Mining and Treatment Plant Practice at the Finsch Mine, De Beers Consolidated Mines Ltd.

Published in the Journal, March, 1969

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Discussion:

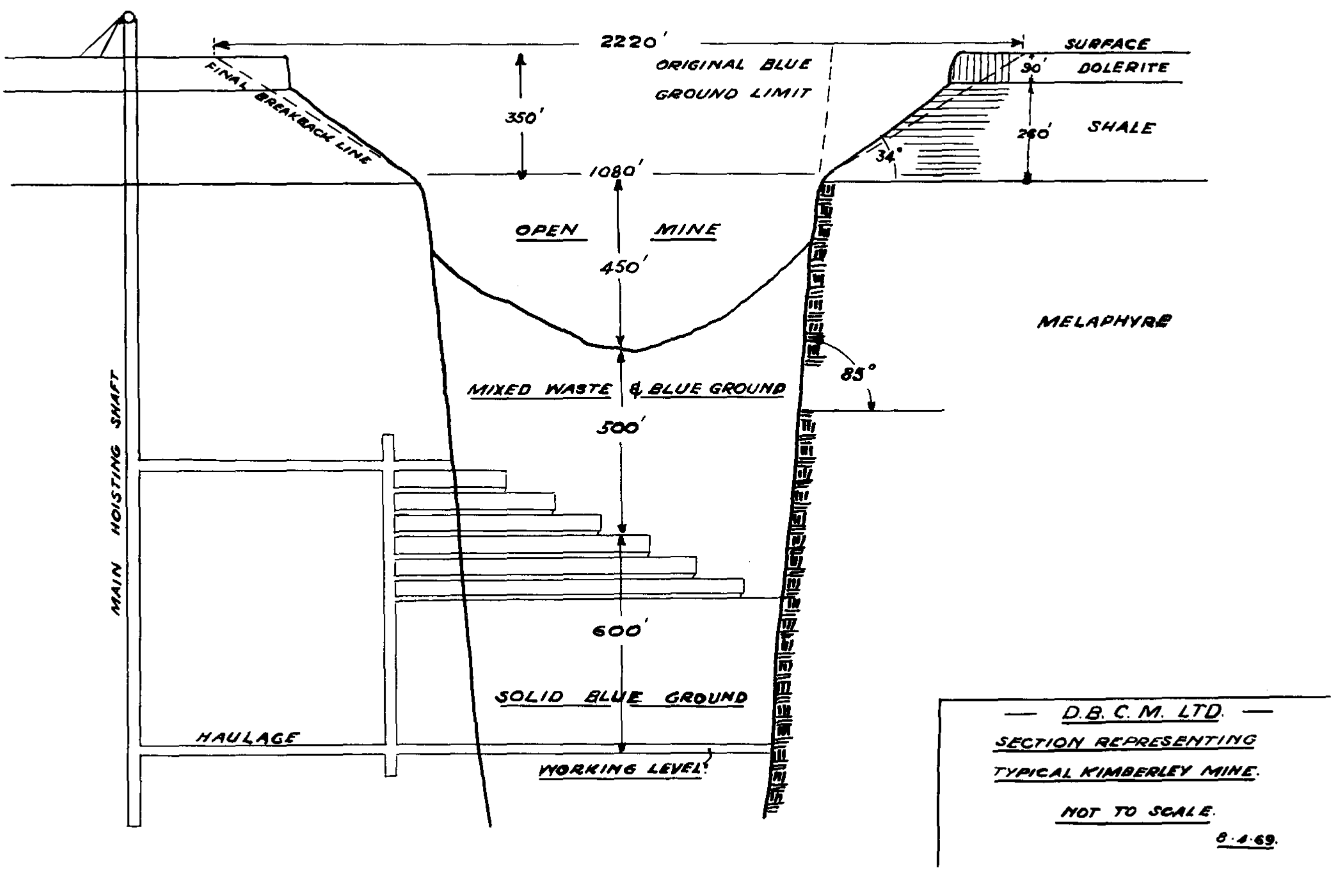
H. J. Wright (Member): The authors are to be congratulated on presenting such a comprehensive and lucid account of bringing a large diamond mine into efficient production from the grass roots.

When one considers the vast amount of planning and hard work that had to be put into the project the authors have done a praiseworthy job in condensing the story, so as to present all the important technical aspects, the logic behind the major decisions and the future thinking on the operations, in a concise and readable form.

This was the first experience of De Beers personnel in starting a large diamond mine as an opencast project *and nor do they employ a large planning staff, most of the planning being carried out by the executive staff themselves.* It is operating most efficiently on a large tonnage and it is neatly and very well laid out. In fact, with its model townships, both for European and Coloured employees, tarred roads, recreation club and its grass golf course, it is like an oasis in the arid climate of the North-Western Cape.

The key operation of opencast mining is, of course, the planning of the geometry of the pit. This has been adequately dealt with by the authors. It is of interest to consider some of the consequences arising from complete lack of open pit planning, based on actual operating mines. I refer of course to the present, four producing mines in Kimberley. The Kimberlite pipes were all discovered in a short span of time from the year 1870, onwards. They soon became a hive of activity with diggers industriously working their claims at varying rates.

The country rock comprises from the surface downwards, some 90 ft of decomposed dolerite followed by approximately 260 ft of soft Karroo shales, which in turn is underlain by a hard competent Ventersdorp lava, known locally as melaphyre. The average water table was just below the dolerite at a depth of approximately 90 to 100 ft. Only the blue ground was mined leaving the waste rock sides almost vertical, and as the depth increased beyond 100 ft, large falls of sidewall started to take place. The shale was unstable and the falls covered



large areas of claims which had to be cleared. In 1881 for instance the average stripping ratio in the Kimberley Mine was estimated to be 3. A fall in De Beers Mine in 1885 is said to have covered the whole pipe and taken four months to clear. Early records do not mention whether there were any casualties.

A gradual changeover to underground mining then took place and in 1888, when all holdings were amalgamated under the control of De Beers Consolidated Mines, all opencast mining ceased and collapsing side walls were no longer considered a problem.

However, a new difficulty soon arose. The water entering the pipe, as well as de-stabilising the side wall, formed a very mobile and highly fluid mud with the shale, which had become finely comminuted. As the head of shale increased with depth a long era of mud rushes commenced. These were quite terrifying, as they would burst through the extraction openings without warning and fill hundreds of feet of tunnel in minutes. In one year it is recorded that 29 mud rushes occurred in one mine which filled 15,000 ft of tunnel.

However, by judicious siting of drainage tunnels in the country rock around the pipe at the water table and below it, the mines were gradually dried out and today the shale overburden is completely dry. Nevertheless it still constitutes a problem when most of the mining in Kimberley is being carried out by Block Caving. Being dry and finely comminuted the shale is fluid and mobile, far more mobile than the broken Kimberlite ore.

From estimates given on the attached table for the four mines a total of approximately 165 million loads of waste (mostly shale) has fallen into the pipes, from surface down to the top of the hard rock melaphyre. The melaphyre has proved to be very stable and falls from it have been negligible. As a comparison the volume of Kimberlite mined for the four mines from surface down to the hard rock is estimated to be approximately 115 million loads, and it is estimated that a further total of 13 million loads of waste are still to fall in, before the shale sidewalls become finally stabilised.

Whilst a large amount of the waste has had to be pulled, the bulk of it still overlies the ore bodies. The sketch attached is a representative section of the four mines, showing the average figures for the four mines. From this it is seen that the average waste capping above the mean level of ore at the start of blockcaving was 500 ft and the total head of broken ground above the extraction level when the cave had broken through was 1,100 ft. The contribution of the extra 500 ft of capping to the weight, which has to be borne by the extraction drifts, cannot be calculated but must be considerable.

In drawing the caved ground, the shale being more mobile than the ore tends to overrun the ore, especially in the later stages when the lower third of the ore is being drawn. This will probably result in the loss of some ore towards the end of a block cave, as once the shale channels through to a draw point it can flow unimpededly through the voids between the lumps of ore. Also if ore is drawn more rapidly in one drift than the adjoining drifts, so that the shale is drawn down to just above the drift, it can migrate at a shallow angle ($\pm 30°$) to the adjoining drifts when they are pulled, causing predilution and cutting off of ore. These phenomena have been clearly illustrated in model experiments as well. Thus every effort has to be made in block caving to maintain an even and uniform draw over the area being pulled.

The Koffyfontein Mine in the South-Western Free State, discovered in 1870, was worked in a similar fashion in the early days and was mined intermittently by open cast until 1931, when the open mine reached a depth of 350 ft and operations ceased. In this case the upper shales of the country rock are extensively intruded

DE BEERS CONSOLIDATED MINES, LIMITED DATA KIMBERLEY MINES WORKING IN MARCH, 1969

		WESSELTON	DE BEERS	DUTOITSPAN	BULTFONTEIN
Surface area	Acres	67·93	65·14 (44·61 before cutting)	91·73	88·90
Area of top of Melaphyre ..	Acres	18·36	10·86	22·38	18·58
Open mine depth at present ..	Feet below surface	800 ft	350 ft	800 ft	800 ft
Depth to top of Melaphyre ..	Feet below surface	370 ft	320 ft	340 ft	340 ft
Axes at top of Melaphyre	Feet	1,200 ft × 700 ft	1,060 ft × 510 ft	2,130 ft × 650 ft	1,010 ft × 980 ft
Thickness of debris	Feet	10 ft	10 ft	10 ft	20 ft
Thickness of dolerite	Feet	90 ft	70 ft	90 ft	40 ft
Thickness of shales	Feet	270 ft	240 ft	240 ft	280 ft
Volume of blueground	Loads	33·2 million	14·7 million	36·6 million	30·9 million
Volume of waste fallen in	Loads	50·4 million	29·8 million	44·1 million	41·5 million
Volume of waste yet to fall in ..	Loads	5·1 million	Nil (already cut back)	4·8 million	3·4 million
Approximate head of overburden at start of cave	Feet	90 ft	750 ft	1,300 ft	1,200 ft
Mean working level	Feet below surface	1,700 ft	1,000 ft	1,900 ft	1,900 ft

by dolerite sills, which have imparted a greater measure of stability to them, as a result of which little fall of sidewall has occurred. The re-opening of this mine will commence early in 1970 and the first step will be to strip back the country rock to a stable angle ($\pm 40°$), involving the removal of some 21 million loads of waste, before the first loads of Kimberlite are hauled. Waste rock stripping will then continue in conjunction with Kimberlite mining so as to maintain a stable wall rock profile.

In conclusion therefore, I am bold to say that, armed with a better knowledge of slope stability criteria and the boon of modern large earthmoving machinery, no unwelcome heritage of waste dilution will be left for posterity at Finsch and Koffyfontein, as was left by our forebears at Kimberley.

I. R. M. Chaston; In the discussion of the waste rock mining rate, the authors suggest that the minimum curve on Fig. 7 will theoretically give the greatest overall profits when calculated on a present value basis.

However, since the maximum stripping rate would then come at the end of the mine life this might well not be so. The number of trucks in the stripping-fleet would have to be increased right up to the end of the life of the mine. It would seem that many of the trucks in this final fleet would not be employed for their full working life.

Thus, for a mine with a projected life approximating to the life of the stripping equipment, it would pay to strip waste at a rate given by the maximum stripping curve. On a mine with a longer life, such as Finsch, the maximum economy will be made by mining, at first, at a rate parellel to, but rather greater than, the minimum curve. This rate curve should intersect the maximum rate curve at a period prior to the end of the life of the mine approximating to the useful life of the trucks. Following the maximum rate line will then enable the fleet to be gradually and economically run down. Although not directly stated, the scheme chosen at Finsch for stripping will follow this principle, and should give the maximum economy in the cost of stripping equipment.

It is noted that the skip-way to be installed will match the present truck size, i.e. 35 tons. However, these skips will not be brought into operation for some years. Do the authors not think it likely that the trucks in use at the mine will follow the present trend in other parts of the world and increase to say, 70 tons, a 100 tons or even more? Will the skip-way construction be such that it can cope with such an increase?

In the description of the treatment plant, it is mentioned that the feed to the tertiary pans is pumped to two 42 in. cyclones used in series. The idea of this being, presumably, that the second cyclone will take out any tramp oversize lost from the first 42 in. cyclone. Experience elsewhere, suggests that the back pressure exerted by the second cyclone on the first cyclone so disturbs the operation of the first cyclone that little advantage is gained by this system. Could the authors give some figures on the operation of these two-stage cyclones at Finsch?

In discussing the Scrubber-Recrush section, it is stated that $+\frac{1}{8}$ in. material is discarded to tailings. I imagine that this is a misprint for $+\frac{3}{8}$ in. material. As stated, the reason for this rejection of $+\frac{3}{8}$ in. material is due to the high proportion of ironstone contained in the upper levels. Although it is not directly stated, I understand that the intention is to crush all the feed material through $\frac{3}{8}$ in. before rejection when these upper levels, with their high ironstone content, have been mined out.

Finally, I should like to congratulate the authors on an excellent paper which gives a clear and detailed picture of the operation of the Finsch Diamond Mine.

G. Schwartz: The authors are to be congratulated on a paper that is of interest to all as the opening of a major diamond mine is a rare occurrence.

In point of fact, Finsch Mine is the first major diamond producer to be brought into production in the Republic since the turn of the century, Premier Mine having started production in 1903.

During prospecting operations at the future Finsch Mine, small samples were made available for test work and results indicated that the material from the pipe would be amenable to treatment by a conventional pan plant. However, it was also apparent that the presence of ironstones, which overlaid the deposit, could complicate washing pan operation and design of the recovery plant if they were present in the plant feed in any great quantity. The ironstone was predominantly of specific gravity greater than 3·5 and as such would form the major portion of the washing pan concentrate. Some of the ironstone was also magnetic. The contact zone material had a high ironstone content and also carried sufficient value to warrant treatment but it was not thought possible to treat this material by itself. Plant design was therefore planned to allow for treatment of this material by mixing with the yellow ground of the actual kimberlite. It was not known, however, how much heavy mineral a washing pan could tolerate and as answers were required to this and how to deal with the high ironstone concentrate in a recovery plant, a decision was taken to construct a pilot plant for this purpose. The pilot plant was designed so that approximately 80 per cent of the equipment could be used in the main plant and thus expenditure for this phase could be minimized.

Due to circumstances which are not relevant here, design and construction of the main plant had to proceed concurrently with test work being done on the pilot plant. In other words the pilot plant test work could not be completed before design work was started for the main plant. This was not as great a drawback as it would appear for with the experience that the company had with washing pans, coupled with some small scale tests that had been done, no serious difficulties were envisaged with the design of the washing plant section but it was recognised that as a result of this policy alterations to the recovery section might be necessary, some even before it was commissioned.

The pilot plant was commissioned in December 1964 with the following five main objectives:

(1) To determine what ratio of the high ironstone contact zone material could be mixed with the yellow ground so that washing and recovery operations would not be compromised.

(2) To obtain information relating to the nature of the washing pan concentrates which would have to be treated by the recovery plant with special emphasis on magnetic separation, heavy media separation and quantities that would require grease belt treatment.

(3) To confirm that the conditioning process chosen would yield good recoveries.

(4) To confirm thickener requirements and to determine more accurate water requirements.

(5) To test a system for grease handling and recovery of grease.

Within a short while of commissioning it became apparent that problems in the recovery section would necessitate all efforts being concentrated there and some of the work planned in the washing plant might have to be sacrificed. Consequently for quite a major portion of the planned test period the washing plant was used solely as a means of supplying concentrates for experimental work in the recovery plant. It also became apparent, as stripping on the pipe proceeded, that ironstone contamination of the yellow ground below the contact zone was far more serious that had been anticipated and it might be necessary to modify and perhaps abandon plans to mix and treat the contact zone material along with the yellow ground.

Two problems had been highlighted in the pilot recovery section:

(1) The quantity of ironstone in the minus $\frac{1}{4}$ in. and minus 7 mesh size ranges was going to be greater than had been planned for and it would be necessary to rearrange portions of the grease belt section in the main plant.

(2) The attritioning and conditioning process was not operating as well as was anticipated and recoveries were poor.

The latter was by far the more serious and intensive testing was commenced early in February 1965. Recoveries were eventually improved to a point where they were acceptable but it was also recognised that a scavenging process for the minus 7 mesh material would have to be worked out, as despite all efforts to improve recovery in this size range, losses were not inconsiderable. It was therefore decided that these grease belt tailings would be dumped for future treatment; this decision was later modified to include all the grease belt tailings.

It was during these tests that a need for additional or third stage of recovery became apparent. Normal grease belt operation produced about 1 per cent concentrate, that is, material which adhered to the grease belt along with the diamonds as a result of the conditioning process. Test results indicated that for acceptable recoveries a minimum of 2 per cent concentrate had to be tolerated and this factor coupled with the finding that the quantities of minus 7 mesh material would be greater than originally planned for, made another section imperative. A third stage of recovery was consequently incorporated into the design of the recovery plant.

Mention has been made of the tests conducted with filtration and water recovery from the waste slimes and an interesting aspect here was the effect that the ironstone had on the whole water recovery problem. Settlement of slimes from the yellow ground presented no difficulty but it was found that contamination by ironstone fines had a most adverse effect and, whereas the yellow ground did not require flocculants, when ironstone was present flocculants were not only necessary but vital if satisfactory settlement was to be obtained. It was found that the guar based flocculants were the most effective and economical of the flocculants available at that time.

Much useful and indeed essential information was obtained from the work done in the pilot plant that more than justified the expenditure for this phase of the operation.

Mention has been made of the installation of X-ray sorters which are to replace grease belts at Finsch Mine and also at Kimberley. The use of these machines might one day be regarded as a milestone in the history of winning diamonds, along with the introduction of the washing pan and the grease recovery process.

The use of grease for the final recovery of diamonds has served the industry well but the very nature of the process and the difficulties relating to defining parameters for control would appear to have made it outlive its usefulness. To anyone trained to putting numbers to things the grease recovery process has always been a challenge. To touch briefly on some of the problems associated with this method of diamond recovery will perhaps illustrate why a search for an alternate method appeared necessary. There are many unknowns with grease recovery and control appears to be more of an art than a science. Much research work has been carried out to improve the process as a whole but the problem of how to evaluate improvements will always be present as assaying for diamonds is not easy. With a tailings product the search is likely to be for one part in 30 or 40 million by weight and statistically meaningful answers can, at times, only be obtained by treating samples of some hundreds of tons.

The so-called greases used are petrolatums and waxes of different grades which are blended to give mixtures of different hardness or more specifically different penetration points, depending on the application. Basically the blend should be such that it is hard enough to resist penetration of the unwanted gangue and at the same time retain sufficient tackiness to hold the diamond which has stuck to the grease, by virtue of its water repellent surface. To date it has not been possible to write a complete specification for a petrolatum for diamond recovery. A penetration value can be specified and there has long been a standard method for determining this; recently the Diamond Research Laboratory developed an instrument that will give a value for tackiness but there is evidence to suggest that there are other properties required which have yet to be defined. This has been brought to light by the fact that petrolatums from different sources, having the same petrolatum specifications, have not recovered diamonds with the same apparent efficiency.

Temperature also plays an important role in the process. The ambient temperature and that of the water used to transport the gravel over the grease are factors which have to be considered and both influence the type of mixture that is used.

The quality of the water can also affect diamond recovery on grease, hard waters being undesirable as they tend to make the diamonds water avid. These are some of the more important factors that have to be considered for successful diamond recovery with this method but even more complications arise when refractory diamonds have to be recovered. These have been defined as stones which do not have a natural water repellent surface. The exact cause of this phenomenon has not been precisely determined but from work that has been done it would appear to be caused by a surface deposit on the diamond, the nature of which varies with the ore being treated. Some further confusion arose with a finding that a freshly cleaved surface on a diamond will also not adhere to grease. In spite of the difficulties that have always been apparent with the grease recovery process, diamond recovery from unweathered kimerlites has been surprisingly good, a tribute perhaps to some skilled operators. But it was when refractory diamonds had to be recovered that the shortcomings of this process really made themselves felt.

A treatment process to render refractory diamonds water repellent and so able to adhere to grease, as has been mentioned, is used at Finsch Mine. This method was developed by the Diamond Research Laboratory and was first used at Oranjemund in South West Africa.

The foregoing are then some of the reasons why work was started on using other properties that a diamond is known to possess as a basis for a final recovery process. X-ray separators do not appear to be troubled by the type of difficulties that have so long been apparent with the grease recovery process and there is every reason to believe that within a relatively short period they will be widely used for the final stage of diamond recovery.

H. S. Simpson: Since publication of the above paper, one of each of the two types of wide beam X-ray separators became available for test purposes. These were a Diamond Research Laboratory wide beam and the Gunson Sortex XR-11B broad belt machines.

The machines were installed at different times in the recovery plant in Kimberley. As they have a very high capacity they could not be tested fully on run-of-plant flow without the addition of large bins before and after the process and it was therefore decided to set them up in a test rig and to circulate material through them for test purposes. A known number of diamonds were added to this material, these diamonds being of the correct size and incidence. By this means the size range and feed rate of the test material could be varied and a count of the diamonds recovered would determine the efficiency of recovery.

In the test rig, the machine to be tested was mounted on a platform. A two-ton bin, with an outlet leading to the machine, was mounted above it and a similar bin, to collect the tailings, below it. From an outlet in the tailings bin the tailings could be gravitated via a gate to one-ton containers which were then hoisted and tipped into the upper bin. The concentrates spout from each machine was led to a locked container. With this rig quantities of up to seven tons per hour could be circulated through the machines.

As the tests were primarily intended to determine which machine was suitable for Finsch Mine $+\frac{1}{4}$ in. concentrates, several tons of grease belt tailings was transported to Kimberley for test durposes. At the same time, a similar quantity of Kimberley grease belt tailings were collected for parallel tests.

The first machine to be tested was the Diamond Research Laboratory separator, which employs two electro-magnetically vibrated feeders in series to provide a mono layer of particles. As the stream of particles drops in a curtain off the second feeder, it passes through a wide X-ray beam in a darkened cabinet. Any resultant fluorescence is detected by photomultiplier tubes which trigger, through an electronic system, a compressed air operated flopper gate which takes a cut from the stream as concentrate.

Tests were carried out with both wet and dry material, results indicating that, with abrading of the material in handling dust was formed in the dry state, and slime in the wet state. Of the two conditions it was felt that treatment in the dry stage was preferable, as dust generated could be removed continuously by an extractor. It would appear necessary in plant practice therefore, to dry all sizes of feed for this process.

Many teething troubles were experienced with this machine, mostly mechanical, and results were inconclusive. It became obvious during testing however that it would not be suitable for separation of gravels where the incidence of diamonds was high.

The second machine tested was the Gunson Sortex XR-11B broad belt separator. This employes a hydraulically operated vibrating feeder for control purposes, delivering onto a 6 in. wide conveyor belt with resilient flanges which contains a mono layer of particles of material on the belt. The trajectory from the belt passes through a wide X-ray beam in a darkened cabinet. Fluorescence from diamonds is detected by photomultiplier tubes which trigger, through an electronic system, a blast of air from ejectors to divert the diamond from the stream. The detection and ejection system in this machine is split into two sections so that detection of fluorescence in one-half width of the stream results in ejection on that side of the stream only, instead of over the whole width. With low incidence of diamonds this results in a low percentage concentrate being produced. A dust extractor is provided for continuous removal of dust which would otherwise affect the photomultiplier tubes.

The tests were highly successful but revealed that, for best recoveries, two stage operation, and closer sizing of the feed would be necessary. The latter presents no problem as one machine can be used for treating any size range of material, a rapid change over being made by operating a control on the outside of the cabinet to adjust the ejection system. They also revealed that, on Finsch Mine concentrates, it was effective in the -4 $+7$ mesh size range thus encroaching on the field of the XR-21 separator. The machines have a considerably higher capacity than the XR-21 units although a higher percentage concentrate is produced.

Test results at the most effective size ranges were:

Size range	*Capacity T.P.H.*	*Per cent concentrate*	*Per cent recovery*
$-1\frac{1}{4}$ in. $+\frac{3}{4}$ in.	7	Trace	100·00
$-\frac{3}{4}$ in. $+\frac{1}{2}$ in.	5	0·03	100·00
$-\frac{1}{2}$ in. $+4$ mesh	3	0·25	100·00
-4 $+7$ mesh	2	1·19	99·90

On the results of the tests, four of these machines have been ordered for treating $+4$ mesh Finsch Mine material, the -4 mesh to be treated by XR-21 machines. The four machines will also have spare capacity for treating any surplus in the -4 mesh size range that might tend to overload the XR-21 machines. This overloading may be caused by the proposed re-treatment of stockpiled recovery plant tailings.

A series of parallel tests were carried out on Kimberley grease belt concentrates. These revealed the unexpected phenomena of fluorescence of zircon, a constituent of the heavy media concentrates, which fluoresces under the influence of X-rays in the same colour spectral distribution as diamond. The fluorescence causes excessive triggering of the machines with the production of a high percentage of concentrate.

The zircon occurs mostly in the -4 $+7$ mesh size range and elimination of it from feed to X-ray separators is a problem yet to be solved and on which research is being carried out. The lower incidence of zircon in the $+4$ mesh size range presents no problem and on the results of the tests, six of these machines have been ordered to replace the existing grease belts.

Reply by the authors to the discussion

The authors would like to thank the various contributors who have generally added a wider background to the paper itself. The remarks made by Mr Wright have highlighted the problems caused to the Kimberley mines by not stripping the unstable shales back to a safe angle at an early date. The subsequent costs of mud rushes, additional weight and the mining of ore diluted with shale, have undoubtedly been considerable. It is of interest that the early diggers recognised the problem and through their associations made attempts at stripping these shales—however, the lack of capital doomed these attempts from the start. A review of the early history of the 'dry diggings' would make fascinating reading today and it is hoped that Mr Wright will consider producing a paper on the subject at a later date.

With reference to Mr Chaston's contribution, the authors would like to make the following comments:

(*a*) The calculation of the most economic stripping rate is complex and can probably only be done satisfactorily with the aid of a computer, provided reliable future cost estimates are available. Mr Chaston rightly points out that the economic life of equipment is an important factor in these calculations.

(*b*) The trend at Finsch, as with all large stripping operations, will be to purchase larger truck sizes with increased depth of operation. However, if the concept of a skipway is pursued, the design features of such a skipway are such that the ore trucks are likely to be standardised at 35 tons.

(*c*) The original intention of two-staging the 42 in. primary cyclones was to remove tramp oversize lost from the first 42 in. cyclone. Mr Chaston correctly forecast that the back pressure exerted by the second cyclone on the first cyclone would be undesirable and these cyclones have been changed back to single stage operation.

(*d*) In the paper it is stated that $+\frac{1}{8}$ in. material from the Scrubber-Recrush section is discarded to tailings. This is a mis-print and should read $+\frac{3}{8}$ in. With the reduction in the percentage of ironstone with depth, this material will all be crushed to $\frac{3}{8}$ in.

Mr Schwartz, with his personal knowledge of plant practice at Finsch Mine, has added considerably to the value of the paper. As he mentions, the design and construction of the Main Plant had to proceed concurrently with test work being done on the Pilot Plant. As a result, a very high priority was placed on achieving early results to assist in the design of the Main Plant, with the main emphasis being on the recovery section. In spite of improvements made during this period, grease recovery of diamonds was never completely satisfactory at Finsch and thus research was intensified into alternative recovery methods finally leading to the successful development of X-ray sorters. Mr Schwartz is to be congratulated on his review of the grease method of recovery and the disadvantages associated with its use. The shortcomings of this process really became apparent when treating refractory diamonds.

Mr Simpson has supplied figures of test work carried out on the Gunson Sortex XR-11B broad belt machines. Four of these units are shortly to be installed at Finsch Mine to treat the final $+\frac{1}{4}$ in. concentrate. These four machines will operate in two banks of two, the second bank having been installed to increase the rate of retreatment of stockpiled Recovery Plant tailings. Six of these machines are also on order for the Kimberley Recovery Plant and will treat all +4 mesh concentrates. Additional units will be required for the $-4+7$ mesh material if the problem of zircon reporting with the diamonds can be solved.