Opencast coal mining at Kriel Colliery

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SYNOPSIS

In 1969 the Coal Division of the Anglo American Corporation was awarded the tender to supply coal to the new 3000 MW power station to be erected at Kriel, which is situated midway between the towns of Ogies and Bethal in the eastern Transvaal. The coal was to be supplied from the No. 4 Seam in the Kriel Area of the Witbank Coalfield. The No. 4 Seam in this area varies in depth of overburden from 75 m to 6 m, with an average of 55 m.

Initially, the mine was planned as an underground mine incorporating a three-shaft system for transporting some 8 million tons of coal per annum to the power station. As the result of negotiations between Anglo and Escom to improve the volumetric extraction of coal from the new mine, the Kriel Division of Anglo Power Collieries was finally planned to produce 4,1 million tons of coal from an underground mine and 4,5 million tons from an opencast mine, with a resultant increase in volumetric extraction.

This paper describes the opencast operation at Kriel, with particular reference to the water problems encountered and to the operation of the draglines on sloping ground to a depth of 40 m to the top of the coal seam.

SAMEVATTING

Die tender vir die voorsiening van steenkool aan die nuwe 3000 MW-kragstasie wat by Kriel, halfpad tussen die dorpe Ogies en Bethal in Oos-Transvaal opgerig sou word, is in 1969 aan die Steenkoolafdeling van die Anglo American Corporation toegeken. Die steenkool sou voorsien word uit laag nr. 4 in die Krielgebied van die Witbank-steenkoolveld. Die diepte van die bolaag van laag nr. 4 wissel in hierdie gebied van 75 m tot 6 m met 'n gemiddeld van 55 m.

Die myn is aanvanklik beplan as 'n ondergrondse myn met 'n drieskagstelsel vir die vervoer van ongeveer 8 miljoen ton steenkool per jaar na die kragstasie. As gevolg van onderhandelinge tussen Anglo en Evkom om die volumetriese ontginning van steenkool uit die nuwe myn te verbeter, is die Krielafdeling van Anglo Power Collieries uiteindelik beplan om 4,1 miljoen ton steenkool uit 'n ondergrondse myn en 4,5 miljoen ton uit 'n oopgroefmyn te produseer met 'n gevolglike toename in die volumetriese ontginning.

Hierdie referaat beskryf die oopgroefbedryf by Kriel met spesiale verwysing na die waterprobleme wat teëgekom is en die werking van die sleepgrawe op skuins grond tot 'n diepte van 40 m tot aan die bokant van die steenkoollaag.

Introduction

In 1969 the Coal Division of the Anglo American Corporation was awarded the tender to supply coal to Escom's new 3000 MW power station that was to be erected at Kriel, which is situated midway between the towns of Ogies and Bethal in the eastern Transvaal.

The Anglo American Corporation had acquired a large area of proven coal reserves in the No. 4 Seam of the Kriel Coalfield, which was to be mined to supply the coal for the power station. The No. 4 Seam in this area varies in depth of overburden from 75 m down to 6 m, with an average depth of 55 m to the top of the coal seam. The mine had initially been planned to produce 8 million tons of coal from an underground mining operation.

Owing to uncertainty surrounding the coal reserves available in South Africa following the publication of a report entitled *South Africa's Coal Resources*, a commission of inquiry into the coal reserves of the Republic of South Africa was appointed in May 1970 by the State President, under the chairmanship of Dr A. J. Petrick.

Escom and the Coal Division of Anglo had for some time expressed their concern about the extremely low percentage of volumetric extraction being achieved on many of their coal mines. This fact, together with the appointment of the Petrick Commission and the looming energy crisis, led to negotiations between the two parties to consider alternative methods of mining on the new mine that would improve the volumetric extraction.

Subsequent discussions and planning finally led to the establishment of a colliery comprising an underground mine to produce 4 million tons of coal per annum and an opencast mine to produce 4,5 million tons of coal per annum, with a resultant increase in the volumetric extraction of coal.

Geology of the Kriel Coalfield

Topography and Stratigraphy

The Kriel Coalfield covers an area of more than 25 000 ha. The country is generally undulating, with the main Dwars-in-die-Weg/Steenkoolspruit Valley more or less bisecting the coalfield. The few extensive flat areas within the coalfield are largely confined to the crests of ridges and to the floor of the main valley. Apart from a prominent scarp on the north of the Steenkoolspruit and a few pans and marsh areas, there are no distinctive topographic features within the field. Prominent hills, composed of pre-Karoo felsites and granites, occur immediately to the north of the area.

Some 500 boreholes have been drilled throughout the field. The data obtained from these holes and from surface observations have provided a detailed knowledge of the stratigraphy and geology of the area.

The coalfield is underlain by a thin sequence of Dwyka and Middle Ecca strata resting on an undulating floor composed of felsites, granites, and diabase associated with the Bushveld Complex. The Karoo strata have been denuded in the north by the Steenkoolspruit, and, where prominent pre-Karoo hills are present, the lower strata of the Karoo System, including the main coal seams, have not been deposited at all.

A typical stratigraphic column is shown in Fig. 1.

*Kriel Division, Amcol Company, Anglo Power Collieries (Pty) Ltd, Kriel, Transvaal.

The stratigraphic units shown are virtually identical to those found in the Witbank Coalfield, which is situated immediately to the north of the area. The stratigraphy throughout the Kriel Coalfield is remarkably uniform.

The topography of the No. 4 Seam is generally flat to gently undulating. Where a dolerite sill has cut through, the seam is faulted and in some areas slightly tilted along the margins of the fault. The coal is generally burnt in these fault-margin areas.

There are two main dolerite sills in the coalfield, the thickest occurring in the south-west area of the field. The thinner sill, 1,5 to 15 m thick, is very extensive. It underlies the No. 4 Seam in almost half the field, and overlies the seam in the rest of the field. In some areas, the sill has cut through the seam in dome-form fashion, whilst the same sill has intruded through the coal horizon in at least six places. The vertical throw on the faults caused by this intrusion varies from 6 to 25 m. Very few dykes have been located on the surface, probably because they are not very numerous and because the generally thick cover of soil renders them difficult to detect either on aerial photographs or by magnetometer.

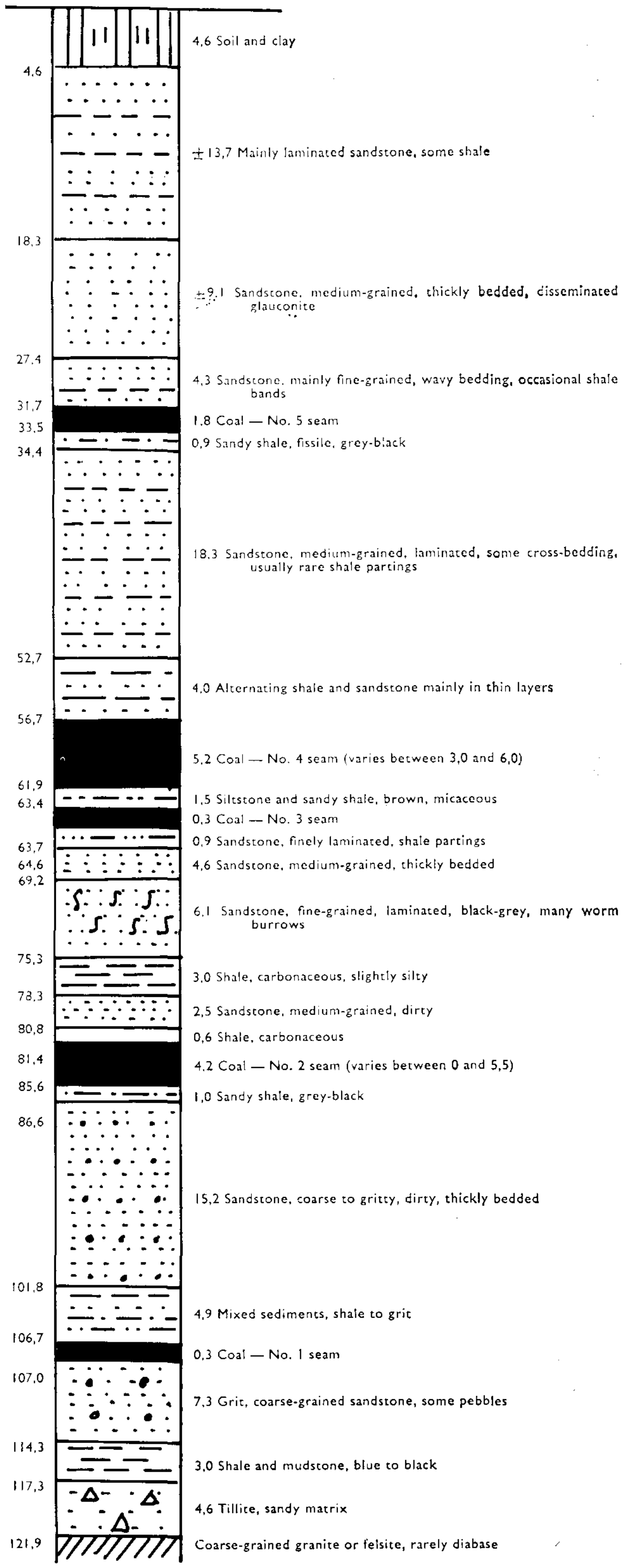

Fig. 1—A typical stratigraphic column, Kriel Coalfield (all measurements in metres)

The No. 4 Coal Seam

As shown in Fig. 1, the main coal seams of the Witbank Coalfield are all present in the area.

The No. 4 Seam occurs through most of the area, and is unworkable only where it has been affected by weathering or burnt by dolerite. A typical graphic log of the seam is shown in Fig. 2. The seam varies in thickness from 3 to 6 m, but is remarkably consistent at about 4,9 m in thickness.

The coal tendered for the underground mine reserves is contained in the lower two-thirds of the seam (usually below the marker shale), except in a few isolated areas where a thick parting occurs near the seam floor. In these exceptional cases, the workable section extends down to the top of the parting. The coal is generally dull and dull–lustrous, with a few bright bands. The top third of the seam is composed of inferior coal interbedded with bands of shale, coaly shale, and carbonaceous shale. The average calorific value of this section is only in the region of 13,3 MJ/kg.

Planning of the Underground Mine

The initial tender to Escom envisaged a production of 8,0 million tons of coal from a completely underground operation.

The method of mining was planned on a bord-and-pillar system, the bord width being accepted at 6,2 m. The working height was accepted as being the shale marker, later termed the *fossil marker*, which has a natural bed separation and a good roof. This mining height varies between 3,2 and 4 m, with an average of 3,5 m, leaving an average of 1,4 m of coal in the roof.

A panel system was to be developed within barrier pillars containing a limited number of entries to the panel, since tests had shown that the coal was liable to spontaneous combustion. Pillar centres were calculated by the use of Salamon's Nomogram. Where the overburden cover was less than 15 m, no mining was planned.

Conventional mechanized mining was envisaged, using cutters, loaders, and shuttlecars with the normal drilling and blasting operations. Back-up equipment would be provided to allow for maximum production at

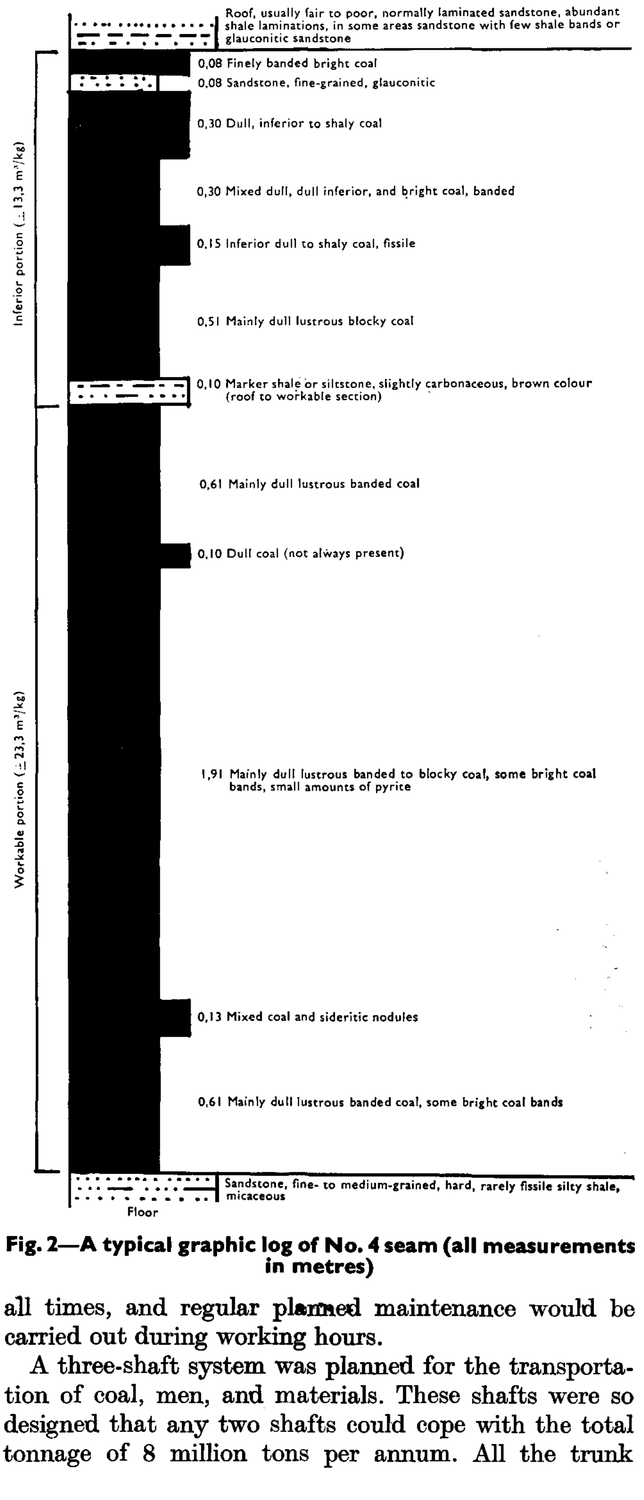

Fig. 2—A typical graphic log of No. 4 seam (all measurements in metres)

all times, and regular planned maintenance would be carried out during working hours.

A three-shaft system was planned for the transportation of coal, men, and materials. These shafts were so designed that any two shafts could cope with the total tonnage of 8 million tons per annum. All the trunk conveyors were designed with 1350 mm belts, whilst the section belts were to be 1050 mm in width.

The total area covered in the tender for the full underground operation amounted to 8826,6 ha. In this area, the coal *in situ* was calculated to be 657,4 million tons. A geological loss of 10 per cent was estimated, which would leave 591,7 million tons of extractable coal. Mining losses due to selected mining of the lower part of the seam, to loss in boundary pillars between the panels, and to loss in pillars in the panels amounted to an extraction percentage of 43,8 per cent. The total reserves to be extracted therefore amounted to 259 million tons.

Planning of the Opencast Mine

Percentage Extraction

When the opencast operations were included in the planning, the mining area changed its boundaries, but the aerial extent remained almost constant and amounted now to 8901,5 ha. The reserves are shown in Table I, which gives only an indication, since the areas selected for underground mining and for the combined operations do not coincide exactly.

Table I indicates an overall increase of more than 14 per cent in the extraction owing to the very large improvement in extraction from underground to opencast mining. Because of the lower calorific value of the coal in the opencast operation, the planned annual production was increased from 8 to 8,5 million tons of coal. It will be noted that this increased demand is more than compensated for in the improved extraction rate. The increased demand amounts to $30 \times 0{,}5 = 15$ million tons, while the increase in extractable coal is $338{,}5 - 259 = 79{,}5$ million tons.

Production Required

In the opencast operation, full seam extraction was planned, which would mean a decrease in the calorific value of the coal delivered to the power station (see Fig. 2). After discussions with Escom, it was agreed that three generating sets of the station would burn coal from the underground mine, while the last three sets would be fired with coal from the opencast operations. Thus, the total coal requirements increased from 8 million tons per annum to 8,5 million tons per annum, consisting of 4 million tons from underground and 4,5 million tons from the opencast mine.

Draglines Required

When opencast mining of the field was first discussed, a stripping ratio of 6:1 was used to determine the

TABLE I
ORE RESERVES AT KRIEL COLLIERY

Method of mining*	Area ha	Seam thickness m	Ore *in situ* $t \times 10^6$	Extractable ore $t \times 10^6$	Extraction %	Extracted ore $t \times 10^6$
Underground	5817,7	4,90	433,3	390,0	43,8	170,8
Opencast pits 1 to 3	1635,3	4,32	108,8	97,9	90,0	88,2
Opencast pits 4 to 6	1448,5	4,40	98,2	88,3	90,0	79,5
Total	8901,5	4,73	640,3	576,2	58,7	338,5

*The geological loss was estimated as 10 per cent for each method of mining.

capacity of the draglines required to remove the overburden, since the long delivery times did not allow the completion of a detailed drilling programme and geological evaluation.

At a production rate of 4,5 million tons per annum, the stripping capacity required amounted to 27 million bank cubic metres of overburden, which is equivalent to a total dragline capacity of 122,3 m^3. On this basis it was decided to purchase two 61,2 m^3 draglines to work in two separate pits, thus achieving standardization with Arnot of a well-proven machine and local manufacture of parts of these units in order to meet the required delivery dates.

The completed drilling programme in the area being worked at present indicated a small decrease in stripping ratio, and, to maximize the utilization of the draglines, it was decided to modify the dragline to an increased boom length of 94,5 m and a bucket capacity of 56 m^3. The effect of this change was to decrease the percentage of rehandled material in the deeper areas of the mine.

The digging capacity of the two 56 m^3 draglines includes a provision to cope with areas of deep overburden where rehandling is necessary. In addition, it provides for operation in out-of-balance situations caused by geological anomalies and quality blending of the coal. In future extensions of the opencast mine, the stripping ratios are to be higher than in Block 4, which is being worked at present, and the maximum capacity of the draglines will be more fully utilized in these areas. The very large variation in stripping ratio (shown in Table II) was the main problem encountered in the initial planning of the opencast operation.

Pit Configuration

In consultation with Escom, it was decided that the opencast operation would be started in Block 4 immediately south of the power station. The haulroad and two provincial roads cutting through the reserves divided the block into three distinctly separate pits. Detailed geological drilling further divided Pit 3 into two separate pits, Pit 3 North and Pit 3 South, as the result of a no-coal strip roughly parallel to the small stream bisecting the field. Mine planning was started on Pit 1 using the computerized dragline simulation programme, which is described elsewhere[1].

The parameters used were based on information available from the opencast operation at Arnot that had been adjusted to the specifications of the Kriel draglines.

During the early stages of the planning it was soon assessed that Pit 1 would be the most difficult pit to mine owing to the shape of the pit, the varying depth of overburden, and the steep slopes that would be encountered. It had originally been decided to follow the American practice of contour mining to exploit the pits, but it soon became apparent that this principle could not be applied at Kriel, where after a few years both draglines would be required to move overburden in excess of their capacity to uncover the required coal for the power station, which amounts to 375 000 tons per month. Once this fact was accepted, use was made of the concept of average stripping ratio, which required the dragline to dig in the same cut in relatively shallow overburden, as well as in overburden as deep as 40 m, which is accepted at present as the limit of opencast mining.

In total, 11 different plans of cut configuration were investigated. The experience gained in the development of the mining plan of Pit 1 proved advantageous because the cut configuration of the other pits was based on the same principle of mining to the average stripping ratio and could be finalized in a relatively short period.

TABLE II

AVERAGE AND MAXIMUM STRIPPING RATIOS IN THE VARIOUS PITS

Area	Pit	Stripping ratio	
		Average	Maximum
Block 4	1	4,2	7,0
	2	3,6	5,0
	3	4,2	7,5
Block 6	4	3,8	6,0
	5	5,6	8,1
	6	5,3	9,0

Stream Diversion

A small tributory of the Steenkoolspruit bisects the two pits where mining would commence. It was therefore decided that this stream should be diverted over the no-coal area, thus minimizing the loss of coal. This diversion, which required the excavation of almost 1 million cubic metres of soil, was started in July 1976, and was completed in December 1977 about six months ahead of schedule.

The diversion was constructed according to the design of the Civil Engineers' Department of Anglo American Corporation, who based the design on the requirement to cater for a 20-year flood in the secondary diversion around Pit 3 North and for a 100-year flood in the primary diversion around Pit 1. Strict control was exercised over the gradient of the floor to prevent excessive erosion, and this required the building of 11 gabions in the diversions, which was pioneered at Kriel. The stone required for the gabions was mined from a small quarry excavated around a defunct adit into old workings in No. 5 Seam. Indications to date are that the gabions have served a very useful purpose. A typical layout of a gabion is shown in Fig. 3.

Construction of Haulroad

The position of the haulroad was planned so as to cut over the coal reserves at the narrowest extent of the pits, but was governed by the position of the ash dam of the power station. As the haulroad ran over very incompetent ground, sub-soil drains had to be constructed under the road over about half its length.

Drilling of Overburden

Since No. 4 Seam is fairly horizontal, overburden-drilling conditions are affected by the surface topography, which varies from a low-lying flood plain to a maximum overburden-stripping depth of 40 m. Sedimentary rock strata (shales and sandstones) overly the coal seam. In the low-lying flood-plain area, these

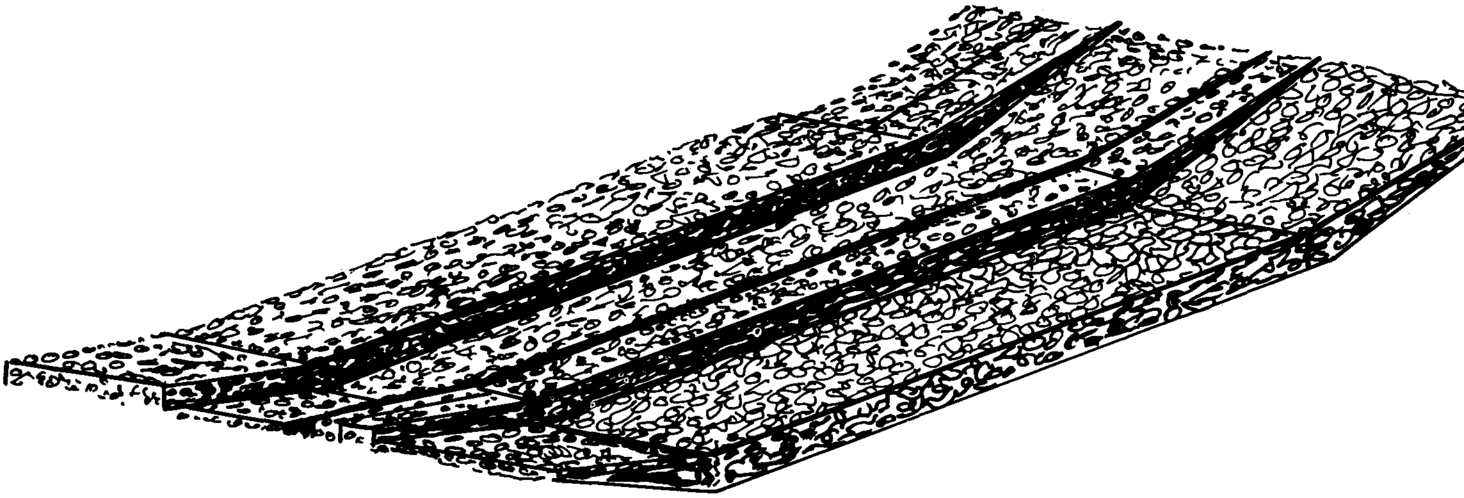

Fig. 3—A typical gabion

strata have weathered into a soily clay material of up to 8 m (Fig. 1) in thickness and overlie approximately 2 m of hard shale.

The deep areas comprise competent hard layers of shale and sandstone. For control of the drilling and blasting operations, all the blast-holes have a specific number. The numbering system was derived by the division of each pit into 100 m-wide zones, which are numbered. The centre of each zone is regarded as row M, and the holes are numbered from the box cut, e.g. 1/8/P/99 denotes Pit 1, zone 8, row P, number 99 (Fig. 4).

Gardner Denver GD 120 blast-hole drills are used for the drilling of overburden, producing a hole 311 mm in diameter. The two conditions require differing drilling operations.

Shallow Drilling

A drill rod of 219 mm outside diameter is used in conjunction with a four-rib stabilizer. A 311 mm steel-tooth tricone bit with a nozzle size of 17,5 mm is run at 100 r/min.

One of the problems encountered concerned the baling of the cuttings. The original equipment bought for the drills consisted of 273 mm drill rods and 311 mm tungsten carbide insert-type stabilizers. Both the annular spacing between stabilizer webs (and drill rod) and hole were insufficient for removal of the clay material, which led to clogging of the tricones of the bits. These problems were overcome by the use of the 219 mm diameter drill rods in conjunction with a four-rib stabilizer. The bit life is now typically 2200 m.

Another problem concerned pad building. Because the area was an old flood plain, a 1 m stone pad had to be constructed to take the weight of the drill. Problems were experienced in obtaining the stone until hard shale was obtained by the mining of a quarry to the west of the proposed Pit 1.

Deep Drilling

The depths vary from 11 to 40 m, and the formations range from soft micaceous sandstone, through shale and a very hard abrasive sandstone, to shale. Water was encountered to within 2 m of the top of the hole. At present, the water table has been reduced to within 8 m from the bottom of the hole. The drilling pattern used at present is 9 m by 9 m square.

In this area, the drills are fitted with 273 mm diameter drill rods, roller stabilizers, and tungsten carbide insert bits. The penetration rate is 1,25 m/min. The average time for the addition and removal of two drill rods is 16,5 minutes, and a hole of average depth (25 m) takes 54,18 minutes including walking time, i.e. 4860 BCM per hour. The rotation of the bits varies between 60 and 95 r/min, with a pull-down pressure between 13 800 and 34 500 kPa. The bit life, which has reached levels as high as 9700 m, averages 7800 m. Bit failures are mainly due to the collapse of bearings as the result of fatigue.

Blasting of Overburden

Sinex F600 slurry is the blasting agent used in all the drilling areas. The explosive is supplied as a 'down-the-hole' service in batches of 12 tons from the on-site factory; 400 g of Pentolite attached to premium Cordtex is used to detonate the blast pattern, and 25 ms relays are used to 'time' the blast-holes.

The close proximity of the opencast workings to the Kriel Power Station (3000 m), and particularly the ash dams (500 m), was cause for concern at the start. It was felt that vibrations, both ground and air blast, might have a damaging effect on these constructions.

Test blasts were carried out so that the maximum or limiting mass of charge detonated per delay could be determined. Calculations resulted in a maximum limit of 6 tons of explosive per delay at a minimum distance of 500 m, which is the closest that mining operations will approach the ash dams.

Shallow Areas

In the initial drilling in the Pit 1 box cut, the top layers of mud slowly collapsed into the hole after the completion of drilling and prior to charging-up operations. The most effective method of charging a hole with explosives was found to involve pumping of the

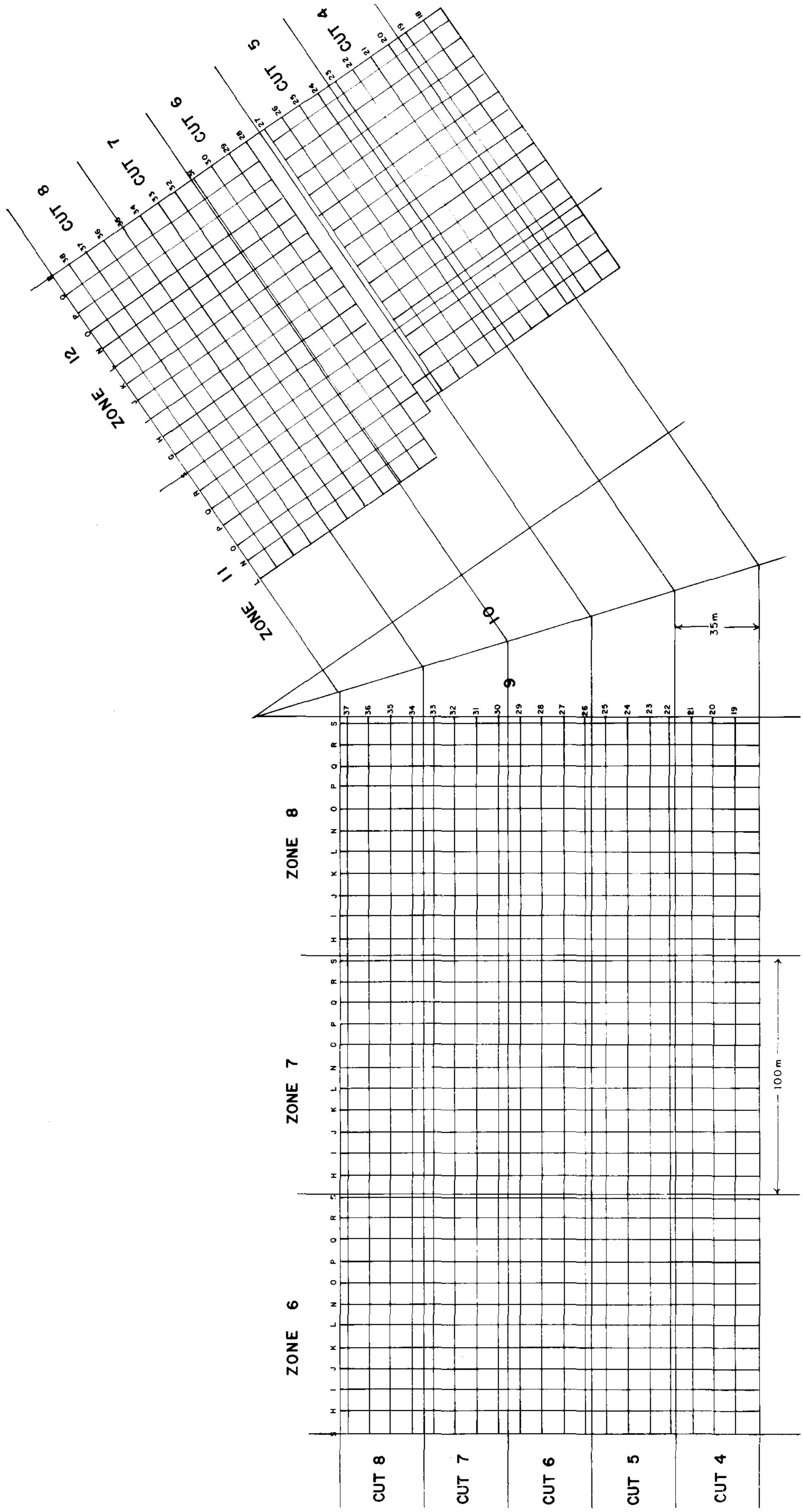

Fig. 4—Opencast overburden drillhole pattern (9 by 9 m)

slurry into the hole immediately after the drill rods had been removed. This required the approval of the Department of Mines for exemption from Regulation 9.33.2.

Where drill holes encounter less than 2 m of hard shale, the minimum charge that can be used is 120 kg of slurry to give a 1,4 m column rise of explosive. This minimum column rise is required for the slurry to reach a stable detonation velocity of 4500 m/s. The blasting ratios obtained vary from 0,15 to 0,25 kg/BCM depending on depth.

Deep Areas

The existence of the very hard, competent sandstone formation, which is some 8 m thick and about 8 m from the top of the coal seam, has caused concern. The sandstone tends to act as a cap to any explosive charge beneath it, resulting in poor fragmentation of both the sandstone and the overlying shale strata. The consequences are very tight digging and low output of the draglines.

The most successful method to fragment the sandstone is deck loading of the explosive charge. A typical hole 30 m in depth would be charged as follows (Fig. 5).

Bottom charge	480 kg of 10% aluminium	5,5 m rise
Deck stemming	Drill chippings	4,0 m rise
Top charge	510 kg of 7% aluminium	6,0 m rise
Overall blast ratio		0,41 kg/BCM.

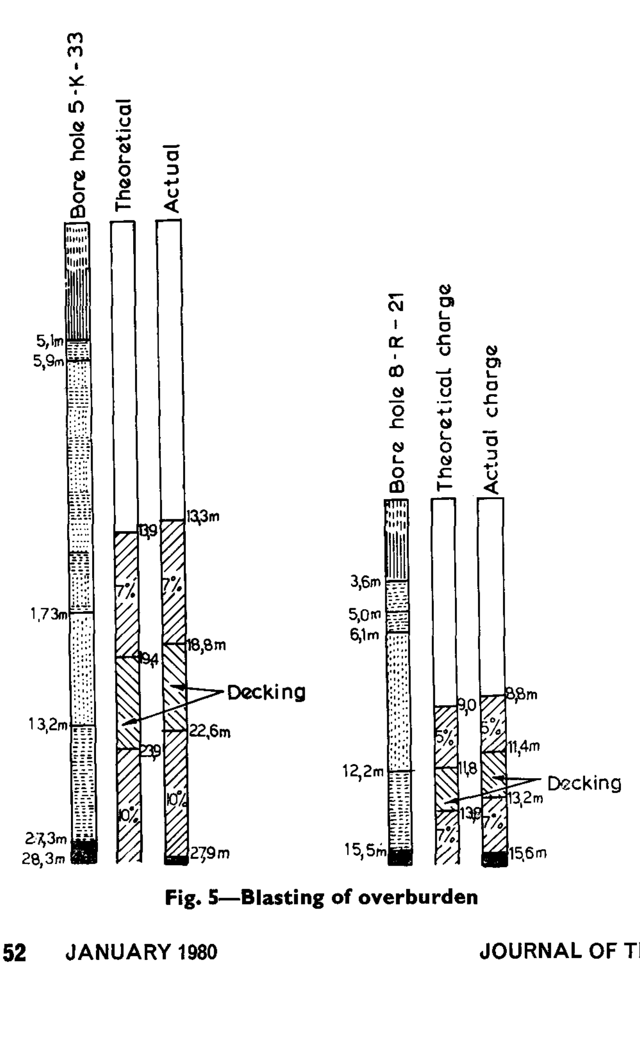

Fig. 5—Blasting of overburden

It must be noted here that the maximum slurry and stemming-column lengths for a formulation of 10 per cent aluminium is 35 m at a critical density of 1,30 g/cm^3, and that for a formulation of 7 per cent aluminium is 25 m at a critical density of 1,27 g/cm^3.

In areas where the depth of the overburden is more than 30 m, buffer drilling is dispensed with and the highwall is blasted into the pit. The reasons for this are discussed in the next section.

Stripping of Overburden

Two Bucyrus-Erie 1570W draglines are used for the stripping of overburden, one in each pit, to a maximum depth of 40 m. The boom lengths are 94,5 m, with an operating radius of 86,9 m at 34,5°. The size of the buckets is 56 m^3. The average stripping ratios in the present two pits are 4,2:1 and 3,6:1, and the width of the pits is 35 m on the top of the coal.

Shallow Areas

Because the shallow areas are very marshy, it has been found essential to construct a good digging pad from shales. The local vegetation is left *in situ* since it acts as a good foundation to the pad. This implies that there is no chop-down for the recovery of top soil by means of the dragline. This practice is acceptable since the Rensburg soils *in situ* are of poor quality and do not warrant recovery.

Normal techniques of dragline stripping require the dragline to move into a key-cut position, followed by a move into the strip-cut position. This has been found unnecessary in areas where there is less than 10 m of overburden. The dragline is located in the key-cut position, from which the whole of the overburden is removed. Swing angles of 60° to 75° are used so that the spoil is spread uniformly.

Deep Areas

The conditions and techniques for the stripping of overburden vary as follows in the present Kriel coalfield:

- normal operations where there is no significant change in the surface topography,
- hill climbing, which requires a series of steps in order to negotiate an increased change in elevation, and
- extended bench, together with a chop-down operation.

In all the operations, a void of 11 m is maintained between the exposed coal and the toe of the spoil. The key cut is 11 m wide at the top of coal position.

Normal Operation

In areas where the depth of the overburden is between 10 m and 28 m and there is no significant change in elevation, normal dragline stripping is carried out. A key cut is dug, followed by a strip cut 30 m in length (Fig. 6).

Hill Climbing

A dragline has to stand on level ground during the digging cycle. The maximum gradient that the 1570W can negotiate during the walk is 7,5 per cent (4,3°).

On backing into a hill, the following dragline sequences take place.

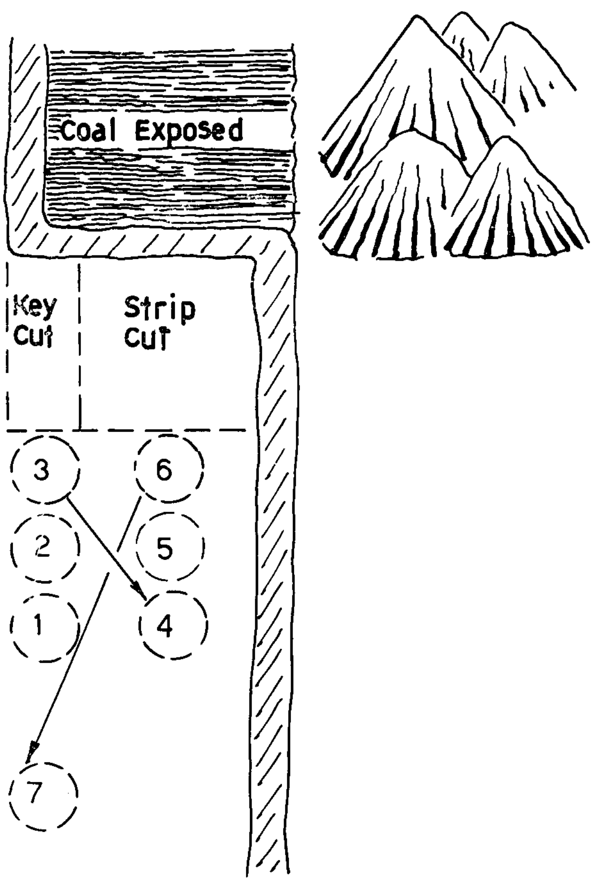

Fig. 6—Normal operation

(a) When the digging face is 87 m away from the point where there is a significant increased change in elevation, the dragline chops down behind in order to make a level digging pad. This is carried on until the chop-down face height reaches the elevation of the next raised bench. The width of the chop-down is 70 m in order to facilitate the dragline's exit roadway. The digging platform of the raised bench is prepared by dozers. (Fig. 7).

(b) While operation (a) is being carried out, dozers prepare the walking ramp so that it does not exceed 7,5 per cent.

The above practice is carried out until the final bench elevation has been achieved. At Kriel, two steps are being taken in order to raise the dragline 16 m (Fig. 8) to 30 m.

Extended Bench

The extended-bench operation is carried out in areas outside the normal operating parameters of the dragline. With a boom length of 94,5 m, this condition is reached at an overburden depth of 28 m. When the dragline walks up to the final bench elevation, the chop-down material is used to extend the bench (Fig. 9) of the first cut and any subsequent cuts. Should the extended bench have to be enlarged, key cut material is used (Fig. 10).

The extent of the extended bench is a factor of overburden depth and the swell factor of the broken ground. With a swell factor of 1,3 and an overburden depth of 30 m, an extended bench equal to 4,18 m in width at the surface is built.

There are two reasons for not drilling and blasting a buffer in areas where the overburden is more than 30 m deep.

(1) It has been found that there is less loose rock in the highwall, which is safer for the personnel working on top of the coal.

(2) When the bench is blasted into the pit, less material has to be handled by the dragline.

Removal of Coal

The mining of No. 4 Seam in the Kriel Coalfield has shown no deviation from any strip-mining operation except for the removal of top-of-coal contamination or the leaving behind of bottom-of-coal contamination. Both conditions require accurate drilling according to predetermined information. The coal seam is contaminated primarily by the existence of bands of carbonaceous mudstones, which are intermixed with thin bands of coal.

The contamination of the top of the seam is primarily a function of the overburden stripping. Once the dragline has removed the contamination to a predetermined thickness, a visual inspection is made of the top of the coal seam. Any further removal of mudstone is done by tracked dozers.

Contamination of the bottom of the seam is controlled

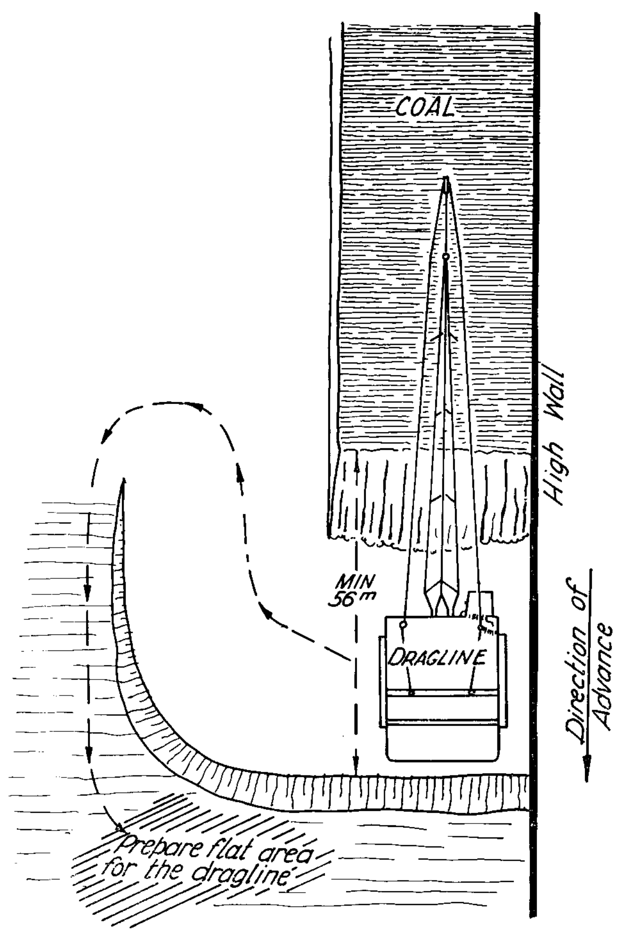

Fig. 7—Preparation of digging platform

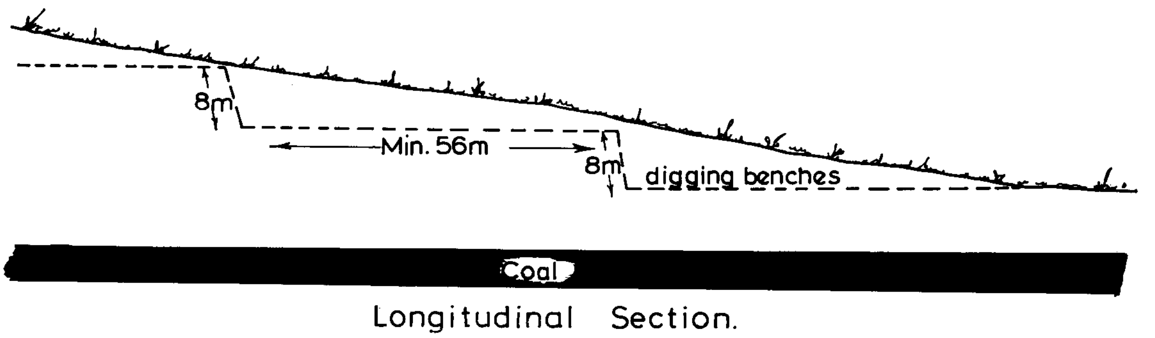

Fig. 8—Steps to raise the dragline to 30 m

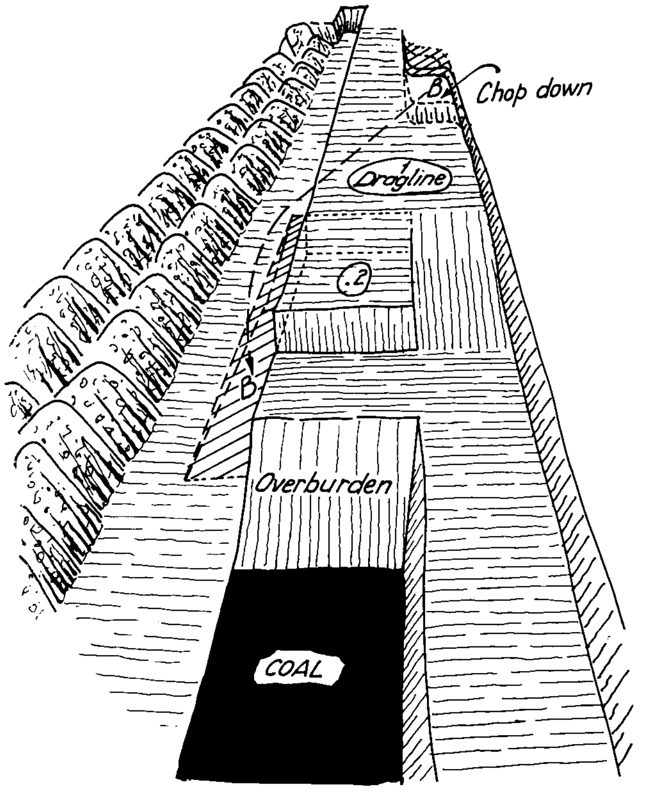

Fig. 9—Extension of the bench with chop-down material

by drilling to an elevation shown on the geological sections. All the contamination is undrilled, unblasted, and hence left *in situ*.

For the coal drilling, two Ingersoll-Rand Damco 3000 machines are used to drill holes 108 mm in diameter by means of four-bladed drag bits. The drilling patterns are 3 m by 3,5 m.

Sinex F600 (5 per cent aluminium) slurry is the blasting agent and it is detonated by one 160 g Pentolite booster, which is initiated by ordinary Cordtex. Blasting ratios of 0,2 kg/t are being obtained.

Two P & H 1900 AL/CL electric rope shovels, one in each pit, with 12 m^3 buckets are used to load 91-ton bottom-dump trucks. The loading times are 2 minutes, and the round-trip cycle time for the truck is 21 minutes. Continuous face clean up is carried out using a rubber-tyred dozer.

Rehabilitation

The rehiabilitation work has concerned three areas: borrow pits, stream-diversion levees, and strip-mining spoils.

Borrow Pits

Borrow pits were excavated in order to obtain Ferricrete (Ouklip) for the construction of the roads. A total of 99,2 ha was disturbed, of which 60 ha has been totally reclaimed and planted with grass, an additional 27 ha has had the topsoil replaced by dozers, and a further 12 ha has still to be reclaimed.

Of the areas that have been planted to grass, a fertilizer treatment of 3:2:0(20) at 650 kg/ha has been used. Of these areas, 30 ha has been planted with *Erragrostis Curvula*, 20 ha with *Erragrostis Tef*, and 10 ha with a combination of the two.

Despite the drought, the results have been excellent and there are only a few bare batches. These may be

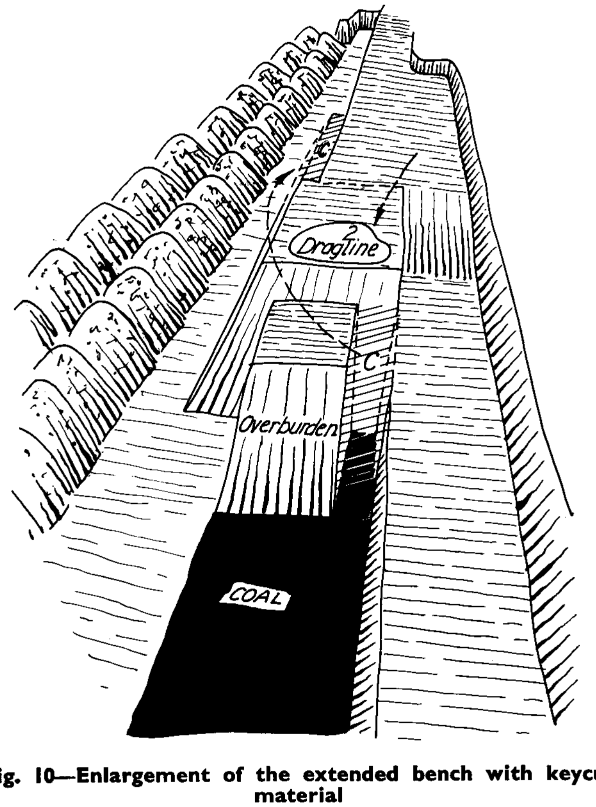

Fig. 10—Enlargement of the extended bench with keycut material

due to shallow topsoil, or to a low pH in conjunction with aluminium toxicity. The counter measures being taken include liming of the area and ripping of the surface to break up any hard pan that has formed.

Stream Diversion

The diversion of the Rietspruit from the mining areas is 6,8 km long. The levees formed in the construction have required the revegetating of 61 ha; of this area, 50 per cent has been successfully put to grass. The grass has failed in the remaining area because of three interrelated problems: poor rainfall in the 1978/79 season, poor-quality soils that are low in nutritional values, and steep slopes. The methods being used to overcome these problems include the planting of grass seed in niches, followed by intermittent irrigation.

Strip-Mining Spoils

At the full production rate of $4,5 \times 10^6$ tons of coal per annum, 61,7 ha of land surface will be disturbed and is going to have to be rehabilitated. To date, an area of some 18,7 ha has been disturbed, and 5,1 ha has been rehabilitated to the stage where the spoil has been completely removed or flattened and the area fertilized and seeded. The 4,5 m^3 utility dragline and two D9 dozers are used to remove the spoils.

Conclusion

The decision to change the scope of operations at Kriel from a completely underground mine to separate underground and opencast operations has resulted in an increase of at least 14 per cent in the volumetric extraction of coal in the Kriel I Coalfield.

Acknowledgement

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Contribution to the above paper

by M. F. Pleming* Pr. Eng. (Fellow)

Optimum Collieries, the 'oldest' large-scale opencast coal mine in the Republic, has made significant progress in its restoration programme, and purchased a hydraulic excavator in October 1978 to improve the efficiency of partings removal. In common with other opencast mines, Optimum has been hard hit by the oil crisis and is developing solutions to the problem.

Restoration

Optimum embarked on a programme of levelling spoil and planting pinetrees on mined-out land as early as 1971. Activity during the first few years (until early 1977) was mainly experimental, and small areas were planted with various species of pinetrees. Growth was stunted, and, after the drought of 1976/77, it was concluded that these trees would not prove successful in the long term. A policy of grassing was therefore adopted instead.

The Marion dragline exposes an area of between 3 and 6 ha per month. By March 1977 some 10 ha had been recontoured, and pines had been planted on 6 ha of this recontoured land. A total of 190 ha remained at that time to be contoured. With the purchase of 3 additional bulldozers (total 4) in March 1977, the job of recontouring gained momentum, and by February 1979 a total of 299 ha of opencast land had been recontoured. Of this area, some 200 ha are under grass.

We have now experienced two rainy seasons on the recontoured land. Drainage has been good, but in the season 1978/79 the rainfall was very low and, as a consequence, the good progress achieved earlier in the project was not maintained.

Use of a Hydraulic Excavator for Partings Removal

Three coal seams are mined at Optimum. These seams are divided by a shale or sandstone parting, which varies in thickness between 0,1 and 0,7 m. Right from the start of mining the removal of the partings has proved problematic.

Where the parting is thin (i.e. no more than 0,3 m thick), it is generally possible to rip the partings with a D9 bulldozer blade. Partings thicker than this were previously drilled and blasted.

In 1977 some investigational work was done on the feasibility of using a hydraulic excavator for the removal of partings. Two machines were taken on trial, and both showed that the use of the hydraulic excavator principle is valid. However, the quantities removed by these small machines were inadequate. As a result, it was defined that a machine having a break-out force of approximately 50 t would be required, and a machine in this range, the O and K RH 75 with a 7,5 m^3 bucket, was defined as acceptable. Such a machine was purchased and arrived on site in October 1978. After some minor teething problems, the machine proved itself capable of moving partings at a rate of more than 200 m^3/h, and is proving more economic than the previously used method of drilling and blasting. The labour force on this operation has been reduced by 10, and it is no longer necessary to use drilling equipment together with the Michigan 475B front-end loader.

Conservation of Diesel Fuel

The success of opencast mining is due in no small measure to the flexibility provided by the diesel-powered equipment. On some operations the whole power-motive

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force depends on the diesel engine. With the advent of the large dragline, this has been somewhat limited, and today a lot of the hard work of moving overburden, and in some instances loading coal, is achieved by electric power. There are particular historical reasons why diesel power should be used. In the main, although electrical equipment is more costly in capital terms, it offers the advantages of low working cost. The movement of electrical equipment has invariably been somewhat slow compared with its more flexible counterpart, the diesel-powered machine. However, the diesel machine, although often less in initial cost, has had the disadvantage of higher operating cost. With the most recent energy crisis the whole situation has changed radically. We are now faced not only with the very high cost of diesel fuel but also with the possibility that diesel fuel will not be available in the quantities required in the years to come.

At Optimum Collieries diesel is used for the loading and drilling of coal, the loading and drilling (where necessary) of partings, the transportation of coal, and to a large extent the restoration of mined land. Until very recently the cost of diesel fuel for transporting coal was subsidized, but on 1st May, 1979, this subsidy was withdrawn. The total dieselene consumed at Optimum before April this year was more than 500 000 litres per month, which represents approximately 1,3 litres of diesel fuel per ton of coal mined.

Since early in 1979 the whole question of using diesel in opencast operations at Optimum has been under review. Some economies have been realized, but in the main these have had no great effect on the total problem. In particular, maximum use is now made of the small 150 RB dragline and 150 RB shovel. Whenever possible, use is not made of the diesel-powered coal-loading equipment. The diesel-powered hydraulic excavator (O and K RH75) is at present being converted for electric operation, and investigations are in progress to convert the coal-drilling operation to electric power. The two areas in which the problem remains to be solved are the haulage of coal and restoration.

In the short term (the next four years), the whole mining operation is to be moved closer to the plant area. This will radically reduce all distances for the Dart haul fleet and, although final figures are not yet available, it is anticipated that the consumption of dieselene on the coal haul will be cut by 40 per cent. At present, there is an element of catching up on a back log in restoration. Because of our particular responsibility in restoration, there has been no effort so far to reduce progress in this area. Once the pace of restoration matches that of dragline exposure, the diesel consumption in this area will fall by approximately 50 per cent.

In the longer term, investigations are directed into the utilization of inpit conveyors and crushing equipment. Planning exercises are at present in progress to lay out the pit for the maximum possible length of cut. These cuts would be as long as 4 or even 5 km. A wide conveyor belt would be installed in each cut following the progress of coal exposure. The coal-loading method would probably be by hydraulic excavator or electric shovel loading into a fairly small hopper. The hopper would discharge into a McNally- or Bradford-type breaker, and any waste would be sorted automatically at the loading point. The conveyor would be equipped with an extensible take-up, and additional conveyor belting would be added to the extensible conveyor, say once every 4 to 5 shifts. Coal conveyed out of the cut would be discharged onto a transverse conveyor, which in turn would lead the coal to the plant areas. The problem in restoration appears to be more serious. There is seemingly no way in which a bulldozer can be electrified. Nevertheless, efforts are at present being made to use a small dragline to 'drag down' the spoils and so minimize the amount of work to be undertaken by a bulldozer.

These are just some of the problems and some of the solutions that have been experienced at Optimum at present. We have plans for expanding production from the opencast in the short and medium term. It is vital that our efforts in conserving fuel, not only at Optimum, but at other opencast mines in South Africa, should be speeded up unrelentingly. Unless this problem can be solved, and solved quickly, the advantages that prompted the expansion of opencast operations in this country will be seriously questioned.