent by siting refrigeration plants on surface. Chilled water passing down the shafts is used in energy-recovery systems at the point of intake to the levels; this reduces its rise in temperature. Air down-casting in shafts undergoes auto-compression which increases wet-bulb temperatures by 0.5°C per 100 metres. The effect is reduced by bulk air cooling on surface, with secondary cooling underground on the main intake airways.

The quantity of air required depends empirically on the amount of rock broken and the average virgin rock temperature encountered. It is expressed in cubic metres per second of air supplied per monthly kiloton broken, and for mines with a maximum depth of 1500 metres would range between 2 and 6 m³/s. For deeper mines operating under varying conditions, a set of computer programs has been developed to predict air requirements. This has become a major determinant of shaft diameters.

Pathogenic dust is formed at all mining and rock-handling processes. Control measures are aimed at keeping dust production to a minimum, diluting it, filtering air and removing personnel. Water sprays are most effective at source. By government regulation there is a specified period of re-entry to a working place after a dust and fume-creating blast.

Heat stroke and heat exhaustion in underground workers are today reasonably well understood and can be prevented by heat tolerance testing and acclimatisation procedures. These have been subjects of extensive mining industry research in South Africa.

Productivity and Mechanization

A single rational comparative index of productivity in South African gold mining does not exist in a form simple enough to be applied in a meaningful way. By the most commonly used measure, tons mined per underground employee, no really significant improvements have taken place in a decade. Given that increasing depths, deteriorating environmental conditions, and longer working times to working places have combined to counter employee productivity, the fact that this measure has not decreased might be regarded as an achievement. Inevitably, increased employee productivity is accompanied by an increase in the use of other resource inputs such as energy (for machines and refrigeration) and capital (mainly for shaft systems but also for refrigeration). Mechanization brings with it a need for more skilled maintenance and additional supervision of the unskilled labour. Fig. 28 is a scattergram plot of tons mined per underground employee per month against operating costs per ton milled for 30 Witwatersrand mines; no adequate statistical conclusion can be drawn from this picture, which covers the calendar year 1985. Capital inputs and the effect of waste sorting have also been ignored. Therefore, neither cost per ton milled nor tons mined per employee can be regarded as acceptable comparative measures of overall productivity.

This in no way reduces the need to continue efforts at improving the productivity of current mining methods. Joughin has analysed the benefits to be expected in conventional mining from achievement of a blast per panel per shift, and from mechanized mining without explosives. The limitations imposed by the underground environment on the necessary mechanical equipment are severe. They include the size of the stopping excavation; the aggressive nature of mine service water, which causes gross corrosion; and the highly abrasive nature of quartzite rock. Despite massive research effort, no methods of acceptable cost have as yet been developed for narrow-
By contrast, rapid strides have been made in the introduction of mechanized mining in wide orebodies\(^7\) and in trackless rockhandling outside the plane of the reef\(^8\). These developments have made it possible to considerably reduce the number of employees on particular mines\(^9\).

**Organization of Manpower**

The average number of employees at work on South African gold mines in 1984 was 465,047. Of these, 358,714 (77 per cent) were employed underground, and 425,824 (91 per cent) were classified as non-White, most of them being migrant Black workers\(^5\).

Procedures for recruiting, training, housing, and administering the migrant workforce have been developed to a high level of efficiency. Concurrently, conditions of service have improved greatly; the skills acquired by the miners are available to the mining companies for longer periods of time, and remittances by migrants to their home countries have increased dramatically\(^7\). On the mine level, the introduction of computerized time and attendance systems (Fig. 29) has made it possible to streamline the transportation of large numbers of men through the shaft systems to their designated working places, ensuring that those who entered the mine at the start of the shift leave at the end of the shift, and ensuring that the correct entries are made on the payroll system\(^\)9.

### Gold Metallurgy

**Characteristics of the Feed**

The size distribution of run-of-mine feed depends on the nature of the stoping ore (reef plus external waste) and the stoping method—in particular, the distribution of blast-holes drilled in the face and the type of explosive used. Table X gives typical distributions of particle size and gold content for two reefs on the West Wits Line.

The gold values and mineralogical composition of the Witwatersrand reefs also show considerable variation.

**TABLE X**

<table>
<thead>
<tr>
<th>Fraction mm</th>
<th>Carbon Leader % by mass</th>
<th>Gold, %</th>
<th>VCR % by mass</th>
<th>Gold, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>-3</td>
<td>23</td>
<td>55</td>
<td>15</td>
<td>36</td>
</tr>
<tr>
<td>-25 + 3</td>
<td>32</td>
<td>26</td>
<td>34</td>
<td>27</td>
</tr>
<tr>
<td>-32 + 25</td>
<td>10</td>
<td>2</td>
<td>10</td>
<td>3</td>
</tr>
<tr>
<td>-30 + 38</td>
<td>9</td>
<td>2</td>
<td>6</td>
<td>7</td>
</tr>
<tr>
<td>+30</td>
<td>26</td>
<td>15</td>
<td>35</td>
<td>27</td>
</tr>
</tbody>
</table>

**Fig. 28**—Scattergram of cost per ton milled against tons milled per underground employee for 30 Witwatersrand gold mines

**Fig. 29**—Computerized time and attendance control at a gold mine
Appropriate metallurgical processes must be devised to allow for the expected head grades, sizes of gold grains, and liberation characteristics, and for the deleterious effect of certain gangue minerals that may absorb excessive amounts of reagent or prevent gold particles from reacting to treatment. With a gold content of about 10 ppm, the feed clearly contains relatively massive amounts of barren material, and it may be possible or desirable for particles of waste to be eliminated from processing by sorting at an early stage. Recently it has become profitable to re-treat material from residue sand dumps and slimes dams containing less than 1 ppm of gold and with mineralogical characteristics that are significantly different from those of underground ore. Finally, the feed may also contain recoverable quantities of uranium oxide and pyrite.

Given the variety of material that may be delivered to a plant, there is no single optimum process for the extraction of gold; it is generally advisable for laboratory and pilot-plant studies to be conducted to determine the response of a particular ore to the available treatment methods.

Elements of Gold Metallurgy

The process of gold extraction usually consists of the following: comminution; dissolution; recovery either by filtration and cementation or by adsorption onto carbon or resin, elution, and electrowinning; smelting; and refining. Several secondary processes may have to be adopted to take account of the special nature of a particular ore. A generalized flowsheet incorporating the most important of these processes is given in Fig. 30.

Comminution is the reduction of the feed particles in successive stages to the point at which microscopic gold is liberated from its gangue for further treatment. Sorting of barren waste may take place at certain points during comminution. Thickening reduces the liquid content of the comminuted pulp from 80 percent to around 40 percent. Concentration is either a physical or a physicochemical process, using gravity or flotation to reduce the volume of gold-bearing material by several orders of magnitude and commensurately increasing its gold value. Gold dissolution (leaching) may be followed either by solid–liquid separation (filtration) or by adsorption of gold by carbon or resin. In the case of filtration, the solution is clarified further and the gold precipitated by zinc cementation. The resulting concentrate is smelted. Adsorption is followed by elution, electrowinning, and melting of cathode gold, or by zinc cementation and smelting of gold slime. Gold bars are refined to marketable purity at the Rand Refinery. Additional circuits account for the roasting of refractory ores as found in the Barberton area; production of uranium and acid; and further treatment of filter residues by flotation.

Tables XI and XII summarize the 1982 flowsheets of treatment plants on the Witwatersrand.

Historical Development

Until 1904, the large-scale production of gold on the Witwatersrand was directly dependent on stamp milling, but was supplemented in the early 1890s by screening on bar grizzlies, with the coarse fraction (plus 150 mm) subjected to hand sorting and jaw or gyratory crushing. The stamp-mill product—a coarse sand—was screened and then passed over amalgamation plates for the recovery of about 75 per cent of the gold. This process did not suit the finer-grained, lower-grade pyritic ore encountered at depth. The introduction of the MacArthur–Forrest cyanidation and precipitation process resolved the serious extraction problems that were threatening the viability of the industry around 1890; the recovery reached 90 per cent, of which 75 per cent was attributable to cyanidation.

An obstacle to early processes was the presence of fine (minus 50 μm) material, called slime, which settled very slowly in water and impeded the flow of solution in the leaching tanks. It became necessary to separate the slime and dump it in tailings ponds, together with its gold content of some 4 to 5 g/t. This was, of course, too valuable to be left lying around, and in 1894 J. R. Williams introduced the natural-settlement decantation process, in which the slime, assisted by the addition of lime, was settled in large, conical-bottomed tanks. Clear water was decanted and cyanide solution added to the thickened slime, which was then transferred to a second tank and circulated by a pumping system from the discharge to the entry point at the top. This process dissolved most of the gold. After further settlement, the pregnant gold-bearing solution was also decanted and sent to precipitation. If necessary, the slime was treated a second time in a third tank.
TABLE XI
BASIC FLOWSHEETS FOR GOLD MINES ON THE WITWATERSRAND TREATING MORE THAN 20 kt OF ORE PER MONTH\(^{16}\)

<table>
<thead>
<tr>
<th>Mine</th>
<th>Location</th>
<th>Mining Group</th>
<th>Au content</th>
<th>UO(_2) content</th>
<th>Sulphur</th>
<th>Cyanide treated</th>
<th>Crushing and screening</th>
<th>Milling</th>
<th>Gravity in milling circuit</th>
<th>Flotation in milling circuit</th>
<th>Gold treatment</th>
<th>Uranium treatment</th>
<th>Residue flotation</th>
<th>Sulphuric acid plant</th>
<th>Au leach of calcine</th>
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Location: C Carltonville, E Evander, K Klerksdorp, W Welkom ER East Rand, WR West Rand
Gold content: A 1 to 5 g/t, B 5 to 10 g/t, C 10 to 15 g/t, D 15 to 20 g/t, ? - not available
Uranium content: M 10 to 50 g/t, L 50 to 200 g/t, N greater than 200 g/t
Sulphur content: X 0,1 to 0,5 %, Y 0,5 to 1,0 %, Z 1,0 to 1,5 %, V > 1,5 %
Tonnage treated: 1 20 to 100 000 t/m, 2 100 to 200.000 t/m, 3 200 to 300 000 t/m, etc
Sorting: H Hand sorting, R Radiometric sorting
Milling: Conventional milling, R Run-of-mine milling
Gold treatment: F Conventional cyanidation, filtration, H Dicarbon-in-pulp after cyanidation
Uranium treatment: F Filtration, C Countercurrent decantation, S Solvent extraction, I Ion exchange
Gravity: Gold removed in conventional type separators × Gold recovered from behind specially designed mill liners

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AUGUST 1986

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TABLE XII
PLANTS TREATING ONLY OLD TAILINGS (EXCLUDING WASTE ROCK)\textsuperscript{77}

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<tr>
<th>Plant</th>
<th>Location</th>
<th>Mining company</th>
<th>Au content</th>
<th>( \text{U}_2\text{O}_3 ) content</th>
<th>Sulphur</th>
<th>Tonnage treated</th>
<th>Sorting</th>
<th>Crushing and screening</th>
<th>Milling</th>
<th>Flotation</th>
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* Dump reclamation circuit

Location: E R East Rand, W R West Rand, C Carletonville, K Klerksdorp, W Welkom

Mining company: A A Anglo American, R Rand Mines, GU Gencor, GF Gold Fields, A V Anglovaal

Gold content: A\textsuperscript{*} 0.5 to 1.0 g/t, B\textsuperscript{*} 1.0 to 2.0 g/t

\( \text{U}_2\text{O}_3 \) content: L 10 to 50 g/t, M 50 to 200 g/t

Sulphur content: X 0.1 to 0.5 %, Y 0.5 to 1.0 %

Tonnage treated: 1 20 to 100 000 t/m, 2 100 000 to 200 000 t/m, 3 200 000 to 300 000 t/m, etc.

Milling: * Milling after flotation

Gold recovery: Conventional cyanidation, filtration, and zinc precipitation, C carbon-in-pulp after cyanidation

Uranium recovery: F filtration, C countercurrent decantation, S solvent extraction, I ion exchange

The major drawback of this system was the large quantity of pregnant solution sent for zinc precipitation. It was superseded in the 1900s by the use of Brown's tanks (pachucas) in which a relatively dense pulp was treated with lime and cyanide and agitated for a period by compressed air. Pregnant solution was then drawn off by vacuum filtration. The final development in the system, still widely used today, was the use of tube (or cylindrical) mills, which was introduced in 1904. These mills were capable of a generally finer grind than stamp mills and, by the 1920s, had virtually replaced stamp mills. The treatment of slime by amalgamation and cyanidation had become obsolete, and the 'all slining' process was in general use.

\textit{Comminution}\textsuperscript{7,8}

Each ore has its own gold-liberation characteristics. The degree of grind required for most Witwatersrand ores is about 75 per cent minus 75 \( \mu \)m (200 mesh). Several routes are currently used to achieve this grind, four of which are depicted in Fig. 31 (which, for simplicity, does not show all the screening and size-classification circuits).

The simplest process is direct run-of-mine milling, now being used on at least twelve large gold plants. The hoisted ore is fed direct to large-capacity mills, typically 4.88 m
in diameter by 12.2 m in length. The mills rotate at about 85 per cent of critical speed (the speed at which the contents adhere centrifugally to the walls), and the larger rocks in the ore act as its own grinding medium. Under certain conditions, steel balls may be introduced. If the mining system produces excessively large rocks, primary jaw crushing may precede milling. The mill product is classified by stages in cyclones, the underflow being returned to the mill for further grinding and the overflow, by now reduced to the required particle size, being sent to the next stage of the treatment process.

Rod mills are high-throughput, open-circuit primary grinders that use cylindrical steel rods as the grinding medium. 'Open circuit' implies that there is no return of coarse particles to the rod mill. The size of the feed is about 80 per cent minus 10 mm, and it must therefore have been crushed to that size in secondary and tertiary cone crushers.

The crushing system provides opportunities for the manual, photometric, or radiometric sorting of suitably sized rocks. Conglomerate ore is usually distinguishable from external waste rock, and can be sorted manually by the selection of waste pieces (or reef pieces) or photometrically. This process requires efficient screening and washing of the ore in the crusher circuits. Manual sorting is (by definition) labour-intensive, requires capital input for a sorting plant, and is falling into disuse on new gold plants. When there is a concentration of gold in run-of-mine fines, sorting is sometimes possible by the screening of the coarse fraction, from which the ore is selected by reef picking or photometric means. Recent developments in sorting technology include radiometric sorting, which depends on the association of gold with uranium in certain ores, and detects the uranium by gamma emission.

Ball mills have a lower throughput per unit capacity than rod mills but, used in closed circuit, provide the necessary fine grind for further treatment of the ore. The balls used for grinding are generally steel and are typically 100 mm in diameter. The ball loading, expressed as a percentage of the mill content, and the mill speed together determine the maximum efficiency, which can be assessed experimentally. Pebble mills are also used in closed circuit. Pebbles of ore ranging in size from 100 to 200 mm serve as the grinding medium. Pebble mills are longer than ball mills of equivalent diameter (lengths of 6 m or more are common), but the throughput is considerably lower, as is wear of the mill liners.

A typical mill configuration to treat, say, 60 kt per month would consist of a rod mill and two pebble mills, or composite ball-and-pebble mills, interspersed at appropriate points with size-classification devices such as spiral classifiers, hydrocyclones, and (in older plants) Dorr rake or bowl classifiers. In South Africa, milling is a continuous process 7 days per week, but crushing on Sundays is not permitted by law.

A techno-economic evaluation of various comminution routes carried out at Mintek in 1985 concluded that semi-autogenous run-of-mine milling with or without secondary ball milling holds significant advantages over routes that include crushing.

Concentration Methods

At various points in the milling process, certain ores may liberate large grains of gold in the size range 600 to 30 μm. Under particular circumstances, it is advantageous to separate these grains from the gangue by use of their considerably higher relative density (19.3 g/cm³ as against 2.7 g/cm³). More than a dozen South African plants still employ gravity concentration in the milling circuit. The benefits claimed are that it avoids the excessive dissolution time required for large grains; it recovers coated grains, which would resist dissolution; and it provides an opportunity for particles enclosed in pyrite or uraninite to be released so that they can be subjected to roasting or other treatment, thus releasing gold for subsequent dissolution.

Many devices are available and in use for gravity concentration, and are well described in the literature. They may be stationary (corduroy blankets or plane tables, in which gold is trapped in the riffles) or more elaborate moving devices such as jigs, moving belts, or rotating cylindrical (Johnson) concentrators. The selection of a suitable device usually involves a compromise between simplicity, recovery efficiency, labour intensity, maintenance cost, and the need for security where concentrations of gold accumulate.

One reason for the discontinuation of gravity concentration in a number of plants is the danger of mercury poisoning during the subsequent amalgamation stage. An alternative method in use on several mines is fine milling followed by intensive leaching in the presence of high concentrations of cyanide and oxygen.

A flotation circuit can be introduced at several points in the treatment plant to produce a concentration of pyrite, gold and pyrite, or gold alone. The recovery that gold can be rendered hydrophobic—and thus amenable to flotation—at the pH values prevailing in mill pulps was a consequence of work done in the Anglovaal Group on the flotation of pyrite from gold ores. The method has subsequently been adopted in the following procedures:

- in slimes-dams treatment plants immediately following the uplifting and repulping of the slimes-dam material (Ergo),
- as a primary recovery process in the eastern Transvaal following milling and prior to roasting (Fairview),
- with gravity concentration in the milling circuit (Hartebeestfontein),
- purely for the recovery of pyrite from a particular size fraction of milled material prior to gold dissolution (Venterspost), and
- for the scavenging of both gold and pyrite from filter residues that may otherwise have been sent to the slimes dam (Buffelsfontein).

Dissolution

The large-diameter cyclones used in South Africa operate on a feed of low solid-to-liquid ratio. The overflow contains 10 to 20 per cent solids, and must be thickened to between about 60 and 65 per cent solids before dissolution to avoid both excessive consumption of cyanide and a need for excessive precipitation capacity. The correct amount of unslaked lime added to the mill pulp in the thickener tanks acts as a flocculant, and assists the settlement rate. Diameters of tanks in common use
range between 23 and 61 m.

Thickened pulp is pumped to agitator tanks, which are conical- or flat-bottomed vessels typically 12 m in diameter and 16 m high. Sodium cyanide is added either as a solid or in strong solution to provide a tank solution of 0.025 per cent. Lime may also be added to produce the correct operating alkalinity. The mixture is agitated mechanically or by compressed air introduced at the bottom of the cone for a period that may be as long as 45 hours. Air provides the oxygen necessary for the dissolution reaction to take place. Two possible equations describe this process, which results in the formation of a gold cyanide complex:

\[
\begin{align*}
4 \text{Au} + 8 \text{CN}^- + 2 \text{H}_2\text{O} & \rightarrow 4 \text{Au(CN)}_2^- + 4 \text{OH}^- \\
\text{or}
2 \text{Au} + 4 \text{CN}^- + \text{O}_2 + 2 \text{H}_2\text{O} & \rightarrow 2 \text{Au(CN)}_2^- + 2 \text{OH}^- \\
2 \text{Au} + 4 \text{CN}^- + \text{H}_2\text{O}_2 & \rightarrow 2 \text{Au(CN)}_2^- + 2 \text{OH}^-.
\end{align*}
\]

The consumption of cyanide is about 0.25 kg of sodium cyanide per ton of ore, and that of lime around 1 kg, but the requirements for a particular ore and the length of agitation time are best determined experimentally. Losses of undissolved gold range between 2 and 5 per cent.

Two mines, Libanon and East Rand Proprietary Mines (ERPM), introduced cyanide solution into the milling circuit several years ago with apparent advantage; the method has not been discontinued.

**Filtration and Precipitation**

Gold-bearing solution is separated from the cyanided pulp in rotary vacuum-drum filters capable of recovering 99 per cent of the dissolved gold in a single stage. The drums are divided into twenty panels arranged on their surfaces parallel to the axes, each panel with its own set of suction pipes. A panel of filter cloth emerging from the pool at the base of the drum is coated with a 10 mm thick layer of pulp. For about three-fifths of a revolution, this pulp is washed with a gold-free cyanide solution, and gold-bearing liquor is drawn under vacuum through the filter cloth. For the last 5 per cent of a revolution, just before the panel re-enters the pool, pressure replaces suction on the panel and the pulp is blown loose. Drums as large as 7.3 m in diameter by 6.1 m in length are currently being used. A bank of 6 filters of this size is required per 100 kt treated per month.

The gold-bearing solution, now termed the pregnant solution, requires two more stages of treatment before the gold can be precipitated. The first is *clarification*, in which suspended colloidal particles are reduced to 5 ppm of solids or less by a process of filtration under pressure in Stellar filters. This may or may not follow pre-clarification by sedimentation in hoppers with the addition of flocculant. The second stage is *de-aeration* to remove dissolved oxygen, which would otherwise render the precipitation step inefficient. The equipment used is the Crowe tank, a vertical cylinder 3.66 m high and 2.13 m in diameter, within which the solution is trickled over tiers of wooden grids. A vacuum in the cylinder removes air. One such Crowe tank is able to handle 150 kt of solution a month.

Clarified, de-aerated solution is gravitated to an emulsifying tank. Zinc dust and lead nitrate are added in quantities of between 5 and 12 parts zinc, 0.5 and 1 part nitrate to 1 part gold. The amount of free cyanide and lime in the solution at this stage should not fall below specific minima. Lead in the lead nitrate precipitates on the zinc dust to form a lead-zinc couple that has certain electrochemical properties and enhances the precipitation of gold.

This process (Fig. 32), also known as *cementation*, is a consequence of the relative electropositive nature of zinc in cyanide solution: it dissolves more readily than gold and, when added to gold solution, it displaces the gold from the solution:

\[
\begin{align*}
2 \text{Au(CN)}_2^- + \text{Zn} & \rightarrow 2 \text{Au} + \text{Zn(CN)}_2^-
\end{align*}
\]

or

\[
\begin{align*}
2 \text{Au(CN)}_2^- + \text{Zn} + 3 \text{OH}^- & \rightarrow 2 \text{Au} + \text{HZnO}_2^- + 4 \text{CN}^- + \text{H}_2\text{O}.
\end{align*}
\]

Cementation is a heterogeneous redox process in which zinc is dissolved at the anodic areas of the surface and the electrons that are released serve to reduce the aurozincate ions at the cathode areas.

**Adsorption by Carbon**

Activated carbon is a highly porous material with a large intra-particle surface area. Its adsorptive properties are generally known and have been applied in gold recovery for more than a century. Carbon derived from coconut shells has proved best suited for this purpose. The mechanism of the adsorption of gold from cyanide solution has not yet been fully explained. The following theories have been advanced:

- the \( \text{Au(CN)}_2^- \) ion is itself adsorbed and held by electrostatic or Van der Waal's forces;
- the gold compound is altered in the adsorption process to some other form;
- metallic gold is precipitated on adsorption.

McDougall and Hancock have noted that activated carbon offers little scope for more direct investigation into the nature of the gold species, and leave the matter to speculation.

The carbon-in-pulp (CIP) process—carbon adsorption of gold direct from cyanidated pulp, followed by elution of the gold from the carbon (Fig. 33)—has displaced conventional filtration in every plant built in South Africa since July 1980. By January 1986, 28 plants with an annual capacity of 50 Mt were in operation, and at least six more had been proposed. In several of these plants, CIP was introduced to 'scavenge' the gold from repulped filter residues (e.g. Western Areas); others were designed to treat gold-bearing calcines from sulphide roasters (e.g. President Brand). The experience gained has led to the extensive use of CIP in slimes-dam reclamation plants (e.g. Ergo and Rand Mines Milling & Mining) and in large new plants for the treatment of underground ore (e.g. Western Deep Levels and Beatrix). Its advantages lie in its lower capital costs (up to 30 per cent); lower claimed operating costs; better recoveries in general; and higher efficiency where the ore contains minerals such as copper and nickel that interfere with zinc precipitation. Mintek has estimated that an average additional 0.03 g of gold per ton can be recovered by the process over recovery by filtration and precipitation.

![Fig. 33—Flowsheet for the carbon-in-pulp process (Laxen)](image)

After conventional milling, thickening, and cyanide leaching, the pulp must be screened at around 1 mm for the removal of oversize particles, particularly woodchips arising from milled underground timber that has found its way to the plant. Carbon adsorption takes place in a series of 4 to 8 mechanically agitated tanks. The pulp moves through successive tanks under gravity, the residence time in each tank being 60 to 80 minutes. Carbon particles, sized between 1.2 and 2.4 mm, are introduced at the last tank, and are air-lifted or pumped intermittently so that they move countercurrent to the flow of pulp. A screening step is employed for the removal and advancement of the carbon between the tanks. In this way, up to 99.6 per cent of the dissolved gold is recovered from the pulp.

The loaded carbon is washed to remove residual adhering pulp, and is then stripped of its gold charge by a process known as elution. Most South African plants use the batch AARL procedure which was developed by the Anglo American Research Laboratories, in which the carbon is washed in a hot caustic cyanide solution and then stripped with hot (110°C) de-ionized water in a single pass through an elution column 10 m in height by 1 m in diameter. The elution time is 9 hours for a 2.5 t load of carbon. The stripped carbon is reactivated thermally in a steam environment for re-use in the circuit. Gold can be recovered from the eluate by electrowinning onto steel-wool cathodes—both Mintek and AARL have developed cells for this purpose—or by zinc precipitation and smelting.

In a variation of the CIP process, the carbon is introduced direct into the cyanide-leaching tanks. Such a carbon-in-leach (CIL) plant has been commissioned at Ergo for the treatment of residues from the flotation plant.

**Heap Leaching**

The technique of heap leaching was developed in North America for the treatment of low-grade, shallow gold deposits that are mined by low-cost opencast methods. It is not used for the relatively high-grade ore mined by underground methods in South Africa. However, it is being used increasingly for the recovery of gold from surface accumulations of residue on the Witwatersrand and in small underground mining operations.

Piles of gold-bearing ore on an impervious pad are sprinkled with cyanide solution, which is collected and recycled. Part of the solution is drained off and subjected to a carbon-extraction process until the gold value in the solution falls below a practical limit.

**Pyrometallurgical Processes**

Gold in flotation or gravity concentrates rich in sulphides can be liberated for subsequent cyanidation by roasting at temperatures between 450 and 800°C. Thermal decomposition of the pyrite is exothermic in the presence of atmospheric oxygen:

\[
4 \text{FeS}_2 + 11 \text{O}_2 \rightarrow 2 \text{Fe}_2\text{O}_3 + 8 \text{SO}_2
\]

The calcine contains free gold. When arsenopyrite is present, or when ferrous sulphate (a strong cyanicide) is likely to appear in the calcine, appropriate atmospheric and temperature control eliminate the deleterious product. Roasting is a necessary pre-dissolution metallurgical stage for the recovery of gold from most Archaean and Transvaal-hosted orebodies. Rabbled-hearth (Edwards) or fluidized-bed roasters are used, and off-gas treatment is integral to the process.

Some South African plants continue to employ amalgamation for the recovery of free gold. Retorting is the process of separating and recovering mercury from pressed amalgam by heating it to above 350°C; the boiling point of mercury. Cylindrical retort furnaces, 1.5 m long by 0.3 m in diameter, mounted horizontally, taper at the outlet end to a 75 mm pipe, which is flanged to
take a water-cooled condenser. Amalgam is loaded in moulds, and the inlet end is secured by bolts clamped against an asbestos gasket. Electrically applied heat drives off the mercury vapour, which then condenses with a total loss of about 0.3 per cent. Sponge gold remaining in the moulds can be melted directly into gold bars.

Zinc-gold precipitate (gold slime) cleaned from the precipitate filters is collected into receiving vats in the security area of the smelthouse. Acid-soluble impurities, if present, are removed by acid treatment prior to de-watering by decantation and filtration. Calcining is necessary to remove residual moisture, burn off combustible impurities, and oxidize the remaining base metals, particularly lead and zinc. Electrically heated calcining furnaces may be either of the multiple tray or continuous steel belt varieties. The layer of gold slime (Table XIII) is ideally no more than 75 mm thick. Roasting at between 550 and 700°C for about 16 hours slowly changes the colour of the slime from black through dull red to brown, at which stage it is ready for smelting.

TABLE XIII
COMPOSITION OF GOLD SLIME (IN PERCENTAGES)

<table>
<thead>
<tr>
<th>Element</th>
<th>After clean-up</th>
<th>Acid-treated and calcined</th>
<th>Not acid-treated, but calcined</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>29</td>
<td>25</td>
<td>33</td>
</tr>
<tr>
<td>Ag</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Zn</td>
<td>24</td>
<td>5</td>
<td>17</td>
</tr>
<tr>
<td>Pb</td>
<td>12</td>
<td>19</td>
<td>9</td>
</tr>
<tr>
<td>Si</td>
<td>7</td>
<td>9</td>
<td>7</td>
</tr>
<tr>
<td>S</td>
<td>3</td>
<td>11</td>
<td>5</td>
</tr>
<tr>
<td>Cu</td>
<td>2</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>Ca</td>
<td>1</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>Fe</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
</tbody>
</table>

In the final stage of the metallurgical process, the gold and silver must be separated from the other constituents of the calcine by smelting (Fig. 34), in which the other metals remain as oxides incorporated into the slag. Gold melts at 1063°C, and silver at 961°C. Bullion furnaces operate at between 1200 and 1400°C. Traditional reverberatory furnaces, in which the charge is loaded in crucibles that are poured manually, have generally been replaced by the submerged-arc type. The furnace pots take between 200 and 350 kg of calcined slime with a flux of silica and borax. Three carbon electrodes are lowered into the charge, the required high temperature being generated by an arc of electric current between the electrodes, and being controlled by variation of the current. Fusion takes about 1.5 hours, and the process is completed by the withdrawal of the electrodes, tilting of the furnace, and pouring of the precious metal and slag into a series of moulds arranged in a cascade. The slag is granulated in cold water, and is collected for despatch to the Rand Refinery, where its residual precious metal is recovered. Gold bars of standard shape and mass (30 kg) contain, typically, 86 per cent gold, 10 per cent silver, and 4 per cent base metals.

If electrowinning has been used in the recovery process, the steelwool cathodes on which the gold has been deposited may either be calcined directly or subjected to acid-digestion of the iron followed by smelting.

Process Control

The gold mines in South Africa have been operating since long before the development of commercial instrumentation. The early gold-recovery processes depended almost entirely on manual control, supported by simple plant tests and the assay laboratory. Labour was plentiful and relatively inexpensive, and could be economically used for tending machinery.

Instrumentation was being developed for the chemical and petrochemical industries in the 1930s, but only in the 1950s did it first find its way into South African gold-extraction plants. Although the need had been realized for some time, the aggressive nature of the process streams, with their abrasive slurries, and the harsh plant environments precluded the use of the equipment (sensors) used in other industries. These problems, associated with a shortage of instrumentation engineers and technicians, meant that it was only by the 1970s that proper instrumentation was possible and process control realized.

The most basic task of all process control systems is to gather data from the various process instruments and to present this data in a suitable form to the process operating personnel. Since the decision to automate must be economically justified, the number of pieces of equipment in a plant will have a direct bearing on the overall control strategy. Thus, the development trends in gold plants in the 1970s and 1980s towards simpler flowsheets using large run-of-mine mills, a small number of high-capacity thickeners, and continuous downstream recovery
process led the move towards process control. The development of the on-line measurement of particle size, cyanide concentration and, most recently, gold concentration to join the ranks of flowmeters, density gauges, weightometers, and liquid-level meters has given the process engineer a full armament. Initially, the data from sensors and equipment were hard-wired to the control centres to be displayed as lights on mimic diagrams and alarm annunciators. Today, this information is collected via computers and displayed on VDU screens. The data can be collected, stored, and analysed by any number of process study packages.

The primary objective of any overall process-control system is to ensure that certain key physical variables are kept as close as possible to their target values for as much of the time as possible. It is necessary for the control system to manipulate suitable process inputs to force this to happen in spite of disturbances. The complex nature of the processes in gold extraction defies exact analysis. Therefore, the design of a control system takes place by trial and error and is highly dependent on the experience and ingenuity of the designer. Simple, standard control configurations are used as building blocks. Feedback control reduces the effect of a disturbed input by first measuring its effect on the process output and then calculating the necessary correcting input. Feedforward control attempts to counteract the effects of the disturbance generated by measuring the disturbed input. Cascade control links the other types of control. The equipment used for process control can be analogue (black boxes) or digital (computers). Standard single-variable feedback and feedforward control configurations are grouped together into small sub-systems. Owing to strong interactions between variables within a sub-system, a more sophisticated control strategy is often required. This has led to the development of multivariable controllers. A number of complex mathematical techniques, such as the Inverse Nyquist Array, is available for carrying out the detailed design of multivariable controllers.

Assaying remains an integral part of overall control of the mining and metallurgical process. Fire assay is still the primary method used. A sample of standard mass is pulverized, mixed with a litharge flux, and fused at 1100°C to concentrate the gold content in a lead button, which is then placed in a shallow magnesium oxide cupel and heated to above 850°C. The lead oxidizes and is absorbed by the cupel, leaving a clear gold bead that can be weighed.

Refining

Delivery of Bullion to the Rand Refinery

Bulls of gold bullion from the smelt, with a gold content of approximately 86 per cent, are retained in smelt-house strongrooms until collected once or twice a week by an armoured vehicle for delivery by road or air to the Rand Refinery. Situated at Germiston, the Refinery is conveniently located relative to mines, airports, and the South African Reserve Bank ( Pretoria). It is a cooperative venture of the gold-mining industry, being owned by gold producers who are members of the Chamber of Mines of South Africa. Since 1921 all the gold produced in South Africa has been refined to marketable purity at this Refinery, including the gold produced by non-members of the Chamber, who are charged a small premium for the service. Rand Refinery processes are detailed in Fig. 35.

The Refinery accepts any bullion assaying over 50 per cent gold, preferably in a tapered bar of standard dimensions and with a mass of 30 kg. Prior to refining, the precise gold content of the bars must be determined and must agree with the assay of the source mine (the depositor). Because of possible irregular distribution of impurities, each deposit is melted to a uniform liquid from which dip samples are taken for assay. The deposit retains its source identity until its mass and assay are both agreed with the depositor within specified limits. The assay tolerance is 1.5 fineness units. In subsequent refining operations, unless it has a relatively high platinoind content, the bullion is mixed with that from other sources and its identity is lost.

Chlorine Refining

The Rand Refinery uses the Miller process to produce market-acceptable 995 or 996 fine-gold bars. In this process, base-metal impurities and silver are converted to their chlorides by the bubbling of chlorine gas through the molten metal. The 2000 Hz coreless induction furnaces carry a charge of 450 kg of bullion and 3 kg of flux. Melting takes 45 minutes. With the temperature raised to 1150°C, chlorine is introduced to the melt through graphite tubes. The base-metal chlorides FeCl₃, ZnCl₂, and PbCl₂ all boil at below 1000°C and are thus fumed off, in that order, the fumes being collected by electrostatic precipitators. When fuming is complete, copper and silver chlorides form as liquids that float on the melt, and can be bailed into a crucible. After about 35 minutes, reddish-brown fumes of gold chloride are emitted, indicating the end-point of refining. A sample is taken and analysed immediately for its silver content by X-ray spectrometry. Less than 0.5 per cent silver implies an acceptable gold purity.

Refined bullion is poured into a transfer ladle and then into moulds placed on a platform scale to produce bars of 12.5 kg mass. The cooling of the bars (from the bottom up) is controlled by a soft reducing flame that is directed onto the surface of the bar; this has the effect of eliminating shrinkage cavities as well as polishing the surface. Dip samples are taken from the transfer ladle for assay. The bars must be a minimum 995,0 fine gold and weigh in the range 400 to 410 oz tr. The average fineness of bars produced in 1985 was 996,4.

Fine particulate gold is extracted from the base-metal chlorides by treatment in the Refinery's de-golding section, where sodium carbonate is used to precipitate the silver from molten silver chloride. The silver acts as a gold collector and solidifies into a button containing gold, silver, and base metals, which are recycled to the refining furnaces.

Electrolytic Refining

Platinum-group metals have high melting points and cannot be removed by chlorination. Bullion, which carries abnormal quantities of these elements—notably bullion from the Evander mines—must be refined electrolytically. This process also yields high-purity gold of 999,9 fineness, which carries a certain added market value.

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The electrolyte is acidic gold chloride, prepared either by the treatment of metallic gold with hydrochloric acid in the presence of chlorine gas:

$$2 \text{Au} + 2 \text{HCl} + 3 \text{Cl}_2 \rightarrow 2 \text{HAuCl}_4$$

or by direct dissolution of the gold in aqua regia. The cathodes are strips of 999.9 gold, alternatively pure titanium, and the anodes are 995.0 gold from chlorine refining. Under a potential difference of 0.5 to 0.7 V, the electrolyte ionizes to $\text{H}^+$ and $\text{AuCl}_3^-$. $\text{Cl}^-$ and Au are formed at the cathode, the gold being deposited on the cathode strip. Anode gold is ionized to Au$^{3+}$ and the reaction that follows,

$$\text{Au}^{3+} + 4 \text{Cl}^- \rightarrow \text{AuCl}_4^-,$$

replaces the AuCl$_4^-$ in solution.

Copper, zinc, palladium, and platinum are soluble in the electrolyte, which ultimately causes it to be discarded. Iridium, osmium, ruthenium, and silver chloride are insoluble, and collect on the bottom of the electrolytic cell as a sludge. After electrolysis, the cathodes are washed, melted, and cast into high-purity bars.

**Smelting at the Rand Refinery**

A pyrometallurgical section of the Rand Refinery treats gold- and silver-bearing 'byproducts' from gold plants, producing a gold-silver bullion for further refining. The byproducts are borax slag from mine smelting operations, concentrates, ash, screenings, computer scrap, and any other material that may contain precious metals. The throughput is about 300 t of material per month containing 450 kg of gold and 1200 kg of silver. The process includes careful sampling of deliveries; sintering; blast-furnace treatment in the presence of lead oxide (litharge) and coke, during which the precious metals are collected by lead metal; and cupellation to produce a doré bullion.
assaying about 70 per cent silver and 30 per cent gold. This is blended with mine bullion and refined by the Miller process.

The Refinery is currently replacing the blast furnace with an electric furnace, which is expected to eliminate the need for a sinter plant and bring about environmental and cost improvements.

**Disposal of Gold**

*Gold Bars*

Refined bars are required, in terms of the exchange control regulations, to be sold to the South African Reserve Bank within 30 days of production. The price paid is the average of the last two London gold fixings prior to delivery, and is remitted by the Rand Refinery to the mining companies, using the principle of first-in-first-out. The period between the despatch of unrefined bars to the Refinery and the receipt of payment by the mines may vary between 5 and 18 working days, but is usually around 10.

Bars may be held in the vaults of the Reserve Bank or the South African Mint, both in Pretoria. While the Reserve Bank is the sole selling agent on international markets, the Mint is permitted to purchase bars for resale to licensed gold fabricators and jewellers in South Africa. The Reserve Bank does not declare its specific gold holding at any time, but rather the sum of its gold and foreign-exchange reserves. On occasion it may choose to use its gold as security for a foreign-exchange loan, and several such deals have been executed in recent years.

*Krugerrands*

A special section of the Rand Refinery produces blanks of 22 ct gold coins containing exact designations of 999.9 gold plus a further one-twelfth copper. The designations currently produced are one ounce, half ounce, quarter ounce, and tenth ounce, weighing respectively 33.931 g, 16.966 g, 8.483 g, and 3.393 g. At the Mint, the blanks are struck into Krugerrands, which are returned to the Refinery's vaults for sale through the International Gold Corporation (Intergold). The Mint itself also produces pure gold 'proof' coins for sale to coin collectors at high premiums. The current premiums payable on sales of ordinary Krugerrands within South Africa are 12, 14, 16, and 18 per cent, respectively, on the four coins. These premiums accrue to the gold producers.

Krugerrands can be purchased by private individuals in South Africa from banks, stockbrokers, and financial institutions. They are legal tender in terms of the Mint and Coinage Act, which means that they can be sold at face value and need not be assayed; in effect, they are guaranteed by the State.

As a proportion of the South African gold output, Krugerrands peaked in 1978 at 20.5 per cent. The figure in 1984 was 12.0 per cent and, of all the gold output between 1970 and 1984, they accounted for 11.7 per cent. For political reasons, the production of Krugerrands in 1986 has declined to almost zero.

**Industrial Use and Fabrication**

Intergold estimates that the current annual industrial usage of gold in South Africa amounts to about 230 kg. Electronics consumes 50 kg in such applications as low-voltage connectors, printed circuit edge tabs, gold bonding wire, and thin-film metallizations. Some 30 kg goes to plating applications of a decorative nature, and the dental industry uses about 150 kg annually.

At the end of 1985 there were 250 jewellery manufacturers in South Africa, made up of pure manufacturers who deal with the trade and manufacturing retailers who produce goods for their own retail shops or to order. The sales of gold to the trade by the Mint since 1980 are given in Table XIV.

There are indications that South Africa is at present a net importer of jewellery. An Intergold-commissioned survey of South African carat-jewellery consumption early in 1984 revealed the purchasing profiles shown in Table XV.

**Economics of Gold Production**

*Operating Costs*

The typical distribution of cost elements in a Witwatersrand gold mine is shown in Table XVI.
The most common base for the expression of unit cost is tons milled. Others used for various purposes are tons mined, square metres broken, and gold produced. All have their imperfections. The cost per ton milled, for example, is affected by the amount of development and waste sorting, which depend on policy decisions by mine management. It has, nevertheless, been used as a monitor of cost performance since the early days of Witwatersrand mining, and is plotted for interest in Fig. 36. With labour the largest component of cost, the ratio of total tons milled by the industry to the total number of employees has also been plotted but on a different vertical axis. This ratio reflects movements in labour productivity in a qualitative way, since it does not allow for increasing mining depth, shorter working-shift periods, and deteriorating mining conditions.

The irony disclosed by this diagram is that, during the period of relatively low inflation up to the early 1970s, the labour productivity showed marked quantum increases. The inflation rate moved into double figures in 1972 and since then has not come down; in fact, the real rate of cost increase over this period is around 7 per cent. By contrast, productivity has reached a plateau that has defied every effort at further improvement.

The extent to which the gold price has come to the rescue of the industry is shown in Fig. 37, in which the average cost to the industry per kilogram of gold produced is compared with the revenue received per kilogram since 1970. If the extraneous effects of changes in grade and movements in the rand exchange rate are ignored, the pattern is one in which the cost lags the price by about four years. On average, the cost in any year is about 50 per cent of the price. The rate at which the rand price will be required to increase to maintain this situation would reflect a very unstable economic order. It is clear that the rise in mining costs will have to be checked. One requirement will be another quantum leap in labour productivity. Two avenues are open: further mechanization of stoping and development, which would attack the largest component of the process costs; and better labour utilization. The industry continues to pursue both approaches vigorously, but it remains necessary to ensure that capital and other operating costs do not rise to compensate for savings in labour costs.

Ore Accounting

Control of the mining operation as a whole rests heavily on the system used to account for the flow of ore from the underground stope face to the final separation of its gold content in the treatment plant.

South African accounting systems generally classify sources of underground ore according to reef (where more than one reef is mined), shaft area or zone, and ore-reserve designation. The term ore reserve has a special meaning in gold mining. It refers essentially to blocks of ore that have been made available for mining by completed raise development and to which grades, above a selected cut-off, have been assigned according to the mine’s valuation method. Valuation by skin sampling of the block, possibly adjusted with reference to a historically determined regression curve, is steadily giving way to kriging methods. The cut-off is often imprecisely refer-

---

**Fig. 36—Cost and productivity trends on Witwatersrand gold mines, 1920–1984**
red to as the pay limit, having been calculated from cost and gold-price data. This point is discussed more fully in the next section.

The ore reserve at a certain date is the aggregate of blocks of ore, each representing about 2 years’ face advance, that carry a value greater than the cut-off grade. Sources classified as ‘not in reserve’ are either blocks valued at less than the cut-off, or stope face made available by raise development after the date on which the ore reserve was fixed. This date is usually the end of the mine’s financial year.

Ore from development rock mined on reef and from ‘other sources’ completes the package. Of particular interest are the unmeasured other sources, or survey shortfall, which represents the difference between the surveyed tonnage broken underground and the ore delivered to the mill. It arises largely as a result of convergence of the reef and floor of the stoping excavation: the stoping width is therefore under-measured and this ‘error’ is transferred to the measurement of rock broken. A typical spread of ore broken underground in a given period would be as shown in Table XVII.

The ore-accounting system must now track the ore through surface treatment, taking account of stockpiling and measured losses along the way. It is possible in this way to draw up a ‘sources and application’ statement such as that shown in Table XVIII. In this simplified system, line 2 is calculated backwards from lines 6 and 7 using the metallurgical balance. The ratio of gold content, line 2/line 1, in this case 90 per cent, is the mine call factor, which was mentioned earlier in connection with the grade control.

A balance such as this is of the utmost importance in the calculation of the stope value cut-offs. If, for example, the economics of mining dictates that the gold recovery from the mill should be at least 4.6 g/t, then a

<table>
<thead>
<tr>
<th>TABLE XVII</th>
<th>TYPICAL SPREAD OF ORE BROKEN UNDERGROUND</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Percentage of ore to plant</td>
</tr>
<tr>
<td>Stopping:</td>
<td></td>
</tr>
<tr>
<td>Ore reserve</td>
<td>60</td>
</tr>
<tr>
<td>Not in reserve</td>
<td>20</td>
</tr>
<tr>
<td>Development</td>
<td>5</td>
</tr>
<tr>
<td>Measured other sources</td>
<td>8</td>
</tr>
<tr>
<td>Shortfall</td>
<td>7</td>
</tr>
<tr>
<td></td>
<td>100</td>
</tr>
</tbody>
</table>
TABLE XVIII
A SOURCES AND APPLICATION STATEMENT

<table>
<thead>
<tr>
<th>Sources</th>
<th>Tons</th>
<th>g/t</th>
<th>g</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Total sources</td>
<td>100</td>
<td>4.9</td>
<td>490</td>
</tr>
<tr>
<td>Application</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2. To plant</td>
<td>100</td>
<td>4.4</td>
<td>440</td>
</tr>
<tr>
<td>3. Less: stockpiled</td>
<td>5</td>
<td>4.4</td>
<td>22</td>
</tr>
<tr>
<td>4. sorted</td>
<td>12</td>
<td>0.8</td>
<td>10</td>
</tr>
<tr>
<td>5. To mills</td>
<td>83</td>
<td>4.9</td>
<td>408</td>
</tr>
<tr>
<td>6. Gold recovery</td>
<td>83</td>
<td>4.6</td>
<td>383</td>
</tr>
<tr>
<td>7. To slimes dam</td>
<td>83</td>
<td>0.3</td>
<td>25</td>
</tr>
</tbody>
</table>

knowledge of the elements of the balance makes it possible to determine the equivalent stope cut-off as 5.9 g/t. Alternatively, if the ore-reserve grade is known to be 6.5 g/t, the ultimate gold yield per ton milled can be calculated forwards to be 4.6 g/t.

Payability
Although often misapplied, the concept of payability is fundamental in the linking of economics and geology. In its simplest form, the term describes the percentage of ore in a mine that, when subjected to mining and treatment, would yield at least enough gold to cover the cost of production. If, therefore, we know the price that the gold will fetch and the average cost per ton of ore milled, we can obtain the mill pay limit from the formula

\[
\text{Mill pay limit} = \frac{\text{Cost per ton milled}}{\text{Gold price per gram}}
\]

We could then take this figure through the ore-accounting system to arrive at a stopping pay limit and, if the distribution of block values is known, the payability can be calculated.

Since 1968 this system has been complicated by inflationary movements in average costs as well as in the gold price and, if payability could be calculated in some believable manner, it has also fluctuated widely. But this is not the only difficulty. The actual definition of the costs to be used also generates major problems. A mining area requires a major input of capital before operating costs are incurred in the production of ore. How are these capital costs brought to account in the calculation of payability? How are ongoing development costs treated? What about overheads? Does the concept of incremental mining cost have any validity at all? Can the mine's cost-accounting system isolate these costs in an acceptable way?

Attempts have been made on several mines to calculate the contribution of particular stope panels to the overall payability\(^a\), but no convincing case for the validity of these methods has been published. In any event, 'payability' loses its meaning where the mining method limits the selectivity of blocks, as in longwall mining at depth. In those circumstances, it would be more logical to refer to percentage extraction of ore, rather than payability.

It has been proposed elsewhere\(^b\) that a resource of virgin gold-bearing ore should not be described as 'payable' unless it contains enough gold to enable an acceptable threshold rate of return to be earned on the necessary capital investment after allowance has been made for all operating costs and payments due to the State. Careful application of a criterion such as this would limit the potential for investment loss in new, unproven mining ventures. Krige has developed the concept of 'flotation pay limit' to cover this particular point\(^c\).

One further matter can appropriately be raised under this heading. It is a requirement of gold-mining leases that ore reserves should be depleted in such a manner that the average value of ore-reserve blocks mined in a period does not fall below or rise above the average value of the entire ore reserve. This is nothing more than good traditional mining practice. While it may not optimize the present value of a resource, it does in theory optimize the total profits over a mine's life. Adherence to this stipulation is monitored by the Government Mining Engineer through the ore-accounting returns that are submitted to him. In recent years, variable pay limits have tended to reduce its efficacy. In addition, it is possible for a mine to locate its shafts in the most favourable areas, so that it can 'overmine' the total 'payable' ore reserve in the lease area with impunity—but also out of necessity in order to justify the high capital cost of bringing a new deep shaft system to production. To some extent, then, the economic conditions of the 1980s have tended to overturn traditional methods of control, while at the same time emphasizing the necessity for careful analysis of the available data prior to investment decision-making.

Reclamation Mining
The ratio that defines the mill pay limit remains of fundamental importance in gold mining. The dominant influence since about 1970 has been the behaviour of the gold price, which has served to reduce this ratio consistently despite the equally consistent increase in the cost of underground mining.

There have been two important consequences for the older areas of the Witwatersrand. Firstly, it has become profitable to retreat surface accumulations carrying gold values of less than 1 g/t and, secondly, the flotation pay limit for the re-opening of abandoned underground mines has decreased to the point at which such operations are proliferating in the area between Randfontein and Springs. Reclamation of old properties is aided economically by the pre-existence of shaft and development infrastructure and the use of inexpensive, but efficient, metallurgical processes, such as heap leaching and carbon-in-pulp techniques. A case in point is Modder B, which ceased operations in 1962, was re-opened in 1982 on a limited scale, and is now producing around 15 kg of gold per month\(^d\).

Capital Expenditure, Funding, and Rates of Return
Mining ventures along the West Wits Line provide an instructive case study in the trend of funding required to bring new gold-mine lease areas to production. These are given in the Table XIX in money of the day, and reflect the amounts that were (or are expected to be) invested to the point at which the project generates enough profits to become self-financing. The major component of capital expenditure in each case is that relating to shaft systems; with increasing depths, ventilation and refrigeration have also assumed considerable importance. These
elements have also been manifest in the ongoing expenditure of existing mines faced with the need to replace mined-out shaft areas or to improve environmental conditions. In a very real sense, the gold-mining industry has become capital-intensive while remaining labour-dependent. This trend, already noted in 1974, shows no sign of abating.

## TABLE XIX
### FUNDING OF NEW GOLD MINES ON THE WEST WITS LINE

<table>
<thead>
<tr>
<th>Venture</th>
<th>Period</th>
<th>Funding</th>
<th>Source of finance</th>
</tr>
</thead>
<tbody>
<tr>
<td>Valterspost</td>
<td>1934-39</td>
<td>8</td>
<td>Equity</td>
</tr>
<tr>
<td>Libanon</td>
<td>1936-39</td>
<td>12</td>
<td>Equity</td>
</tr>
<tr>
<td>West Driefontein</td>
<td>1945-51</td>
<td>14</td>
<td>Equity</td>
</tr>
<tr>
<td>Dooornfontein</td>
<td>1947-52</td>
<td>13</td>
<td>Equity</td>
</tr>
<tr>
<td>Western Deep Levels</td>
<td>1957-61</td>
<td>73</td>
<td>Equity, minor loan component</td>
</tr>
<tr>
<td>Kloof</td>
<td>1964-69</td>
<td>48</td>
<td>Equity, minor loan component</td>
</tr>
<tr>
<td>East Driefontein</td>
<td>1967-72</td>
<td>66</td>
<td>Equity, minor loan component</td>
</tr>
<tr>
<td>Deelkraal</td>
<td>1974-80</td>
<td>145</td>
<td>Equity</td>
</tr>
<tr>
<td>Elandsrand</td>
<td>1974-80</td>
<td>312</td>
<td>Equity, temporary loan facility</td>
</tr>
<tr>
<td>Dooornfontein South</td>
<td>1980-87</td>
<td>240</td>
<td>Tax cover</td>
</tr>
<tr>
<td>Western Deep Levels</td>
<td>1980-86</td>
<td>715</td>
<td>Convertible debentures, tax cover</td>
</tr>
<tr>
<td>South Leudorno</td>
<td>1985-91</td>
<td>580</td>
<td>Convertible debentures, tax cover</td>
</tr>
</tbody>
</table>

The traditional source of pre-production capital funding of South African gold mines is equity. The industry has generally adopted a conservative attitude to gearing. Where loans were raised, they were generally small, or made convertible to equity in due course. Another source of funding is tax cover, since capital expenditure is directly allowable for tax in the year it is incurred. Where a new venture can be attached to an existing profitable operation, tax savings could contribute 70 per cent or more of capital investment. However, the State requires that the new venture should be contiguous to an existing one, and by this relatively recent stipulation has prevented tax cover from being used in several new but isolated projects.

Rates of return on capital investment have varied considerably. Table XX reflects the real rates of return earned by equity investors who

A. acquired an investment in mining companies on the West Wits Line on their incorporation, followed all rights, took dividends, and then sold the shares in June 1985;

B. purchased shares in June 1968, took dividends, and sold again in June 1985.

Because the tax status of various classes of shareholders differs, the tax payable on dividends or capital gains has been ignored for this exercise.

Several other, more detailed, studies of return to equity from gold mining have been published. In Table XIX there appears to be a good correlation between rates of return and average gold yield, with Elandsrand an interesting exception.

## TABLE XX
### REAL RETURNS ON THE WEST WITS LINE

<table>
<thead>
<tr>
<th>Mining company</th>
<th>Real return, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>A. Since inception</td>
</tr>
<tr>
<td>Valterspost</td>
<td>4.1</td>
</tr>
<tr>
<td>Libanon</td>
<td>4.6</td>
</tr>
<tr>
<td>Blyvooruitzicht</td>
<td>11.8</td>
</tr>
<tr>
<td>West Driefontein*</td>
<td>10.9</td>
</tr>
<tr>
<td>Dooornfontein</td>
<td>7.9</td>
</tr>
<tr>
<td>Western Deep Levels</td>
<td>15.3</td>
</tr>
<tr>
<td>Kloof</td>
<td>17.0</td>
</tr>
<tr>
<td>East Driefontein*</td>
<td>21.6</td>
</tr>
<tr>
<td>Deelkraal</td>
<td>3.5</td>
</tr>
<tr>
<td>Elandsrand</td>
<td>12.2</td>
</tr>
</tbody>
</table>

* Making allowance for the merger that created Driefontein Consolidated Limited in 1981

### Taxation and Lease

The lease payment is the share of profits due to the State in respect of the granting of a lease of the right to mine precious metal. The formula used is specific to a particular mine, but its form is common:

\[ Y = A - B/X \]

where \( Y \) is the percentage of profit after the deduction of current capital expenditure and a 6 per cent per annum allowance on unredeemed capital expenditure, \( X \) is the percentage ratio of profit (less capital expenditure) to revenue, and \( A \) and \( B \) are constants. The effect of this formula is to allow the mining company to deduct a certain percentage of its revenue from its profits before calculating the lease consideration. The effect for selected gold mines is shown in Table XXI.

## TABLE XXI
### EFFECT OF LEASE FORMULA

<table>
<thead>
<tr>
<th>Mine</th>
<th>Lease formula</th>
<th>Percentage of revenue deductible</th>
<th>Lease as percentage of remaining profits</th>
</tr>
</thead>
<tbody>
<tr>
<td>Harmony</td>
<td>12.5 – 75/x</td>
<td>6</td>
<td>12.5</td>
</tr>
<tr>
<td>St. Helena</td>
<td>11-110/x</td>
<td>10</td>
<td>11</td>
</tr>
<tr>
<td>Deelkraal</td>
<td>15-120/x</td>
<td>8</td>
<td>15</td>
</tr>
</tbody>
</table>

The tax formula is similar in nature and effect, but does not vary for mines that were floated in the same time period. That for pre-1967 mines is \( Y = 60 - 360/X \), and for post-1967 mines is \( Y = 60 - 480/X \). Lease consideration is deductible from profit both for the calculation of \( X \) and for the application of \( Y \). In some cases, a capital allowance is also deductible, but not in the calculation of \( X \). At present, a surcharge of 25 per cent on the tax so calculated is payable.

The formula tax is of benefit to low-grade high-cost mines, which could in certain circumstances pay no tax at all though earning profits. It discriminates against high-grade low-cost producers, which may be required to pay up to 74 per cent of profits to the State. At the time of writing, the Margo Commission is considering represen-
tations on gold-mining tax as part of its overall brief to review the South African tax structure.

A similar formula system has been used for the calculation of the amount of State assistance payable to mines classified under the Gold Mines Assistance Act of 1968 as assisted mines. This provision is also currently undergoing reconsideration.

Financial Accounting

The major asset of a gold mine is its life ore reserve, which is by nature a wasting asset. South African mines have adopted an accounting convention that recognizes this particular characteristic of the business by appropriating annually from profits amounts spent on fixed and other assets, rather than conventional depreciation. This recognizes the inevitability that major mine infrastructure will in due course become both redundant and unsaleable. All gold-mine annual accounts carry a note to this effect. The system is, as far as can be established, unique to South African mining.

Mines that hedge gold sales—a minority of South African producers—may make a point of reporting such hedging transactions with their accounts and in quarterly reports to shareholders.

Designing the Future

The century of Witwatersrand gold mining has come about because of the remarkable continuity of its reefs in both grade and extent. At enterprise level, it is possible now to combine geological and geostatistical interpretation to develop an acceptable pattern of grade distribution over a particular reef in a given lease area. In consequence, it is also possible with confidence to embark upon large-scale programmes of capital expenditure with lead times of 10 or more years before the first bar of gold is poured. The technical risk of not finding the gold or of not being able to mine it has been reduced to manageable proportions. Finance is generally available from the profits earned by existing producing operations.

The technical future of this industry can be designed, and finds its expression in long-term planning departments that now exist within the controlling mining houses.

One very vital factor in the industry still remains subject to severe risk. A long-term plan rests heavily on the assumption that the ratio of cost per ton milled to price of gold per gram—the mill pay limit (Fig. 38)—will not change significantly: put differently, that the rate of cost inflation will be matched by the rate of gold-price escalation. This proposition is acceptable, if only in the negative sense that no strong arguments exist for any other, but there are also strong economic reasons to support it. Two of these are South Africa’s dominant position as a supplier, and gold’s dominant position in the South African economy, which in an era of floating exchange rates makes it possible to adjust the rand price in response to both movements in the dollar price and to mining cost pressures.

Cartwright has identified simple faith in gold as the distinguishing characteristic of South Africa’s mining men. Unscientific though it may be, faith continues to underpin their actions, but the pin is strongly tempered by hard engineering and hard-nosed financial acumen. It is a combination that serves the industry and its people well.

Acknowledgements

The writer thanks his colleagues, in particular Richard Viljoen, Carina Lemmer, Richard Beck, Gerald Gossman, Gavin Martin, and Herbert Nijland and their staffs, for their valuable assistance in the compilation of this paper, and Professor D.A. Pretorius for agreeing to the inclusion of his map of mining activity prior to 1886.

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Fig. 38—Mill pay-limits, 1920–1984


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