A Mobile In-Pit Crushing System for Overburden at Iscor's Grootegeluk Coal Mine

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Summary

In this paper the mobile in-pit crushing system for overburden, which was commissioned in July 1981, at Iscor's Grootegeluk Coal Mine, is described. Special reference is made to some engineering aspects of the system components, operating practice and an economic appraisal of the system.

1. Introduction

Grootegeluk Coal Mine, near Ellisras in the Northwestern Transvaal is located on the coal bearing strata of the Waterberg coal field and was officially opened on April 15, 1981 to produce blend coking coal for Iscor's blast furnaces at Vanderbijlkpark and Newcastle.

2. Geology

Numerous coal seams occur in the Waterberg coal field over a stratigraphic thickness of 129 m (Fig. 1). The lower 31 m is predominantly arenaceous with three well-developed seams (zones) of dull coal, and is considered equivalent to the Middle Ecca. The three zones, numbered 1 to 3 from bottom upwards, have average thickness of 1.35 m, 3.37 m and 7.82 m, respectively.

Zone 1 contains hardly any bright coal, but the bottom 1—2 m of zones 2 and 3 contain some bright coal and yield a low ash (5—6 %) fraction (RD 1.40) that could be suitable for form coke. Overlying the Middle Ecca is a 37 m layer shale, carbonaceous in the upper part, with two zones of dull coal: Zone 4A, 1.52 m thick and zone 4 on average 4.02 m thick. This predominantly shale succession is considered a transition between the underlying Middle Ecca and overlying 61 m of alternating layers of bright coal and shale, subdivided into seven zones (zones 5—11) and correlated with the Upper Ecca. The bright coal developed in zones 5 to 11 is the source of coking coal in the Waterberg coal field.

Drilling on the six Iscor farms proved in-situ reserves of 578 million tons of coking coal and a similar reserve of middlings suitable for steam generation for zones 5 to 11, as well as 1 457 million tons of raw coal in zones 1,2,3,4 A and 4.

3. Coal Products

In the first phase of development, the bright coal occurring in zone 5 to 11 is treated in a beneficiation plant to produce blend coking coal for use as a blast furnace feedstock at Iscor's steelworks. A middlings product is also obtained which in turn will be used as a power station feedstock for Escom's Matimba power station which is currently being constructed approximately 6 km away from the mine (Fig. 2). In this phase the production from the mine will be as follows:

Overburden stripping	8×10^6 t/a
Run-of-mine plant feed	15×10^6 t/a
Coking coal product	1.8×10^6 t/a
Middlings as power station feedstock	3.5×10^6 t/a
Plant waste	9.7×10^6 t/a

In the next phase of development it is planned to expand the mine to provide for an increase in the production of coking coal and the additional production of coal for the direct reduction processes. The middlings arising from these beneficiation plants plus the remaining raw coal will be used as power station feedstock for a total power station capacity of 3,600 MW. In this way optimum use is made of the coal reserves for metallurgical and energy generation purposes.

4. Mine Layout

The mine layout has been done for a 40 year period in order to cater for the demands of the power station over its expected life (Figs. 3, 4).

The Enkelbult pit, which is one of the coal sources, has been designed with an overburden bench height of 15 to 20 m, followed by four benches of approximately 17 m each in bright coal. Below bench No. 5, a separate selective mining operation has been planned for the extraction of raw coal from the lower four seams as a feedstock to the direct reduction coal plants.

Access to the various benches is by means of a double ramp system with a gradient of 1 in 12 and a road width of 50 m on the south side and 35 m on the north side. The ramps were designed to provide the shortest possible distance between the coal faces and the primary crushers.

COKING COAL / MIDDLINGS

	THICKNESS M	%COAL (MASS) IN SITU	% YIELD (RD 1.427)	% ASH	% VOLATILE MATTER	ROGO INDEX RD 1.40	TONS x 10^6	% YIELD RD 1.427–1.70	% ASH	% VOLATILE MATTER	MJ/kg.	TOTAL SULPHUR	
ZONE 11	6.02	38.33	10.78	12.75	36.23	64	29.7999	17.84	29.10	31.50	23.34	1.58	3
ZONE 10	9.37	53.86	13.98	11.19	36.57	65	86.164	21.27	29.99	30.19	23.14	1.30	71
ZONE 9	6.53	48.34	13.24	11.42	36.11	63	70.198	17.43	27.87	31.54	23.90	1.29	5
ZONE 8	9.04	42.82	12.79	10.42	36.14	65	79.101	20.87	29.96	29.59	23.35	1.00	12
ZONE 7	10.15	39.94	21.21	10.62	36.40	67	104.715	16.65	28.23	30.80	23.95	1.15	13
ZONE 6	6.54	32.17	23.37	9.68	36.73	65	79.914	10.24	27.16	31.35	24.32	1.36	5
ZONE 5	13.54	21.97	15.05	9.97	36.05	57	127.890	6.31	27.56	30.13	24.39	1.02	7
TOTAL 9-11		35.91	14.30	10.63	36.30	64	577.781	14.33	28.85	30.30	23.80	1.10	5

RAW COAL

	THICKNESS M	% ASH	% VOLATILE	MJ/kg	TONS x 10^6
ZONE 4	4.02	35.42	21.08	21.25	314.807
ZONE 4A	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
ZONE 3	7.82	24.31	21.89	25.28	610.392
ZONE 2	3.37	23.23	22.84	26.19	297.376
ZONE 1	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
TOTAL 1-4					1456.860

Fig. 1: Grootegeluk coal bearing seams

Since it is possible that the plant waste, which contains some 15% carbonaceous material, will burn when stockpiled, it was decided to blend the plant waste with the overburden from the pit in order to increase the ash content of the mixture, and thus prevent spontaneous combustion of the plant waste dumps.

Overburden and plant waste will initially be dumped east of the Daarby fault, outside of the possible open pit mining area. Back filling of the overburden and plant waste will commence as soon as sufficient pit floor has been exposed.

5. Mining Methods

5.1 Overburden Stripping

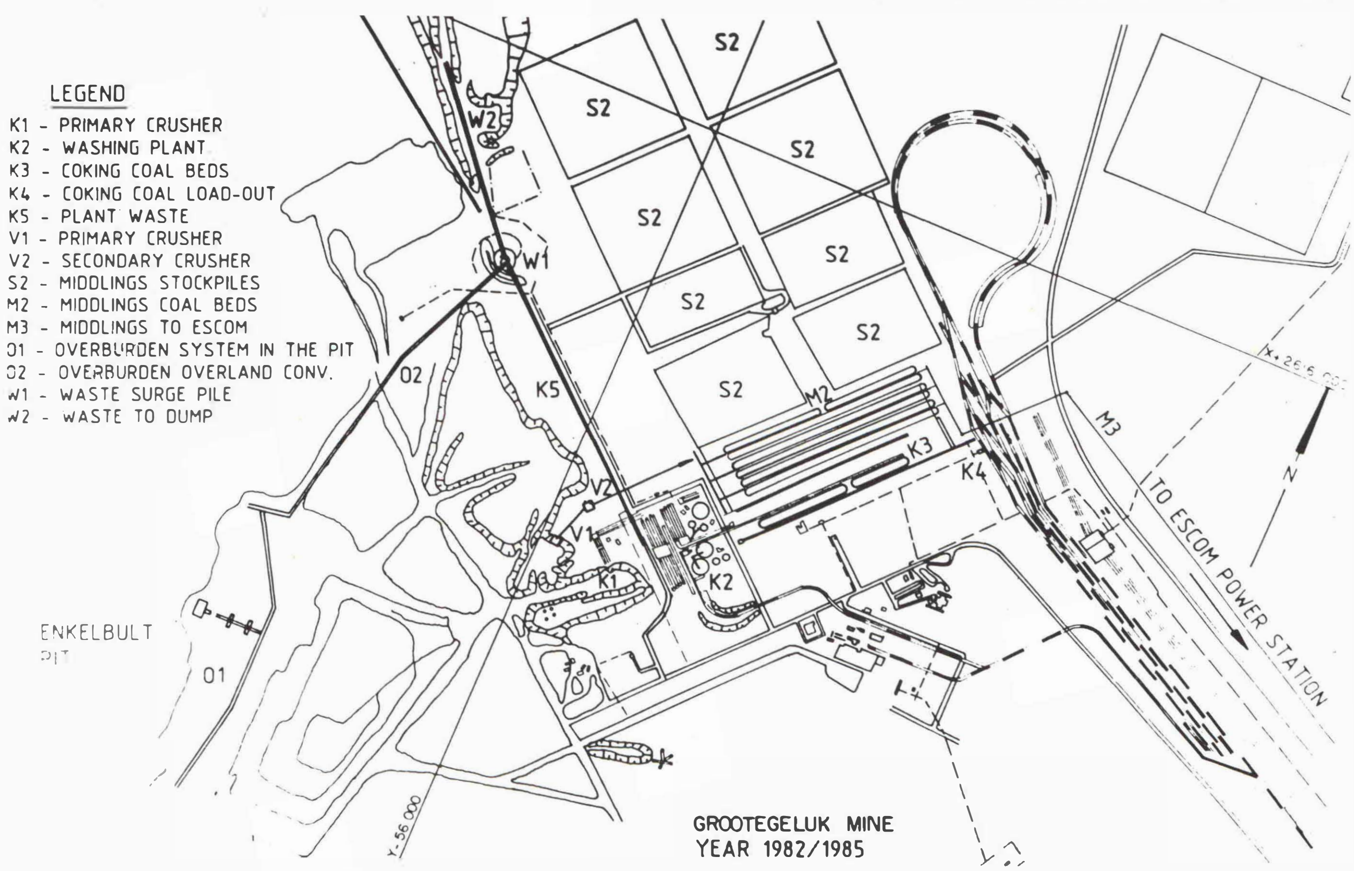
The original pit layout was based on the use of 150 t diesel-electric rear dump trucks for the transportation of overburden.

A feature of the pit layout is long, straight mining faces — plus the crushing and mixing of the overburden with plant waste. The fuel costs involved in using dump trucks would have proved prohibitive and it was clear that alternative stripping methods had to be investigated.

The following alternative mining, crushing and transport schemes for in-put overburden stripping were studied with regard to practicability, capital and working costs:

- shovel and trucks (scheme A)
- shovel, trucks and fixed gyratory crusher (scheme B)
- shovel, trucks and fixed impact crusher (scheme C)
- shovel, trucks and semi-mobile gyratory crusher (scheme D)
- semi-mobile crusher with conveyors on coal surface (scheme E)
- fully mobile crusher with shiftable conveyors (scheme F)

Fig. 2: General layout of the Grootegeluk mine area (year 1982/1985)



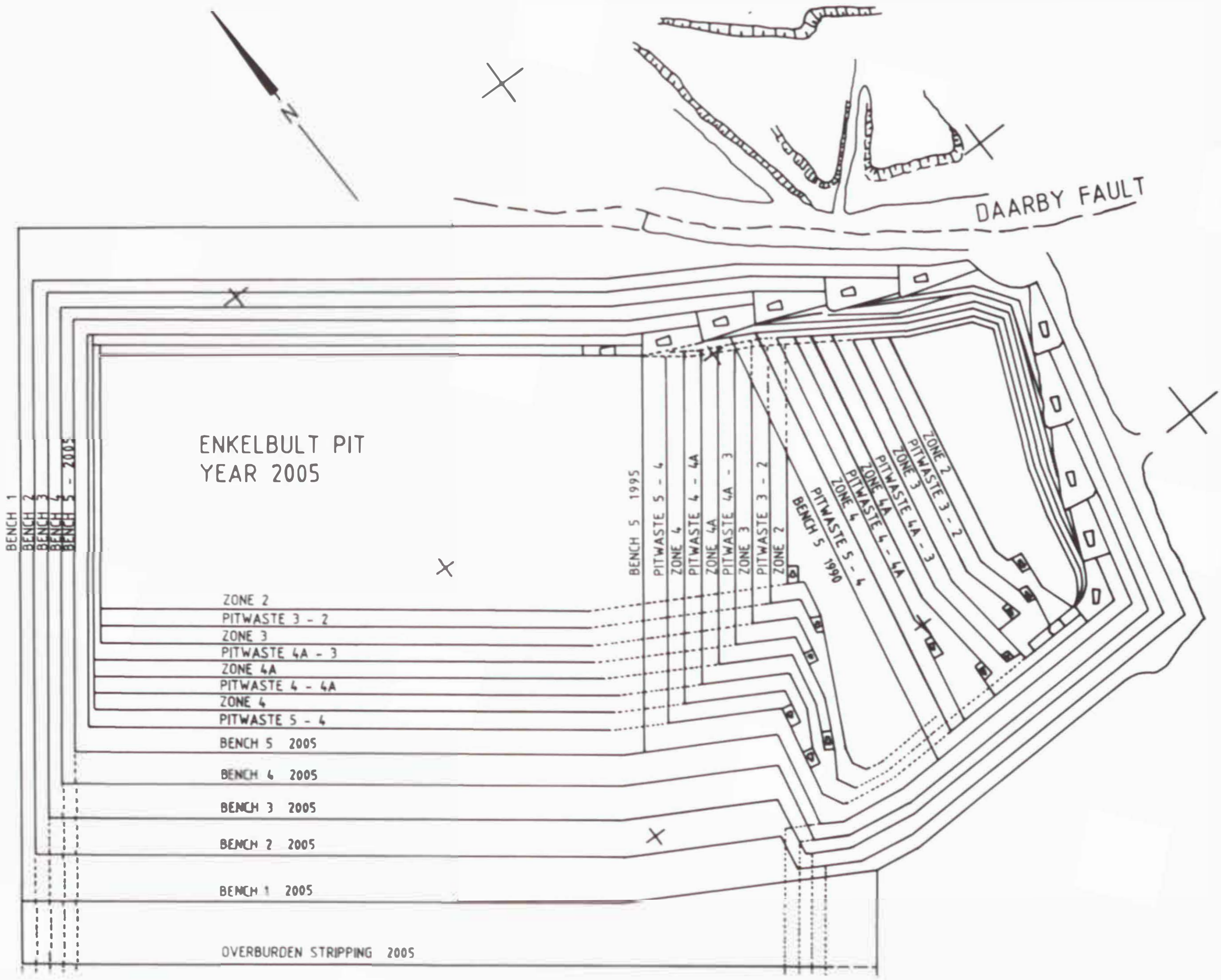
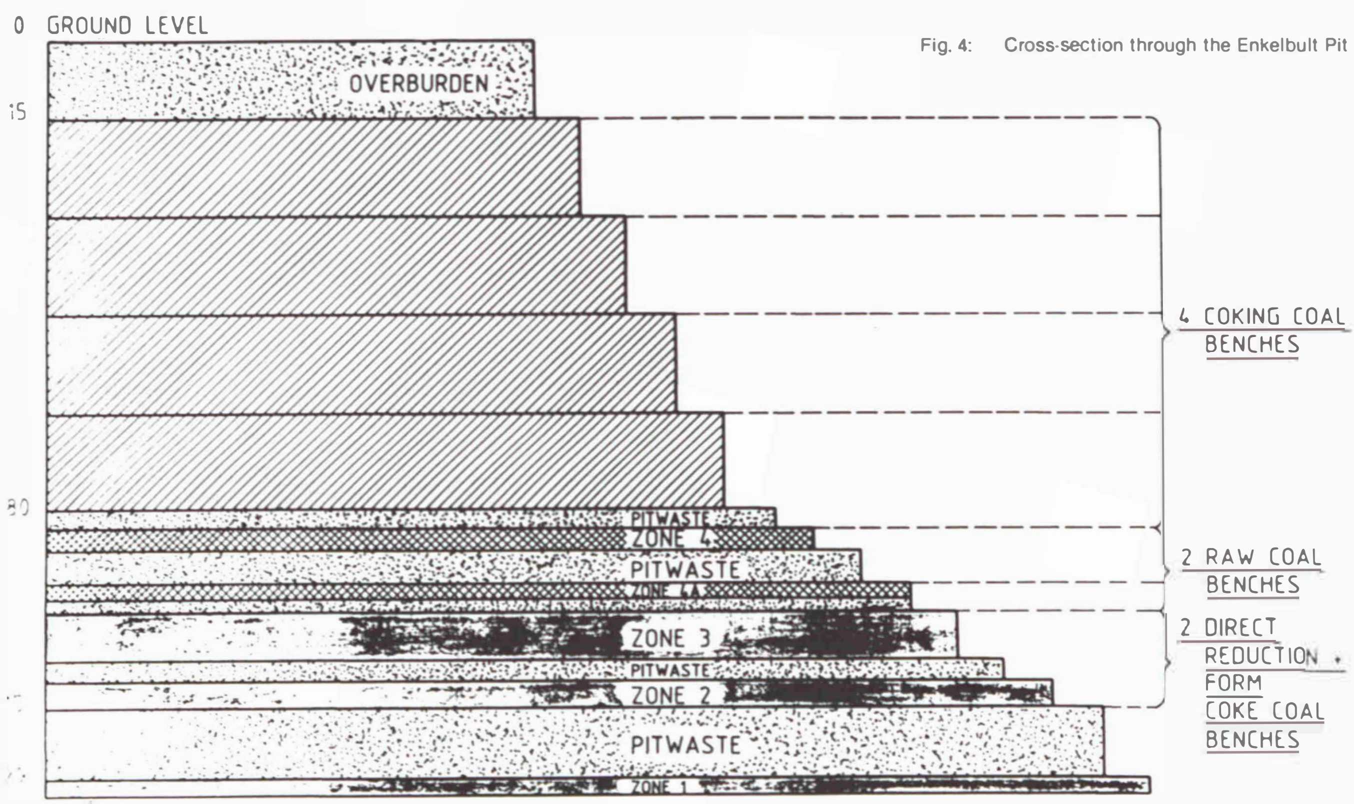


Fig. 3: Layout of the Enkelbult Pit (Year 2005)

Fig. 4: Cross-section through the Enkelbult Pit



The straight-forward shovel-and-trucks scheme had the advantages of flexibility and low initial capital outlay. However, as the pit working face moved further away, haul distances would increase and more trucks would be required to cope with the required production rate. The main disadvantage — and the deciding factor, therefore — was the amount and cost of fuel required and the total operating cost.

Other schemes also had plus and minus factors and it was finally decided that fully mobile crushers and shiftable conveyors would be the most feasible and least costly.

The strategic and cost implications of diesel fuel, in addition to other technical aspects of the system, indicated that a healthy return on the capital investment would be realised when compared with a conventional shovel/truck mining system (Fig. 5).

After drilling and blasting, the broken rock is loaded by means of a 17 m³ electric rope shovel directly into the feed hopper of the mobile crusher (Fig. 6). A 110 m long crawler mounted bridge conveyor provides a safety distance between the blasting area and the shiftable conveyors and furthermore increase the effective operating block width and reduces the shifting frequency (Fig. 7). The rest of the transport system consists of 3 shiftable face conveyors, 1 extendable conveyor and 2 overland conveyors (Fig. 8). Mixing of overburden and plant waste and stacking by means of a crawler mounted slewing and luffing stacker is considered to form part of the plant waste stacking system. Figs. 9 and 10 show the relative movement of the main items of equipment.

SCHEME		ESTIMATED COSTS AFTER ESCALATION (COSTS ESCALATED FROM A BASE DATE OF JUNE, 1979					
		A	B	C	D	E	F
TOTAL CAPITAL	R/ton	0,089	0,214	0,196	0,214	0,222	0,289
	Present Value	2 702	6 527	5 967	6 495	6 777	8 808
OPERATING AND MAINTENANCE COST							
Fuel	R/ton	0,203	0,133	0,133	0,155	0,117	-
	Present Value	6 184	4 059	4 061	4 723	3 556	-
Tyres and Belting	R/ton	0,094	0,090	0,090	0,080	0,078	0,041
	Present Value	2 862	2 737	2 737	2 449	2 375	1 261
Other	R/ton	0,158	0,183	0,232	0,168	0,160	0,198
	Present Value	4 809	5 577	7 078	5 099	4 880	6 010
TOTAL OPERATING AND MAINTENANCE	R/ton	0,455	0,406	0,455	0,403	0,355	0,259
	Present Value	13 855	12 373	13 876	12 271	10 811	7 271
REPLACEMENT COSTS AND REMAINING VALUE	R/ton	-0,020	-0,093	-0,085	-0,098	-0,112	-0,149
	Present Value	- 588	-2 856	-2 597	-2 939	-3 424	-4 527
TOTAL COSTS	R/ton	0,524	0,527	0,566	0,519	0,463	0,379
	Present Value	15 969	16 044	17 246	15 827	14 164	11 552
Savings in Cost expressed as a D.C.F. Return (% p.a.) in respect of incremental capital expenditure.		21,4% 17,6% 44,8% 22% 28,2%					

Fig. 5: Economic evaluation of various schemes for overburden stripping

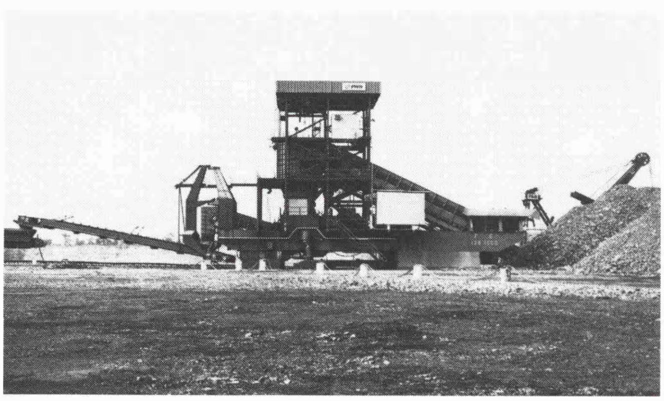


Fig. 6: Mobile crusher/face shovel

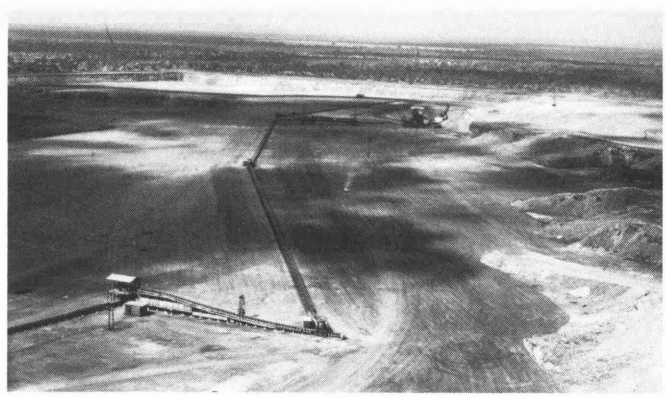
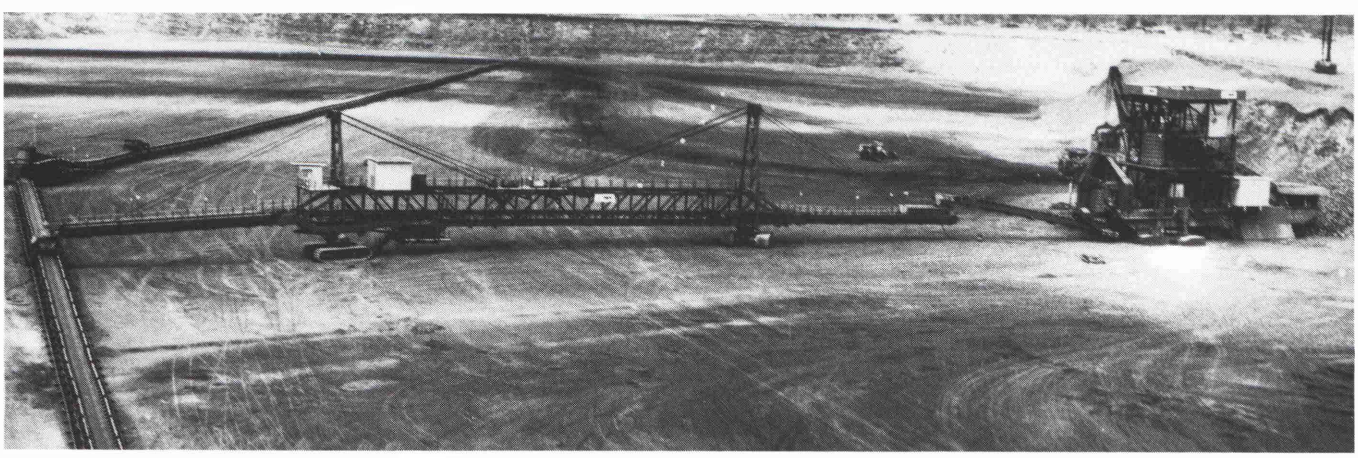


Fig. 8: In-pit crusher belt conveyor system

Fig. 7: Mobile bridge conveyor



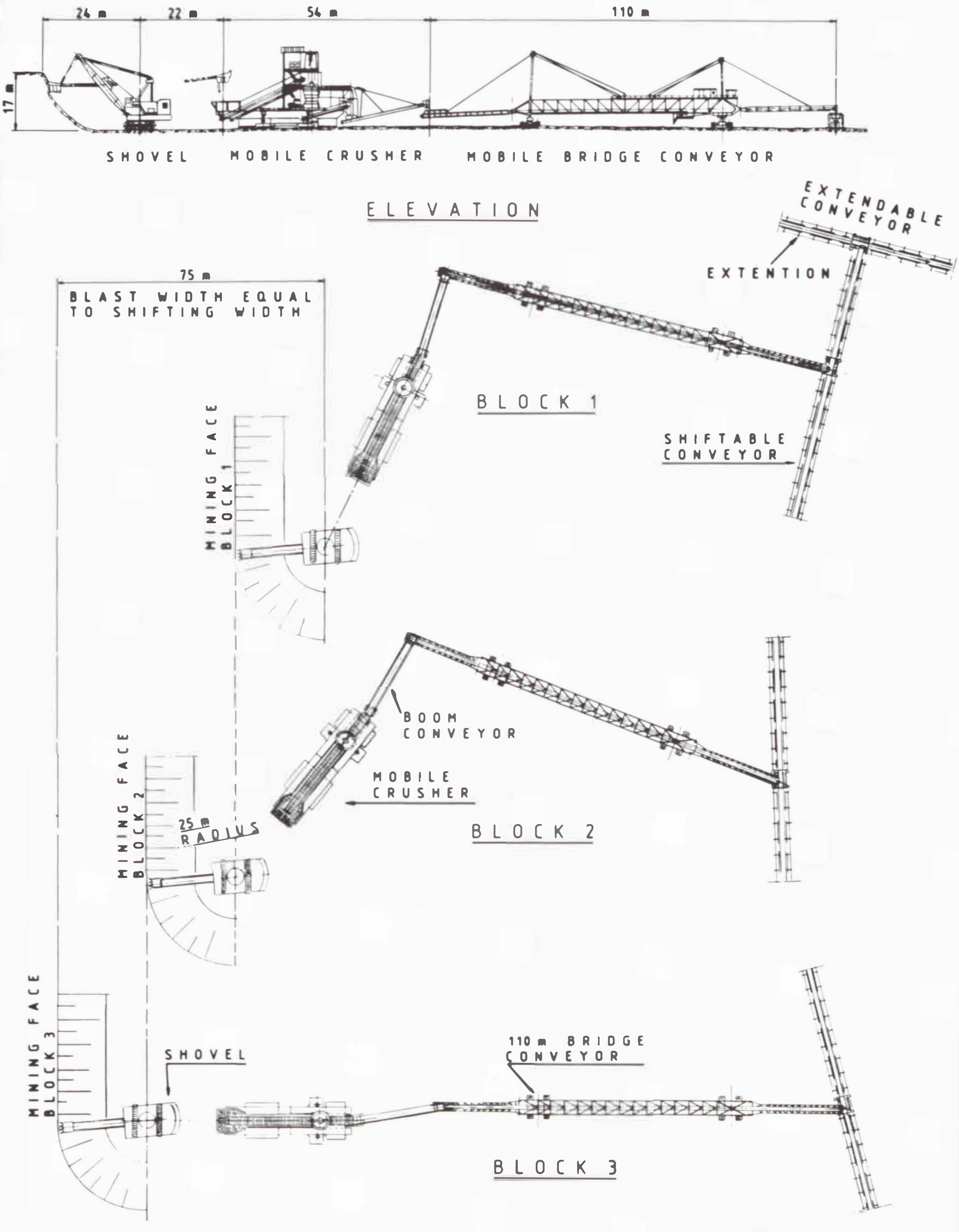


Fig. 9: Relative position and movements of the main components during overburden stripping

Fig. 10: Relative position of mobile crusher to working face

5.2 Bright Coal

Mining of the bright coal is carried out on four benches, approximately 17 m each, by means of drilling and blasting, loading with 22 m³ electric rope shovels and transporting to the beneficiation plant with 150 ton diesel electric rear dump trucks.

5.3 Dull Coal

The mining of the four seams of dull coal and their respective partings will be a selective mining operation with smaller equipment, for example 5 to 8 m³ shovels and 50 ton rear dump trucks.

6. Main Features of the In-Pit Crushing System

6.1 Shovel

The shovel is a P&H model 2300 LR fitted with a 19.8 m boom. The longer boom was chosen to maximise the amount of rock that could be loaded before the crusher has to be moved to a new position. The dipper size had to be reduced from the standard 22 m³ which is used in the coal to 17 m³ (struck). This machine is fitted with the swing drive of the next larger P&H shovel i.e., the 2800 to compensate for the longer boom. The hoist has also been increased in capacity with the result that the cycle times of this unit is similar to the standard unit.

Initial calculations indicated that the loading capacity would be in the order of 2,400 t/h based on the performance of the standard unit in combination with rear dump trucks. In practice it was found that an instantaneous loading rate of some 3,000 t/h can be achieved, mainly because no delays are experienced as is the case with spotting of trucks.

6.2 The Mobile Crusher Unit (Fig. 11)

The mobile crusher is a PHB-Weserhütte design with gyratory crusher type 54/74 and hydraulic walking mechanism fitted with 3 x 200 kW hydraulic pump drives.

Working weight:	1,100 t
Installed power:	approx. 1,300 kW
Material:	shale and sandstone, 0—1,800 mm 1.4 t/m³
Throughput:	3,000 t/h, 0—200/250 mm
Feed Hopper:	Capacity 60 m³
Apron Feeder:	2,400 mm wide, centre distance 24 m, incline 25°, 0.08 to 0.24 m/s variable speed drive, required power 2 x 200 kW
Spillage chain conveyor:	2,800 mm wide, centre distance 17.8 m, motor 2.2 kW
Crusher:	Allis-Chalmers gyratory crusher type 54/74, motor 400 kW, air cooling 75 kW
Crusher outlet Apron Conveyor:	2,600 mm wide, centre distance 12.1 m, motor 75 kW
Spillage Conveyor:	2,800 mm width, centre distance 9.1 m, motor 2.2 kW
Slewing Conveyor:	1,800 mm wide, centre distance 23.2 m, motors 2 x 45 kW
Mounting:	Raft — type coordinated walking mechanism speed up to 0.9 m/min., ground pressure when walking 1.5 kg/cm³, when working 1.0 kg/cm², motors 3 x 200 kW, 1 x 45 kW
Other features::	Maintenance crane, dust suppression.

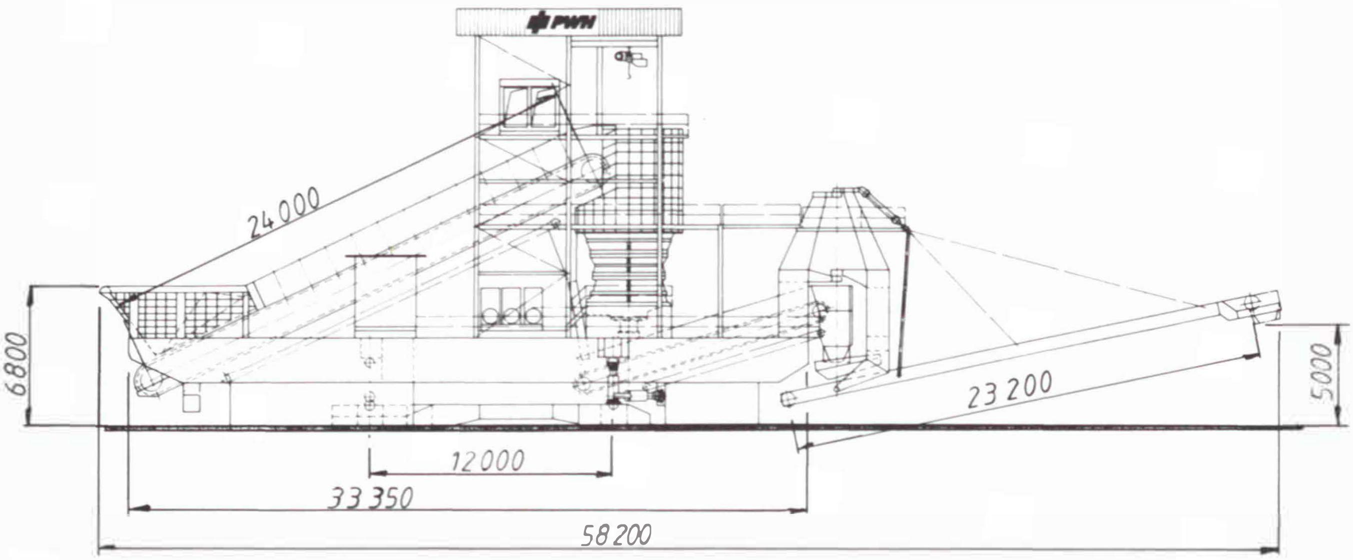
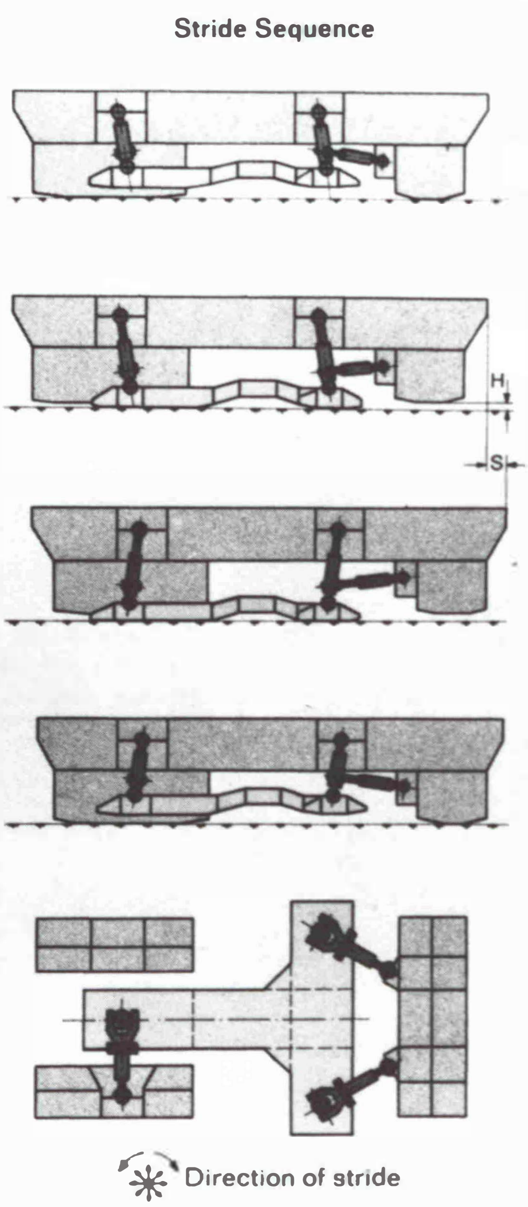


Fig. 11: The mobile crushing unit

Fig. 12: Operation of walking mechanism



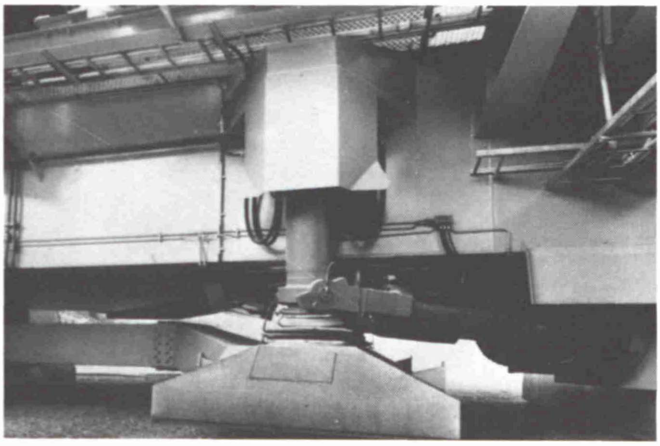
Run-of-mine rock is loaded into a 60 m³ feed hopper directly onto the apron feeder, which has been designed to absorb shock loads in the loading area such that no permanent deformation of apron feeder flights can take place. The feeder is a PHB-Weserhütte design fitted with a Caterpillar type D10 double chain and is driven by two variable speed hydraulic motors to ensure a controlled feed to the crusher. The whole feeder unit can be moved backwards or forwards to enable control of the rock trajectory into the crusher.

The mobile crusher is propelled by means of a coordinated hydraulic walking mechanism, consisting of a T-shaped walking pad, three lifting cylinders and three striding cylinders (Figs. 12 and 13). Through the press of a button any one of eight travel directions can be selected and in addition the installation can be turned on the spot. This high degree of manoeuvrability facilitates accurate repositioning which takes place once per day. During the crushing operation the mobile crusher rests on pontoons so that the walking mechanism is completely relieved.

Invaluable experience has already been gained with this installation whilst certain aspects can only be finally judged over a longer timespan, for example:

— Whether the simpler oil to air crusher lubricating oil cooling system selected (instead of more costly and complicated refrigeration units) is adequate for the worst operating conditions must still be seen.

Fig. 13: Typical part of walking pad with one lift and one stride cylinder



— Oversize boulders are easily loaded into the feed hopper because of the shovel dipper size. A rock breaker on a hydraulic articulated boom, as used in other similar installations, would reduce downtime due to crusher blockages.
— The hydraulic circuit has been designed to operate the three sets of walking cylinders in the walking mode as well as the variable speed motors of the apron feeder in the crushing mode. This design eliminates the need for an additional power source to the apron feeder, but results in a relatively complicated hydraulic circuit which has worked well in practice but still needs to stand the test of time before a final evaluation can be made. The alternative which had been considered was to install a variable speed electric drive to the apron feeder.
— The ground bearing pressure to which the walking pad and the pontoons are designed, needs special thought. If too low a value (conservative value) is selected, the bearing areas are designed unduly large. If the low value does not materialize in practice, undue stresses are induced due to point loading over the larger area. In addition the machine does not bed down as well for proper stability.

6.3 The Bridge Conveyor

The crawler mounted bridge conveyor is a PHB-Weserhütte designed unit with hydraulic travel drives.

Length between pulley centres:	110 m
Belt width:	1,350 mm at 2.62 m/s
Belt drive:	1 x 132 kW

The unit is fitted with an operator's cabin from where the crawlers are controlled. An improvement here would be to install a pendant mounted remote control unit which would allow the operator to relocate the machine from ground level. Accurate positioning is necessary, especially at the discharge end and at least two people are required to carry out this move with the present arrangement.

6.4 Shiftable Face Conveyors

Three shiftable conveyors are used, each having a length of 410 m, 2 x 132 kW drives, 1,350 mm belting at a speed of 2.62 m/s. Careful consideration was given to a single conveyor vs the three conveyor system. Advantages of the three conveyor system are the following:

(i) A straight mining face need not be maintained, which increases the flexibility of the system. During start-up a curved mining face had in fact to be contended with, and future planning necessitates the use of curved mining faces.
(ii) With three conveyors, the shifting operation can commence whilst the system is producing on the conveyor nearest to the extendable conveyor thereby reducing the total conveyor shifting downtime.
(iii) Upstream conveyors not in use can be switched off thereby reducing unnecessary wear and power consumption on the empty belt portions.

The conveyors are shifted by means of a shifting head which is mounted on a crawler dozer (Figs. 14 and 15).

Conveyor old to new positions are normally between 50 and 100 m apart. At present the shifting time is two days per conveyor with no shifting done during the night, mainly because of a shortage of personnel.

As experience is gained, the time required to move a conveyor is expected to decrease. The major problem in shifting

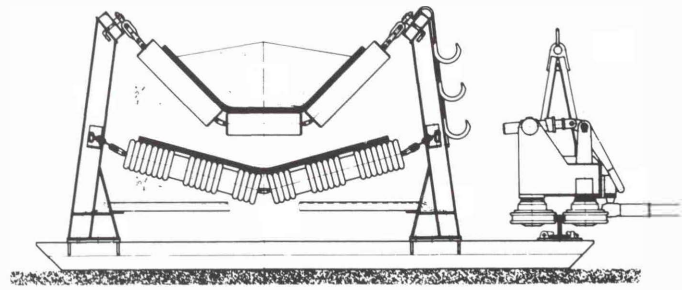


Fig. 14: Typical cross-section through a shiftable conveyor with shifting head

Fig. 15: Typical shiftable conveyor being shifted

the snaking conveyors is to avoid moving the conveyor more than approximately 1 m per traverse. Although larger increments may appear to be quicker, it usually results in subsequent breakages of fishplates, bolts, etc. and bending of the rails connecting the conveyor tables. An even more problematic result is the creep of the conveyor tables along the rails, resulting in uneven spacing of the tables.

6.5 Extendable Conveyor

This conveyor will be extended from its minimum length of 80 m as installed to a maximum of 875 m. The drive station is fitted with 44 m of belt storage, which means that new belt sections will not be spliced in with every extension of the conveyor. One 250 kW drive has been installed initially and two more of 250 kW each will be added as the conveyor length is increased.

6.6 Overland Conveyors

Two overland conveyors deliver the overburden onto the plant waste system where it is mixed with plant waste. Each is fitted with 3 x 250 kW drives and the conveyor lengths are 845 m at 26 m lift and 650 m at 35 m lift respectively.

6.7 Waste and Overburden Stacking

Plant waste and overburden is currently stacked by means of a temporary dual stacking system which had been designed with the object of creating so-called test stockpiles to test the combined stacking of plant waste and overburden. This part of the system is planned to be replaced by the final design in the near future (Fig. 16).

Fig. 16: PHB Weserhütte is building a similar crawler mounted luffing and slewing stacker as part of the final design of the system

7. Operating Practice

7.1 Drilling and Blasting

Blast holes are drilled at 200 mm dia with 6.8 m x 7.3 m burden and spacing, by means of a Bucyrus Erie 40R rotary drilling machine. Penetration rate is approximately 25 m per hour and rotary bit life in the order of 12,000 m.

Sub drilling of 1.5 m is normally applied for adequate floor control and a stemming height of 3.5 m, after filling with Anfex is found to give optimum results.

Flyrock was initially expected to be one of the main problems associated with this continuous mining method. It has been found in practice that the shiftable conveyors could be safely moved to as close as 110 m away from the face to be blasted without undue damage to the equipment.

7.2 Loading

With proper setting up of the shovel/crusher positions instantaneous loading rates of up to 3,000 t/h over short periods have been attained, provided that a short boom swing arc can be maintained. Approximately 65,000 tons can be loaded before a crusher move becomes necessary. Careful planning is therefore essential in order to minimize the number of equipment moves.

7.3 Crushing and Conveying

The crusher has been performing well and it has been found that the very small buffer space between the crusher outlet and the discharge apron conveyor is adequate, due to the controlled feed obtained with the input variable speed apron feeder.

With certain types of overburden the dust created can be a problem. Dust is suppressed by means of water sprays on top of the crusher, but this water addition creates severe problems downstream, due to build-up in chutes and on idlers.

Modifications to chute designs had to be carried out to prevent spillage. The average feed rate is normally set at 75 % of full capacity in order to ensure long runs without blockages or excessive spillage, which results in an average production rate of 1,800—2,000 t/h under present conditions.

8. Economic Evaluation

On the basis of the results of the feasibility study (Fig. 5) the mobile in-pit crusher system was installed at Grootegeluk. After more than a year of production, the question arises — was the right decision made? In order to answer this question, the system as implemented was again evaluated on the basis of the actual capital and working costs as found in practice. Of the systems considered in the original feasibility study, the second best alternative was as described as scheme D, i.e., trucks and semi-mobile crusher (outside the pit). The two systems were again compared on a "with escalation" basis. The waste tonnages used in this study are considerably higher than for the original feasibility study, due to the anticipated increase in the coking coal production from Grootegeluk. On a July 1982 price basis the comparison of the two schemes are as follows:

	Trucks and Crusher Outside Pit	Mobile In-Pit Crusher
Present value of Capital Expenditure (in million Rand)	9.1046	13.702
Total unit cost (after escalation) (Rand/ton)	0.674	0.558

From these results it is apparent that the capital cost of the mobile crusher scheme was higher than anticipated but still shows a D.C.F. rate of return advantage of 22.1 % on the differential capital between the two schemes.

9. Future Applications

An inspection of the final pit shape as shown in Fig. 3 indicates that the haul distance for coal extraction to the primary crusher increases to approximately 6 km and it appears that in-pit crushing offers one alternative method for this part of the operation. In this instance one needs to contend with four mining benches, and an in-pit crusher evaluation would probably resolve around the decision to install a mobile crusher on each bench to eliminate trucks altogether or alternatively, a semi-mobile unit installed at the pit edge to reduce the truck haulage distance. Another alternative method would be to install an electric assist line for the rear dump trucks. A decision to proceed with this latter system was recently taken and it is currently being installed at the mine.

A study was recently carried out for an open pit application with a 60 to 90 m thick overburden layer in the Waterberg coal field. In this study it was found that in-pit crushing of overburden and around-the-pit conveyors and waste stacking compared favourably with a mining system comprising shovels and trucks in a pre-benching operation, together with a dragline in the bottom portion of the overburden. The main advantage of the in-pit crusher in this instance, is that the width of the coal pit could be increased to any given size, whereas the working space becomes very limited at this depth, when using a dragline.

In general it can be stated that, given the correct application, mobile in-pit crushing offers a viable alternative to shovel/truck operations in opencast mining.

10. Conclusions

At the time when the decision was taken to go ahead with the mobile in-pit crushing system for overburden, no other application of similar magnitude and operating conditions existed with which a parallel could be drawn. Now after twelve months of limited operating experience, it can be safely said that most of the objectives have been achieved. The system design capacity has been demonstrated over limited periods and the shifting of components are being carried out within reasonable time with the prospect of further improvement. The question of total system availability over long periods, under full load conditions, is a function of the maintenance and operating stoppages which will only be known once the final design for the stacking system has been installed. There are no indications, however, that the required availability standards could not be achieved and we are of the opinion that this continuous opencast mining system has been proven to be technically and economically viable.

11. Acknowledgement

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