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Copper in South Africa—Part II

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SYNOPSIS

This, the second and final part of a review on the subject (the first part was published in the March issue of this *Journal*), deals in detail with the 9 copper-producing companies in the Republic of South Africa. These are The Phosphate Development Corporation, Prieska Copper Mines (Pty) Ltd, Black Mountain Mineral Development Co. (Pty) Ltd, Rustenburg Platinum Holdings Ltd, Impala Platinum Holdings Ltd, Western Platinum Ltd, The O'okiep Copper Co. Ltd, Messina Ltd, and Palabora Mining Co. Ltd. The following are described for each company: background, development, geology, mining, concentration, and current situation. The last-mentioned company, being South Africa's largest producer, is dealt with in greatest detail.

SAMEVATTING

Hierdie tweede en slotdeel van 'n oorsig oor die onderwerp (die eerste deel het in die Maart-uitgawe van hierdie *Tydskrif* verskyn) handel in besonderhede oor die 9 koperproduserende maatskappye in the Republiek van Suid-Afrika. Hulle is die Fosfaat-ontginningskorporasie, Prieska Copper Mines (Pty) Ltd, Black Mountain Mineral Development Co. (Pty) Ltd, Rustenburg Platinum Holdings Ltd, Impala Platinum Holdings Ltd, Western Platinum Ltd, The O'okiep Copper Co. Ltd, Messina Ltd, en Palabora Mining Co. Ltd. Die volgende word met betrekking tot elkeen van die maatskappye bespreek: agtergrond, ontwikkeling, geologie, mynbou, konsentrsie en huidige posisie. Die laasgenoemde maatskappy, wat Suid-Afrika se grootste produsent is, word in die meeste besonderhede bespreek.

THE PHOSPHATE DEVELOPMENT CORP. (FOSKOR)[1]

This Company was formed in 1951, on Government initiative, to exploit the apatite (phosphate) resources of the Phalaborwa Igneous Complex (Fig. 1) and make South Africa independent of imported phosphates. It established the township of Phalaborwa (23,9°S 31,3°E), in the Lowveld of the eastern Transvaal, which then provided a secure base for further population growth and mining and industrial development[2]. Foskor is primarily a producer of apatite concentrate and has contributed greatly to the development of appropriate technology, but it is also a significant producer of copper concentrates.

Originally, Foskor established an open-pit phosphate mine near a celebrated outcrop named Loolekop, which has now been engulfed by the larger open-pit mine that was established subsequently by Palabora Mining Company (Palabora). However, some of the copper-bearing phosphatic ore mined by Palabora arises in areas to which Foskor holds title. In accordance with the terms of a co-operative agreement, Palabora delivers that ore to Foskor for further processing, which involves grinding in closed-circuit rod mills, and the flotation recovery of the copper minerals in cells arranged in rougher and cleaner banks. The tailings from these cells are then treated for the recovery of magnetite and phosphate.

The bornite-rich copper concentrate has a copper content of about 35 per cent, and is currently trucked to Palabora for smelting and refining under a toll agreement. The quantity of copper concentrate produced by Foskor is influenced by the exact source of its feed within the orebody mined by Palabora, but it approximates 36 kt/a. In 1982 and 1983, metallic copper contained in Foskor concentrates amounted to 11,5 and 14 kt respectively.

Foskor concentrate contributes significantly to the load on Palabora's smelter and refinery, and also the annual ouput of metallic copper from the Phalaborwa Complex.

PRIESKA COPPER MINES (PTY) LTD[3]

This Company was established in the late 1960s to mine an orebody of copper, zinc, and pyrite that lies 64 km south-west of Prieska (29,7°S 22,8°E) in the north-western Cape Province (Fig. 1). It is owned jointly by Anglo Vaal and United States Steel, but is managed by Anglo Vaal. It produces copper, zinc, and pyrite concentrates.

A local gossan occurrence had been known for many years and was thoroughly investigated by Anglo Vaal in the late 1960s. By 1969 diamond drilling had indicated the presence of about 60 Mt of ore containing about 30 per cent pyrite and grading 1,74 per cent copper and 3,87 per cent zinc, with 8 g/t silver and 0,4 g/t gold.

Prieska Copper Mines has installed an underground mine and a concentrator. The Company also arranged the construction of road and rail connections to Prieska, a water supply from the Orange River, and a connection to the Escom national power grid.

Geology

As discovered, the Prieska orebody had a strike of about 1800 m and an average thickness of 10 m, extending to a depth of at least 900 m. This massive sulphide tabular orebody dips to the north-east at about 85°, the abutting rocks being competent. The main orebody is on the north-eastern limb of an overturned synform, the hinge of which is about 1000 m deep and almost horizontal.

The mineralization is part of a stratabound zone of chemical sediments in a sequence of gneisses, and the upper 85 m of the deposit has been altered by weathering. The main sulphide minerals are chalcopyrite, sphalerite (ZnS), pyrite, and pyrrhotite, and sulphides oc-

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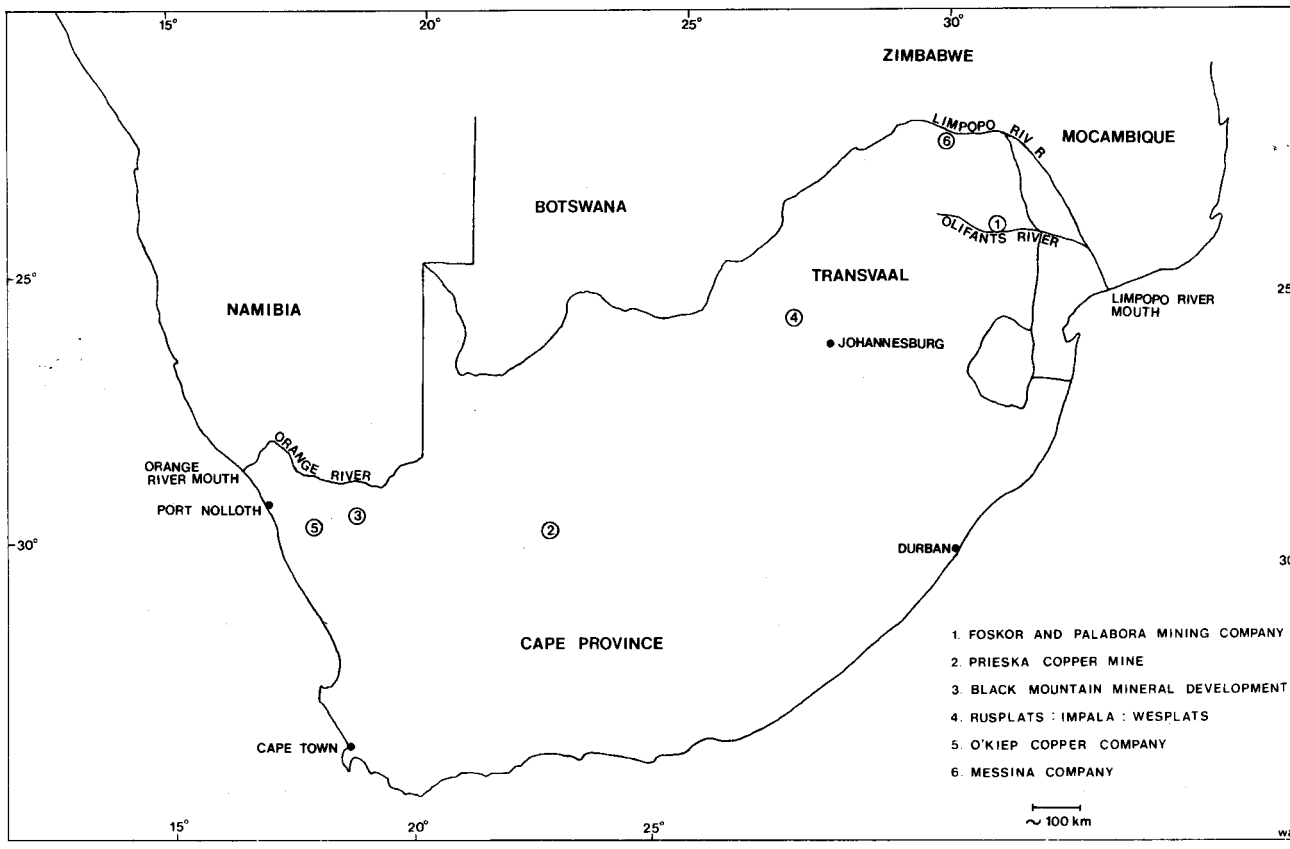

Fig. 1—Location of South African copper deposits

cupy about 55 per cent of the volume of the ore. The gangue contains calcite and dolomite. Three main types of ore can be distinguished visually: pyrite-rich, silicate-rich, and carbonate-rich.

Underground Mining

Mining started in 1974. The orebody is accessed from a main hoisting shaft, 8,84 m in diameter and now 1024 m deep, placed in the footwall about 350 m from the orebody. Haulage ways placed at 116 m levels are run from this shaft towards the orebody. A spiral roadway has also been installed, placed in the footwall between the hoisting shaft and the orebody.

Ore is blasted from holes drilled parallel to the orebody from stripping crosscuts. It is then drawn from the stopes and transported by diesel-driven load–haul–dump vehicles to tips supplying a tracked electric haulage system with 20 t bottom-discharge hoppers. This system transfers the rock to the main ore-passes, which supply a 150 mm gyratory crusher, from where the rock is hoisted to the surface.

In 1983 2,96 Mt of ore was crushed and hoisted, and, at the end of June 1983 the reserves of fully and partly developed ore amounted to 6,3 Mt.

Concentration[4]

The design of the concentrator is based on several years of laboratory and pilot-plant investigation, which was conducted in the late 1960s and early 1970s. The investigations were directed firstly to the successful flotation of chalcopyrite from depressed sphalerite and pyrite, followed by the flotation of reactivated sphalerite and the recovery of pyrite. An important objective of this sequential scheme was to ensure that the zinc content of the copper concentrate would be less than 6 per cent.

Secondary crushing and screening are controlled to produce pebbles of 25 to 100 mm for use as a grinding medium, and a rod-mill feed smaller than 25 mm. Grinding of the ore to a size suitable for flotation is accomplished in two parallel systems, each consisting of a rod mill in open circuit feeding two pebble mills in closed circuit.

The pH value of the pebble-mill slurry is adjusted with lime to 10,4 to suppress the pyrite and sphalerite during the copper-flotation step. Subsequently, the sphalerite is activated in the zinc-flotation step by the addition of copper sulphate.

In 1983, 2,96 Mt of ore was treated at an average grade of 1,1 per cent copper and 3,3 per cent zinc, yielding, *inter alia*, 95 kt of copper concentrate at 29,1 per cent copper and about 4 per cent zinc. The copper concentrates are thickened and filtered, and are then transported to O'okiep for smelting or to Saldanha Bay for export.

Current Situation

In the poor economic climate of the past few years, the exertions of Prieska Copper Mines in developing an efficient mine and a tricky differential flotation system have not been well rewarded in terms of financial revenues. The Company now judges that ore below the 957 m level would not adequately compensate the investment required to recover it; furthermore, no new and accessible orebodies have been discovered. Accordingly, in September 1983, the *Annual Report* announced that

mining operations would cease during the latter half of 1985.

BLACK MOUNTAIN MINERAL DEVELOPMENT CO. (PTY) LTD

This Company operates an underground mine and concentrator at Aggeneys (29,20°S 18,8°E) in the remote, arid Bushmanland area of the north-western Cape Province (Fig. 1). It was formed originally by the Phelps Dodge Corporation of the U.S.A., and now produces argentiferous copper and lead concentrates and also zinc concentrates.

The farm Aggeneys is only about 100 km to the north-east of the O'okiep copper district: consequently, local superficial indications of copper mineralization were recognized many years ago and attracted recurrent interest. The results of all the investigations conducted there were disappointing, and, like the foreman miner in the passage from Simon van der Stel's diary quoted in Part I, such prospectors and mining companies 'expressed the opinion there would be no result and . . . lost heart'. Fortunately, the discovery of the Prieska orebody encouraged continued prospecting in the north-western Cape Province, and Phelps Dodge elected to examine the Aggeneys prospect, which others had rejected.

Development

A well-sited drilling programme was initiated in 1971 and by 1973 had revealed three complex orebodies of lead, zinc, and copper identified as Broken Hill, Black Mountain, and Big Syncline. Copper occurs as chalcopyrite, but not by itself in payable quantities. An ore tonnage of about 175 Mt was indicated, exploitable by either open-pit or underground mining.

The Broken Hill orebody held most promise, being richest in lead, zinc, and silver, and an adit was driven in 1974 to verify drilling results and obtain a sample for bulk testing. These tests were successful. In 1977, Gold Fields of South Africa acquired a 51 per cent interest in Black Mountain, and thereafter managed the development and operation of the mine and concentrator. Lucid and comprehensive accounts of these activities, providing valuable material for future case studies, have been published by Gold Fields executives[5,6]. Supporting facilities that were established include an airfield and township, a water-supply pipeline from Pelladrift on the Orange River, and a connection to Escom's national power network. In addition, a 150 km heavy-duty road for the export of concentrates was built to a new siding on the Sishen–Saldanha railway. Production started in January 1980.

Geology

Two conformable orebodies have been found at Broken Hill, an upper and a lower, separated by barren schist that is 5 to 30 m thick. The orebodies are composed of high-grade, massive sulphide lenses and mineralized magnetic schists. The sulphides include galena and pyrrhotite, with lesser amounts of sphalerite, chalcopyrite, and pyrite. The upper orebody is about 25 m thick, and the lower about 4 m thick. In general, the orebody plunges at 23° in a direction N 64°E, and the dip varies from 0 to 55°. The length on strike is about 1600 m.

The metamorphic rocks of the area have been highly folded. The hanging wall of the upper orebody is generally composed of schists, quartz–micas, and magnetic quartzites, with schists and quartzites constituting the footwall. In the lower orebody, the hanging wall and footwall are rich in magnetic material.

In 1980, the *in situ* ore reserves were estimated as 38 Mt at 6,35 per cent lead, 2,87 per cent zinc, 0,45 per cent copper, and 87 g/t silver.

Underground Mining

Although a potential for open-pit mining has been recognized, initial efforts have been directed to the exploitation of the Broken Hill orebody by underground methods. The orebody has been diamond-drilled on 40 m centres. The approach to the orebodies has been provided by an 1800 m 11° spiral decline, with access levels at 35 m intervals, and a 5,5 m by 4,40 m three-compartment vertical shaft.

Two mining methods have been used: large blast-hole stoping for the large, steeply-dipping high-grade orebodies containing about 3,5 Mt, and, at a later stage, cut and fill for the lower, flatter formations. The stopes are backfilled with mixtures of classified tailings and dune sand, with or without cement as appropriate.

Ore is transferred from the stopes to an underground jaw-crusher sited near the pit bottom by Wagner ST 8 loaders and Wagner MT 425 25 t haul trucks, which deliver to ore-pass systems. As the ore passes are located in a comparatively soft footwall and have to handle large dense, abrasive rocks, much thought was given to their design and reinforcement[6].

The task of the concentrator is aggravated by the complex mineralization and by its occurrence as massive sulphides or disseminated in magnetite amphibole. Accordingly, thorough grade-control procedures and blending facilities have been installed in an effort to stabilize the character of the mill feed. Experience has disclosed that any backfill finding its way into the ore-stream can adversely affect material-handling and flotation control[7].

All facets of the mining operation are monitored and controlled from a master control centre, which is located on the surface near the shaft bank.

Concentration

The concentrator is designed to produce, by sequential flotation, copper, lead, and zinc concentrates in that order, with a high recovery of silver into the lead and zinc concentrates. The initial production targets are shown in Table I.

TABLE I
INITIAL PRODUCTION TARGETS AT BLACK MOUNTAIN

Material	Target, kt/a	Remarks
Feed	1130	Easy potential for 50 per cent expansion
Copper concentrate	24	Mainly chalcopyrite
Lead concentrate	132	
Zinc concentrate	34	

The complex regime for reagents, and the flowsheet, which includes on-stream analysis and a control system, were developed after extensive bench and pilot-plant tests

on drill cores and bulk samples. The inherent difficulty of the concentration process is aggravated by several local factors, including a tendency for pyrite and pyrrhotite to oxidize and the occurrence of two types of ore in the mill feed: a massive sulphide with a lead content of up to 50 per cent, and a much harder, lower-grade mineralized magnetite amphibole.

Ore is crushed in an underground jaw-crusher through 150 mm, and crushing is completed on the surface with closed-circuit secondary and tertiary cones. A 3,2 by 4,9 m open-circuit rod mill, followed by a 4,9 by 5,2 m closed-circuit ball mill, grinds the crushed ore to about 72 per cent smaller than 75 μm, and then supplies it to the copper-flotation system as a slurry with a solids content of 33 per cent.

In the flotation of copper, sulphurous acid is added to the first conditioner to activate the chalcopyrite and depress the galena. Provision is also made for the regrinding of any low-grade rougher concentrates arising from areas of unusually fine mineral dissemination.

The design objectives and the results achieved after the first six months of operations are compared in Table II.

TABLE II
RESULTS AT BLACK MOUNTAIN AFTER SIX MONTHS' OPERATION

Item	Design	Operation
Copper, %	23,0	24,15
Lead, %	3,00	14,06
Zinc, %	3,00	2,52
Silver, g/t	1 000	1 598
Copper recovery, %	76,00	37,00
Concentrates, t	11 338	4 620

The operational results in respect of copper were very disappointing in terms of production rate, contamination by lead and zinc, and recovery. It was discovered that the performance of the collector was affected adversely by the unexpectedly low values of redox potential, which were probably attributable to corrosion of the grinding medium and oxidation of the pyrrhotite. Other negative factors arising from the operation itself included the appearance of cemented backfill in the concentrator feed, which was indirectly responsible for weak froths, and the use of a high proportion of water reclaimed from the tailings dam. The original expectation was that the retention time in the tailings dam would be sufficient to allow the residual reagents to decompose; however, in the early stages of operation, this water was found to contain troublesome organic material.

The following factors were identified as both critical and capable of manipulation to improve the recovery of copper and reduce the contamination by lead:

(a) increased redox potential of pulp through aeration and the use of a corrosion-resistant grinding medium,
(b) reduction of the proportion of recycled water and contamination of the feed by backfill,
(c) a modified regime for collectors and frothers, and
(d) the judicious use of pulp heating together with the addition of sulphurous acid to depress the galena.

These measures continue to receive attention, and have increased the copper recoveries to at least 67 per cent, with the combined lead and zinc contamination approaching the original expectations. Equally important, the revenues derived from copper per ton of ore processed have now met the original expectations[7].

The copper concentrates are dewatered on drum filters, and are transported by road to a siding on the Sishen–Saldanha railway, from where they are exported through Saldanha Bay. The lead concentrates are also exported, but the zinc concentrates are consumed in South Africa.

Current Situation

As long as the Company receives reasonable prices for its products, its long-term future seems assured, possibly including opportunities for open-pit mining and local smelting.

In these connections, it is worth noting that metallic lead and copper, and zinc concentrates, are currently produced from a lead–copper–zinc deposit at Tsumeb in South West Africa. However, the mineralization at Tsumeb is much more complex and difficult than at Aggeneys.

PLATINUM COMPANIES PRODUCING COPPER

Rustenburg Platinum Holdings Ltd (Rusplats), Impala Platinum Holdings Ltd, and Western Platinum Ltd (Wesplats) produce copper, although their principal objective is the recovery of platinum-group metals (PGM) from the Merensky Reef of the Bushveld Complex, which outcrops over more than 130 km in the Rustenburg area of the Transvaal and Bophuthatswana, about 110 km north-west of Johannesburg (Fig. 1). Iron, nickel, and copper sulphides are associated with the platinum minerals.

Rusplats, a subsidiary of Johannesburg Consolidated Industries, and Impala, a subsidiary of Gencor, produce some metallic copper in local refining plants that they have erected. Wesplats, which operates on a smaller scale, is a subsidiary of Lonrho and Falconbridge Nickel. At the time of writing, its pyrometallurgical concentrates are processed in Norway, but the erection of a refinery in South Africa is under study.

The copper produced from these three sources is accessory to the platinum output, which is adjusted from time to time according to market conditions. Competition is strongly entrenched in the platinum industry, and producers, both here and abroad (with the exception of Wesplats), do not publish much information on their processes, production, and costs. However, copper revenues, unlike nickel revenues, play only a minor part in their economics[8].

Geology

The Bushveld Complex is an extraordinary assemblage of igneous rocks, which covers an area of about 650 000 km^2 in the central Transvaal, and contains immensely valuable quantities of platinum, chromium, vanadium, nickel, copper, tin, fluorspar, andalusite, magnesite, and asbestos. Much of the complex is buried, but the wide noritic rim that is exposed in the Rustenburg area is notable for a number of distinct and persistent bands or reefs.

The Merensky Reef is one of these bands and can be traced in outcrop over 375 km. It is composed of a narrow pseudo-stratified group about 95 cm wide, typically including a dark pyroxene gabbro underlain by a coarse

pyroxenite and a thin sheet rich in chromite. It contains PGM minerals and associated iron, nickel, and copper sulphides, most commonly pyrrhotite, pentlandite, and chalcopyrite, which grade about 0,1 to 0,2 per cent nickel and 0,1 to 0,15 per cent copper. Cobalt is sometimes present in significant amounts.

The UG-2 Reef, found about 150 m below the Merensky Reef, also contains PGM and is mined on a comparatively small scale. However, its copper content is negligible.

Mining

The Merensky Reef dips towards the centre of the Complex at an angle of 10 to 25°, tending to become shallower at depth. Its limit has not been established. The thermal gradient is high.

Its regularity in all respects guarantees ample reserves for many decades to come, and facilitates systematic mine planning and operations. The shallower areas of the Reef are selected for development, and labour-intensive stoping methods are used, which resemble those developed on the Witwatersrand goldfields. Inclined haulages to higher areas of Reef are spaced along strike, and are succeeded beyond depths of about 200 m by vertical shafts.

By any standard, the scale of underground mining is immense: Rusplats alone has the capability to produce about 17 Mt of ore annually from three major mines. Impala operates four mines, all in Bophuthatswana.

Concentration[9,10]

Rusplats operates a number of concentrators at their various mining sections, but Impala and Wesplats have each installed one central concentrator to serve all their mining operations.

Some platinoid minerals are recovered by gravity methods. In general, however, crushing, grinding, and flotation are directed to the production of a bulk sulphide concentrate containing part or all of the PGM and the iron, copper, and nickel minerals. By the standards of the copper industry, the copper content of the concentrate is low and the gangue content is high. The following is a typical concentrate analysis:

Cu	2,3
Ni	4,0
Fe	15,0
S	10,0
SiO_2	39,0
MgO	15,0.

The gangue usually includes refractory olivine.

Substantial milling capacity is installed; for example, the Impala mill includes nineteen 4,3 by 4,9 m closed-circuit ball mills, each with its own flotation section. The total installed milling capacity for ore from the Merensky Reef is estimated to be more than 30 Mt/a, i.e. about one-third of South Africa's total grinding capability for gold ore.

Recovery of Copper[8,9]

As the first stage of their concentrate treatment, all three companies process flotation concentrate to matte in electric furnaces[11]. Electric furnaces have replaced blast furnaces, and are preferred because of their negligible production of waste gas and their ability to achieve and control the high temperatures required by the local slags. The slags may be quenched, reground, and floated for the recovery of the entrained values.

The smelter matte collects most of the PGM, and typically contains 10 to 15 per cent copper, 13 to 25 per cent nickel, 45 per cent iron, and 30 per cent sulphur.

Although the Orford process was used previously for the separation of the nickel from the copper, smelter mattes are now blown in Peirce–Smith converters to produce an iron-free converter matte containing about 28 per cent copper, 48 per cent nickel, 21 per cent sulphur, and the PGM. This matte is solidified and, after being ground, part of the PGM can be recovered physically as a magnetic phase. The matte, with or without magnetic treatment, is then exposed to an agitated pressure leach to dissolve the nickel and copper.

After various processing steps, the copper is recovered by electrowinning, i.e. by the electrolysing of a purified copper sulphate solution in a cell between a lead anode and a copper starter cathode.

Copper cathode derived from Rusplats mines is produced at a base-metal refinery that was constructed by Matthey Rustenburg Refiners at Rustenburg[10]. Impala produce a matte in Bophuthatswana containing about 28 per cent copper and then refine it at Springs, near Johannesburg. As mentioned earlier, Wesplats is considering constructing a local refinery.

The total installed cathode-production capacity of the three companies will probably exceed 25 kt/a within the next few years.

Current Situation

The production of copper by the platinum companies will be influenced by the demand for PGM and by the degree to which mill feed is drawn from the UG-2 Reef rather than from the Merensky Reef. Even so, electrowon cathode from these sources, which is suitable for a variety of mechanical applications, is likely to be a reliable supply for many years to come.

THE O'OKIEP COPPER CO. LTD

This Company was established in 1937 as a fully owned subsidiary of the Newmont Mining Corporation of the U.S.A. At the time of writing, the Newmont interest has been reduced to 49 per cent and Gold Fields of South Africa has acquired 25 per cent. The Company mines copper ores and produces blister copper for export near the town of Springbok (29,7°S 17,0°E), in that part of the north-western Cape Province known as Namaqualand[12] (Fig. 1).

As mentioned earlier, the copper ores of Namaqualand attracted the attention of the Dutch East India Company in the seventeenth century, but the isolation of the area and its aridity frustrated development. Towards the middle of the nineteenth century, an expanding demand for copper in Europe prompted further interest in Van der Stel's discoveries and in the mineral resources of the Richtersveld area, which is further north towards the Orange River[13,14].

In 1846, the South African Mining Company, the first of its kind in South Africa, was formed in Cape Town

with a capital of £1000 to exploit the copper ores of Namaqualand. In 1850, Philips and King initiated successful mining at Springbokfontein in Namaqualand, and many prospectors bent upon developing copper mines then entered the district. By 1854 a speculative fever had spread through the colony, and many new mining companies had been created, all dedicated, nominally anyway, to a search for copper ores and their commercial development. Public excitement prompted the Cape Government to send Commander Nolloth to survey the coast of Namaqualand, and a geologist, Mr A. Wyley, to appraise the copper district.

The former identified Robbebaai as a potential port for the shipment of ore, and it was named Port Nolloth in 1855. (Regrettably, it cannot accept large ships safely.)

The latter reported pessimistically on the cost of labour and transport, recognizing, however, that 'almost entire hills in the Concordia country contain a greater percentage of ore than average produce at the Cornish mines'. As to the utility of lower-grade ores, Wyley was sceptical: 'Much has been said and written about inventions for concentrating or reducing these poorer ores, so as to put them in a more transportable form, but I have no faith in any of the plans hitherto proposed'[14]. Evidently he did not foresee froth flotation!

In 1863 the Cape Copper Mining Company started exploiting the deposits near Springbok. They exported high-grade sorted ore over a narrow-gauge railway to Port Nolloth and from there to Swansea, in Wales, for smelting. Later on, eschewing Van der Stel's use of local charcoal, this Company imported Welsh coke and used it to produce a matte with a copper content of about 50 per cent for final treatment in the U.K. This is probably the first South African example of the export of 'processed minerals', a policy that is much favoured today.

Operations of this kind continued until 1919, when the post-war slump forced a closure, and 'distress and poverty reigned supreme through Namaqualand'[14]. Operations were restarted in the 1920s when flotation and leaching processes for oxidized ores and tailings dumps were introduced to the district, and some blister copper was produced. By 1939 all the significant properties and operations in the area had been acquired by the O'okiep Copper Company Limited.

This Company subsequently conceived and implemented many new projects, all directed towards the beneficial utilization of the local copper resources. A water supply from the Orange River was provided, and a connection made to Escom's national power grid. The old railway to Port Nolloth was closed in 1942, to be replaced by road transport to the railhead at Bitterfontein, about 200 km away.

Geology[15]

The copper district in which the Company operates covers an area of about 1500 km^2. Early prospecting and mining were focused mostly on outcrops exhibiting copper staining and regions bearing evidence of ancient working.

It is now known that the copper mineralization is associated with basic rocks (hypersthenite, norite, or mica diorite) that have transgressed, in the form of dykes or plugs, a sequence of folded metamorphic rocks and younger granite. Unfortunately, not all the basic bodies have been mineralized to ore grade, but are irregular in form and mass, many of the larger extending below a depth of 1200 m. Their distribution and copper grade do not follow a simple, regular pattern, although it is now recognized that their emplacement was influenced by a series of shear folds, which are mainly antiform and locally termed 'steep structures'.

The common sulphide minerals in these mines are chalcopyrite, bornite, chalcocite, pyrrhotite, and relatively sparse pyrite and galena. Oxidized minerals occur in outcrops but generally disappear below a depth of 40 m. In the past, massive sulphide lenses were mined and surficial oxidized ores were leached on a comparatively small scale. More recently, operations involve the mining of ore in which the sulphides are finely disseminated. Magnetite and ilmenite are common minerals in the ore.

Exploration

The discovery and evaluation of such orebodies, once the obvious outcrops have been exploited, require skilled professional attention and advanced exploration techniques. Accordingly, the O'okiep Copper Company has established a strong, well-equipped exploration department. More than 1500 bodies of basic rock have been identified within 15 km of O'okiep, many of them buried, and by 1975 more than twenty orebodies had been discovered, ranging in size from 0,14 to 21,5 Mt of ore.

Development Strategy

Because of the wide, irregular distribution, mass, and grade of the many orebodies in the district, the Company has dispersed its operations over a large area. Thus, concentrators have been erected at or near long-life mines, and ore is road-hauled to them from smaller mines. The concentrates are then trucked by road to one central smelter.

Mines have been closed, new ones reopened, and equipment relocated as circumstances required. In 1981, eight mines were in operation, collectively producing 1,7 Mt of ore and organized in two divisions: Nababeep and Carolusberg. Each division possessed a concentrator, and both concentrators supplied a smelter at Nababeep. The installations were connected by the Company's own road system, on which it operated a fleet of 370 vehicles.

Mining

Three mines are in operation at the time of writing: Hoyt, Spektakel, and Carolusberg. Because of the current difficult trading conditions, developmental work on the first two has been suspended, and their operations will terminate in 1984. Mining will then be restricted to Carolusberg.

At Carolusberg, a valuable 'deep' orebody has been discovered below the higher levels that were worked in the past. This orebody is understood to contain about 15 Mt of ore, averaging about 2 per cent copper, and grading from about 1,5 per cent copper at the top, 850 m below the surface, to between about 5 and 6 per cent at the 1500 m level[16].

The development of Carolusberg entails exacting rock-mechanics studies and the installation of a special refrigeration facility. A 1690 m single-life shaft system

ed in Table III. The mill grades vary somewhat, but at Carolusberg they lie in the range 1,3 to 2,0 per cent. At Spektakel they varied between 0,75 and 1,8 per cent copper according to the extent of oxidation.

TABLE III
CONCENTRATION PLANT AT O'OKIEP

Location	Rating, kt/m	Status
Nababeep	90	Closed down
Spektakel	40	To close down
Carolusberg	120	Operational

Crushed ore from underground is passed through closed-circuit cone crushers on the surface. Fine crushing is followed by two-stage closed-circuit grinding in ball mills. The dimensions of the mills at Carolusberg are 2,8 by 3,6 and 2,8 by 3,0 m, and the product size is about 100 per cent smaller than 300 μm and 10 per cent smaller than 75 μm. Lime is added to the milling circuits to adjust the pH value to 10 or 11, and the slurry is then subjected to froth flotation. The concentrates are then thickened and filtered, and trucked to the smelter at Nababeep.

Pertinent concentration statistics are given in Table IV.

TABLE IV
CONCENTRATION AT O'OKIEP

Item	1983	1982
Ore treated, Mt	1,58	1,67
Copper, %	1,49	1,41
Concentrate produced, kt	63	63
Copper, %	34	32
Copper recovery, %	89,6	86,6

Smelting

Since 1940, the Company has operated a smelter at Nababeep, employing conventional processes. The principal equipment includes a coal-fired reverberatory furnace for the production of matte, two Peirce–Smith converters, and a casting machine that produces 850 kg cakes of blister copper.

O'okiep copper has a gold content of about 6 g/t and a silver content of 100 g/t, and is generally free of deleterious impurities. The ingots are trucked to Bitterfontein, from where they go by rail to Cape Town for containerized export to Japan and Europe.

Pertinent smelter statistics are shown in Table V. The toll concentrates are obtained from Prieska Copper Mines, about 400 km to the west, and provide almost half the feed to the smelter. This supply will terminate in 1984. operation, the Company planned a reduction in the labour force from 2855 to 1500 by the end of 1984, with future mining and concentrating operations based solely on the Carolusberg Deep Ore Project. The substantial capital requirements of this Project have entailed heavy borrowing, with Government guarantees and onerous obligations to the financial institution involved.

Although the Company investigated the erection of an electrolytic refinery near Cape Town, the results were negative. The Company has a 27,5 per cent interest in a zinc deposit at Gamsberg, but judges that higher prices are necessary for project viability.

The Company recently announced the development of a process to make cement from slag, which is expected to reduce the cost of backfilling at Carolusberg by 2,5 million rands a year.

MESSINA LTD

In 1905, the Messina (Transvaal) Development Company was incorporated in London to exploit the copper deposits at Messina (22,3°S 30°E) (Fig. 1). In 1950 it was reconstituted in South Africa under the name Messina Limited, since when it has mined and concentrated copper ores. These ores were first smelted at Swansea, in Wales, but for many years now the Company has produced high-quality fire-refined ingot in its own smelter at Messina. Messina Limited is now a diversified industrial and mining group, and is no longer dependent upon copper mining at Messina itself.

The copper deposits of Messina were discovered and extensively mined and smelted by native people long before the arrival of Europeans[17,18]. Such operations ceased in the latter part of the nineteenth century. The word *Musina*, meaning copper, or spoiler because of copper's weakening effect on iron, survives in the name of the town and the Company.

In 1903, rumours of the abandoned workings attracted an exploring party under Colonel J.P. Grenfell, who subsequently formed the Company and acted as Chairman for many years. Between 1914 and 1916, Mr Herbert Hoover, later to become President of the United States, was a member of the board[19].

The town of Messina, which was established by the Company, is the most northerly in South Africa. It is now an important regional centre connected to the South African rail and power network. The Great North Road to Zimbabwe, a few kilometres to the north, runs through it. Reliable water supplies are drawn from the nearby I impopo River.

Geology[20]

The copper orebodies of the Messina area are associated with a linear fault, about 16 km long, that runs in a north-easterly direction and cuts through metamorphosed and highly folded gneisses of varying composition, which constitute the Limpopo Mobile Belt. The principal oxidized mineral is malachite, and the sulphide minerals are chalcopyrite, bornite, and chalcocite.

Mining[21]

Five orebodies have been identified and mined: Messina itself, which lies near the centre of the fault, and Spence and Artonvilla, which are fairly close together on the north-eastern limb, and Harper and Campbell, which are spaced along the south-east extension. The orebodies are irregular, featuring lodes disposed in various directions and wandering shoots, to which mining methods have to be adapted. The orebodies at Messina, Harper, and Campbell are associated with mineralized breccia pipes. Large irregular orebodies were previously mined by shrinkage stoping, but more recently bench-and-trail stoping has been practised.

In the near future mining will be restricted to the Harper orebody. The Harper deposits are at the intersection of the Messina fault and the western limb of a syncline of metaquartzite and metasediments surrounded by granite gneiss. The ore shoots follow irregular breccia bodies along the upper contact of the metaquartzite.

Electric and diesel locomotives transport the ore to the main shaft, the lowest hauling level being 884 m below surface.

In 1983, 719 kt of ore was produced at an average grade of 1,59 per cent copper. At the end of 1983 the proven ore reserves amounted to 2,9 Mt at 1,37 per cent copper.

Concentration

A central concentrator has been erected at Messina. Waste is rejected from the material larger than 38 mm by hand-sorting, to give about 80 kt of mill feed per month, grading about 1,15 per cent copper.

A flotation feed sized at about 15 per cent larger than 500 μm and 30 per cent smaller than 40 μm is produced by grinding in six closed-circuit ball mills.

Concentrates are recovered by flotation thickening and filtration, and typically grade between 36 and 40 per cent copper at a recovery of about 92 per cent and a moisture content of about 10 per cent.

Metal Production

The smelting plant was erected many years ago, and was designed to permit operation without heavy cranes to handle the ladles of matte, metal, or slag. The principal items of equipment are as follows:

(1) a 6,1 by 21,8 m coal-fired reverberatory furnace, fitted with a waste-heat boiler and capable of producing about 100 t of matte per day at a copper content of about 50 to 55 per cent;
(2) a stationary converter, equipped with pulverized-coal burners and waste-heat boilers, capable of producing about 35 t of blister copper per day;
(3) a 65 t refining furnace of reverberatory pattern using airblowing and poling, followed by a copper-ingot casting wheel.

Slag from the reverberatory furnace is skimmed into sand beds, from which it is removed as a solid to the waste dump. Matte is delivered to the converter via a connecting launder. The tuyères on the stationary converter cannot be lifted above the bath; accordingly, a special oil-cooled plugger system was installed to prevent the furnace contents from flowing out when the blast is turned off. The converter slag is skimmed to a bed, and then crushed before being recycled to the reverberatory furnace.

The fire-refining is regulated by reference to the appearance and internal porosity of solidified test samples, and is directed to the production of sound finished ingots with an oxygen content of about 400 g/t. The high purity of Messina's fire-refined copper has established an excellent international reputation.

In 1983, 8710 t of ingots was produced from Messina concentrate and scrap, and 2856 t from reverts and toll concentrates.

Current Situation

Inflationary cost pressures and poor prices have injured Messina severely during the past six years, and losses were recorded during four of them. State assistance in the form of a repayable interest-free loan was provided in 1978, and the final instalment was drawn in 1983. Economic pressures continue; for example, in 1983 the production costs averaged R2007 per ton, as against an average selling price of R1668 per ton.

As defensive measures, Messina ended operations at Artonvilla in 1983, and scheduled the termination at Campbell and Messina itself for June and September 1984. Ore will then be drawn from the Harper shaft alone. The smelter has also been closed, and arrangements have been made to deliver the concentrates to Palabora Mining Company.

The *Annual Report* for 1983 mentions that negotiations with the Government for further State assistance in respect of mining at Messina are proceeding, and that their outcome will influence plans for the curtailment of operations.

PALABORA MINING CO. LTD

The Palabora Mining Company Limited (Palabora) was registered in 1956 to acquire title to a copper deposit at Phalaborwa (23,9°S 31,1°E) in the north-eastern Transvaal (Fig. 1). Its promoters were The Rio Tinto Mining Company of South Africa Limited and the Newmont Mining Corporation.

The word *Phalaborwa* is a local native word, generally interpreted as meaning 'better than the south'. This odd phrase is thought to signify the satisfaction of a tribe that, while wandering from the south, discovered and were well pleased with the local iron and copper resources, which they then mined and smelted. Much evidence of their activities survives, but production ceased in the last century when European articles became available[20,22,23].

The Phalaborwa area always attracted attention, both from natives and from Europeans. In an otherwise flat sea of bush, it was distinguished by an isolated hill, known as Loolekop, and a number of nearby prominent rocky koppies. Karl Mauch wrote of the area, and in this

century the celebrated name of Dr Hans Merensky is intimately woven with the discovery of its vermiculite and phosphate resources.

Development

European development was hindered until the 1950s by its isolation, hot summers, and endemic fevers. In 1951, however, Foskor was established to attempt large-scale, profitable development of the igneous phosphates on the flank of Loolekop, and established a settlement from which has grown a flourishing and healthy town[2].

In 1952, the Geological Unit of the Atomic Energy Board discovered the radioactive mineral uranothorianite on Loolekop, and then initiated thorough investigations. These investigations were completed in 1956, indicating that the exploitation of the uranium resources alone would be uneconomic but suggesting that Loolekop, with its old copper workings, might well be the outcrop of a significant copper orebody.

As indicated earlier, Rio Tinto and Newmont then formed Palabora Mining Company to acquire and further explore this prospect. By 1962, Palabora, which is managed by Rio Tinto, had completed an extensive drilling and bulk-sampling campaign. The results expanded geological knowledge of the structure, and confirmed the existence of at least 286 Mt of ore at an average grade of 0,69 per cent copper at a cut-off grade of 0,3 per cent copper, mineable by open-pit methods to a depth of 366 m. A pilot plant had been erected, and a copper flotation concentrate had been recovered successfully from the bulk sample.

By June 1963 these findings had been audited by the Bechtel Corporation and Western Knapp Engineering Company of the U.S.A., who together confirmed that profitable exploitation was feasible. In 1963 a prospectus was issued inviting capital subscriptions for the establishment of a major open-pit mining and reduction complex. The invitation was successful, and construction began soon after, together with complementary arrangements with the appropriate authorities for a water supply from the nearby Olifants River and for rail and power connections to the national networks. Copper production started in 1966.

The concept underlying the design of the production complex was the production of blister cakes, or fire-refined anode, for electrolytic refining elsewhere. At that time all of South Africa's requirements of electrolytic copper were being imported from a potentially risky source—Zambia. Accordingly, the Department of Commerce asked Palabora to consider building a local refinery, and in due course this request was accepted. Furthermore, the electrolytic refinery was complemented by melting and rolling facilities for the production of continuously cast copper rod[24].

Since its start-up, the scale on which Palabora operates has been expanded greatly, and the product mix has also been changed[25,26] (Fig. 2). In 1967 26,4 Mt of rock was loaded and hauled, and 76,6 kt of anodes was produced for sale, whereas in 1983 101 Mt had been loaded and hauled, and 139 kt of cathode produced, of which about 68 kt was processed further to continuously cast rod.

An important element in Palabora's original financing and all its subsequent operational plans has been an export contract with Norddeutsche Affinerie for the supply of anode, and since 1976 of cathode, to its facilities in Hamburg, West Germany. Palabora also processes copper concentrates from other sources and produces many valuable byproducts.

Fig. 2—Surface plant at Palabora Mining Company (refinery, smelter, and concentrator in left foreground, and open pit in right background)

About 4000 people are employed by Palabora at Phalaborwa.

The character, and indeed the existence of the Company, owe much to the leadership and inspiration of its first General Manager, Mr E.W. Hunt.

Geology

The Phalaborwa Igneous Complex is a vertical volcanic pipe of roughly kidney-shaped section that intruded the Archaean granite country rock more than two billion years ago. The north–south axis is about 8 km long, and east–west about 3 km; the exposure covers about 2000 ha.

The primary pipe and two subordinate pipes within it are of great economic importance for their contents of phosphorus, copper, uranium, zirconium, sulphur, iron, silver, gold, and vermiculite. Pyroxenite, containing up to about 17 per cent apatite, constitutes about 90 per cent of the exposure; the phosphorus reserves are enormous.

One subordinate pipe includes a serpentine–phlogopite–pegmatoid in which, to a depth of 50 m below the surface, the phlogopite has weathered to the valuable mineral vermiculite.

The other subordinate pipe, the Loolekop Complex, covering about 5 per cent of the total outcrop area, is more relevant to this paper[27]. It contains several valuable minerals, principally copper, and its 60 m high outcrop formed the celebrated hill Loolekop. It is elliptical in section, with axes about 1400 m east–west by 800 m north–south. It is composed of three principal rock types disposed in an annular sequence (Fig. 3).

The outermost rock is foskorite, consisting mainly of partially serpentinized olivine, titaniferous magnetite, phlogopite, and about 25 per cent apatite. The apatite is economically significant and prompted the formation in 1951 of the Phosphate Development Corporation.

Foskorite grades inwards to banded carbonatite, which is composed of dolomitic calcite and important accessory apatite, with magnetite lenses of various sizes roughly

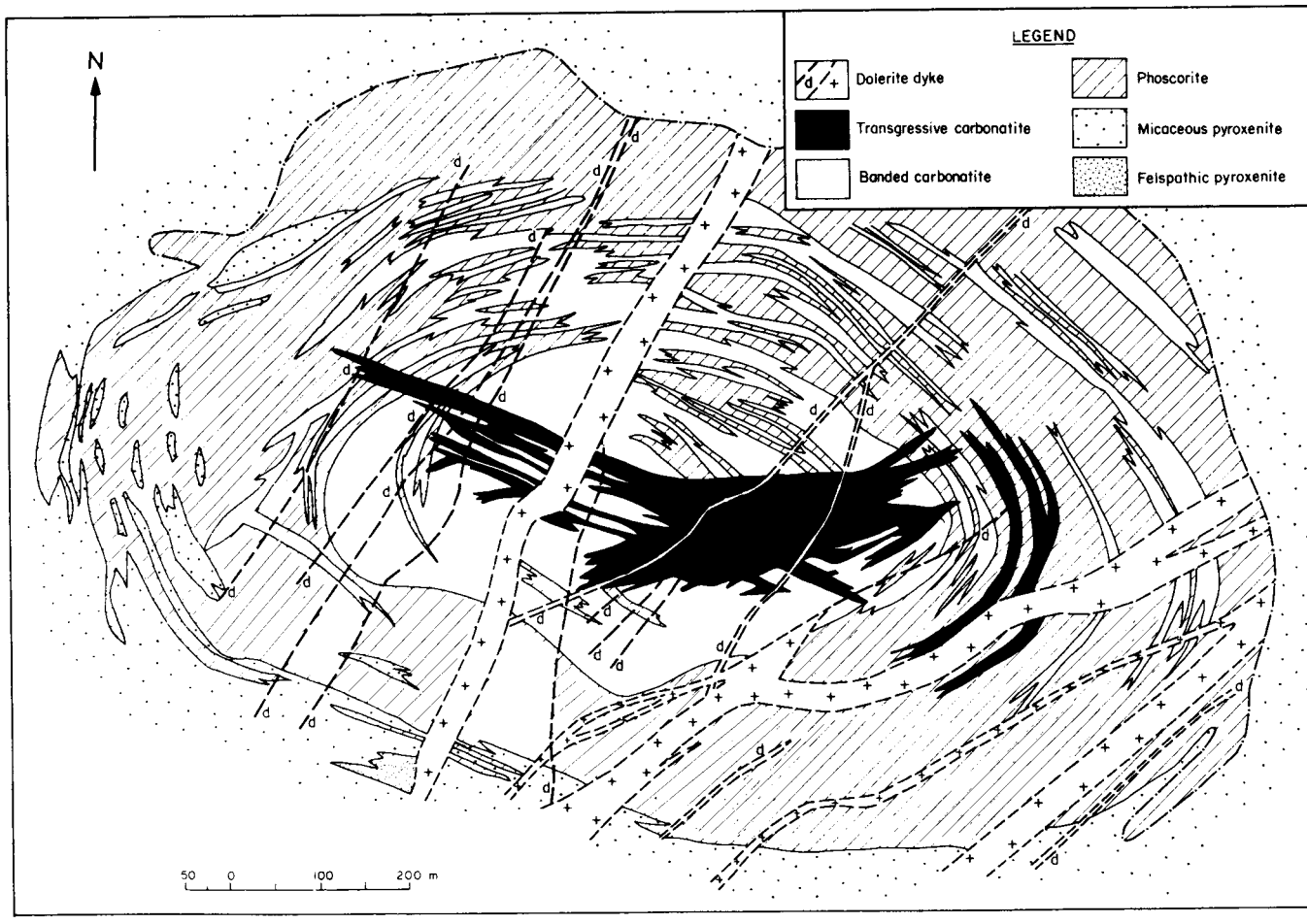

Fig. 3—The Loolekop Complex, geological plan of the 120 m level (*phoscorite* is now usually spelt *foskorite*)

distributed in a banded annular pattern.

Transgressive carbonatite constitutes the core of the pipe. It is similar to the banded carbonatite in composition, but is distributed differently. Its emplacement seems to have been controlled by pre-existing fracture patterns within the other rock types, since it provides a distinct core at the central axis of the whole structure, with large dyke-like projections radiating outwards and transgressing the other rocks. It has also infiltrated many parts of the orebody along a multitude of irregular veins, varying in section from millimetres to metres. Two substantial arcuate bodies of transgressive carbonatite are located within the foskorite of the eastern zone of the pipe.

The disposition of these three main rock types can be explained in terms of successive intrusions, all more or less directed upwards about the same vertical axis. Following their emplacement, the Phalaborwa Complex was intersected by a swarm of dolerite dykes running towards the north-east.

The Loolekop Complex is notable for its near-vertical dip and the insignificant variation of its cross-sections in shape, lithology, and mineralization. The orebody has been intersected by drilling to a depth of 1400 m, and the results confirm its continuity at that level. No evidence is available at the time of writing beyond that depth; however, there is also no geological evidence of imminent bottoming (Fig. 4).

Mineralization

Disseminated copper sulphides are found throughout the Loolekop orebody, with only traces of pyrite and pyrrhotite, and are generally recoverable by flotation. Copper sulphides are most abundant in the transgressive carbonatite, including the veins, in which chalcopyrite is the predominant mineral, sometimes with intergrowths of cubanite. The highest copper grades, approximating 1 per cent, are associated with the transgressive carbonatite. Bornite, sometimes associated with chalcocite, is the predominent mineral in the foskorite[27].

Valleriite is an interesting and important local copper mineral, although it contains only about 3½ per cent of the total copper in the Phalaborwa orebody. It consists of layers of $(Cu, Fe)_2S_2$ alternating with layers of $(Mg, Al, Fe)(OH)_2$, and grades about 22 per cent copper. This structure is not cohesive, and the mineral slimes easily and is accordingly difficult to float. Valleriite seems to have been the last copper sulphide emplaced at Phalaborwa, and is irregularly distributed, sometimes coating other copper minerals, or intergrown with them and with magnetite and gangue. It is suspected of adversely affecting the flotation of any ordinary sulphides with which it might be associated.

Other significant sulphides include Mooihoekite and nickel-bearing pentlandite. Small but valuable quantities of silver, gold, and PGM are associated with the copper

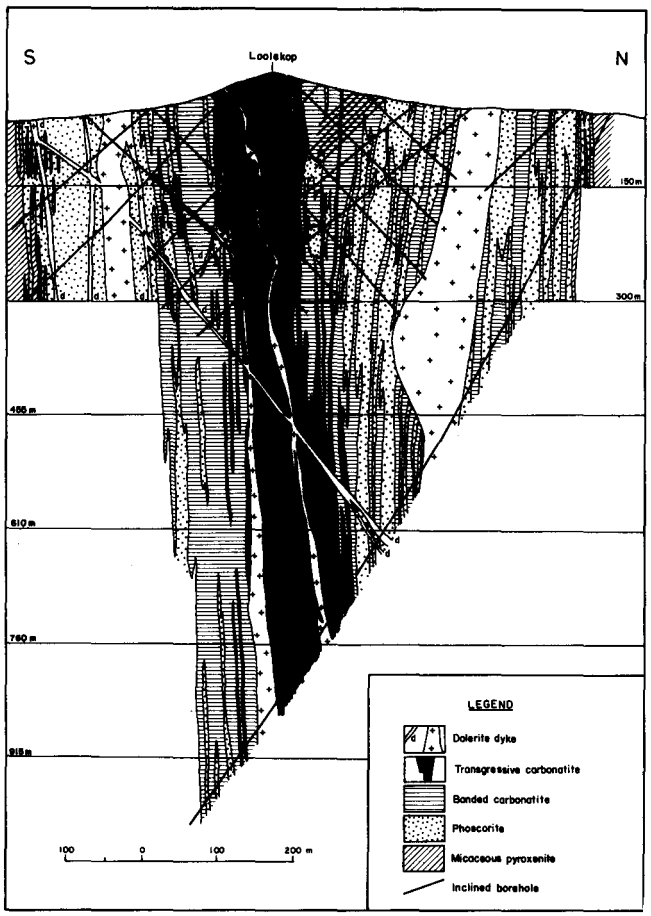

Fig. 4—Vertical section through the Loolekop Complex

sulphides, which are conspicuously free of objectionable impurities.

Open-pit Mining

As a basis for pit design and long-term planning, the upper part of the orebody was subdivided into about 647 000 co-ordinated 15 m cubes. Specific estimates of important factors, such as copper content and relative density, as inferred from the original drilling campaign were assigned to each cube. All these data are stored on a removable disc cartridge, which can hold 60 M characters. At least three back-up sets are kept, and they are accessed through an ICL ME 29 computer by means of a suite of COBOL programs.

A mine-planning group uses this data bank for ongoing strategic studies to define an optimal pit shell, and new factors and changing values are continuously brought into updated account. These include co-operative mining agreements with Foskor, and new geotechnical data such as those on stable pit-wall angles and economic cut-off grades. Post-pit mining concepts are also formulated and evaluated. This group also develops medium-term plans, covering, in increasing detail, prospective periods of one year and three months for the assistance of the open-pit management.

The primary responsibility of the pit management is to prepare and execute operating plans, commensurate with long-term parameters, that guarantee a daily supply to the concentrator of about 80 kt of feed segregated into two distinct streams according to phosphorus content and averaging about 0,5 per cent copper. (However, as time passes and mining operations centre more closely on the comparatively rich core of the orebody, the head grade will rise somewhat above that copper grade.)

A vital associated responsibility is the execution of a massive waste-stripping programme designed to expose ore in the near, medium, and remote future according to a carefully prepared schedule.

The open pit operates six days a week on three shifts a day. In this time, it must deliver about 560 kt of feed to the concentrator, i.e. enough to support its operations for seven days.

The list of production equipment has changed over the years in favour of bigger units, both to reduce traffic congestion in the pit and to reduce unit costs. The present complement is as follows:

- 12 rotary drills (30 cm)
- 83 150 t trucks equipped with pantographs for electric operation on adverse ramps
- 3 11,5 m^3 shovels
- 4 19,0 m^3 shovels
- 2 22,8 m^3 shovels.

A slurried explosive is used, composed essentially of aluminium powder, ammonium nitrate, and fuel oil. About three blasts are scheduled each week, each moving about 0,7 Mt and carefully planned for optimal breakage and minimal fly rock[28].

Bench heights are standardized at 12,2 m down to bench 16, and at 15,2 m after that. Bench 28 is the lowest developed at present, but current pit designs provide for an operation of 56 benches. Double-benching is practised at the final pit limits. Efforts are made to mine four benches concurrently in the mineralized area.

The Geology and Grade Control Department samples and analyses drill-hole chippings to facilitate short-term operational planning. In addition, broken ore is appraised visually by the geologists, who can make remarkably accurate estimates of grades and minerals, and who issue daily bulletins on these aspects for the guidance of the mine and concentrator managements[29].

The ramp gradient is 8 per cent, and a length of about 4,5 km is electrified. In 1983 this facility reduced diesel usage by 26,4 Ml at a saving of 8 million rands in energy costs. Electrification also improves truck productivity[30,31] by raising the speed on adverse ramps from 13 to 21 km/h (Fig. 5). The relative density of the ore can vary significantly with the magnetite content, making it difficult for the mass of a truck load to be standardized.

An active development programme is directed to the minimization of truck deadweight and to the further reduction of diesel-fuel consumption. A comprehensive and highly rewarding on-line computer system has just been installed to monitor the performance of equipment, to schedule maintenance, and to control spares.

Traffic patterns within the pit are controlled from an observer's station, which is in radio contact with all the shovels, bulldozers, and maintenance vehicles. The movements of the trucks are monitored, and a computer operates an annunciator board at the pit rim that signals returning drivers to go to specific shovels, which are selected by the computer so as to minimize truck-queuing time[32].

At the time of writing, the dimensions of the open-pit are about 1,79 km by 1,58 km on surface, and its depth below the original Loolekop beacon is about 360 m. The planned 56 benches by the end of the century will correspond to a depth of 805 m.

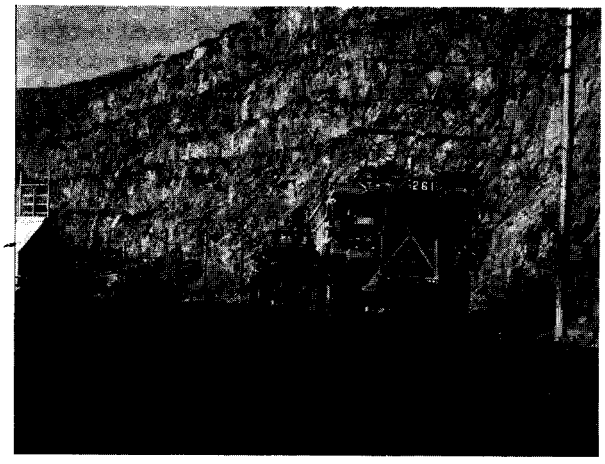

Fig. 5—Haul trucks with trolley assist

Since operations began, about 1,16 Gt of rock have been loaded and hauled, including about 385 Mt of ore delivered to the Palabora concentrator. The current annual haulage duty is about 500 Mt·km. According to the present plans, it will rise somewhat and then decline to about 380 Mt·km in the late 1990s. At that time, the ramp from the pit bottom to the rim will be about 10 km long. In 1983 a total of 101 Mt was loaded and hauled at an average daily rate of 326 kt, of which 29 Mt was delivered to the concentrator. Some of this total, containing significant amounts of copper and phosphorus, arises in areas to which Foskor has title and is delivered to their stockpile.

By any criterion, the Palabora open pit deserves to be ranked among the world's greatest.

Concentrator

The Palabora concentrator can process up to 80 kt of mill feed per day. Its primary purpose is the efficient liberation of the copper sulphide minerals and their subsequent recovery as a flotation concentrate. The pursuit of this primary objective is complicated by peculiar local factors, including the following:

(1) the presence in the mill feed of about 25 per cent magnetite, which effectively forms a heavy medium in cyclones and adversely affects the classification of copper minerals;
(2) the erratic occurrence in the mill feed of dolerite, which is hard to crush and grind, and therefore tends to accumulate in the autogenous and rod mills, adversely affecting grinding; and
(3) the somewhat erratic occurrence of valleriite and other slowly floating minerals in the mill feed, such as cubanite and mooihoekite, which adversely affect recovery (seven copper sulphide minerals have been recognized in the Phalaborwa orebody).

The concentrator has the following additional responsibilities:

(a) the obligation to process two ore-streams, which are distinguished by their phosphorus contents, having regard to the subsequent pumping of phosphorus-rich tailings to Foskor; and
(b) the need to recover titaniferous magnetite for stockpiling and also a heavy-mineral concentrate containing uranothorianite and baddeleyite for further processing.

The original crushing and grinding system comprised a conventional three-stage crushing circuit, followed by rod- and ball-mill grinding. The grinding duty is organized into six parallel sections, one 3,66 by 4,87 m rod mill and two 3,66 by 3,66 m ball mills to each section. Optimal grinding and classification are achieved when the rod mill is operated in open circuit with a ball mill, which then feeds the other ball mill in closed circuit with cyclones. The capacity of this system is about 54 kt per day.

This conventional system was supplemented in 1976 by two 9,76 m 7200 kW closed-circuit autogenous mills fed with primary crushed ore. These mills eliminate the need for additional secondary and tertiary crushing and for rod- and ball-mill grinding equipment because they produce flotation feed in one step. The capacity of this system is about 26 kt per day.

Flotation feed is produced at about 18 per cent greater than 300 μm and 75 per cent greater than 45 μm, and is presented to the cells in a slurry with a solids content of 56 per cent. The currently favoured primary collector is Princol 505, a dithiophosphate mixed with cresylic acid and a C12 mercaptan as a scavenger collector. Betafroth (FP 8) is used as a frother, as required. The flotation cells are arranged in a rougher–scavenger–cleaner combination or in a rougher–cleaner–recleaner combination. The rougher concentrates are reground. The cell capacities permit average residence times of 20 minutes, which have been shown to be more than adequate. The optimal management of flotation conditions is difficult to achieve and maintain because the mixture of copper minerals in the feed can vary significantly from day to day according to the areas mined.

Tailings with a copper content of about 0,07 per cent are discharged to an impoundment in the basin of a small tributary of the Selati River or, if rich in phosphorus, are directed to Foskor. Water is not permitted to escape from the tailings basin.

About 330 kt of concentrates are produced every year, sized at 90 per cent minus 45 μm and grading 35 to 38 per cent copper according to the mineral mixture. The concentrates are first dewatered on disc filters, and then in a rotary dryer fired with pulverized coal, to a moisture content of about 8 per cent, and are then delivered by conveyor belt to the nearby smelter.

Historically, copper recoveries into the concentrate average about 84,5 per cent, but in 1983 the recovery was depressed to 82,63 per cent by refractory minerals.

The Smelter

The purpose of the smelter is to process Palabora concentrates and a comparatively small amount from other sources, together with recycles and revert materials, so as to produce the following:

(i) good-quality anodes at a satisfactory copper recovery,
(ii) sulphuric acid at a satisfactory sulphur recovery, and

(ii) a waste slag low in copper.

The smelter embodies the principles and equipment of batchwise, multistage operation that were developed during the first half of this century and were widely adopted as good conventional practice all over the world.

The efficient and integrated operation of the various units calls for careful scheduling and is facilitated by a constantly updated record of their status. A close check is also kept on the distribution of copper and sulphur throughout the plant by sampling and analysis.

Production of Matte[33]

The first element in the flowsheet is a 36 by 10,7 m reverberatory matte furnace, with a suspended basic roof. It is fired at one end by six pulverized-coal burners consuming about 240 t of coal per day. The secondary combustion air is preheated to 240°C and is enriched by 28 t of oxygen per day. Waste gases are passed through two waste-heat boilers, generating 18 000 kg of steam per hour, and are then discharged via an electrostatic precipitator to a scrubbing plant or to a 152 m stack. The steam is used in a 10 MW turbo-generator to drive converter blowers and to preheat the combustion air.

The solid feed is charged along both sides of the furnace, and comprises damp concentrate, quartzite flux, and various copper-bearing dusts. Liquid converter slag, containing about 4 per cent copper and 25 per cent Fe_3O_4, is also recycled to this furnace.

Particular care is directed to the condition and performance of the reverberatory furnace, which must perform a heavy smelting duty continuously and successfully, and is one of the few major units in Palabora's operations that is not duplicated. If excess Fe_3O_4 is recycled with the converter slag, it will form an objectionable accretion on the floor. Its accumulation is monitored by dipping with an iron rod; if necessary, cold pig iron is added to an accreted zone, promoting the formation of ferrous oxide, which can then be fluxed to the slag.

Slag with a copper content of about 0,6 per cent is skimmed from both sides and dumped; matte is also obtained from two opposite tap holes as required by the converters. Matte and slags are transported in ladles carried by overhead cranes or trains. The mattes have a copper content of about 50 per cent.

Production of Blister Copper from Matte[34]

The second element of the smelter section comprises three basic-lined Peirce–Smith converters, each of 3,96 m in diameter and fitted with automatic tuyère-punching machines. Two converters are usually operational, with one on standby. A complete converter charge yields about 120 t of liquid blister copper.

The converters are supplied with liquid matte and various copper-rich arisings (such as recycled anode stubs and electrowon cathode from the electrolytic refinery, and ladle skulls). Siliceous flux can be added from an overhead feeding system. Blowing air is supplied by three 50 000 m^3/h blowers and can be enriched with oxygen if required. The blowing rates vary between 560 and 720 m^3/min.

Exhaust gases containing between 3,5 and 8 per cent sulphur dioxide are collected in water-cooled hoods before being ducted to an acid plant for the production of sulphuric acid.

In the operation of the converters, care is taken to avoid the excessive formation of Fe_3O_4. The preservation of a small quantity of ferrous sulphide in the vessel during successive slag blows favours the formation of ferrous sulphide instead, which can then be fluxed successfully with silica provided that flux is added when the temperature is at least 1230°C.

Certain critical metallurgical operations are controlled successfully by traditional methods rather than by rapid analysis. Thus, converter blows are regulated by the appearance of the flame and a temperature trace from a radiation pyrometer.

Liquid slag is recycled to the reverberatory furnace.

Fire Refining[34]

The third element in the smelter section is fire-refining, which uses two fire-refining (anode) furnaces. These are of the same dimensions as the converters, and are fired by pulverized coal.

Air is introduced to the bath through consumable steel lances during the blowing stage, and eucalyptus logs are used for poling. The small amount of slag that is formed is rabbled to a ladle and recycled. The duration of the blowing and poling phases of refining is selected by skilled reference to the appearance of the surface and fracture faces of small solidified spoon samples[35].

About 240 t of blister copper constitutes one furnace charge; its preparation for anode casting takes about six hours.

Anode Casting

Anode casting is the last operation in the smelter flowsheet. Refined copper from the anode furnaces is poured via a controllable spoon to heavy anode moulds made of copper, twenty-two of which are placed on a rotatable casting wheel. The casting of one anode furnace takes about eight hours. When set, the hot anodes are transferred to a water bosh, in which they are cooled prior to inspection. The anodes are then railed to the electrolytic refinery.

If the operations have been conducted correctly, the anodes weigh 315 kg and satisfy strict specifications in respect of internal soundness, dimensions, and surface condition.

Typical Statistics

The following are typical statistics relating to the smelter operation at Palabora.

Concentrate treated	1030 t/d
Copper grade	36%
Silica flux consumed	165 t/d
Oxygen consumed	28 t/d
Coal consumed	280 t/d
Reverberatory slag produced	660 t/d
Matte produced	805 t/d
Scrap copper charged	140 t/d
Converter slag produced	470 t/d
Anodes produced	445 t (99,5 Cu)
Smelter copper recovery	98%
Sulphuric acid produced	355 t/d
Sulphur capture	75%

Electrolytic Refining

The main purpose of the electrolytic refinery is to produce high-quality cathodes from smelter anodes, and also to recover the byproducts—nickel sulphate and anode slimes rich in precious metals. Palabora's electrolytic copper refinery is the only one in South Africa, and it makes the country independent of foreign supplies of high-quality cathode. It can produce about 142 kt of cathode annually[36].

Typical anode and cathode data are given in Table VI. The nickel content of the anodes reflects the presence of pentlandite in the orebody.

TABLE VI
ANODES AND CATHODES PRODUCED BY PALABORA

Item	Anode	Cathode
Mass	315 kg	135 kg
Cu	99,5%	99,998%
Ni	0,35%	2 g/t
O_2	1300 g/t	NA
S	15 g/t	5 g/t
Se	30 g/t	< 0,1 g/t
Fe	30 g/t	1,0 g/t
Bi	15 g/t	< 0,5 g/t
Pb	15 g/t	< 1,0 g/t
As	5 g/t	< 0,1 g/t
Au	4 g/t	NA
Ag	70 g/t	6,0 g/t
Te	25 g/t	< 0,5 g/t
Sb	NA	< 1,0 g/t

NA = Not available

The tankhouse covers an area 140 m by 120 m, and contains 1000 cells with 65 electrodes in each. The cells are lined with 6 per cent antimonial lead and are supplied with an electrolyte produced by the digestion of copper scrap with strong sulphuric acid agitated by air. The electrolyte is adjusted to 40 g of copper sulphate, 185 g of sulphuric acid, and 0,03 g of chloride per litre, and contains certain organic reagents that promote a smooth and coherent copper deposit on the cathode. The electrolyte is warmed to about 65°C by a coal-fired boiler system, and is circulated at a rate of 18 l/min per tank.

Starter cathodes are produced in a group of 'stripper' cells by the plating of copper onto titanium blanks, from which the deposits are stripped when they are 0,5 mm thick and weigh 5 kg. These starter cathodes are then side-trimmed, flattened, and fitted with suspension loops and copper hangers.

Anodes and starter cathodes are then suspended in the 'commercial' electrolytic cells according to the Walker multiple-contact system, the cells being connected in series and the electrodes in parallel. A steady 16 000 A d.c. current is provided by silicon-controlled rectifiers. It is passed, without any reversals, through the electrode system at a rather high density of about 280 A/m^2, favouring high productivity. High current densities are feasible because of the good quality of the anodes produced at Palabora, and because of the continuous attention that is paid to the avoidance of short circuits and to all the factors promoting the formation of smooth, occlusion-free cathodes.

The electrolytic tanks always contain about 8 kt of copper in process. The anodes are consumed in about twenty days, and the anode stubs and similar arisings are washed and returned to the smelter.

The current promotes the transfer of copper from the anode to the cathode, disengaging it from the impurities in the anode. Impurities, like nickel report to the electrolyte and remain in it, and those like gold and lead are deposited as insoluble slimes on the cell floor.

The nickel concentration in the solution is regulated to about 14 g/l by the electrolysing of a bleed stream between insoluble lead anodes and a copper cathode to first remove the copper, followed by crystallization of the nickel sulphate monohydrate in an evaporation and centrifuge plant. The somewhat impure electrowon cathodes are returned to the smelter.

Slimes are produced at the rate of about 1,5 kg for every ton of cathode; the slimes are removed from the cells, filtered, washed, and sold for the recovery of the precious metals.

The following are details of the slimes produced in 1983:

Element	*Troy ounce*
Ag	433 766
Au	18 909
Pt	2 369
Pd	3 042
Revenue	R16 000 000

Generally, current-utilization efficiencies of 95 per cent or more are achieved, and the power consumption is about 250 kW·h per ton of cathode: in 1983, 139 kt was produced. A few kilotons was sold locally, and about 50 kt, strapped in 3 t bundles, was exported through Durban. The remainder was used in an adjacent plant for the production of continuously cast and rolled rod.

Melting and Casting Plant

The primary purpose of this plant is to convert cathodes into tough-pitch high-purity copper rod for sale to South African manufacturers of electric wire and cable.

Melting

The cathodes are melted in an ASARCO shaft furnace fired with sulphur-free naphtha near the bottom, and charged with cathode at the top[37]. High-intensity burners are used, and furnace atmospheres are analysed constantly to ensure that mildly reducing conditions are maintained. The heat transfer within this countercurrent shaft is rapid and efficient. A convenient feature of the furnace is the rapidity with which it can be started and stopped: it has no holding capacity.

A typical melting rate at Palabora is 22 t/h. Molten copper, at a temperature of 1100°C, is run down a launder under charcoal cover to a 10 t holding furnace. Combustion conditions in the furnaces and the charcoal covers on the launders etc. are adjusted so that the oxygen content of the molten copper withdrawn from the holding furnaces averages about 200 g/t. At this oxygen level, copper solidifies with a level set, and surviving traces of reactive impurity metals are precipitated as oxides, with benefit to the electrical and thermal conductivity.

The oxygen content can be determined rapidly, or retrospectively by analysis. In practice, however, the

operating controls are adjusted with reference to the physical and electrical properties of the finished rod, which are determined rapidly, the rod being fairly sensitive to the oxygen content of the molten copper from which it is produced.

Continuous Casting and Rolling

The continuous casting and rolling facilities, with a capacity of 103 kt/a, are owned by the Transvaal Copper Rod Company, a joint venture between Palabora and the Union Steel Corporation. Palabora manages the company[24,36].

Molten copper from the holding furnace is fed as a continuous controlled stream to a Southwire Continuous Casting System. The stream is directed to a channel of 32 cm^2 CSA cut in the perimeter of a large rotating copper wheel. (The copper wheel, with high thermal conductivity, replaced the original steel wheel when the capacity of the plant was expanded.) The stream is retained in the channel by a steel band; the enclosure is cooled by water sprays as the wheel rotates, and the copper it contains solidifies into a bar.

The bar, at 680°C, is peeled from the wheel and fed continuously to a Morgan rolling mill, which, in 14 passes, delivers rod sized at diameters from 6,35 to 20,00 mm. The greatest tonnage is produced at a diameter of 7,9 mm. The rod is cleaned in hot water and isopropyl alcohol to produce a clean bright surface, and is given a thin wax coating. It is then tested in various ways to verify its suitability for subsequent wire-drawing and cable-making. Coils, about 10 km long and containing about 4,5 t of copper, are then mounted on pallets, wrapped in plastic, and railed to South African customers. The quantity of rod produced varies with local demand; in 1983 68 kt was delivered locally.

CURRENT MARKET SITUATION

The deterioration of the economic climate in which South African copper companies now have to operate is illustrated, with some simplification, by the statistics for Palabora given in Table VII.

TABLE VII
PRODUCTION COSTS AND REVENUE OF PALABORA*

Year	Revenue per t of Cu sold R	Production cost R/t	Production cost as % of revenue	Consumer price index
1966	815	238	29,2	100
1972	780	343	44,4	128
1978	1139	935	82,1	241
1981	1537	1 374	89,4	358
1982	1615	1 426	88,3	410
1983	1799	1 519	84,4	450

*Taken from *Annual Reports* of Palabora

It will be seen that in 1983 the production costs amounted to 84,4 per cent of the revenue, as compared with only 30 to 45 per cent during the first six years of operation.

This uncomfortably narrow margin dictates an ongoing search for innovative cost-saving measures, such as electrified haulage, which over the past few years have succeeded in keeping annual cost increases at Palabora below the corresponding change in the consumer price index[30].

In 1983, Palabora earned a net profit of 31,9 million rands. As indicated earlier, its open-pit mining operations are scheduled to continue to at least 1999, and South Africa's need for high-purity electrolytic copper seems guaranteed until that time.

THE FUTURE

As to operations in the next century, it will be recalled that the bottom of the orebody has not yet been discovered: and, in any case, lies far below the lower limit of any conceivable open-pit mine[38]. Accordingly, various long-term post-pit mining studies have been initiated, complementing investigations that are directed towards the more profitable use of byproducts and other resources. The economic feasibility of any schemes developed will depend upon cost and price levels in the future.

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Metallography

The Deutsche Gesellschaft für Metallkunde Metallography Conference will be held from 9th to 11th October, 1985, in the Europhalle in Trier. It is being organized by the DGM Technical Committee responsible for metallography, together with the Verein Deutscher Eisenhüttenleute, the Institut für Metallkunde und Werkstoffprüfung—Montanuniversität Leoben, the Technisch-Wissenschaftlicher Verein Eisenhütte Austria, and the Deutsche Keramische Gesellschaft.

The theme of the Conference covers the metallography of metallic and non-metallic materials. It will mainly concentrate on the following subjects:

- Metallographic testing of protective coatings against corrosion and wear.
- Metallography of welded, soldered, and glued joints.

In addition workshops are planned, i.e. discussion groups on selected topics: 'Preparation of metal/metal adhesive joints and protective coatings', and 'Hardness testing in metallography'.

An exhibition of equipment and accessories for metallography and a photomicrograph competition are also to be held in conjunction with the Conference. Persons interested in taking part should obtain particulars from the address below.

The Programme Committee invites applications to give short papers (15 minutes). Applications should be made on the special form provided, which is obtainable from Deutsche Gesellschaft für Metallkunde e.V., Adenauerallee 21, D-6370 Oberursel, West Germany. Telephone 06171/4081.

Residual stresses

An International Conference on Residual Stresses: Origins—Calculations—Measurements—Evaluation is to be held in Garmisch-Partenkirchen, Germany, from 15th to 17th October, 1986.

Both in regard to the engineering and materials-science aspects, the Conference will provide an up-to-date and comprehensive survey of recent progress and problems still outstanding in the field of residual stresses. A unified approach will be taken to all the main arreas of residual stresses in metals, ceramics, polymers, and composites. The principal topics covered will be:

- Modern methods of residual stress measurement
- Calculation of residual stress states
- Effects of heat treatment, forming, surface treatment, coating, machining, and welding on residual stresses
- Effects of residual stresses on fatigue and fracture
- Optimization of the benefits of residual stresses in manufacturing processes and design
- Safety and reliability problems including residual stresses.

The Conference is being organized by the Deutsche Gesselschaft für Metallkunde in association with the Arbeitsgemeinschaft für Warmebehandlung und Werkstofftechnik, the Deutsche Verband für Materialprüfung, and other international societies. The Conference languages will be English and German.

Further particulars are available from Deutsche Gesellschaft für Metallkunde e.V., Adenauerallee 21, D-6370 Oberursel 1, West Germany.