# Strip mining at Arnot Colliery

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**SYNOPSIS**

Operations of the strip mine at Arnot Colliery are described, and the improved volumetric extraction of the *in situ* coal reserves that has been obtained by the application of strip-mining techniques is highlighted.

An account is given of the major problems encountered during the early stages of the operation with particular reference to overburden removal, coal and parting handling, and transportation.

The techniques used in dealing with the problems and the current degree of success in overcoming them are outlined.

**SAMEVATTING**

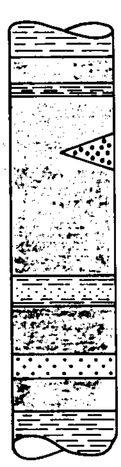
Die werksaamhede van die strookmyn by die Arnot-steenkoolmyn word beskryf en die beter volumetriese ekstraksie van die *in situ*-steenkoolreserwes wat met die toepassing van strookmynboutegnieke verkry is, word benadruk.

Daar word geslag gedoen oor die belangrikste probleme wat tydens die vroeë stadiums van die werksaamhede teëgekom is met spesiale verwysing na die verwydering van die bolaag, die hantering van die steenkool en skeilaag en die vervoer.

Die tegnieke wat gebruik is om die probleme te hanteer en die huidige mate van sukses met die oorkoming daarvan word in hooftrekke behandel.

## Introduction

The surface strip mine at Arnot has been producing coal since August 1975, although the mine has been supplying coal from underground workings to the Arnot power station since May 1971. The present Arnot coalfield extends over an area of 23 000 ha, with current reserves of 900 million *in situ* tons situated at depths between 5 and 75 m from the surface. The coalfield forms part of the eastern section of the Witbank Field, and four seams are present (Fig. 1).



| | AVERAGE THICKNESS m | AVERAGE RD | AVERAGE CV MJ/kg |
|---|---|---|---|
| SHALE | | | |
| No.2A SEAM | 0,70 | 1,65 | 18,30 |
| SHALE | 0,27 | | |
| | 0,95 | 1,50 | 20,85 |
| FLOATING STONE | 0,90 | | |
| No.2 SEAM | 3,50 | | |
| | 1,65 | 1,50 | 24,35 |
| SHALE & SANDSTONE | 0,70 | | |
| No.1 SEAM | 1,18 | 1,45 | 26,20 |
| GRIT | 0 61 | | |
| No.1 LOWER SEAM | 0,83 | 1,50 | 24,70 |
| SHALE | | | |

**Fig. 1—Stratigraphy of the coal zone at Arnot Colliery (RD = relative density, CV = calorific value)**

The middle seams (the No. 2 and No. 1 seams, with average thicknesses of 3,5 m and 1,2 m respectively) are those of the most significance and exist throughout the field. In addition to these seams, there are areas where two other seams are available for mining, the upper No. 2A seam and the No. 1 lower seam.

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Analyses of material from prospecting boreholes have indicated that in some areas the major seam, the No. 2 seam, is split by a sandstone parting into an upper and lower section. In some areas, therefore, the strip-mining operation will be handling five separate coal horizons.

When the mine started operating, all the coal was produced from two independent underground areas mining only the No. 2 seam and using conventional trackless mechanized units. During 1972 investigations were carried out into the feasibility of opencast mining, and in 1973, with the approval and support of Escom, the decision was taken to establish a major strip mine on the western side of the coalfield. The decision to proceed with this method of working was influenced by the following:

— over 30 per cent of the reserves occur at depths of less than 30 m from surface,
— the volumetric extraction rate of No. 2 seam was low by underground mining methods,
— it was not possible to exploit all four seams in the underground mine,
— labour costs were becoming increasingly high,
— geological conditions in the underground mine are poor.

One underground shaft remains in operation to mine the No. 2 seam. The surface mine extracts the coal seams to a cut-off depth of 30 m at an average stripping ratio of 3,2 bank cubic metres (b.c.m.) to 1 sales ton.

The coal-tipping point of the surface mine is near the mine surface buildings and screening plant, which are situated approximately in the centre of the coalfield in an area of thin coal. Strip mining started, and is still continuing, at one extremity of the field, and this necessitates a maximum haulage distance of 14 km to the tipping point. As the surface mine continues, this haulage distance will reduce progressively. To date the coal production has been from the No. 2 and No. 1 seams, only one parting being handled, but mining is now approaching areas where No. 2A seam and the split No. 2 seam are present.

### Handling of Overburden

The overburden typically consists of some 50 per cent soft soils and sub-soils and 50 per cent hard competent rocks. The stratigraphy mainly comprises sandstone, siltstone, and shales of the Middle Ecca Coal Measures overlying rocks of the Dwyka group (Fig. 2). The shale is the predominant rock, and has a compressive strength of up to 236 mPa.

A fundamental prerequisite to acceptable dragline production is good fragmentation of the digging material, and in this regard three main problem horizons are apparent. The first is associated with an irregular hard sandstone band, varying in thickness from 1 to 7 m, which is located high in the bank, even outcropping at the surface in some areas. The other two problems are associated with hard layers of shale. In some areas, unbroken shale up to 3 m in thickness remained above the upper seam after the blast, and in other areas, especially where the shale is thick, an unbroken band of hard shale was encountered high up in the highwall face.

One important consideration in dealing with blasting techniques and problems in strip mines is that, when highwall buffer blasting is practised (as at Arnot), the dragline fully excavates a blast only some four months after it is initiated. Clearly, the ultimate proof of a successful blast can be ascertained only when the dragline has completely excavated the area, and the considerable delay between cause and effect makes innovation and trouble-shooting in drilling and blasting onerous and time consuming.

Buffer blasting is used to stabilize the highwall, and refers to the process of having a constant buffer of broken ground parallel to the highwall between the highwall and the blast pattern. The buffer is created when the original boxcut is drilled and blasted, and is carried through in each successive cut.

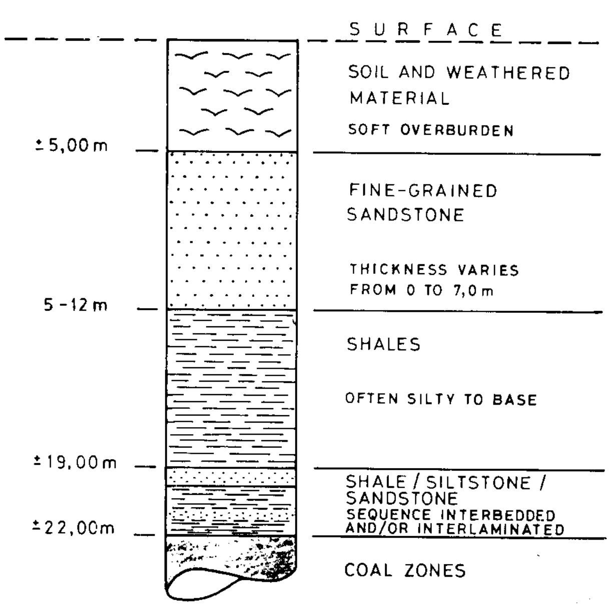


**Fig. 2—Stratigraphy of the overburden at Arnot**

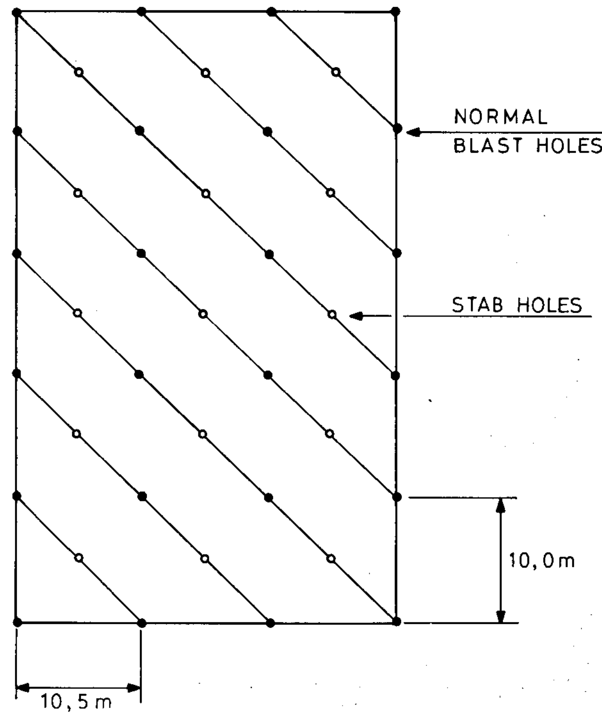


**Fig. 3—The stab-hole grid**

The three difficult zones are discussed in more detail below.

*Hard Toe above the Upper Coal Seam*

This results primarily from lack of control of hole depths, and a series of operational checks was developed to ensure *inter alia* that the blast holes are loaded with explosives to the correct depths. The following information is recorded to the nearest 0,25 m, and operating personnel have specifications of variance tolerances:

(a) the driller's hole depth, showing depths of 'hard' and 'soft' material,
(b) the loaded depth,
(c) the depth of water in each hole,
(d) the collar prior to the addition of tamping material,
(e) the collar after the addition of tamping material,
(f) the collar just before blasting,
(g) the dates the holes were drilled, loaded, and fired,
(h) the location of any potential problem areas, hard toe, etc., which, as well as hole loads and tie in, is recorded on the original blast plan,
(i) the priming practice, the design mass of the explosives charge, the number of Cordtex downlines, the trunklines.

*Upper Portions of Unbroken Shale*

The primary drilling equipment are two Bucyrus Erie 61R rigs, each with a maximum pull-down force of 500 kN. These units were originally equipped to drill a 381 mm-diameter hole using 273 mm-diameter drill pipes.

Good results were obtained until deeper areas were encountered where hard digging was found towards the top of the shale above layers of well-blasted material. This problem was identified, and it was considered that, while the total mass of explosives was sufficient for good fragmentation, an improvement could be obtained by an increase of the column height of the explosive charge. To this end, trials were carried out with 311 mm-diameter holes, which give 50 per cent increase in column height for slurry explosives. These experiments were successful, and both drill rigs are now equipped with 219 mm-diameter rods to drill 311 mm-diameter holes.

*Sandstone Bands*

It was found that the sandstone layers were not well fragmented when the existing drilling pattern was used, even with an increase in the explosive charge, because of its elevation in relation to the charge. Stab holes with the explosive charge located within the sandstone band were found essential for these areas, and a complete intergrid pattern is required to provide the necessary cratering effect (Fig. 3).

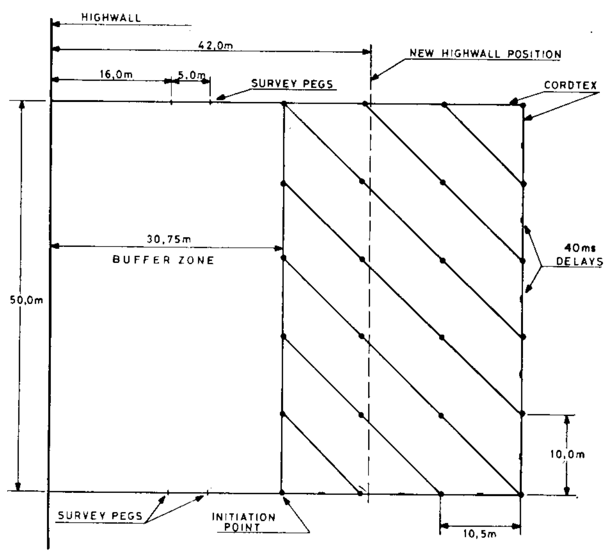


**Fig. 4—The drilling pattern and initiation for blasting of the overburden with slurry**

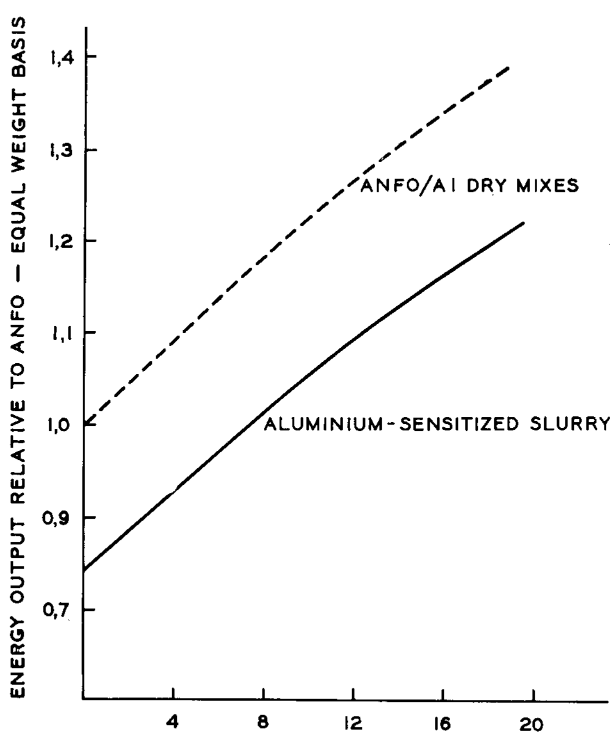


**Fig. 5—Mass strengths for dry mixes of Anfo and aluminium and for aluminium sensitized slurries containing different percentages of aluminium**

## Drill Patterns and Initiation

The original cut width was 35 m, and several drill patterns, all giving about 80 to 90 m$^2$ of cover per hole were used. The cut width was then widened to 42 m mainly to improve the loading and hauling conditions, and to optimize the overall dragline operation. At this width, the drilling pattern has been steadily developed and increased to the current 10 m by 10,5 m grid.

A single Pentolite booster with a double Premium Cordtex downline is used, and the holes are initiated, using 40 ms delays, at 45° to the buffer zone (Fig. 4).

## Blasting Agents

Owing to the presence of widespread ground water, a contract was signed with AECI in October 1975 according to which the supplier would give a 'down the hole service' to the mine. This service was all-embracing in terms of the supply, manufacture, and delivery of bulk explosive into the blast hole, and this explosive could be either waterproof slurry (Iregel 600 with varying percentages of aluminium) or Anfo. The advantage of this arrangement was that the mine had immediate access, with no development or capital cost, to bulk delivery of slurry for the charging of wet holes and of the cheaper Anfo, which could be used where conditions proved suitable.

This system has worked well, and there have been no major administrative or operational breakdowns. Until very recently, the vast majority of primary blast holes were very wet, containing up to 10 m of water, and slurry with 5 per cent aluminium was used exclusively.

*Slurry*

The slurry will not detonate when its critical density, which depends on the percentage of contained alu-

TABLE I

CRITICAL DENSITIES AND COLUMN HEIGHTS FOR VARIOUS ALUMINIUM CONTENTS

| Aluminium % | Critical Density | Maximum column height, m | |
|---|---|---|---|
| | | Slurry + water | Slurry + stemming |
| 0 | 1,15 | 5 | 4 |
| 3 | 1,21 | 10 | 9 |
| 5 | 1,24 | 20 | 15 |
| 7 | 1,27 | 25 | 20 |
| 10 | 1,30 | 30 | 25 |

TABLE II

VARIATIONS IN ALUMINIUM CONTENT

| Depth of hole m | Aluminium % | 1978 cost c/kg |
|---|---|---|
| 0–16 | 3 | 27,80 |
| 16–22 | 5 | 30,24 |
| 22–25 | 7 | 32,72 |
| 25–30 | 10 | 36,36 |
| | Anfo | 23,44 |

minium, exceeds a certain value. This first became a problem in the middle of 1977 as deeper areas of the pit were mined. Table I lists the critical densities for slurry containing various percentages of aluminium, and the total column heights permissible before the stated critical density is reached.

To overcome this problem, the following control measures were introduced, but they resulted in an overall increase in cost in that higher percentages of aluminium were being used as deeper areas were being worked.

(a) The overall column height was controlled to within predetermined limits.

(b) The percentage aluminium was varied as shown in Table II.

Additional quality controls were also introduced by AECI, and the mine started carrying out destructive tests on each batch of slurry. To date there has been no rejection of the slurry mix.

*Anfo*

Although, as mentioned, complete facilities for the use of Anfo existed as part of the contract, trials with Anfo as a blasting agent did not get under way until early in 1978 because of the severe water problems. The primary reason for considering the use of Anfo was its economic advantage in allowing better digging for less money.

A hydraulic blasthole-dewatering pump mounted on a $1\frac{1}{2}$-ton truck was hired for the trials, and this unit is able to pump 10 m of water from a 20 m hole in under 5 min. In the trials, 800-gauge plastic liners are used, a handmade double knot being sufficient to water seal the end. This sleeve is lowered into the hole, using drill chippings as a weight. Anfo is poured into the plastic liner direct from the 6-ton Anfo truck, and there are few operational problems, such as tearing of the plastic associated with the loading.

The results to date are extremely encouraging, and indicate that Anfo has a distinct economic advantage over slurry; for instance, a 10 per cent aluminium slurry has only 7 per cent more energy than Anfo, although it costs 55 per cent more (Fig. 5). In addition, the lower bulk strength of Anfo gives a 21 per cent increase in column height compared with slurry, with the resultant advantage in deep areas; and there is no critical density problem with Anfo.

## Dragline Operation

There are four major phases of the dragline operation (Fig. 6).

### Key Cut

This is the cut that forms the new highwall, and it is taken down uniformly to the top of the coal seam. The width of the bottom of this first cut is fixed at $7\frac{1}{2}$ m, but the width at the top varies with the depth.

### Strip Cut

Once the key cut has been excavated and the new highwall exposed, the remainder of the cut is moved. This portion is called the strip or main cut. Productivity from this cut is higher than that for the key cut owing

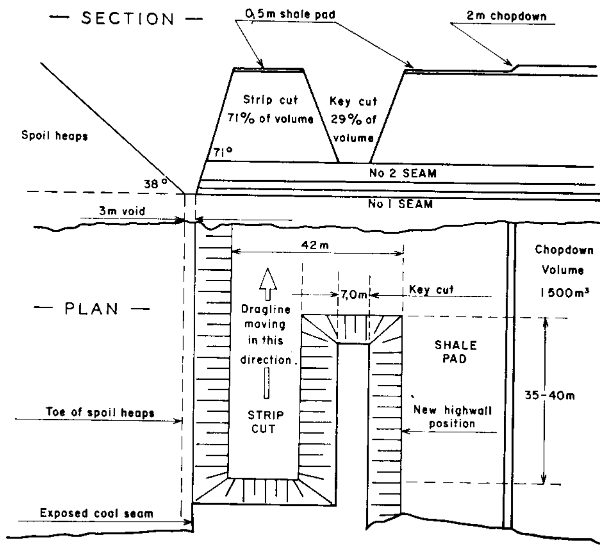


**Fig. 6—Sequences of the dragline operation**

to the shorter dumping swing required, and the fact that, during the digging of all but the surface portions of the key cut, the operator must hoist *prior* to swinging.

### Chopdown and Pad Building

Chopdown is the sequence in which the dragline, working 180° from the dumping position, 'chops in' on virgin unblasted ground, thus lowering the ground profile. A 2 m chopdown is taken, and subsequently a $\frac{1}{2}$ m shale pad is placed on the area where the chopdown has been taken. At present, chopdown and pad building take some 5,2 per cent and 1,5 per cent respectively of the total dragline operating time.

These two processes have the following advantages and disadvantages.

*Advantages*

(1) The chopdown soils are placed between the overburden spoil heaps, and are then recovered to form a cover over the flattened spoils, giving the rehabilitation effort a good start.

(2) As the original ground profile is reduced, the effective digging depth of the dragline is increased. This technique is essential in areas near the critical depth.

(3) A shale pad placed by the dragline and levelled and compacted by the dragline dozer provides a uniform, solid, level base on which the dragline, drills, and other associated equipment can operate.

*Disadvantages*

(a) The chopdown sequence is on a long-swing, slow-bucket loading cycle, and is the least efficient portion of the dragline operation.

(b) Pad building clearly involves rehandling of materials.

### Void

One of the key objectives of the dragline operation is to create and maintain a 3 m gap, or void, between the toe of the spoils and the edge of the exposed coal seams. The void ensures that contamination is kept to a minimum, it provides a very convenient sump for pumping operations, and it creates a space into which a section of the parting between the seams can be sidecast. This concept of parting handling has resulted in a considerable cost saving and is discussed in detail later.

### Dragline Output

The orignal planned production and the actual production in 1977 are compared in Table III.

### Bucket Overloading

During the very early stages of the dragline operation, the centre pin showed signs of heat build-up and the tub hooks were scraping on the rear of the machine. The first site diagnosis was that the ballast weights were insufficient and the dragline was therefore not balanced.

As a result of a detailed investigation carried out by Bucyrus Erie engineers, it was found that the relative density of the upper soil and sub-soil layers, particularly when moist, was far higher than the figures that had been used in the design parameters. This factor caused occasional serious overloading to take place when the operator loaded the bucket to capacity while digging in clayey materials.

Once the problem had been identified, it was clear that there was a need to monitor the bucket loads. To this end, based on the principle that motor torque is proportional to armature currents, meters were installed to monitor the hoist and drag loads. The system uses millivoltmeters to measure the armature currents of the hoist and drag circuits via existing shunts. The maximum limits are clearly marked on the meters, which are conveniently mounted in front of the operator. He uses the meters to assist him in controlling the loading while in the upper portion of the cut.

## Coal Loading and Parting Removal

### Coal Cleaning

Strict control is exercised on the overburden blasthole depths, and this results in a good break between the top of the No. 2 seam and the overburden such that the 1570 W dragline can leave a clean surface. Final cleaning of the top of the upper seam is carried out routinely by a wheeled dozer, which pushes the material towards the dragline digging pile.

The pit reserve figure is not a critical aspect and, although the present pit length allows a total of 750 000 t of coal to be exposed before the dragline becomes coal bound, the coal-exposed objective is currently set at 500 000 t. The 'exposed' value includes the lower seam, which technically is not fully exposed.

TABLE III

DRAGLINE OUTPUTS

| | Planned (1975) | Actual (1977) |
|---|---|---|
| Mechanical availability, % | 84 | 78,5 |
| Digging time, % | 80 | 72,9 |
| No. of swings per hour | 53 | 49,2 |
| Bucket factor, b.c.m. | 45,3 | 46,3 |
| Virgin b.c.m. per hour | 2400 | 2280 |
| Overall production (million b.c.m.) | 13,477* | 13,113† |

*Originally planned on a cycle of 6 days of 24 h per week.

†In 1977 a 7-day week was worked for 10 months. An additional relevant factor was the necessity to train a new crew of 5 operators during the middle of the year since the original contract operators were repatriated.

### Coal Removal

As already stated, the coalfield contains up to 5 seams, but only the 2 major seams are being mined. The No. 2 seam consists of 3,5 m of hard, lustrous coal, and is separated from the softer, better quality No. 1 seam by a hard (164 MPa) dark grey, silty to carbonaceous shale containing finely interlaminated sandstone. All three zones require blasting (some 30 per cent of the No. 1 seam can be ripped by a large dozer), and the pit is equipped with a medium-sized rotary drill and small twin-boom percussive unit.

*No. 2 Seam*

This seam is drilled with an Ingersoll Rand Damco 3000 rotary-drill unit with a rod length of 8,5 m and utilizing drag bits of 82 mm diameter. Because most of the holes are wet, Dynagel is currently being used, although the use of Anfo in sleeves is being investigated. As all the coal is pulverized prior to being burnt in the power station, there is no sizing problem to be considered, and the pattern used is 3 m by 3 m with a powder factor of 0,13 kg/t.

The coal is loaded with Bucyrus Erie 155 face shovels on a full-face method, with the coal hauler reversing into the loading position next to the shovel. There is insufficient room for a double back-up loading system, but, as there are only six haulers, queuing rarely occurs, and the time taken to reverse into the loading position is not excessive — some 60 seconds.

*The Parting and No. 1 Seam*

As soon as the upper seam has been removed, the whole width of the parting is drilled and blasted. This operation is maintained as close as possible to the No. 2 seam loading. Blasted parting, once it has been levelled by a dozer, broken, compacted with a grid roller, and finally graded, provides an excellent road, superior to the natural undulating floor of the parting.

The parting, which has an average depth of 0,7 m, is drilled with an Atlas Copco ROC 601 twin-boom rig with a rod length of 3,3 m and button bits of 75 mm diameter. Holes are drilled at 70° to the horizontal, and are charged with Dynagel. A powder factor of 0,33 kg/b.c.m. is obtained.

### Parting Handling

The original concept of parting handling was that it would be loaded out with a small dragline backed by a loader/truck operation. To this end, a 5 yd dragline, two medium-sized loaders, and three 35-ton dump trucks were bought as capital equipment. The system envisaged was that the small pit dragline would spoil some 30 to 50 per cent of the blasted parting against the major spoil piles created by the 1570 W dragline, and the remainder of the parting would be hauled out of the pit by the dump trucks.

From this drawing-board scheme, the operating staff developed parting handling through several stages to the current method, which involves the following (Fig. 7).

(1) The pit is divided lengthways into two sections, the

one nearest the spoils being 14 m wide. This section is measured, and clearly marked with red plastic roadside cones.

(2) The blasted parting from the 14 m section is side-cast by the face shovel into the 3 m wide primary void between the toe of the spoils and the edge of the exposed parting.

(3) The No. 1 seam thus exposed in the 14 m section is ripped (or drilled and blasted) and loaded with the same face shovel. During this operation, the coal haulers travel on the prepared blasted parting of the remaining 28 m section.

(4) The parting from the 28 m strip is now side-cast by the same shovel into the 14 m strip from which the No. 1 seam has been removed.

(5) On completion of this step, the newly exposed No. 1 seam can be removed.

If a void has not been established by the 1570 W dragline, this method is modified by spoiling the 14 m spoil section towards the highwall side. This obviously creates double handling and emphasizes the necessity for the void.

This method of parting handling has several advantages.

(a) It is by far the most economical. Electric shovels, although very expensive capital items, are extremely reliable and, with good maintenance and use, return low operating costs per hour for high production outputs.

(b) The 5 yd dragline was no longer required for pit work and became available for use on surface rehabilitation work.

(c) A combination of loader and dump truck is now required for ancillary work only, with the result that one loader and one dump truck became redundant and will not be replaced.

## Back-up Drill Rig

For some time, the in-pit drilling operation was a critical aspect since the drilling rigs could not act as a back-up to each other. The rotary drill is unsuitable for parting drilling because of slow penetration speeds and excessive bit consumption; it is unsuitable for the drilling of No. 1 seam because of the large number of very short holes required. The percussive drill is not efficient in the deeper No. 2 seam because of rod-handling problems.

To overcome this problem, a major drilling-equipment company was approached to design and manufacture a dual-purpose rig. This self-contained unit has separate rotary and percussive drilling systems, and it takes 30 minutes to convert from one method to the other. The unit has a single boom but, if it proves successful, it will be uprated to a twin-boom design.

## Coal Handling

Studies carried out in 1972 confirmed that coal haulers would be better for Arnot Colliery than a conveyor-belt system. (Changing circumstances have indicated that the situation requires re-examination, and a detailed study of various options is now being made.)

Six Unit Rig B.D. 180-ton coal haulers were purchased at a price of approximately R600 000 each. The body of the unit was manufactured locally at Middelburg, but all the other components were imported. The specification of the unit is as follows:

| | |
|---|---|
| Installed Power | 900 kW General Motors 149 Series diesel engine |
| | 2 × 450 kW General Electric wheel motors |
| Mass | 144 tons |
| Payload | 178 tons |
| Minimum turning radius | 23,4 m |
| Tyres | 6 size 3000 × 51 |
| | 4 size 3300 × 51 |
| Maximum speed | 52 km/h. |

Mechanical and structural problems associated with

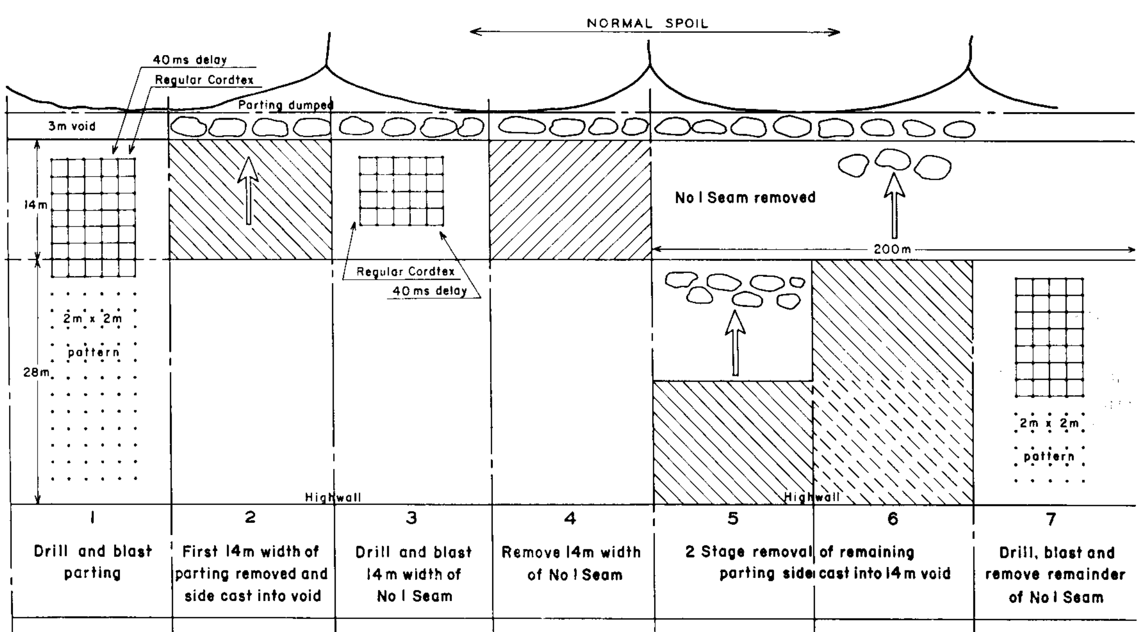


**Fig. 7—Sequence for parting and No. I seam removal**

TABLE IV

TYRE RATINGS (IN TONS PER KILOMETRE PER HOUR)

| Wheels | U.S. and Japanese manufacturers' specifications | Calculated requirement |
|---|---|---|
| Front | 555/645 | 522 |
| Drive | 555/645 | 651 |
| Trailer | 599/645 | 663 |

TABLE V

CONSUMPTION OF TYRES

| Quarter | 1976 | 1977 | 1978 |
|---|---|---|---|
| 1st | 12 | 33 | 5 |
| 2nd | 19 | 14 | |
| 3rd | 47 | 7 | |
| 4th | 19 | 20 | |
| | 97 | 74 | 5 |

the coal haulers centred around the 'goose neck' and torque tube. These failures resulted from truck overloading (it is possible to overload the trucks by some 30 per cent), poor in-pit conditions, undulations in the haulage road, and possible under-design for Arnot conditions.

The first three problems were eliminated through improved standards, tighter controls, and increased experience of the opencast crew, and the fourth was overcome by extensive structural reinforcement to the trucks, which the suppliers undertook with considerable urgency.

Other problems concerned the tyres, and involved heat build-up, ply separation, and impact fractures. These resulted in very high tyre usage, the average life being less than 2000 hours. The high tyre failures arose from the same problems as the mechanical and structural failures, and were all quickly eliminated when those problems were solved, except for the tyre-design problem, which has not yet been completely resolved.

The following improvements have been made since the start of operations at Arnot.

(1) In-pit conditions have been improved considerably by greater use of the grading and compaction equipment.

(2) The strip width was increased from 35 to 42 m, which allows easier in-pit maneouvres.

(3) Impact fractures have been reduced to a minimum by an increase in the grading capacity and insistence that *all* personnel in the opencast mine are responsible for removing rocks from the haul road and in-pit roads.

(4) The trailer tyres carry the highest load, and modifications were carried out on the trailer hubs so that larger tyres (3300 compared with 3000) with a higher rating (tons per kilometre per hour) could be fitted.

(5) The steering tyres of the horse have the lowest load and yet register the quickest rubber wear. All new tyres other than the larger 3300 tyres for the trailer wheels are fitted to the steering wheels and remain there for some 1500 hours, at which stage some 75 per cent of the rubber remains. These tyres are then redistributed to the drive wheels. In this manner, problems arising from heat build-up due to flexing of the rubber are minimized.

(6) The top speed of the haulers, which is rated at 52 km/h, has been governed to 48 km/h.

The tyre problem, while being alleviated to a degree, has not been overcome, and trials are continuing with several makes of tyres of varying ply ratings and amounts of rubber. Table IV indicates the required rating of tyres for the duty at Arnot compared with the specifications to which the tyres are made.

The obvious way to alleviate tyre problems is to use tyres that have a specification close to, or higher than, the calculated requirement. If this is not done, tyre failures occur as a result of heat build-up and ply separation.

It is of interest to note that the small increase in the required duty of the trailer wheels has resulted in a consumption twice as high as for the drive wheels when tyres of the lower specification are used.

Table V indicates the tyre usage per quarter since the start of the operations. (The total number of tyres in use is $6 \times 10 = 60$.)

## Conclusion

The major problems encountered with the strip mine at Arnot Colliery have been overcome, and strip mining, although a capital-intensive operation, has been shown to offer lower unit costs with significantly increased extraction of the coal reserves. As a consequence, feasibility studies are at present being carried out to increase the mining depth from the present limit of 30 m to 50 m.

A code of practice for surface rehabilitation has been drawn up and, within the framework and the spirit of this code, Arnot has developed restoration standards, the overall aim of the programme being to return areas disturbed by surface-mining activities to agricultural use within a reasonable period of time. The end result will be that, once mining has finished in the area, the restored land can be returned to the farming sector.

## Acknowledgement

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