An integrated mining and extraction system for use on Witwatersrand mines*

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SYNOPSIS

An account is given of the development of a process to concentrate gold and other valuable minerals and to produce a discardable waste. In this process, the ore is coarsely milled, a bulk flotation concentrate is prepared, and the flotation tailing is scavenged in a hydrocyclone, which returns coarse gangue, moderately coarse valuable minerals, and locked valuable minerals back to the mill. The engineering developments required by this circuit in order that it will fit underground, and the progress made towards the achievement of those requirements, are reviewed.

If the waste is to be suitable for use as a backfill in a deep-level, flat tabular excavation, it should be capable of being placed hydraulically, and should have both low voidage and high rates of drainage. Such a material could be prepared by the fine screening of the overflow from the cyclone in the proposed circuit.

Possible methods for the handling of the concentrate and the recovery of its valuable constituents are described. The paper concludes with a discussion of the possible benefits from the use, in deep-level Witwatersrand mines, of a system involving concentration underground, extensive backfilling using the tailings from the concentration process, and hydraulic hoisting of the concentrate.

SAMEVATTING

Daar word verslag gedoen oor die ontwikkeling van 'n proses om goud en ander waardevolle minerale te konsentreer en afval te lewer wat weggegooi kan word. In hierdie proses word die erts grof gemaal, 'n massa-flottasiekonsentraat berei en die flottasie-uitskot gespoel in 'n hidrosikloon wat die growwe aarsteen, matig growwe waardevolle minerale en opgeslote waardevolle minerale na die meul terugstuur. Daar word 'n oorsig gegee oor die ingenieursontwikkelings wat in verband met hierdie kring nodig was om dit ondergronds te laat pas en die vordering wat daar met die voldoening aan hierdie vereistes gemaak is.

Om geskik te wees vir gebruik as 'n opvulling in 'n diep gelyk tafelvormige uitgrawing, moet die afval hidrolies gestort kan word en 'n lae ruimteverhouding en 'n hoë dreineertempo hê. So 'n materiaal kan berei word deur die oorloop van die sikloon in die voorgestelde kring deur 'n fyn sif te stuur.

Moontlike metodes vir die hantering van die konsentraat en die herwinning van sy waardevolle bestanddele word beskryf. Die referaat sluit af met 'n bespreking van die moontlike voordele van die gebruik, in diep myne aan die Witwatersrand, van 'n stelsel wat ondergrondse konsentrasie, grootskaalse opvulling met gebruik van die uitskot van die konsentrasieproses en hidrouliese hysing van die konsentraat behels.

Introduction

At present, most of the capital demanded by a gold mine on the Witwatersrand is spent on shafts and development underground. Similarly, most of the working costs are incurred in the mining and hoisting of rock. The extraction of gold from the ore once it is on surface is both relatively cheap and highly efficient.

However, as mines go deeper, the costs of mining increase inexorably, while surface costs remain effectively constant. It thus becomes relevant at a certain point to enquire whether there might not be merit in sacrificing some of the efficiency of the metallurgical process, and possibly simultaneously increasing the costs of extraction, in order to alleviate mining problems and thus reduce mining costs.

Such an enquiry has been undertaken in the Metallurgy Laboratory of the Chamber of Mines of South Africa in recent years, and this paper reviews the findings thus far.

At present it appears that there may be great merit in concentrating gold and other valuable minerals underground, pumping the concentrate to surface, and using the tailing for backfill in stages. Accordingly, consideration is given in this paper to the results of recent investigations, which suggest that an efficient concentration process can be developed; the philosophy of the design of a circuit embodying the process so that it could be placed underground is described; engineering developments called for by the design are outlined; and some thoughts on backfilling and hoisting are presented. The paper closes with a discussion of the possible benefits of the system.

Earlier Metallurgical Findings

Comparatively recently it was found possible, by chemical attack upon the gangue, to determine the mass–size distribution of gold particles liberated undisturbed from reefs. The results obtained by a number of investigators[1–3] are summarized in Fig. 1. Two measures of size are used: a square mesh size and a maximum linear dimension. It appears that the shape of the gold particles may be approximated by a parallelepiped of dimensions 1:0,4:<0,4, so that mesh sizes can be multiplied by about 2,5 to give the maximum linear dimension of the particles.

On average, about 1 per cent of the gold is finer than 30 μm in mesh size or 75 μm in linear size, suggesting that it is necessary to grind finer than 75 μm in order to release the gold. However, it is well-known that breakage is preferential round the gold and heavy-mineral assemblages in the reef. For instance, simple screening of primary crusher products suffices to produce fine high-grade fractions and coarse low-grade fractions on many mines. In a study of visible gold in mill circuits, Feather and Koen[4] found that most of the gold was free in the fine products fed to most mills. Some of their findings are presented in Table I.

It transpires that, when ore is milled to minus 75 μm, the gold particles are reduced to minus 30 μm, at which size they dissolve within about 24 hours[5] in the dilute solutions employed in cyanidation. Indeed, much of the undissolved gold in the residue is present as relatively large particles[4]. The results of the intensive cyanidation

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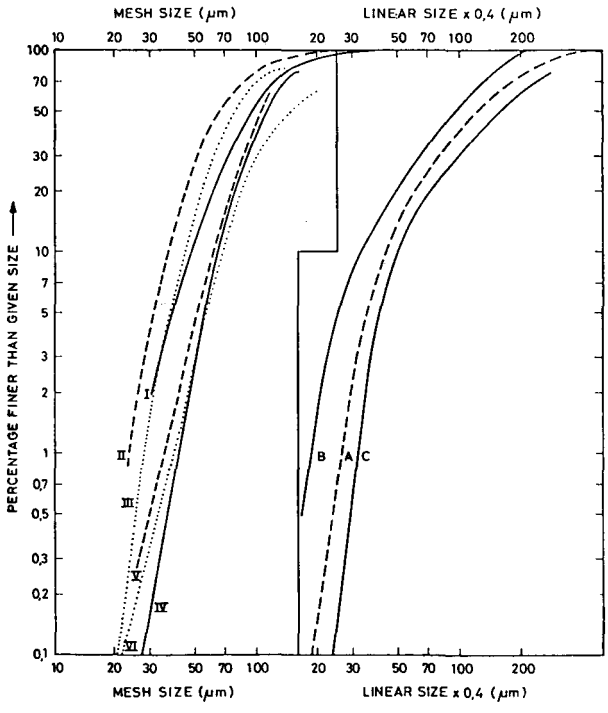

Fig. 1—Mass-size distributions for gold particles recovered undisturbed from a variety of reefs

I Basal Reef[1], Harmony Mine
II Basal Reef[2], St. Helena Mine
III Ventersdorp Contact Reef[2], Klerksdorp Mine
IV Vaal Reef[2], Klerksdorp Mine
V Ventersdorp Contact Reef[2], East Driefontein Mine
VI Carbon Leader[2], Blyvooruitzicht Mine
A Ventersdorp Contact Reef[3], Venterspost Mine
B Kimberley Reef[3], Vogelstruisbult Mine
C Carbon Leader[3], Doornfontein Mine

of a sample of Vaal Reef crushed to minus 2,5 mm are presented in Fig. 2, from which it is apparent that there is considerable dissolution of particles smaller than 600 μm and essentially complete dissolution of particles smaller than 300 μm. From these findings it can be concluded that the gold in the Witwatersrand reefs is relatively coarse, over 99 per cent by mass of the particles being larger than 75 μm, and that these particles are released relatively readily from the reef, the true release size being about 300 μm mesh.

A second finding of interest is that gold is an extremely floatable constituent of the reef. Advantage has already been taken of this fact on several Witwatersrand mines[6]. Fig. 3 shows the results of a rougher batch flotation test in which efforts were made to maximize the gold recovery by floating over 30 per cent by mass from a minus 1 mm feed. Over the range 100 to 450 μm the gold is far more floatable than the gangue.

The decrease in gold recovery in the region 50 to 100 μm shown in Fig. 3 is believed to be real, because it has been observed repeatedly. It is proposed that at particle sizes above 100 μm the gold floats as gold, but that at particles sizes below 50 μm fine gold is entrained along with all the other fine minerals in the water removed in the froth in such a massive float. Indeed, a direct correlation between the fines and the water in a concentrate was reported recently[7]. It is apparent, therefore, that, in a rougher flotation step such as that given in Fig. 3, flotation can achieve fairly efficient classification.

The final finding of interest is that hydraulic classifiers can be efficient concentrators of gold. Advantage has already been taken of this phenomenon on several

TABLE I

RELEASE OF GOLD FROM CRUSHED PRODUCTS*

Sample	Size		Free gold†		Locked gold‡
	μm	Mass %	μm	Mass %	Mass %
—¾ in screen product, Freddies Cons. M.L.	+600	94	+150	—	10
	+150	3	+75	78	
	—150	3	—75	12	
Crusher rake classifier underflow, Freddies Cons. M.L.	+600	82	+150	—	8
	+150	16	+75	76	
	—150	2	—75	16	
Ball-mill feed, Freddies Cons. M.L.	+600	85	+150	—	< 1
	+150	12	+75	95	
	—150	3	—75	5	
Ball-mill feed, Western Deep Levels low grade	+400	89	+200	77	> 1
	+200	6	+150	20	
	+150	3	—150	3	
	—150	2			
Composite ball-mill feed, Western Holdings	+6 000	21	+150	68	2
	+3 000	29	+75	28	
	+1 000	10	—75	2	
	—1 000	40			

*After Feather and Koen[4].
†Free gold is defined as that 'sufficiently exposed to permit ready dissolution or amalgamation'. The masses are approximate and were calculated from size and number.
‡Excluding gold contained in thucholite.

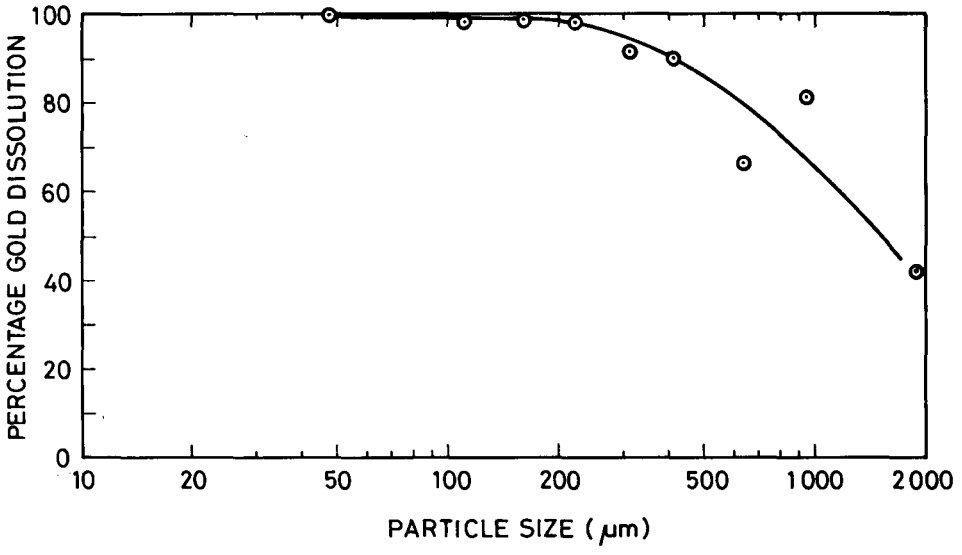

Fig. 2—The dissolution of gold as a function of particle size after prolonged cyanidation at high cyanide concentrations

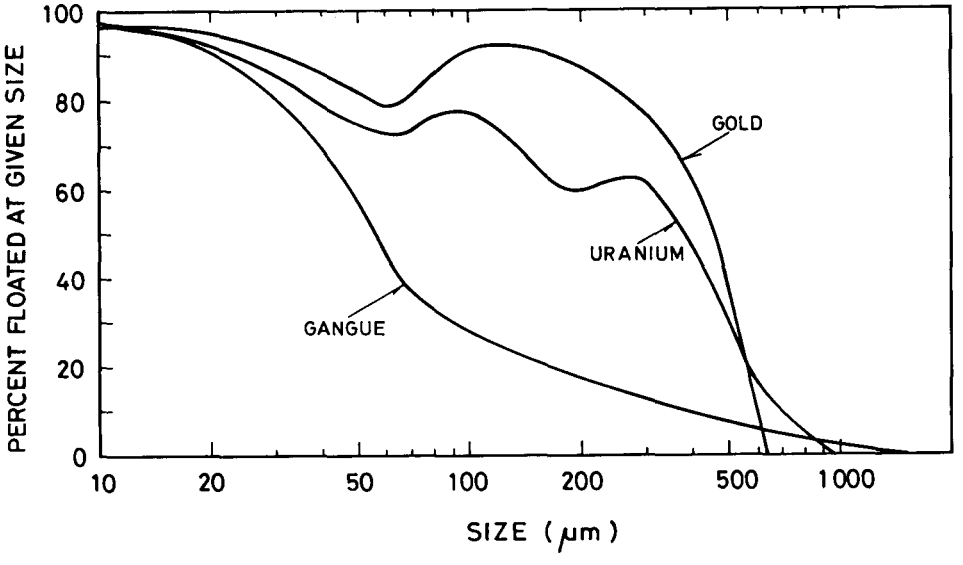

Fig. 3—The flotation of gold, uranium, and gangue as a function of particle size. The flotation was conducted under the following conditions: minus 2,4 mm Vaal Reefs high-grade secondary mill feed, 35 per cent solids, 100 g/t potassium amyl xanthate, 25 g/t Af70 frother, 12 min retention

Witwatersrand circuits. For instance, hydrocyclones have been used to prepare a high-grade gold and uranium stream on several plants from a feed of minus 12 plus 2 mm[8], although in this case the concentration effect probably results more from the density of the sulphide and uranium minerals than from that of the gold. Where gravity concentration is practised, the concentrator is invariably installed within the mill circuit to take advantage of the high concentration of gold in the circuit.

On mines in the Union Corporation Group, the mills are fitted with liners that stand clear of the shell, so that the gold concentrated and recirculated by the hydrocyclones is caught behind the liners and recovered along with significant quantities of osmiridium during relining of the mills. Other results obtained by Feather and Koen are presented in Table II, together with calculations of the circulating load of gold if the circulating ratio is 3:1. This is a low ratio in Witwatersrand practice, thus making the calculation conservative. Plainly, in mill circuits, hydrocyclones can be very efficient concentrators of gold.

An Underground Concentration Process

The following may be seen as the constraints in the design of a metallurgical concentration process for use underground:

(i) the physical volume taken up by the circuit should be as small as possible;
(ii) the recovery of valuable minerals should be as high as possible;
(iii) the waste should be suitable for backfill both as

regards its physical properties and its volume; and

(iv) the process should be as robust as possible, in the sense that considerable deviation from the design capacity should be possible without undue loss of recovery.

The first of these constraints is to a large extent satisfied by the choice of equipment of minimum size for the chosen capacity. For instance, there would be a tendency to avoid concentrators such as tables, which have a comparatively low throughput per unit volume. It may be necessary to develop special equipment, and some work in this direction is summarized in the next section. It may also be desirable to reduce the throughput of units as far as possible, which could be done by the avoidance of recycle streams. However, other constraints make a layout without any recycle difficult to achieve, as described below.

The second constraint, that of maximum recovery of valuable minerals, suggests that it would be advisable to place little emphasis on grade, but to accept that, by operating with a high recovery of both the valuable minerals and the gangue, the grade of the concentrate must inevitably be lower than could otherwise be achieved. In addition, a two-stage concentration step by means of a primary concentrator and a secondary scavenger seems advisable, which in turn immediately suggests that some form of recycle is essential.

There are two aspects to the third constraint, that is the suitability of the waste as backfill; these are volume and quality of the waste.

TABLE II

CONCENTRATION OF GOLD BY HYDROCYCLONES*

Sample	Gold concentration g/t	Gold recirculation† %
Freddies Cons. M.L.		
Ball-mill overflow	14,7	96,66
Ball-mill underflow	141,6	
Secondary overflow	18,1	96,92
Secondary underflow	190,1	
Vaal Reefs West		
Primary low-grade overflow	13,1	81,78
Primary low-grade underflow	19,6	
Secondary low-grade overflow	7,9	96,45
Secondary low-grade underflow	71,5	
Primary high-grade overflow	10,95	85,90
Primary high-grade underflow	22,23	
Secondary high-grade overflow	10,5	89,74
Secondary high-grade underflow	30,6	
Western Deep Levels		
Secondary low-grade overflow	10,0	94,38
Secondary low-grade underflow	56,0	
Tertiary low-grade overflow	45,0	84,21
Tertiary low-grade underflow	80,0	
Secondary high-grade overflow	27,0	94,08
Secondary high-grade underflow	143,0	
Tertiary high-grade overflow	58,0	92,54
Tertiary high-grade underflow	240,0	
Western Holdings		
Tertiary cyclone overflow	40,6	96,92
Tertiary cyclone underflow	426,4	

*After Feather and Koen[4].
†Ratio of gold in underflow to gold in feed.

A definite limitation can be set on the volume of the backfill in that consideration of the packing density of broken materials immediately suggests that not much more than 60 per cent of the tonnage mined can be replaced in the excavation. This in turn means that the concentrate should comprise about 40 per cent of the feed tonnage, again stressing that, in this application, grade is of little significance.

Consideration of the constraint of quality of the fill requires a summary of the findings of the section devoted to the properties of fills (see later). In brief, the ideal fill is believed at present to be a material with a size between approximately 40 μm and 1 mm, and size distribution with high proportions of both the finest and the coarsest fractions. This implies that some means for preparing a gold-free minus 1 mm waste must be sought, and that any classification used must offer efficient desliming, which in turn implies two-stage classification.

The last constraint, that of robustness, implies both that the capacity to accept a varying feed tonnage should be built into the circuit, and that the circuit should have considerable feedback in order to induce stability. The first of these requirements is considered briefly in the following section, and the second leads again to the need for recycle.

A variety of circuits incorporating the above requirements have been examined, and one in particular shows special promise (Fig. 4). The feature of this circuit is concentration by flotation immediately after comminution. This has the disadvantage that the circulating load must pass through the flotation step, thus increasing the physical volume required for this step. However, it has the advantages of offering

(a) a two-stage concentration, because the classifier acts as a scavenger of heavy minerals, thus ensuring high recoveries;

(b) a two-stage classification, with primary desliming

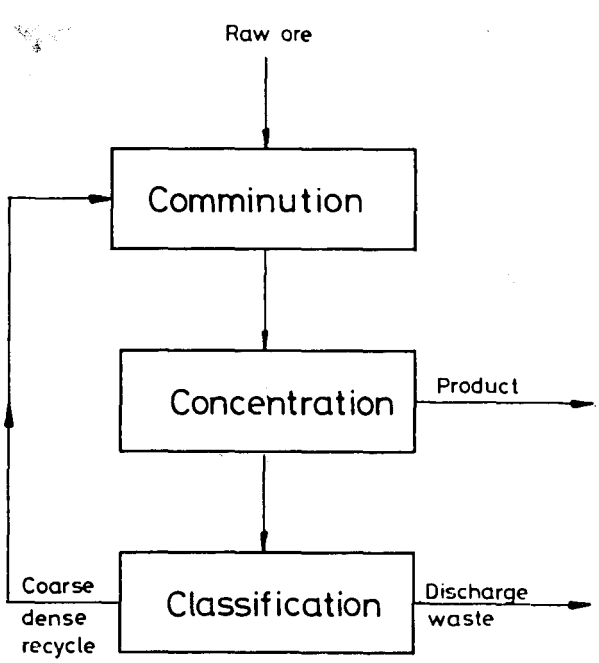

Fig. 4—Diagrammatic flow-sheet of the proposed circuit

in flotation and bulk classification in the classifier, which in turn permits the classifier to make a relatively coarse cut and thus provide a suitable backfill; and

(c) an adequate recycle for stability.

In view of these advantages, this circuit was selected for further study, which involved the development of a computer model of the various units. From a simple sensitivity analysis conducted on the model, critical parameters in the model could be identified and, if need be, could be measured on full-scale equipment operated under simulated conditions. Alternatively, the literature could be scanned for improved data to use in the model. In addition, it was possible to study the dynamics of the circuit on the model, both in order to check the stability and potential operating range of the circuit, and to evolve possible control strategies.

In this way it was hoped to avoid any pilot-plant stage, and merely to check the relevant parameters (and *only* the relevant parameters) on the full-scale equipment that would finally be linked together in a demonstration plant. If this approach proved successful, it would speed the attainment of the objective very significantly.

Fig. 5 shows the results of a typical equilibrium run of the model. The following are some of the noteworthy features.

(1) The flotation step recovers only 76 per cent of the gold fed to it, the balance being scavenged by the cyclone. In spite of this, the overall recovery of gold is 98 per cent.

(2) The size of feed to flotation is minus 3 mm, which is very coarse by conventional standards, and may require careful engineering to avoid sanding.

(3) The recovery of minus 50 μm fines in flotation is high, thus relieving the cyclone of a burden of fine material.

(4) The concentrate is relatively fine and should be easy to pump.

(5) The cyclone makes a relatively coarse cut and yet recovers over 90 per cent of the gold fed to it.

(6) The quantity of uranium recovered is reasonable, although it should be pointed out that this is based on limited data. Nevertheless, these data are encouraging in that, for ores high in thucholite, the recovery of the thucholitic fraction is nearly quantitative in flotation, while for other ores the species recovered are those that are leached relatively readily. In addition, the uranium tends to be concentrated in the finer fractions, which are floated well if not selectively.

(7) The effective recirculation ratio is only 2:1 because of the 'bleed' of product from the circuit. Such low ratios are normally difficult to achieve by hydrocyclone classification. This in turn means that the additional flotation volume required by placing the flotation step in the mill circuit is not excessive.

In the light of this study, therefore, the following were given high priority for investigation:

(i) means for the comminution of run-of-mine ore to minus 3 mm in equipment of minimum volume;

(ii) equipment for the bulk flotation of minerals from a minus 3 mm feed; and

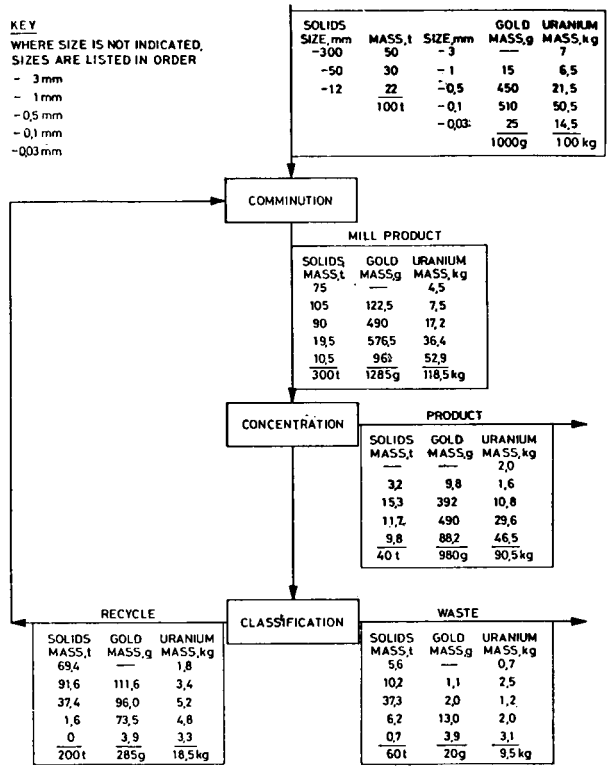

Fig. 5—Estimated equilibrium performance of the proposed circuit

(iii) the performance of cyclones as concentrators of dense minerals when cuts are made at relatively coarse sizes.

In the next section, therefore, these three priorities are emphasized.

Engineering Developments

The size of 3 mm for the product of the comminution step was set partly by considerations of liberation and classification, and partly by limitations on the size of particle that can be handled in the concentration step. It seems most unlikely that adequate liberation would be achieved at a larger size. Some form of screening would be necessary at the end of the comminution stage to remove material larger than 3 mm reliably; screening at 3 mm is perfectly feasible, whereas the use of much finer screens presents definite problems. In particular, they require considerable quantities of wash water, which would make a dewatering step necessary between comminution and concentration if the volume necessary for concentration were to be minimized. On these grounds, therefore, the choice of 3 mm as the product size after comminution seems reasonable.

The comminution of particles of such a size is just within the capabilities of fine crushers, but the water in the recycle stream makes the use of fine crushers undesirable unless some considerable dewatering of the recycle can be achieved, which also seems difficult. Some form of mill accordingly seems desirable.

Of the various mills that offer high throughputs per unit volume, the so-called 'centrifugal mill' seems particularly well suited to this duty. Such a mill has been

developed to an advanced stage by the Metallurgy Laboratory, and most of the parameters that determine its design and scale-up are now fairly well understood[9–11].

The mill comprises a tube similar to that used in the conventional ball mill, which is turned on its own axis while the axis itself is rotated about an axis of gyration. It has been found particularly beneficial to design the mill so that the tube rotates on its own axis at the same frequency as the rotation about the axis of gyration, but in the opposite direction. In this case the mill behaves as a large-amplitude, low-frequency vibration mill, although the charge within the mill is not fluidized as it is in the vibration mill, but moves round the mill much as does the charge in a conventional ball mill. An important feature is that mills of this design operate at 'zero per cent critical', so that the limitations on mill speed are set primarily by the physical strength of the machine. It has been found that the power drawn by the mill varies with the cube of the speed of rotation of the mill, so that, unlike conventional mills, variations in feed tonnage can be accommodated by varying the speed over quite narrow ranges. This feature of centrifugal mills could permit the circuits incorporating them to cope with considerable variations in the feed.

The feeding and discharge of centrifugal mills were studied intensively. It was found that a spiral delivering to the centre of the mill suffices to feed any desired tonnage; several designs of feeder that deliver the feed to the periphery were tested, and all displayed a marked tendency to choke.

The discharge is more complex. Conventional flat screens were found to be adequate when the feed was relatively fine, but the presence of particles of near-screen size restricted throughput drastically, and screen wear was excessive under all conditions. A 'top-hat' design, a screen with bars arranged parallel to the mill axis but on a circle of smaller diameter than that of the mill, wore far less and gave generally better performance. However, with a high tonnage of near-mesh material and a ball grinding medium, the balls tended to be entrained in the pulp flowing through the mill and so choked the outlet screen. Reverse spirals and conical liners were tried without any marked improvement, and eventually it was found necessary to arrange most of the liner of the mill as a screen to achieve an adequate throughput. Because of these problems, it was very necessary to reduce the recycle tonnage to a minimum, as noted in the previous section.

Fig. 6 shows a 20 kW prototype mill that was used for testwork for about eighteen months. The mill tube is 200 mm in diameter and 240 mm long, and the diameter of the circle of gyration is 80 mm. Operating on a minus 12 mm feed and with minus 16 mm balls, it ground nearly 3 tons per hour to 30 per cent minus 75 μm in open circuit at a net power input of 19 kW. Steel wear and power demand in terms of product tonnage were equivalent to those found in a conventional ball mill operated on the same feed.

Fig. 7 shows a 1000 kW prototype machine that was designed in collaboration with a German engineering concern. The mill has a variable-speed drive, a mill tube 1 m in diameter and 1,2 m long, and a gyration diameter of 0,4 m. The mill tube complete with feeder and discharge is designed so that it can be removed rapidly and a fresh tube substituted. Because of this it is believed that the down-time for the replacement of worn parts will be no greater than that for conventional

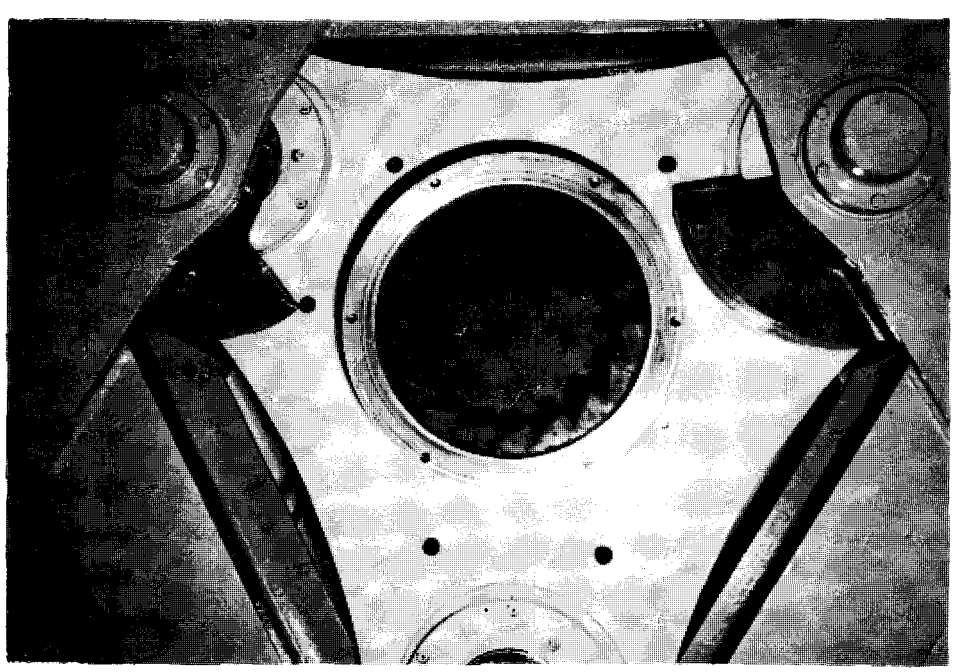

Fig. 6—A view of a single-tube 30 kW centrifugal mill. The mill tube is held at its centre in a triangular frame that is driven by cranks at its apexes. The counterweights on the cranks move through larger circles than the mill to facilitate balancing. The end of the mill has been removed in the photo graph

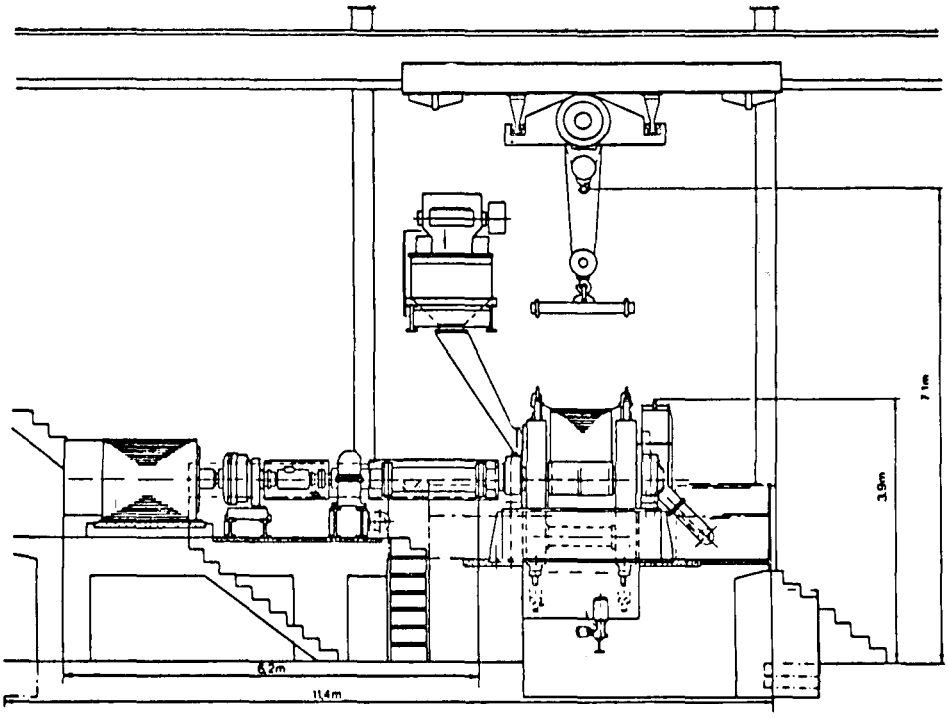

Fig. 7—A view of the proposed 1 MW centrifugal mill. The drive is on the left and the mill tube is below the crane in the centre

mills. It is intended to test this machine on surface on a Witwatersrand mine during 1978.

It is apparent from the size of this machine that it should be easy to install underground. A single jaw crusher should suffice to reduce the size of run-of-mine feed to minus 75 mm, and it is believed that the machine, if operated semi-autogenously, should be capable of reducing this to minus 3 mm at a rate exceeding 100 tons per hour, drawing about 370 kW. These predictions can be confirmed only by full-scale tests.

The recovery of gold and other valuable minerals from material ground to minus 3 mm was the subject of a number of studies, from which it was concluded that flotation is the most efficient process. Processes studied and abandoned in favour of flotation included jigging, riffling, tabling, dense-medium cycloning, and drum concentration. The basic disadvantage with all these methods is the loss of gold in very light minerals such as thucholite. Loss of very fine gold, which tends not to be wetted, and the relatively poor performance of these concentration processes at throughputs above design, were secondary reasons for their being discarded.

However, flotation from feeds containing minus 3 mm particles is unusual, and it was found necessary to design a cell that would achieve adequate recoveries. The disadvantages of the conventional mechanism were as follows:

(a) inadequate turbulence to suspend particles above 1 mm;
(b) undue emphasis on the generation of fine bubbles by shear between the rotor and the stator, which led both to excessive power demand and excessive wear, particularly as relatively large bubbles are needed to float particles of gold as large as 500 μm;
(c) inadequate transfer of coarse pulp between stages; and
(d) insufficient length of overflow weir for bulk flotation.

The cell was designed specifically to overcome these deficiencies. The features of the design (shown in Fig. 8) are as follows:

(1) the pulp is agitated by a conventional four- or six-bladed impeller placed relatively close to the bottom of the vessel to provide good scouring;
(2) the vessel is baffled from some distance above the impeller to the surface to ensure a quiescent pulp–froth interface;
(3) air is injected from a ring at the base of the baffles close to the wall in the turbulent zone created by the radial flow from the impeller, which provides some shear to reduce the size of the bubbles, and the rotational flow created by the impeller helps to carry the air bubbles towards the centre of the vessel, thus improving the distribution of air;
(4) a flat disk is mounted above the impeller to prevent air from entering the impeller;
(5) the froth overflow weir is circular to maximize the potential length of the weir;
(6) pulp is taken off from the stagnation point of the radial flow from the impeller, the take-off pipe leading down to the next cell to below the stagnation point, thus providing a high degree of turbulence to assist in keeping the pulp suspended, and a small head to overcome resistance to flow in the transfer pipe; and
(7) pulp is taken off from the final cell in a chain by means of an air-lift, thus preventing the pulp from settling out and providing a positive means for interface control in the chain.

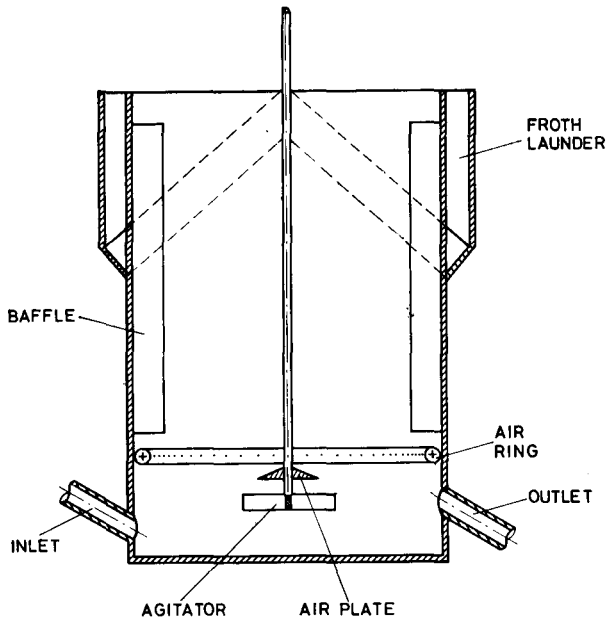

Fig. 8—Design of a flotation cell for use with a feed containing very coarse particles

Thus far, the tests have been restricted to cells of capacities up to 20 litres. However, the basic design principles have been proved, and gold and other valuable constituents of the reef have been floated successfully from minus 3 mm material over pulp densities ranging from 15 to 55 per cent solids by mass without sanding out. High pulp densities are desirable in order to minimize the cell volume in underground applications. Varying the pulp density in this type of cell has been found to have only a very slight effect upon air demand, the power requirements, and the rate of flotation. The results of a typical batch test are shown in Fig. 3. Studies on the scale-up of this type of cell are now planned.

No detailed study has been made of the use of hydrocyclones for relatively coarse separations such as those required by the circuit shown in Fig. 5. Accordingly, a closed-circuit test rig was designed and is in the course of construction (Fig. 9). The circuit incorporates a cyclone of 760 mm diameter that was designed on the basis of an extrapolation of Plitt's model of cyclone performance[12]. Fig. 10 shows one of a series of typical predicted performance curves, which suggest that a corrected d_{50} of over 350 μm should be achievable for an inlet of 150 mm, a vortex finder of 175 mm, and a spigot diameter of 84 mm operated at 50 kPa pressure drop with a feed of 35 per cent solids by volume.

It has proved difficult to predict the precise behaviour of cyclones operating with minerals of different densities. This matter has only recently became the subject of close study[13, 14]. Accordingly, one of the prime topics for study on the test rig will be the split of gold and other valuable minerals between underflow and overflow. Difficulties are expected in the interpretation of the results because of the presence of thucholite, which is expected to be a significant carrier of gold into the overflow of the test rig. However, in the circuit shown in Fig. 5, the thucholite should be removed by flotation ahead of classification. Accordingly, this circuit was modelled on the assumption that the ratio

$$d_{50\,\text{gold}}/d_{50\,\text{gangue}} = \left(\frac{\rho\,\text{gangue}^{-1}}{\rho\,\text{gold}^{-1}}\right)^{1,5}$$

in accordance with the latest findings on relatively coarse cyclone feeds[14].

Properties of Backfill

The stresses that are generated in the deep, tabular excavations on the Witwatersrand are probably greater than those imposed on any other structures in which people work. The effects of stress are experienced as both elastic deformation (which leads to predictable movement of the walls of the excavation) and as inelastic events (which vary in magnitude from falls of hanging with the release of a few kilojoules of energy to 'rockbursts' in which the release of energy can exceed 10^9 J). The inelastic events are essentially unpredictable, but it has been found statistically that the frequency of large events is related to the stress on, and the geometry of, the excavation. In the idealized case, the stress is merely proportional to the product of the depth below surface, d, and the average density of the rock over that depth, ρ_a, while the only geometric parameter of interest is the width of the excavation, S.

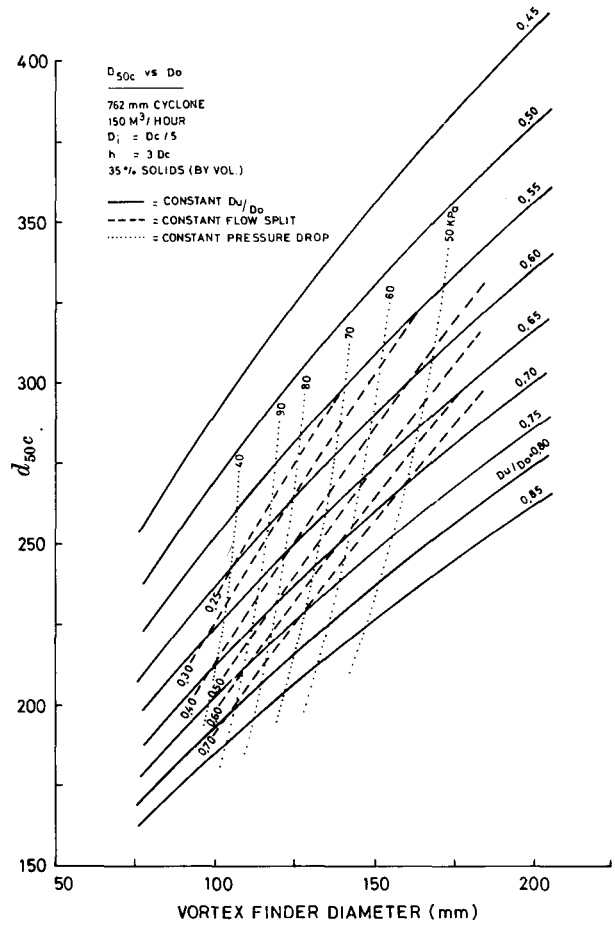

Fig. 10—Variation in d_{50c} as a function of vortex finder diameter for a 760 mm cyclone

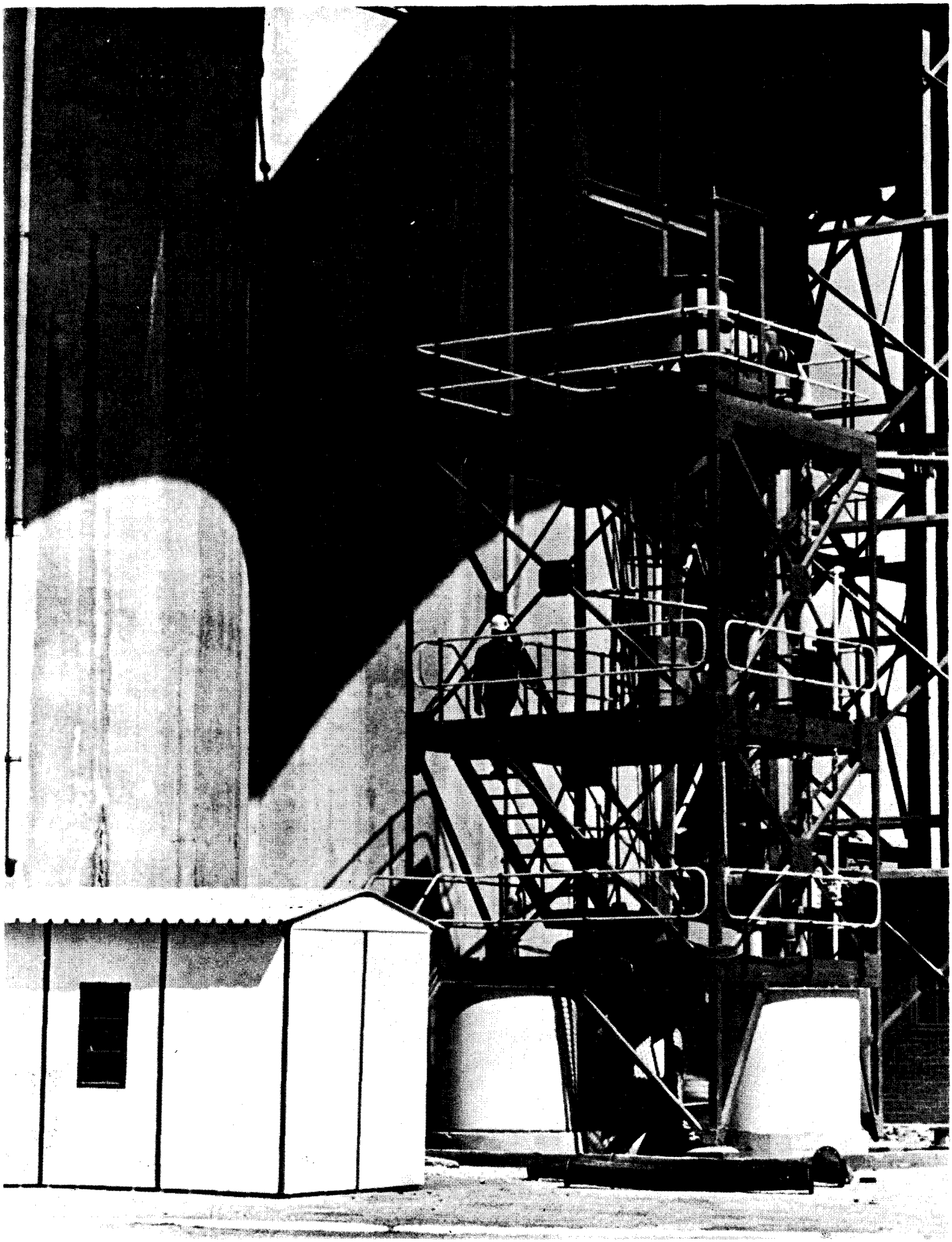

Fig. 9—A view of the cyclone test-rig

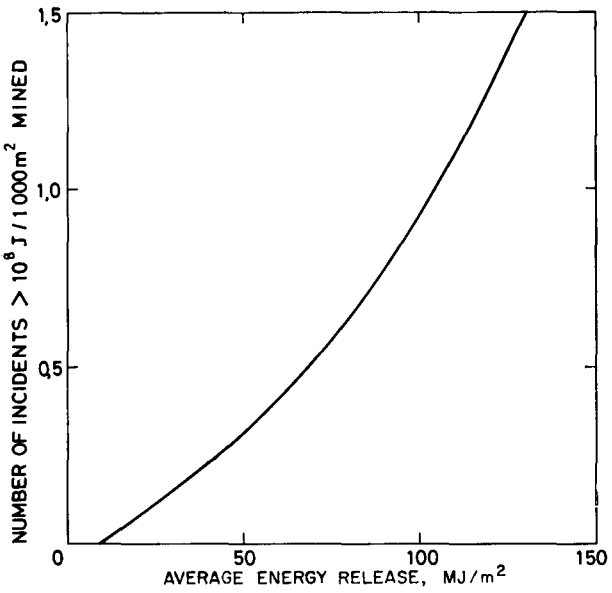

Fig. 11—Variation of the incidence of large inelastic events as a function of the energy release in mining

The product $d\rho_a S$ determines the amount of potential energy that will be released as the excavation collapses, and it has been found that the frequency of large inelastic events is related to the energy release as shown in Fig. 11.

Plainly, it is thus desirable for the fill material to be as stiff as possible, i.e. to deform as little as possible. Accordingly, a study was made of the properties of a variety of materials that might be generated from reefs as a waste free of valuable minerals.

The study took the form initially of triaxial compressive tests of the various materials, either with or without additions of cement. The compressive tests were carried out in a 'stiff' testing machine, in which the strain rate could be controlled and the stress determined. The stresses normal to the principal stress were also controlled to prevent expansion of the sample in that direction, thus simulating the behaviour of a core of material in a fill restrained laterally.

Table III shows typical results for conventional plant slimes (about 75 per cent minus 75 μm, 20 per cent minus 10 μm), tailings from the flotation of a material grading about 75 per cent minus 75 μm (grading of tailing about 65 per cent minus 75 μm, 5 per cent minus 10 μm), and a grit with a linear grading between 37 μm and 2 mm. It is plain from these results that the behaviour of the material is determined largely by the voidage. The low angles of internal friction for the slimes in particular were a cause for concern.

The effect of cement additions to the slimes was tested, and it was found that over 5 per cent cement was necessary to improve the angle of internal friction, and that the presence of 25 per cent water by mass had a very deleterious effect, so that it would be necessary to dewater the slime to a certain extent if cement were to be added.

At this point it became clear that a gold-free waste with a particle size of about minus 1 mm could be produced by a circuit such as that described previously, and further work on slimes was therefore abandoned. However, some success has been achieved with the use of dewatered slimes containing 30 per cent cement[15].

As earlier work had demonstrated the beneficial effect of reducing the voidage of the fill material as far as possible, a study was undertaken of methods to achieve this. The chief determinant of voidage in metallurgical residues was found to be the shape of the size distribution curve, with the actual shape of the particles having a secondary effect.

This is illustrated in Fig. 12 and Table IV. The size-distribution curves for all the metallurgical products (curves a, b, and c in Fig. 12) have the same shape, and the voidage is the same for all these materials. Curve c in Fig. 12 is the curve for the product of the circuit shown in Fig. 5. A hammer-mill grit shows a significantly lower voidage. An artificially prepared 'gap-graded' material gave the lowest voidage of all, and responded very readily to compaction by vibration. Two materials having the linear size distribution shown in curve f in Fig. 11 were prepared, the one from angular crushed quartzite, and the other from the same material after it had been tumbled for several hours in a mill without grinding medium in order to round all the particles. Obviously, such rounding causes the particles to pack more readily.

It is clear from Table IV that the voidage is very strongly determined by the shape of the curve for particle-size distribution. It is of interest to note that, if the product of the circuit were to be screened at about 0,5 mm, the minus 0,5 mm fraction would have a size-distribution curve with a shape close to that of the curve for hammer-mill grit as shown in curve g.

TABLE III

DRAINED TRIAXIAL COMPRESSION TESTS ON POSSIBLE FILL MATERIALS

Material	Voids %	Strain at failure %	Variable stress at failure MPa	Confining stress at failure MPa	Angle of internal friction degrees
Slimes	47	15,3 18,4	3,2 4,1	1,2 1,5	27 28
Flotation tailings	41	5,8 9,4	2,9 5,2	0,9 1,2	33 39
Grits	35	6,7 10,0	3,6 5,8	0,6 0,9	46 47

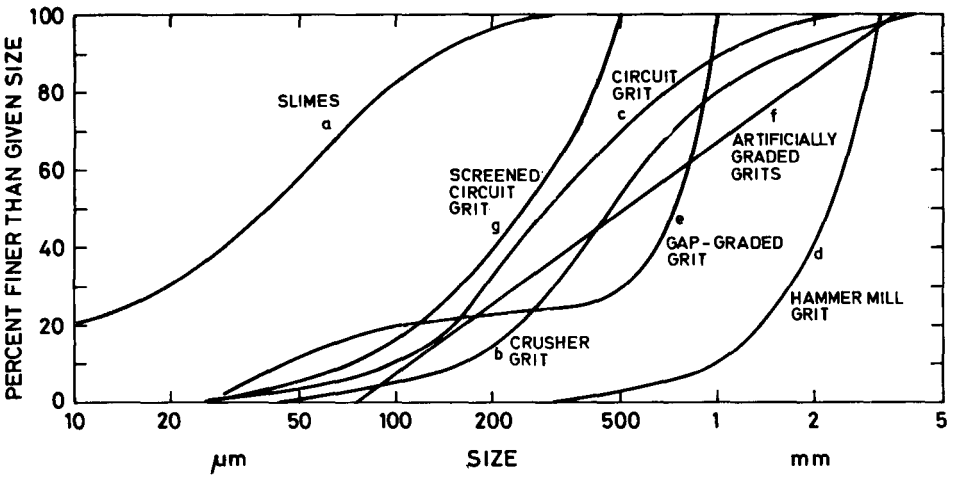

Fig. 12—Particle-size distributions of potential fill materials

TABLE IV

PACKING BEHAVIOUR OF POTENTIAL BACKFILL MATERIALS PLACED HYDRAULICALLY

Material	50% passing size μm	Voids %
a. Slimes	40	47
b. Crusher grit	480	45
c. Circuit grit	300	44
d. Hammer-mill grit	2 200	30
e. Gap-graded grit	720	30
e. Gap-graded grit after vibration	720	25
f. Angular sized grit	510	37
f. Rounded sized grit	510	32

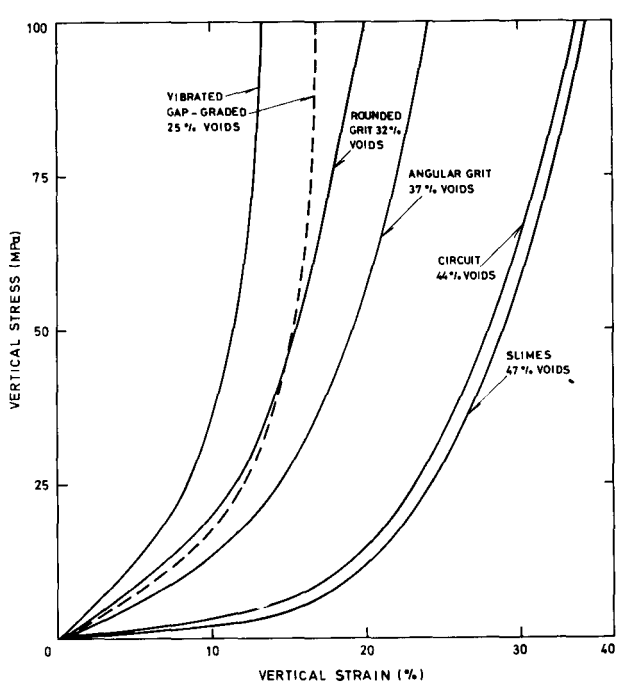

Fig. 13—Comparison between the confined stress-strain curves for various potential fill materials

In Fig. 13, the results of confined compressive tests are given for some of the materials shown in Fig. 12. Again, it is clear that the compression behaviour of a material is related closely to its voidage. Fig. 13 shows a further interesting phenomenon, namely, the effect of the rate of compression. Most of the results shown were obtained during a test that was completed within several hours. However, the results shown by the broken curve in Fig. 13 were obtained over a period of 60 hours, and significantly larger strains were observed than when the same material was tested over a period of only 2 hours. Under extremely slow compression, micro-cracks in the individual particles have a chance to propagate and result in a greater degree of compaction. To some extent, this phenomenon appears to occur less frequently in materials of smaller particle size, as would be expected from theories of brittle fracture. The phenomenon is of particular importance in mine fills, however, because the stress on the fill builds up over periods measured in months.

A further dynamic effect observed was liquefaction under shock loads such as would be expected to occur during a rockburst. This phenomenon was particularly apparent in material that was not prestressed and in materials that had low angles of internal friction. In the experiments, cylindrical specimens of 10 cm diameter and 2 cm thick were placed between platens. They were gradually preloaded hydraulically along the axis of the cylinder as desired, and were then subjected to a load of 40 t for a period of approximately 500 μs. A typical trace showing the stress wave and the convergence of the platens is given in Fig. 14, and some typical results are presented in Table V.

Tests were also conducted in which a sample was loaded triaxially, and the confining stress was then varied cyclically up to about 10 per cent on either side of the preset stress, at cyclic frequencies of up to 80 Hz. These tests confirmed that, for materials with angles of internal friction above 35° and not saturated with water, loss of cohesion during shock loading was unlikely to occur, particularly in materials that were already under some stress.

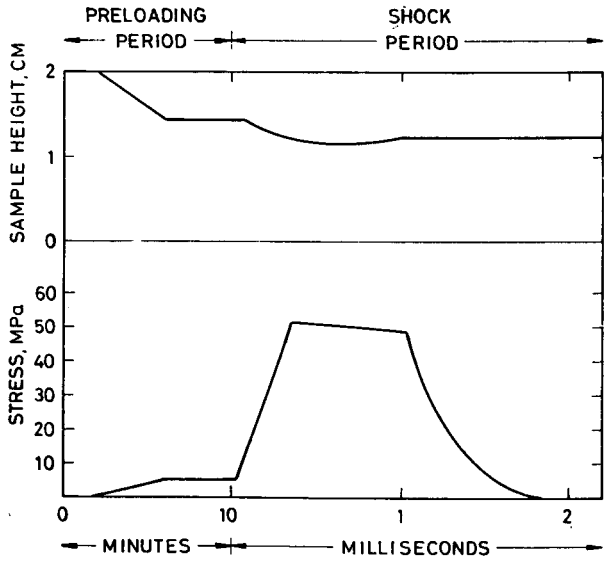

Fig. 14—The effect of a vertical shock load of 0,4 MN on an unconfined cylinder of grit, 100 mm in diameter and 20 mm high, after a preload of 40 kN

TABLE V

SHOCK COMPRESSION OF SPECIMENS OF FILL MATERIALS

Material	Prestress MPa	Platen convergence* mm
Slimes + 2% cement, cured 14 days	0	20,0
Slimes + 2% cement, cured 14 days	5	16,0
Slimes + 2% cement, cured 14 days	15	12,0
Slimes + 5% cement, cured 14 days	5	16,6
Crusher grit (sample b, Fig. 12)	0	10,8
Crusher grit	5	8,0

*The initial distance between the platens before prestressing was 20 mm.

Consideration of the actual placement of fills leads to further demands on the material properties of the fill. It appears that it is most convenient to place the fill hydraulically, but this requires that the material should settle reasonably rapidly and be relatively free-draining.

Settling rate is to a large extent a function of particle size. Qualitatively, the settling rate of particles that have the size distributions shown in curves a, b, and c in Fig. 12, which are all of similar shape, can be compared by considering the Stokes settling rate of their 50 per cent passing sizes (see Table IV). For material of a density of 2,7 g/m^3, these rates are 0,15 mm/s, 9,1 mm/s, and 4,6 mm/s, respectively. Obviously, a relatively coarse grit is to be preferred.

Similar conclusions can be drawn from a consideration of the drainage rates. The coarser materials shown in Fig. 12 are all relatively free-draining in the sense that, when they are placed in a saturated state, water will drain from them under the influence of gravity, and air will be drawn in until very little of the volume placed is still saturated. Slimes, on the other hand, will tend to remain saturated under these conditions, particularly when placed in narrow tabular excavations at low dip angles. As indicated previously, water-saturated materials should be avoided if stability under load is to be achieved in the fill.

It can be expected that the drainage properties of the fill will vary with the voidage of the fill. From the studies undertaken to date, it does not appear that drainage is markedly affected by voidage for materials within a given range of sizes, and for voidages in the range 45 to 25 per cent. However, under the influence of stress, the voidage of the fill is reduced and the particle-size distribution becomes finer. Accordingly, it can be expected that, once the fill is under significant stress, the permeability will be reduced markedly. Studies have been initiated on the importance of this effect in the design of fill systems.

Finally, it is desirable for operational reasons that as little constraint as possible should be placed around the fill in stopes. The behaviour of an unconstrained fill subjected to a known stress pattern can, in theory, be calculated if the material properties are known, but the problem is rather intractable for the varying boundary conditions that arise in practice in mining. Plainly, however, the more extensive the fill, the less will be the fraction of support lost at the unconstrained boundaries; or, the larger the width of the fill relative to its height, the better the support.

A few model tests were conducted in large compression testing machines. In Fig. 15 some results are compared with published results on cemented slimes[15]. It is clear that the performance of the fill increases markedly as the diameter-to-height ratio increases, and a comparison of the results at a diameter-to-height ratio of 4 shows that a grit of low voidage offers significantly better support than a slime of relatively high cement content.

In summary, therefore, it appears that the ideal

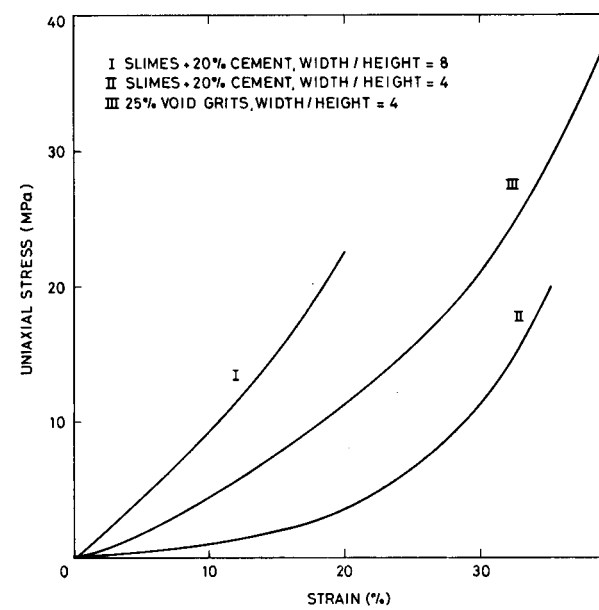

Fig. 15—Comparison between various potential fill materials in unconfined compression at various diameter-to-height ratios

material for backfill in large tabular excavations under high stress is a material with

(i) a 50 per cent passing size in the approximate range 0,2 to 1 mm; and
(ii) a size distribution chosen to minimize voidage and to maximize permeability, which implies a material with a high proportion of coarsest and finest size fractions, and a size distribution with a minimum size of about 40 μm.

Such a material should be capable of being placed hydraulically without requiring the addition of cement, be sufficiently stiff to minimize closure in stopes where the virgin stress may be as high as 100 MPa, and be stable under most of the dynamic stresses impressed on it. Detailed large-scale studies of this pumping and placement of such materials have been initiated.

Handling of Concentrates

The concentrate from the proposed circuit is 40 per cent of the feed tonnage, and has a size distribution of 46 per cent plus 0,1 mm and a top size of 1 mm, which suggests that to transport it hydraulically should not be difficult. Considerable experience has been gained in the pumping of fines from great depths, and the pumping system having the lowest capital and operating costs appears to be a displacement system using as a pressure vessel a chamber drilled vertically in rock by raise-borer[16]. Some adaptation may be necessary for the handling of materials coarser than normal, but at present no work on this aspect is planned.

The recovery of metals from the concentrate was studied only in the laboratory. The results can be summarized as follows.

(a) A regrind of the concentrate may not be necessary, although recoveries improve marginally in some cases, and it may be desirable to regrind in order to ease the handling problems that arise with the existing equipment.
(b) A hot (over 60 °C) leach for the recovery of uranium ahead of that of gold destroys the flotation reagents, which otherwise interfere with cyanidation. As the concentration of uranium in most concentrates is approximately double that of the ore, most mines that adopt the suggested process would probably find it economic to recover the uranium.
(c) It would almost certainly pay to increase the cyanide concentration in cyanidation to 2,5 times the normal concentration, which would not increase the actual consumption of cyanide but would increase the gold dissolution significantly[17].

Discussion and Conclusions

The new process developed for the concentration of gold and other valuable minerals from Witwatersrand ores as described here reduces the volume of equipment to the point where it is feasible to consider placing the plant underground. The tailing from such a plant can be made sufficiently low in gold and other valuable minerals to justify its being discarded as waste; for such a waste to be suitable as backfill in large flat tabular excavations under high stress, it should be capable of being placed hydraulically and should have low voidage and high rates of drainage.

It can be concluded that a possible system for the close integration of mining and extraction has been identified. It is therefore desirable to enquire whether sufficient benefits might flow from the integration to justify the adoption of such a system.

Among the primary advantages of such a system are the benefits in strata control that would follow from the backfilling of worked-out areas. For extensive backfilling carried to within 5 m of the face and using a low-voidage material, it seems probable that the energy release at a depth of 4 km could be reduced from about 80 to about 20 MJ/m^2. As shown in Fig. 11, this would reduce the rockburst hazard from about 0,7 events to less than 0,1 events per 1000 m^2 mined.

If the fill were generated underground, there would be every incentive to backfill as extensively as possible, thus saving the cost of hoisting waste; if the fill were generated on surface, the incentive would be to minimize the volume taken underground, thus minimizing the benefits of backfill. This is one excellent reason for generating the fill underground.

The main benefits to be derived from filling other than those of strata control, include better control of ventilation air, less surface area for the pickup of heat, and a marked reduction in the timber demand with a consequent reduction in the fire hazard.

However, underground extraction applied widely could lead to further benefits. Many mines have limitations on hoisting capacity; the capital cost of shafts is high, and the benefit to be obtained from the incremental tonnage that might be hoisted through a new shaft system is often less than the cost of servicing the capital required for the new shaft. Hydraulic hoisting of most of the value mined could permit a delay in the construction of new shaft systems on existing mines, with a marked improvement in the cash flow.

Similarly, the reduction in the amount of timber to be lowered would increase the time during which 'men and materials' cages could be utilized for man transport. Advantage could be taken of this by creating ventilation districts underground (something that can be done on few Witwatersrand mines at present, but that could become feasible if backfill were employed widely). This would permit a staggering of shifts, the importance of which is obvious when it is realized that at present to get from the bank to the working place in some of the deepest mines takes over two hours. The possibility exists of avoiding the use of sub-vertical shafts altogether[18]: and using high-speed incline shafts for men and materials transport, which would be the factor determining the layout of the transport system if most ore movement were hydraulic. There is a further possibility that extensive backfill could provide sufficient support to make on-reef haulage ways practical on some mines, with savings in development costs and a simplification of mine layout.

However, the greatest economic benefit of extensive backfill would result from the improved support, which would make practical both the extraction of the remnants that inevitably result from present mining tech-

niques, and the pillars that at present are left in deep-level mines to stabilize the excavations. On some mines the reef left in remnants and various pillars exceeds 10 per cent of the reef available within the claim area, and it is nost likely that such reef could be mined if a reliable, stiff backfill could be widely used.

Finally, a further benefit is the possibility of increasing the present depth of mining on the Witwatersrand. As will be apparent from the previous discussion, the various systems at present employed on the Witwatersrand for strata and thermal control and for the transport of rock, men, and materials suffer from identifiable limits, which are being reached at the greatest depths achieved on the Witwatersrand at present. The gold values are known to persist to even greater depths in some areas, and the system proposed in this paper may remove the present limitations on the various systems in a mine to such an extent that mining of the gold at greater depths becomes practical.

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