

The Use of Non-Flameproof Vehicles Underground in a Fiery Mine, Outside Potentially Dangerous Areas

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SYNOPSIS

INTRODUCTION

It is always intriguing to read and decipher papers of a technical nature. This paper may surprise you as being totally different in that respect. It deals with application engineering and has very little to do with design and development. It further deals with simple, straightforward engineering principles, with the emphasis on a total systems approach. Instead of looking at a clinical technical solution, more time is spent on all areas that might have a long-term influence on the project, i.e. operation, people unions, legal requirements, safety, health influence on durability, etc.

The paper makes use of comparisons as a major basis, between what was usually done as opposed to what becomes possible if lateral thinking is applied. Instead of reverting to technical solutions, as most engineers tend to do, simple solutions requiring more discipline (self-discipline) were investigated.

For many years, since the introduction of regulations that enforced the use of flameproof diesel engines underground in fiery mines very little had been done in terms of health, safety, reduction of capital, operating and maintenance cost and intensity as well as reliability of flameproof equipment. Numerous innovations were put onto the market. Engineers and managers accepted what was offered to them by suppliers in terms of new equipment. Understandably, the equipment was very expensive, due to special design, low volume uniqueness. Reliability was not very good, due to poor system integration and natural trial and error methods

of design evolution and development.

On numerous occasions various individuals, especially employees relatively new to the coal mining industry would ask the naïve question; Why do you use these "monsters" underground, and not normal LDV's?. The "normal" answer was: if it works underground, it must be flameproof because its safe, and the heavier and thicker the steel, the better. Well, it would last longer!

A bit more than one year ago all this changed, someone asked once too much. This sparked an in depth investigation into the current state of affairs and the possibility and feasibility for change.

THE CURRENT ISSUES

As many as possible parameters were considered for the currently used means of transportation for supervisors and personnel underground. Although this paper is limited to this kind of transport, the long-term scope is not limited.

The currently used machines were analysed in terms of some key aspects. These aspects are discussed in no specific order:

Safety

Flameproof diesel equipment is generally considered safe due to the principles employed:

- The cleaning of air entering the engine, and
- The cooling and dilution of exhaust gases.

At closer investigation, the above was found to be true but that very little time had been spent on:

- Standardisation from vehicle to vehicle
- Reliability of safety systems and equipment.
- Maintainability of safety systems and equipment
- Safety system probes.
- Flameproof surfaces.
- Fuel systems.

- Monitoring systems.
- Frequency of maintenance required.
- Brake systems.
- Lighting systems.
- Suspension systems.
- Effects of accidents as a result of robustness
- Systems integrations.

Health

It was found upon investigation, that very little or no consideration was given to health factors such as:

- Smoke levels.
- CO emission levels.
- Fuel consumption and subsequent emission of particulate matter from the exhaust systems.
- Levels of NO emissions.
- Noise levels.
- Ergonomics of driver and passengers.
- Lighting efficiency (no dim/bright facility and inadequate light distribution).
- Effect of smoothness of ride on drivers and passengers.
- No positive control of speed (self discipline or enforced discipline).
- Crash ability of vehicles? Protection for passengers in case of accidents (passive accident absorbing materials).

Standardisation

Very low levels of standardisation due to low volume manufacturing resulted.

Effects of non standardisation:

- Poor availability.
- Poor reliability.
- Expensive stockholding.
- Long downtimes.
- Inadequate safety system designs.
- Difficult maintenance.
- Poor flameproof system and component integration practice.
- Flexibility to use artisans on various pieces of equipment.

Cost

Extensive capital and operating cost resulted, due to low volume, non-standard equipment.

- Flame proofing (frequency).
- Non-standard.
- Mass to power ratio.
- Braking systems.
- Suspension.
- Under carriage.
- Wheels/tyres/axles/fixtures.

Other Risks + Design Factors

- Lack of speed indication.

- No control of speed.
- General lack of visual and/or audible monitoring of operating systems.
- Nuisance factor of starting the vehicle.
- Practical implications with regard to operation
 - ♣ Refilling of scrubbers
 - ♣ Refuelling intervals (size of fuel tanks).
 - ♣ Reliability of safety devices (uncontrolled deterioration).
- Poor system integration:
 - ♣ Power transfer of engine to flywheel.
 - ♣ Power transfer of flywheel to gearbox.
 - ♣ Gearbox to differential.
 - ♣ Differential to wheels.
 - ♣ Power head to load box.
- Mass to payload ratios.
- Mass to braking capacity (Brake distance).
- Poor paint specification.
- Fuel consumption:
 - ♣ Extreme fuel consumption figures for the job done (consumption to payload ratio).

OBSTACLES

A number of obstacles were identified with the initial investigation:

Legal requirements

It is believed, by perception, fact or reference, that it is required by regulation that any diesel engine used underground in a fiery mine, must be flameproof or "semi-flameproof".

Culture of the average user

The average user of a diesel powered vehicle underground believes that it must be "big and strong and heavy and of thick steel" to last underground.

Every piece of equipment used underground must, from the outside look as if it is indestructible.

Underground equipment must be designed to withstand abuse. Users must be able to bump machines against one another without damaging the externals. Vehicles must be designed to push start one another if required, without proper means of protection.

There are no means of monitoring the use of these vehicles. Therefore apart from severe physical damage, nobody could ever determine if it was abused or not.

Discipline

- Poor discipline of operators/drivers exists, due to non-reliable /non-existing monitoring systems.
- Systems would be required to restrict operators exceeding parameters, as well as preventing them from entering potentially dangerous areas.
- Proper training of operators/drivers, as well as teaching them self-discipline.
- Proving a new system before mass implementation.

Other Obstacles

- Persuading colleagues, management and operators/users that another system can actually be implemented successfully.
- To prevent incidents/accidents, by implementing a totally integrated system approach.
- Setting of proper baseline assumptions to measure the degree

of success after implementation.

- Designing systems to limit surface and exhaust gas temperatures of engines to within acceptable limits (no existing regulations).
- Possible theft of components.
- Choice of vehicles.
- Setting up new parameters to satisfy current non-existing regulations to ensure that safety is still the highest priority.

POSSIBLE SOLUTIONS

- Standard vehicles:
 - Inexpensive.
 - Mass produced.
 - Total system integrations.
 - Well designed power to mass ratio.
 - Well designed mass to payload ratio.
- Acceptable fuel consumption
- Showroom to underground concept (as standard as possible)
- Comprehensive monitoring system:
 - Driver identification.
 - Vehicle identification.
 - Speed and RPM.
 - Fuel consumption.
 - Start time.
 - Stop time.
 - Idling time.
 - Manifold temperature.
 - Harsh acceleration.
 - Fast deceleration.
 - Accidents.
 - Distance, etc.

- Prevention of the excessive corroding of the much thinner materials used in mass production vehicles by daily compulsory washing.
- Dedicated parking areas for every vehicle.
- Orderly traffic flows

FINAL GOALS OF THE PROJECT

- Safe, healthy and efficient transport for supervisors and personnel.
- Improving availability, reliability and maintainability of equipment.
- Improving mobility of personnel.
- Significantly improving cost of maintenance of equipment
- Significantly improving health risks and not negatively influencing safety aspects.
- Reduce fuel consumption figures significantly .
- Keep stockholding to an absolute minimum.
- Reducing the nuisance factors in the following respects:-
 - ✍ Starting (limited starts with hydraulic and compressed air methods).
 - ✍ Refuelling (after every trip due to poor consumption).
 - ✍ Scrubber refilling (after every trip, numerous recharging points required).
 - ✍ Too heavy to change wheels by ordinary means (i.e. one-man operation).
- Self-discipline of operators/drivers through proper monitoring and enforcement of rules.

- Very strict rules for use, enforced to the letter (all drivers under the same rules, drivers of vehicles are always known due to the identification system).
- Punctual services.
- All drivers to take full responsibility for the vehicle they operate at any time.
- Vehicles employing proper system integration.
- Continuous system monitoring.

MISSIONARY WORK TO CHANGE THE CRURRENT APPROACH

- Unions.
- Colleagues.
- Management (top to bottom).
- Department of Mineral and Energy
- Users.
- Supervisors.
- Passengers.
- Vehicle Manufactures, etc.

APPLICATIONS TO THE DEPARTMENT OF MINERAL AND ENERGY

The following documentation was included:

- Application for approval of non-flameproof vehicle for the regular conveyance of persons in terms of Mine Health and Safety Act (Regulation 10.25.2 (b)).
- Applications for the regular conveyance of persons underground by means of personnel transporters (Mines and Works Regulation 18.3.2).
- Minutes of meetings between all affected parties.
- Test reports of accredited facilities.
- Layout and specification of trailer for personnel transport.
- Layout and specification of canopy for normal LDV.
- Operators handbook and brake system details.
- Statutory requirements as set out by the Chief Director Machinery and Safety.

WHAT HAD TO BE DONE TO COMPLY WITH THE APPROVAL IN TERMS OF REGULATION 10.25.2 (B)

Markings

- ✍ The vehicles had to be distinctly marked NON-FLAMEPROOF (200mm high) on both sides.

Routes

- ✍ Operators/drivers are being trained to use these vehicles only in intake airways, which are clearly indicated on a plan approved by the manager.

- ✍ The road surfaces along the approved routes are kept in a condition that no coal dust can be gathered other than negligible quantities on potentially hot surfaces.
- ✍ All entrances to hazardous areas on approved routes are adequately and effectively indicated by clearly visible means to ensure that inadvertent access of vehicles equipped with these engines do not accidentally proceed beyond these points.
- ✍ The air velocities along the approved routes are continuously monitored and if ventilation weakens, or if flammable gas accumulates, the driver is warned (at the moment travelling way lights switch of in case of ventilation weakening). The drivers are trained to switch off the vehicle and only restart it when permitted by the manager, mine overseer, engineer or shift boss and then only if he has determined by testing that it is safe to do so (same as any other vehicle).

Record Keeping

- ✍ The logbook referred to in Regulation 2.15.6 must be filled in daily by the operator or an official of more senior rank to ascertain that all conditions had been observed and measures taken to comply therewith.
- ✍ This logbook must be examined and countersigned by the responsible manager at least once in every month.
- ✍ In addition to this every vehicle is equipped with a comprehensive self monitoring system as described elsewhere in this document, logging all events. The data of the monitoring system is downloaded on a personal computer at the end of each shift. This on board system accumulates data for up to six months.

Temperature Control

- ✍ The exhaust gas and surface temperature of the engine was set at a maximum of 200°C. A device is installed on the exhaust manifold that cuts the fuel supply to the engine if the temperature exceeds 190°C. The system logs every time the temperature reaches the 150°C level for reference purposes.

- ✍ The engine is de-rated by 20% in respect of diesel injection and maximum revolutions. Both control adjustments are kept sealed at all times. The diesel pumps are distinctly marked "De-rated pump". The differential ratios were changed to compensate for the de-rated engine in an attempt to maintain original power at the wheels (Vehicles operate at fairly low speeds).

Maintenance

- ✍ The engines are maintained to operate at maximum brake thermal efficiency.
- ✍ A comprehensive vehicle washing system is to be introduced to reduce the risk of accumulation of combustible materials on hot surface (Every vehicle has to be washed at least once in every 24 hours. This system forms an integral part of the continuous monitoring system).
- ✍ A competent person or persons are appointed by the engineer to carefully examine the engine once every working day, in order to maintain it in a safe working condition. Every operator/driver is trained in this respect.
- ✍ A record book is kept for each vehicle, where all details are recorded with regard to maintenance, inspections, services, repairs and tests. Such a record book, must be scrutinised and countersigned by the engineer at least once in every month. This record book must further contain the following:
 - ⇒ The names of competent persons appointed to examine daily.
 - ⇒ A copy of the approval granted by the Department of Minerals and Energy in terms of Regulation 10.25.2 (b)

WHAT HAD TO BE DONE IN TERMS OF MINE HEALTH AND SAFETY ACT REGULATION 18.3.2 FOR THE REGULAR CONVEYANCE OF PERSONS UNDERGROUND

- ✍ A specific vehicle (power head) and personnel carrier (trailer) was identified that conform to requirements of normal road ordinance regulations.
- ✍ Every vehicle used the conveyance of persons is clearly marked with an identification number.
- ✍ The number of persons for the specific application was limited to 20, including the driver (the maximum number of persons allowed to travel in the vehicle is clearly marked on each vehicle).
- ✍ A safe speed limit of 40km/h was introduced for constructed roads (the continuous monitoring system takes care of this by alarming at the set limit and logging, it).
- ✍ The manager or engineer who is responsible to appoint a competent person, for the safe loading and unloading and seating of passengers.
- ✍ No materials except light articles, which hold no danger to persons, are allowed to be transported with personnel.
- ✍ The braking system of the vehicle complies with the requirements of Regulation 18.6.1.
- ✍ The responsible engineer must appoint a competent person or persons, whose duty it is to examine carefully at least once a day, all the components of the vehicle, upon which the proper working thereof and the safety of persons, depend (in our case all licensed drivers).
- ✍ No vehicle may be used, before defects that have been found are rectified.
- ✍ A record book is provided for each vehicle in which the following are entered:

- ⇒ The names of the competent persons for safe loading, unloading, inspection, etc. (in our case the drivers with special training and licenses).
 - ⇒ A copy of the approval document in terms of Regulation 18.3.2.
 - ⇒ A true report of every examination.
- ✍ The responsible engineer must scrutinise and countersign these reports at least once per month.

CAPITAL APPLICATION FOR THE PILOT PROJECT

Assumptions for setting up a baseline case:

- Safety & health aspects
 - ✍ Smoke (particulate matter) levels
 - ✍ CO emission levels
 - ✍ NO emission levels.
 - ✍ Exhaust surface temperature levels.
 - ✍ Exhaust gas temperature at exit.
 - ✍ Braking efficiency
 - ✍ Headlight efficiency
 - ✍ Driver visibility
 - ✍ Horn efficiency.
 - ✍ Noise levels (driver and passengers).
 - ✍ Ergonomics (drivability and passenger comfort).
- Integrated management system.

- ✍ Rules and control measures.
- Cost analysis.
 - ✍ Capital cost.
 - ✍ Life maintenance cost.
 - ✍ Fuel consumption.
 - ✍ Trade in values.
- Mass to payload ratios.
- Power to mass ratios.
- Speed versus braking distances.

MEASUREMENT OF CURRENT PERFORMANCE VERSUS ASSUMPTIONS

Current performance exceeded all the initial assumptions on the pilot project (vehicles running for four moths).

In the table below are some key aspects and comparisons:

LDV	A	B	$\left(\frac{B}{A}\right) * 100$	$\left(\frac{B}{\text{Mine cruiser}}\right) * 100$
	LDV (a)	LDV (c)	LDV (c) versus LDV (a)	LDV (c) versus Mine
Cost/Month/Unit * (R)	1,000	29	2.9%	0.72%
Fuel consumption (km/l)	11	20	181.8%	500%
Total cost/km (c/km)	100	16.5	16.5%	3.65%

Key: (a) - Assumption
 (b) - Current
 (c) - Oil and diesel excluded

PONI	A	B	$\left(\frac{B}{A}\right) *100$	$\left(\frac{B}{Bus}\right) *100$
	Poni (a)	Poni (c)	Poni (c) versus Poni (a)	Poni (c) versus Mine
Cost/Month/Unit * (R)	1,100	29	2.6%	0.28%
Fuel consumption (km/l)	10	15	150.0%	1364%
Total cost/km (c/km)	110	16.78	15.25%	1.36%

Key (a) - Assumption
 (b) - Current
 (c) - Oil and diesel excluded

CAPITAL APPROVAL FOR THE REMAINDER OF THE PROJECT

Supervisors in the following categories were identified as potential users:

- All production sections.
- Power reticulation.
- Road building.
- Stonework and general
- Pumps and water reticulation
- Training.
- Central transport.
- Surveyors.
- Telemetry department.
- Quality control.
- Condition monitoring.
- Main conveyor systems.
- Diesel fleet service vehicles.
- One spare unit.

A bus for every production section and one spare unit.

Notes :

Approval of the design and construction of a diesel engine used

in fiery mine (Mine Health and Safety Act: Regulation 10.25.2(b)), was received on the 12th of May 1998.

Approval in terms of Mine Health and Safety Act (Regulation 18.3.2), for the regular conveyance of persons underground, was received on the 18th of May 1998.

CONCLUSION

It is the sincere belief of the authors of this paper and the people that are successfully implementing it, that this is a major breakthrough in terms of safer, healthier, more reliable, less expensive transportation of supervisors and personnel underground in collieries.

Anyone that wishes to attempt a similar venture is welcome to enquire, but shortcuts are not recommended. Rather go through the total process to ensure success:

- Investigate (what you have versus what is required, follow a scientific approach).
- Persuade (ensure buy in of all stakeholders).
- Total system approach (i.e. traffic flow, parking, daily washing, road construction rules, training, special licensing, etc.).
- Communicate (peers, Management, Unions, Department of Minerals and Energy, SABS, etc).
- Control (consider a comprehensive monitoring system).
- Foster self-discipline (multiple drivers, individual control, smart card system, etc.).
- Involve accredited third parties (SABS, Universities, suppliers, etc.).
- Consider strict rules (one page only).
- Enforce these rules (publish these and make it part of the

training exercise).

- Invest in training (compile and implement a comprehensive training program, K53, practical testing, etc).
- Don't take anything for granted.
- Special licences.
- Agreed maintenance programmes:
 - ⇒ Manufacturer and mine.
 - ⇒ Accredited workshop.
 - ⇒ Extended guarantee.
 - ⇒ Service intervals.
 - ⇒ Record keeping.
 - ⇒ Comparisons.

Improvements in Dragline Productivity at a Witbank Colliery the True Story



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IMPROVEMENTS IN DRAGLINE PRODUCTIVITY PRACTICAL SOLUTIONS TO PRODUCTION PROBLEMS

SYNOPSIS

The formation of Ingwe Coal Corporation in 1995 saw an increased demand for coal from Douglas Colliery. Mine management was aware that the underground operation was facing diminished pit room, therefore the increased tonnage had to be produced by the opencast mine. It was known that this operation was not performing well and that an increase in productivity was needed. Management's initiative was to address all areas having a negative impact on productivity and to raise production with available skills and without capital expenditure. This paper attempts to show how a practical, structured approach to productivity improvement can achieve the desired results. The Douglas Opencast Bucyrus Eyrie 1570 dragline has achieved a productivity improvement of 56.7% over a three and a half-year period and now ranks amongst the top performers both locally and internationally.

LOCATION AND STRUCTURE OF DOUGLAS COLLIERY

Douglas Colliery is situated in the Witbank coalfield and is located 23 Kilometers South East of the city of Witbank, Mpumalanga. The mine complex is situated 550 kilometers from the Richards Bay Coal Terminal. The mining complex is large and consists of three collieries, Douglas, Vandyksdrift and Wolwekrans, consolidated into a single administrative unit. The mining area extends for 20 kilometres on an East-West axis: and for 15 kilometres from North to South. The mine employs 2980 people and currently produces 14.3 million run of mine tons of coal per year. 57% of this coal is produced from underground mining and 43% from opencast. Run of mine coal is sent to three destinations.(1) to a processing plant to produce a 28 mega-joule/kg product for both export and local markets: (2) to a two stage processing plant to produce low ash and premium grade coal for export and (3), as a raw feed to local power stations. The formation of Ingwe Coal Corporation in 1995 resulted in an increased demand for coal from Douglas and, to meet the demand, a new project was approved by the board of directors. This six hundred million Rand opencast operation will produce six million run of mine tons per year from pillars

and in situ coal from the year 2001 onwards. The production profile for the mine will rise by 23,5 million tons in the year 2003. The opencast contribution will amount to 63% at that time. Figure 1 shows the annual tonnage produced, from formation of the mine in 1949, up to the planned tonnage in 2003.

GEOLOGY

Stratigraphic Column

Shown below is a typical stratigraphic column of the Douglas Colliery coal reserve. The three main seams mined are the 4 Lower seam, 2 seam and 1 seam.

Geological Description

The Douglas Colliery reserves are located in the Witbank Coal field. All that remains of the Karoo system in this coal field is the bottom part of the Middle Ecca (Vryheid Formation) along the Dwyka tillite. There are seven coal seams in the areas where the sequence is fully developed. The strata in which the coal seams occur consist mainly of fine, medium and coarse grained sandstone together with mudstone, shale and carbonaceous shale. The Karoo sediments are abundantly intruded by dolerite dykes and sills. A major feature along the Northern boundary is the Ogies dyke, which is a post deposition feature striking East - West.

HISTORY

Douglas Colliery was commissioned as an underground mine in 1949, with a planned production rate of two million tons per annum. By 1981 the international coal trade was growing rapidly and in order to exploit this opportunity, Douglas decided to introduce opencast mining on a larger scale. A Bucyrus Eyrie 1570W dragline was commissioned in 1982 in order to produce an additional plant feed of three million tons per year.

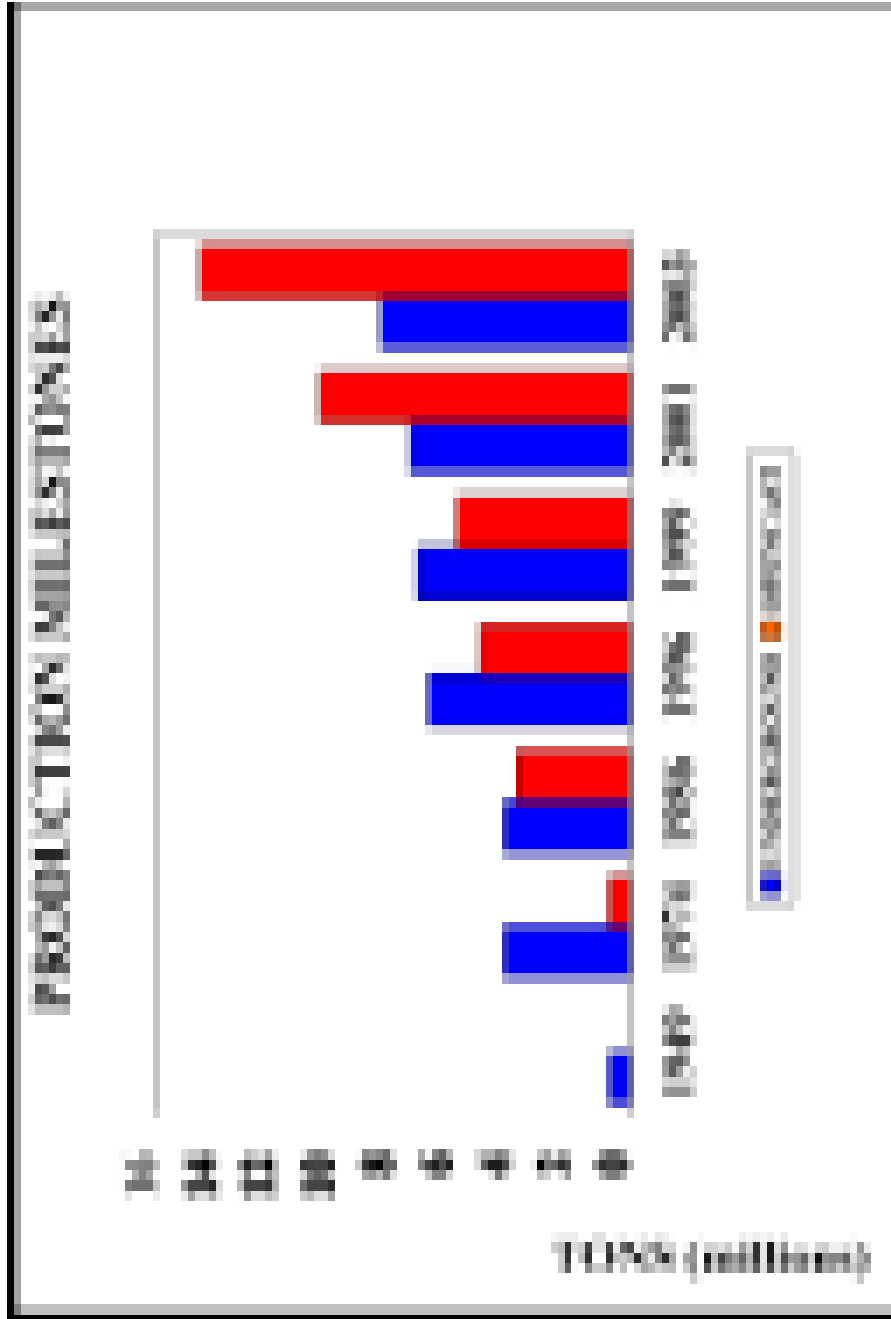


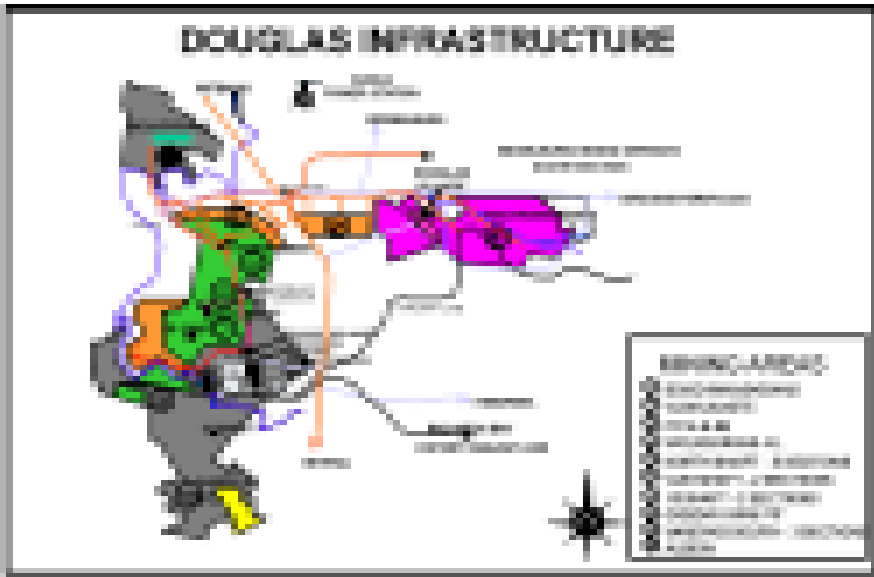
FIGURE 1

Figures 2 and 3 show, respectively, the location and infrastructure of the Douglas mining complex

FIGURE 2 - LOCALITY



FIGURE 3



By June 1994, the underground operations were producing two thirds of the mine's output. With the formation of Ingwe Coal Corporation in 1995, the mine's tonnage throughput was increased by a further 20%. Due to ever-decreasing pit room, the underground sections could not meet the additional tonnage requirement. Attention was then focussed on the opencast mine to see how the tonnage could be produced.

INTRODUCTION

During 1996 the dragline was mining at a rate of 1500 BCM per hour, which resulted in a coal exposure rate of 270 000 tons per month. The new tonnage profile required an exposure rate in excess of 400 000 tons per month, by 1998. To increase coal exposure by this large amount would require a major investigation of all issues impacting on dragline productivity. A senior management-led team, consisting of production and technical personnel, was tasked to carry out the productivity improvement study and investigations began in July 1996.

Figure 5 shows dragline performance at 6-monthly intervals for the period January 1996 to June 1999.

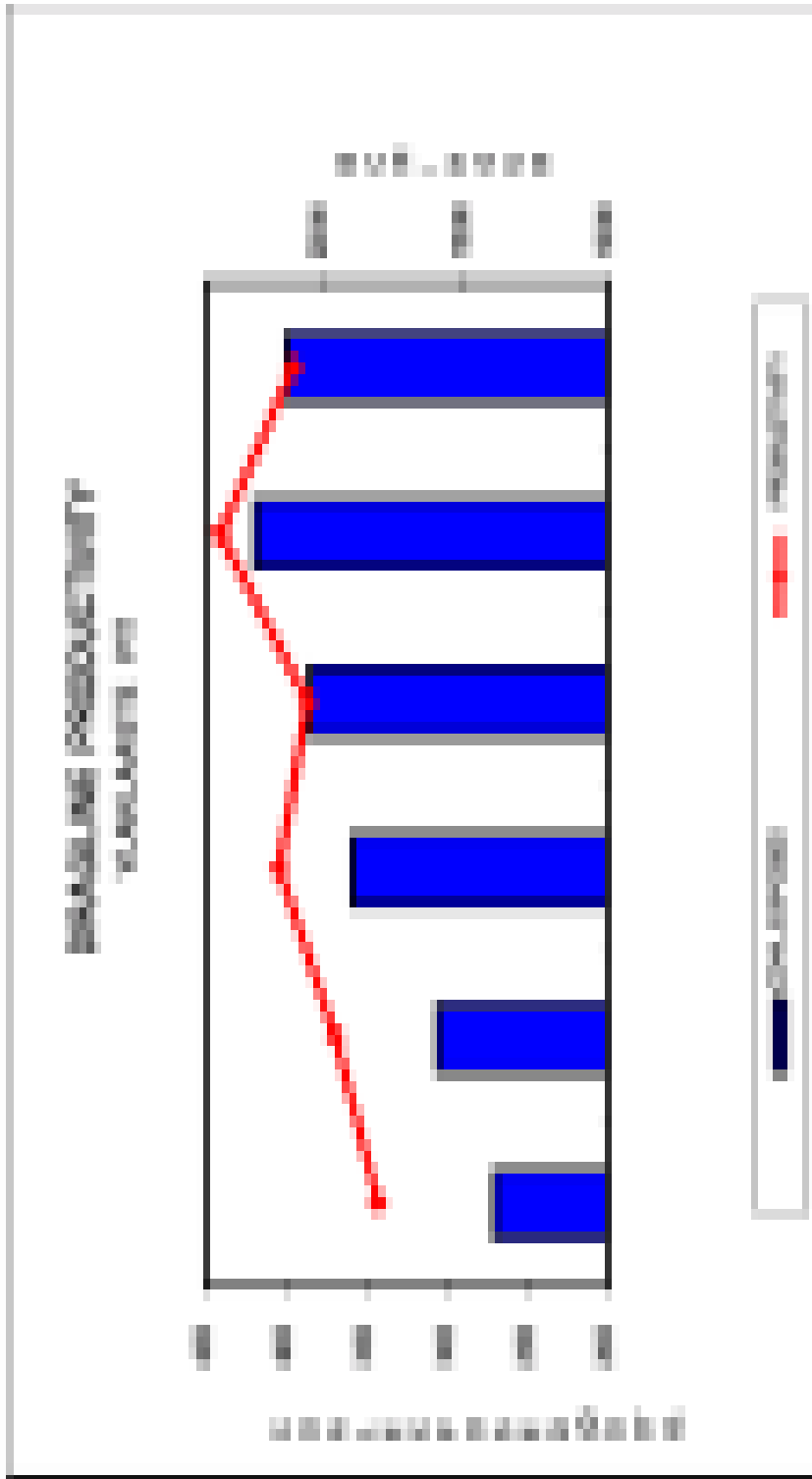


FIGURE 5

DRAGLINE PRODUCTIVITY

The task team identified three areas with the most significant impact on dragline productivity:

- Drilling practices
- Blasting efficiency
- Dragline methodology

Drilling Problems

After investigation, the following problems were identified:

Field Plans

There were no formal drill plans, which resulted in poor hole placement and lack of depth control. This in turn caused inefficient explosive distribution and therefore the fragmentation and blast cast¹ -was inadequate.

Reconciliation

There was no means of checking the correctness of hole depths, spacing or burden because there were no drill plans to refer to.

Re-drilling

There were no procedures in place to ensure cleaning of drill chipping² around the blast holes, with the result that backfilling occurred without being monitored. Blast holes were not closed for protection when left uncharged for lengthy time periods and chipping was washed into holes during rain showers.

Front row burden

The negative impact of large front row burdens was ignored. Because of this easier holes³ were not drilled and casting and fragmentation of the rock was adversely affected.

Figure 6 shows the effect of poor drill control, where the pre-

split line for the midburden bench no longer has the required 2.5-metre standoff distance from the highwall.

To rectify the above problems the following steps were implemented:

Drilling Solutions

Wire - line logging

Prior to in-fill drilling, every sixth pre-split blast hole (i.e. every 24 meters) has to be wire-line⁴ logged in order to provide exact depth to top of coal, as well as strata identification. (Location of sandstone bands etc.)

Minescape plans

The wire line data is incorporated into the Minescape model⁵ with other relevant inputs such as drill pattern and standoff distances. Geological information in the form of seam elevations and dyke and sill location, are also input. Minescape generates an accurate drill plan from which the surveyor places control lines in the field.

Survey control

Because the surveyed positions of the highwall and toe are shown on plan, easier holes are accurately placed to reduce front row burdens. Previously measured coal horizons are used in the model to further improve the accuracy of predicted hole depths.

Strata logging on drill rig

Where the Ingersol Rand DMM2 drill is used, the on-board stratalogger⁶ data is incorporated into the model.

Reconciliation

Every drill plan makes provision for two measurements of drill depths, one by the drill crew and the other prior to charging, by the blasting technician. The completed plan is kept on file for reference. The result is that re-drills are identified before charging.

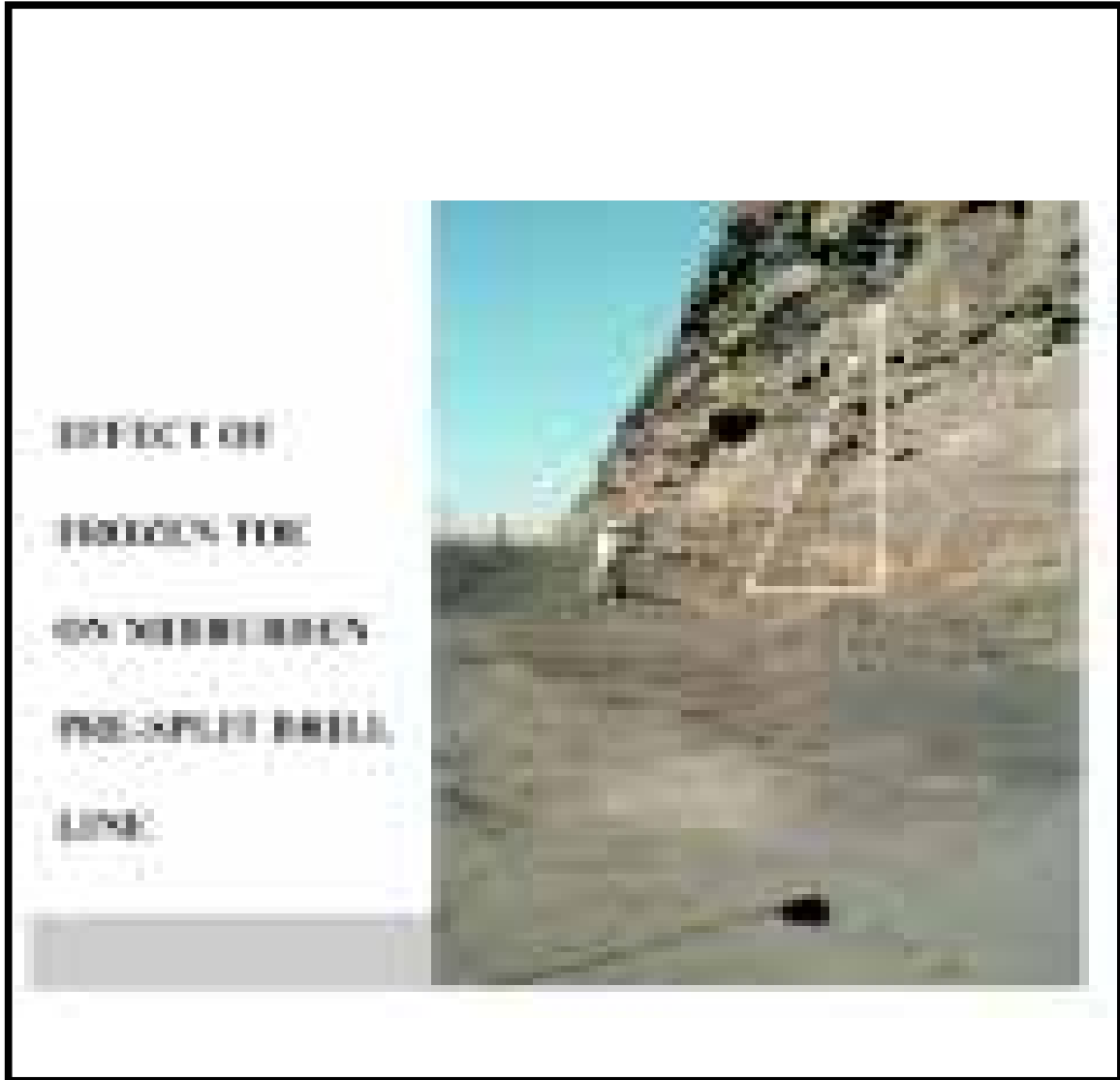


FIGURE 6

The closer supervision has resulted in the implementation of standards such as cleaning of hole perimeters as well as covering of holes by means of plastic bags. Figure 6 shows a Minescape generated drill plan with initiation tie-up and delays for the blast.

The plan shows hole positions with data boxes, which are used for recording original planned depth, actual drilled depth and the re-measured depth prior to charging. The tie up configuration⁷ indicated on plan is an additional and user friendly feature, which assists field personnel in correctly tying each blast. Any changes made to the planned tie-up and accessories are shown (and countersigned) on the plan for reference.

The red arrows indicate an inter-hole delay of 17 milliseconds per hole. The reason for this delay period is to ensure that the holes detonate at separate intervals. This reduces air blast and ground vibration and is used when blasting takes place close to surface and underground structures. The grey arrow shows the delay period between rows and forecast blasting. The delay periods are as follows:

Row A to B = 125 milliseconds`
Row B to C = 125 milliseconds
Row C to D = 125 milliseconds
Row D to E = 125 milliseconds
Row E to F = 75 milliseconds (row adjacent to pre-split)

Blasting Problems

Having successfully implemented the drilling program over a period of three months, the next step taken was to focus on the blasting operation. Certain areas of dragline performance such as frequency of double digs⁸, hard digging and capping on coal, indicated that blasting efficiency could be improved. Other areas of concern were:

Sub-standard blast design

The absence of drill plans required field personnel to adapt blast designs to suit existing conditions. For example, tie-ups varied from blast to blast and since there were no records kept, it was extremely difficult to identify problem areas and to ensure repeatability

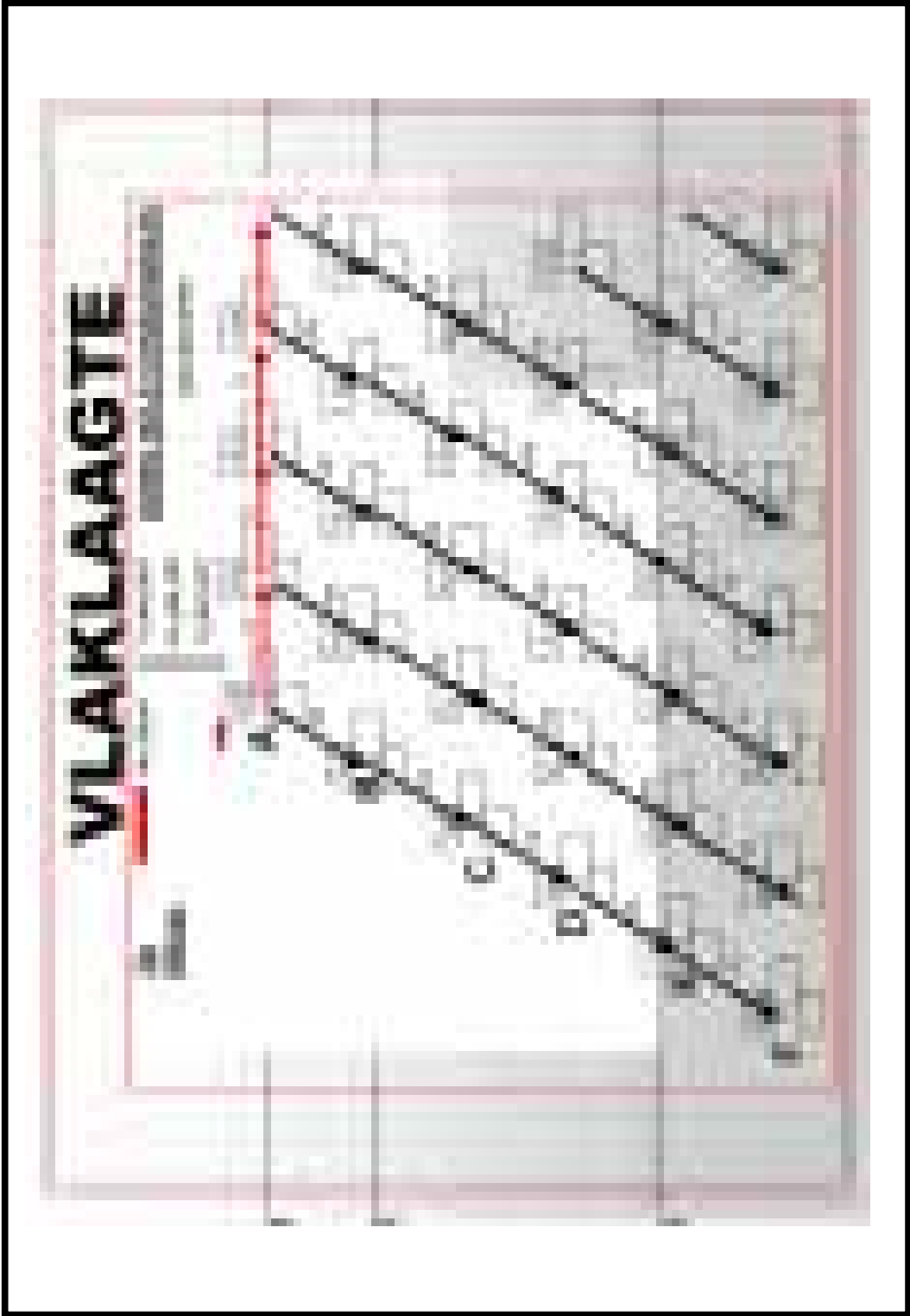


FIGURE 7

of good blasts.

Timing

Although an effort was being made to cast blast, no records of high-speed photography or timing of face movement were available. Furthermore, cast blasting of the midburden was being attempted despite the inherent width to height ratio⁹ not being suitable for cast blasting. This caused unnecessary explosive costs to be incurred.

Frozen highwall

Frequent freezing¹⁰ of overburden highwalls indicated ineffective blast design.

Midburden backbreak

The 2,5 meter stand-off distance¹¹ from the pre-split¹² on the midburden resulted in excessive backbreaking. This caused unstable highwall conditions and increased toe burdens for the next strip. The photographs below illustrate the extent of the excessive toe burden and back-break at the time of the investigation.

Figure 8 indicates that the front row burden for the next blast will be double the required distance. The unstable highwall prevents the drilling of easer holes, which would to some extent reduce the toe burden. The overall effect is that cast blasting is reduced by as much as 43 %; that is, from a 35 % desired cast, down to 20 %.

Figure 9 clearly shows the absence of pre-split blast hole barrels. This is normally the result of too little standoff distance between the last row of blast holes and the highwall pre-split line.

Having identified the problem areas, the following corrective actions were taken:

Blasting Solutions

Blasting Consultant

FIGURE 8

FIGURE 9



A reputable Consultant was employed to assist with the development of a standard design for both overburden and midburden, suited to the current conditions. The Consultant was also tasked to analyze burden movement with the aid of high-speed photography and to obtain a "T-min"¹³ which could be utilized in the timing of rows to improve blast cast.

Revised spacing and burden on midburden

Calculations showed that the existing midburden powder factor¹⁴ of 0.5 kg/m³ could be reduced to less than 0.4 kg/m³ due to the move from cast blasting to in-place blasting in this area. The cast blast design curves given in Figure 10 clearly show that the current midburden height of 12m is insufficient for effective cast blasting. The 8.5 metre x 9 metre drill pattern was not changed. However, the explosive product was changed from a dense emulsion to a less dense anfo, which could still optimise the full column rise¹⁵.

The overburden drill pattern was changed from 8.5 metres x 9 metres to a smaller 7 metre x 9 metre pattern, resulting in an increase in powder factor from 0.55kg/m³ to 0.75kg/m³. The increased column rise, together with optimised energy distribution, has resulted in improved fragmentation as well as casts of up to 40%. These results conform to the predicted parameters shown in figure 10 for depths in excess of 30m. The overall explosive consumption remained unchanged as a result of redistribution from midburden to overburden. However the blast-over¹⁶ increased from 25% to 35-40% on overburden and as shown later, the improved fragmentation had a positive effect on dragline performance. Figure 11 shows the effectiveness of the redesigned overburden criteria, where a cast of 40% was obtained.

Sandstone capping

One of the management innovations, which proved to be effective, concerned capping above the 4 seam horizon. The analysis showed that it was necessary to intersect the coal seam with the blast holes, but no back filling was to be done. The purpose is to ensure that the competent overlying sandstone band is fragmented to

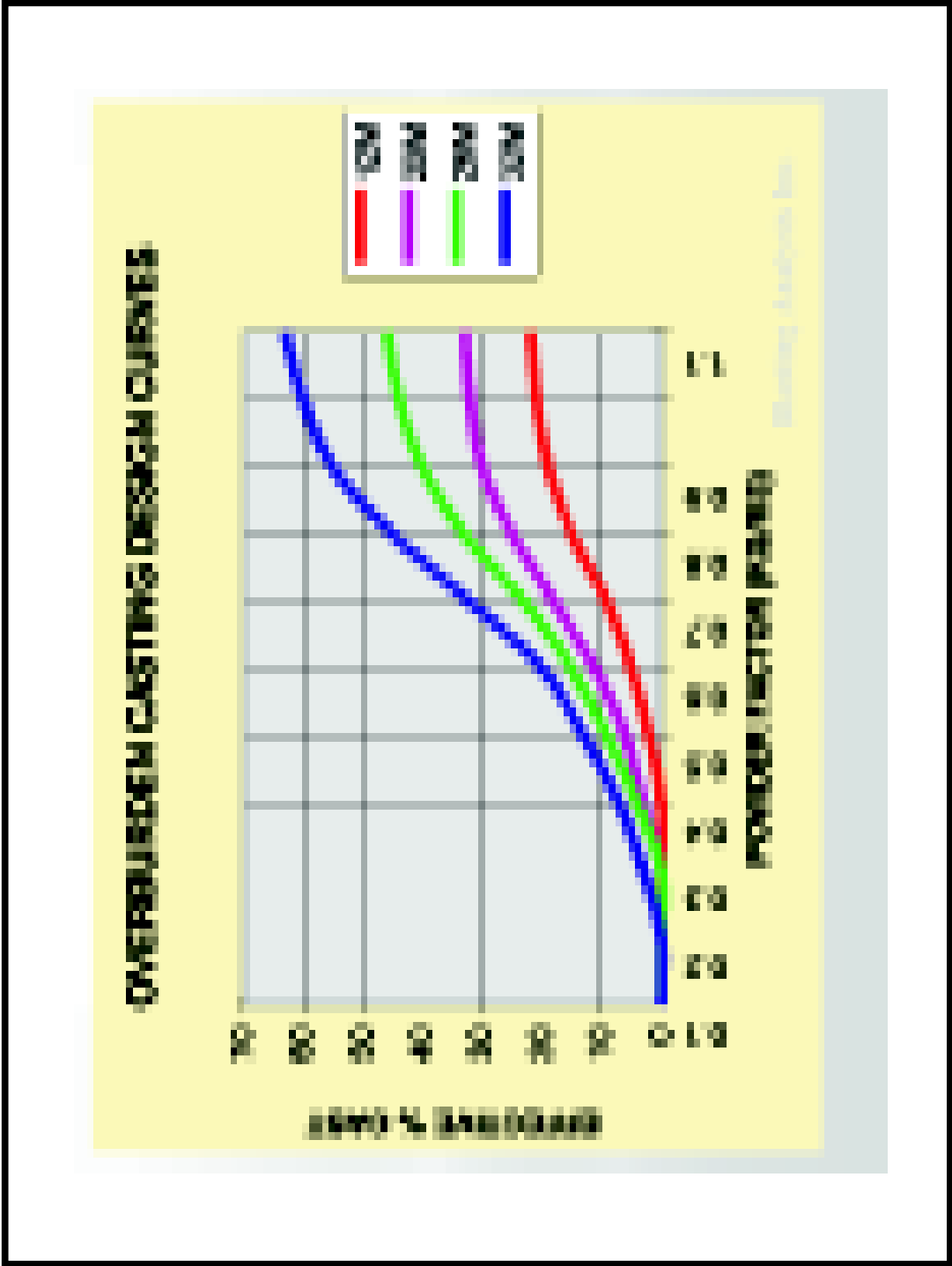


FIGURE 10

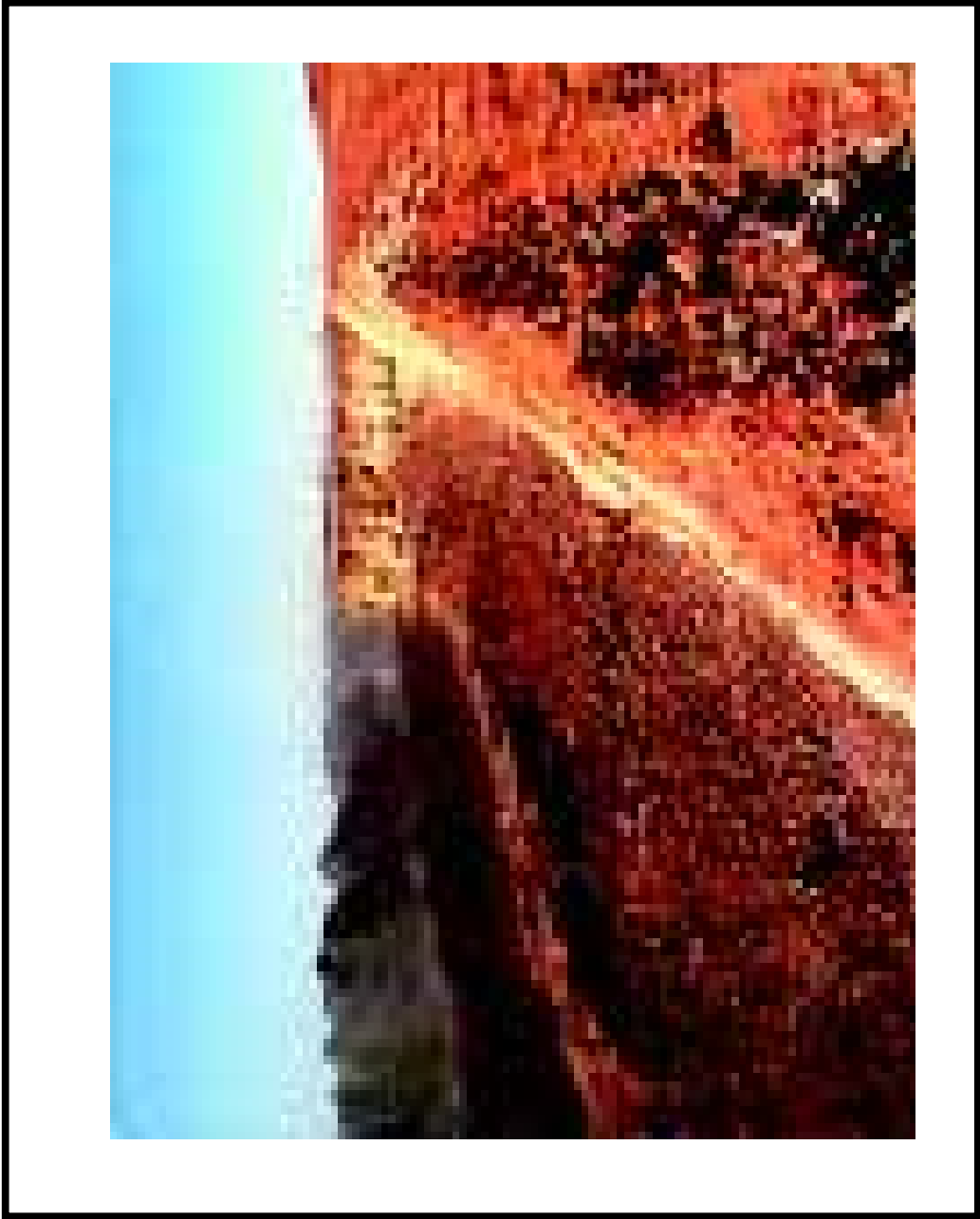


FIGURE 11

top of coal. In those areas where shale overlies the coal seam the above practice is not recommended, because it will result in extensive coal damage.

Pre-splits

The overburden pre-split design (4 metre spacing at 2,5-metre standoff) remained unchanged because of the predominant sandstone strata, but the holes are drilled to the floor of the coal seam. To maintain a clean highwall, concerted efforts are made to ensure correct placement of the explosive charge in the sandstone and not in the softer materials, as shown in Figure 11. This shows air bags and retaining rope attached before the bags are charged and primed and then lowered to the pre-determined depth. The depth is determined from both tape measurement and Wire-line logging data.

The effect of tight control on pre-split charging and depth placement is clearly shown in Figure 13 below:

The midburden pre-split design was changed with regard to spacing and standoff distances. To prevent back-break through the pre-split line the standoff distance is 2.5m. To create an effective split on the shallow midburden the spacing was reduced from 4 metres to 3 metres between holes. Figure 14 shows the effectiveness of the redesigned presplits. Note the visibility of pre-split barrels. (Compared to figure 9).

Dragline Problems

While the drill and blast investigations were being done, studies to improve dragline productivity were also carried out. Initial efforts were directed primarily at the dozer pushover¹⁷ operations and then later extended to include the performance of buckets, excavator assistance, operators and computer monitoring systems and the installation of a DC 2000¹⁸ system.

Dozer push over

It was found that final bench elevations were too high. This was caused by inadequate elevation control and resulted in rehandle of up to 30% for the dragline, whilst at the same time the dozer



FIGURE 12

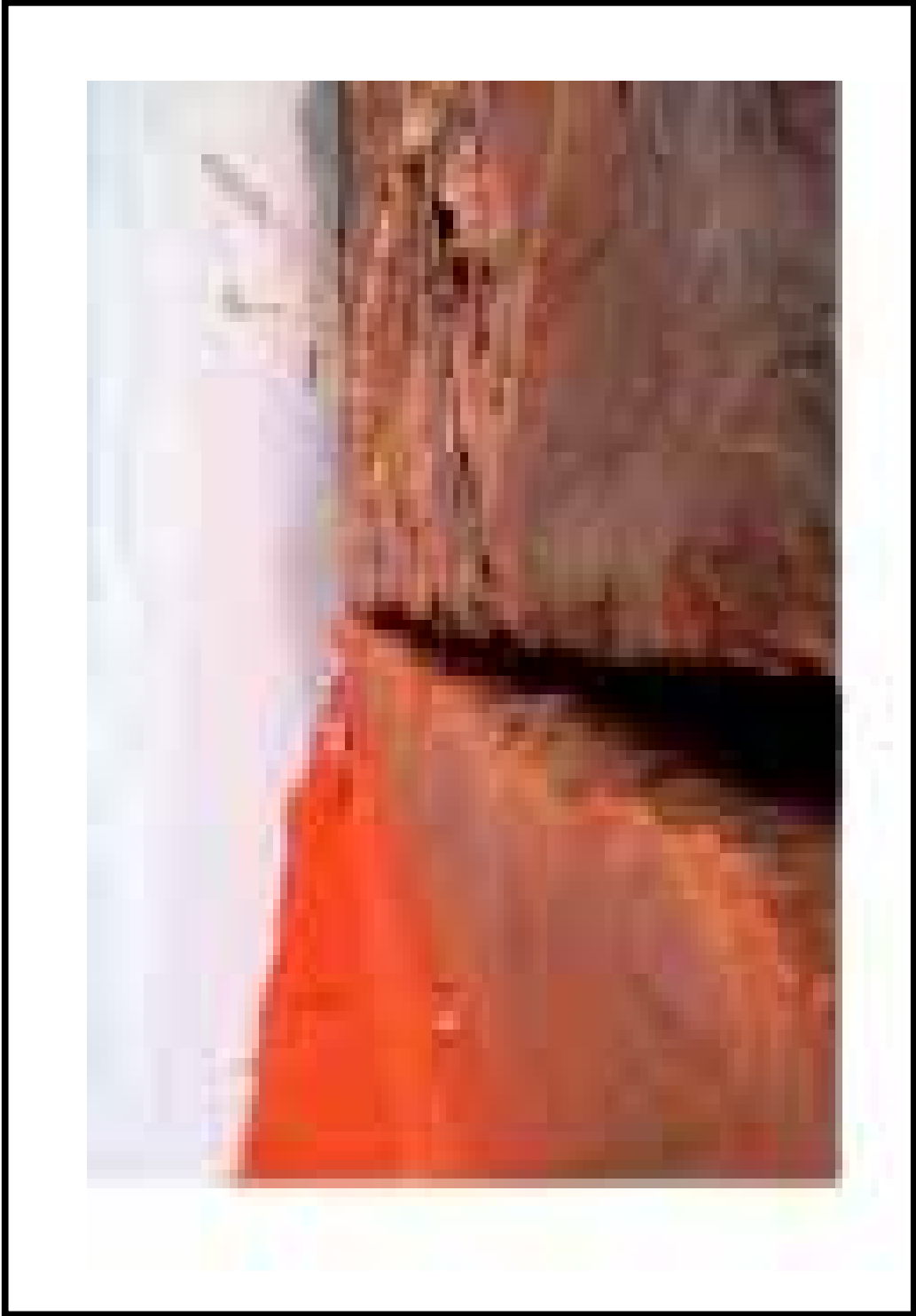


FIGURE 13 - EFFECTIVE PRE-SPLIT



FIGURE 14
PRE-SPLIT BARRELS ON OVERBURDEN AND MIDBURDEN

advance was well ahead of the dragline. Because the dozers were far ahead, they were used for other applications. The overall effect was that dozer capacity was under utilised in establishing a lower bench and the dragline was left to handle extra material. At this time dozer productivity was 250 Bcm/hr.

Buckets

Although the benefits of using a lightweight bucket¹⁹ were well known in the industry, Douglas Colliery was still using the smaller 56m³ standard buckets. This was mainly due to the existing digging conditions. The progressive optimisation of the drill and blast operations opened the way for the application of a lightweight bucket.

Highwall chop down

The double bench stripping²⁰ method requires a chop down²¹ operation on the overburden high wall, with the dragline positioned at the 2 seam key cut²². Although this method was effective, it resulted in rope damage and where hard digging was experienced against the highwall, productivity dropped. This was aggravated by use of the lightweight bucket.

Operators

Skill base and performance levels of the Douglas dragline operators appeared to be below Ingwe Group standards.

Systems

Contel system²³

This on-board facility was not being utilised and no alternate monitoring system was available.

Pit Boss

This software produced Range diagrams²⁴, which were supplied by planning personnel on a monthly basis only and with very little input from operations. A need existed for much shorter-term planning and control (ownership of design application) by field personnel.

The manually set electrical and mechanical drive systems on the dragline were not always clearly defined. As a result operating speeds could not be optimised within allowable limits. Clearly, the dragline was not operating at optimal capability and during the course of the investigation it became evident that the dragline's performance could be substantially improved. The solutions to the above problems were implemented as follows:

Dragline Solutions

Dozer push over

To maximise the dozer fleet productivity, an optimal bench elevation had to be defined without compromising dragline performance. Range diagrams were produced at various depth intervals to establish an ideal balance between dragline performance and dozer bench elevation. A computer program "Pit-Boss" was used to generate and evaluate range diagram data, which was incorporated into a spreadsheet. From this a diagram was generated on the different depth contours to establish optimal bench elevation. Figure 15 is an example of a combined dragline and dozer performance curve relative to dozer bench height above 4-seam elevation.

The X-axis represents the distance, in metres, above the 4-seam coal and the Y-axis is the number of dozers required. For example, if the selected bench height is 5 metres above 4 seam, then for 3 Dozers the calculated dozer productivity is 450 Bcm/hr (top line). This indicates a dragline productivity of 2483 Tcm/hr and a forecast coal exposure of 475 508 tons per month. For a bench 3 metres above 4 seam and for the same number of dozers, coal exposure increases to 536 959 tons per month. However, dozer productivity must increase to 600 Bcm/hr and dragline performance will drop to 2267 Tcm/hr as a result of increased dump height. This exercise was carried out over the full pit length of 3 km. By applying the correct dozer bench elevation it was found that the dozer productivity has increased from 250 Bcm/hr to in excess of 450 Bcm/hr. Furthermore, to remain ahead of the dragline the dozers are fully dedicated to push-over. This means that the dozers are no longer available for other, unproductive, applications. Figure 16 shows the pushover operation 200 metres ahead of the dragline.

Buckets

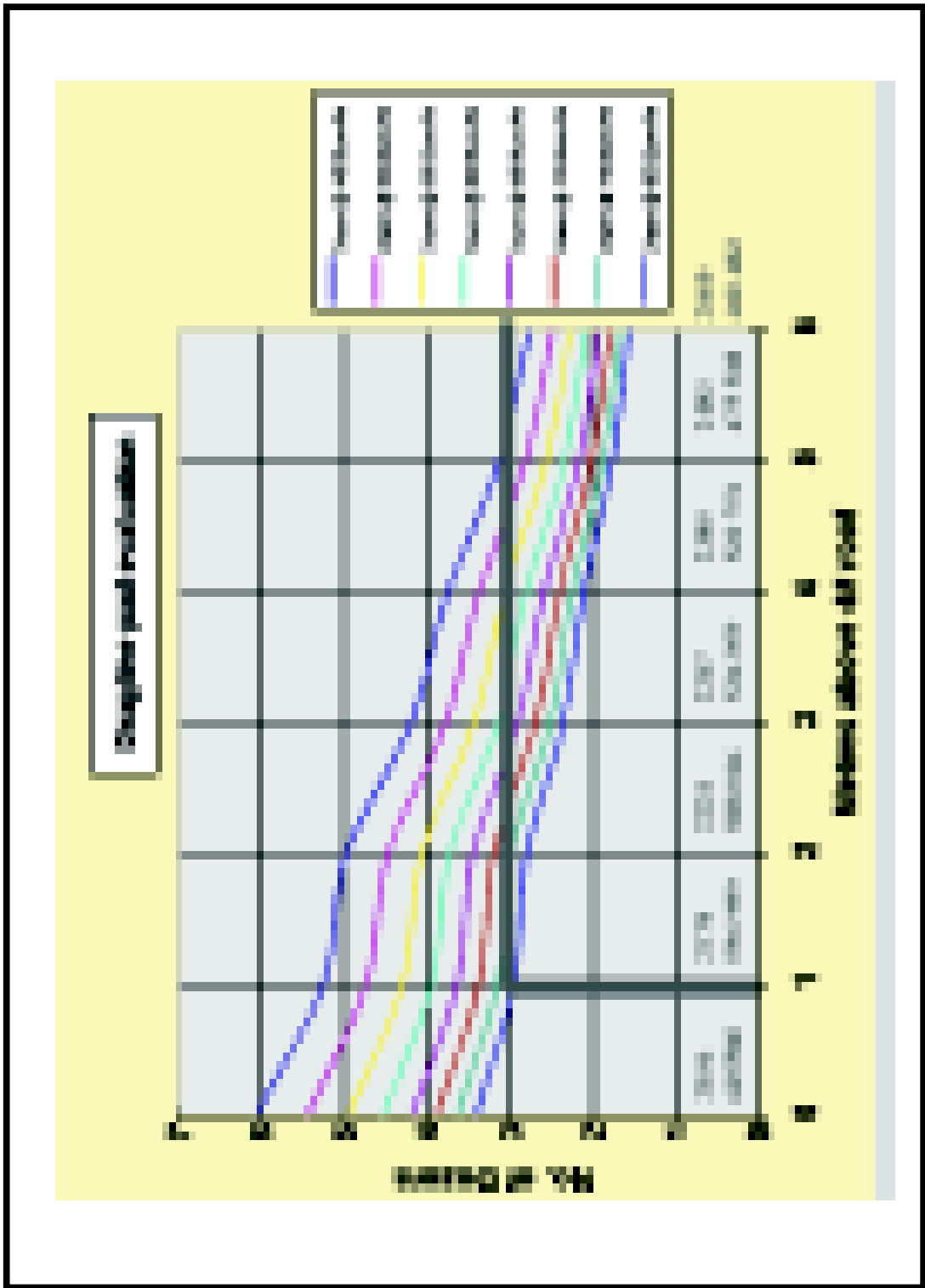


FIGURE 15

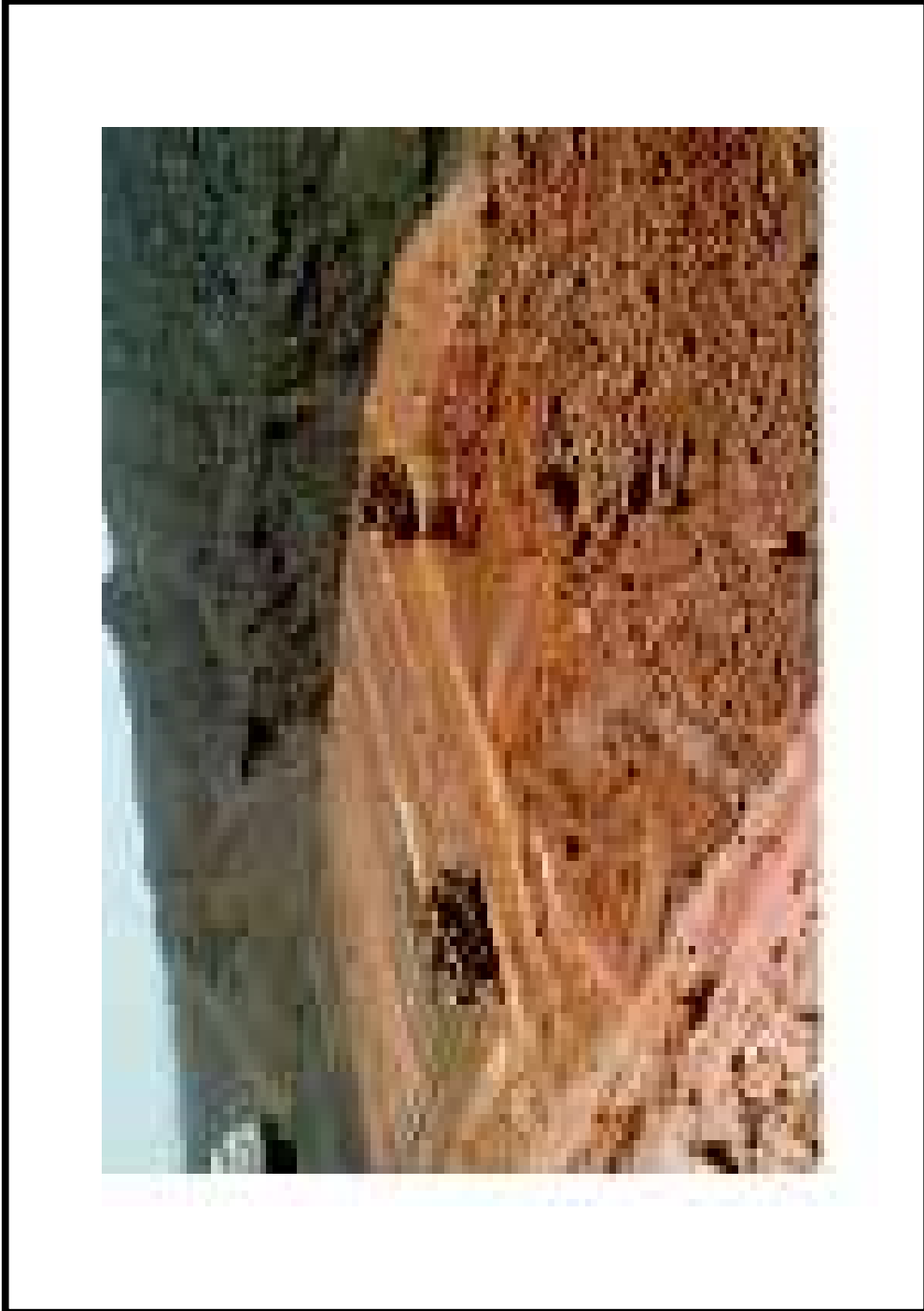


FIGURE 16
DOZER PUSHOVER

The improved digging conditions favoured the purchase of a Wright lightweight bucket. The new bucket has a rated capacity of 61 m³ - an increase of 8.9% over the standard bucket. This bucket has been used since December 1997 and although the precise change in productivity could not be established, it is estimated that performance improved by 100 BCM / hour. This is an increase of 5 % over previous performance. Figure 17 shows the two types of buckets.

Lessons learned with the lightweight bucket are:

- continual hard digging results in extensive damage (ripping out of lip)
- bucket fill is reduced in tight digging conditions (the bucket tends to skip)
- More susceptible to damage on highwall chop-down than standard bucket

Highwall chop down

The highwall chop down was causing damage to the buckets. To minimise this damage it was necessary to carefully maneuver the bucket, which led to slower cycle times and decreased rope life. A study was done to determine the viability of applying an excavator to cut a slot on the highwall. The purpose was to create a void for the bucket to cut into. The proposed method was tested using a hired excavator, comparing 7 identical, 100 metre long panels over two strips. The hired machine was used on the second strip. The results obtained are shown in Figure 18.

Coal exposure has increased by approximately 300 000 tons per year. This is conservative because the comparison took place in the deeper, slower part of the pit. Figure 19 shows the highwall slot created by the excavator.

The excavator also creates a safe batter on the soft overburden and cleans loose material off the pre-split line. The application is described below.

Figure 20 shows three steps:

STEP 1

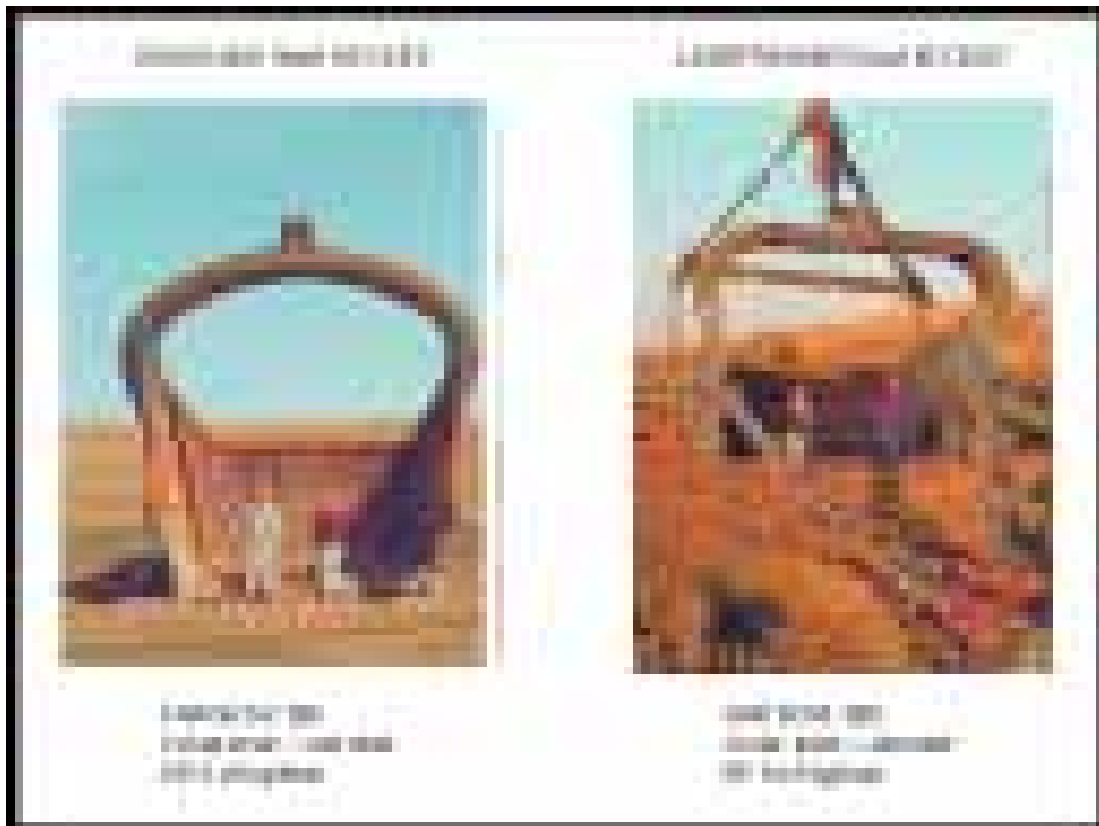


Figure 17

	Pre-Excavator Method	Excavator Method	Variance	%
Days taken to complete	36	31	-5	-14%
Strip width	45	45	0	0%
Total Bcm's	1307634	1284562	-23072	-2%
Total coal exposed	323904	304684	-19220	-6%
Hours taken	735	623	-112	-15%
Bcm/hour	1779	2062	283	16%
Coal exposed per day	8997	9829	831	9%
Coal exposed per annum	3239040	3538266	299226	9%

FIGURE 18



FIGURE 19
HIGHWALL SLOT

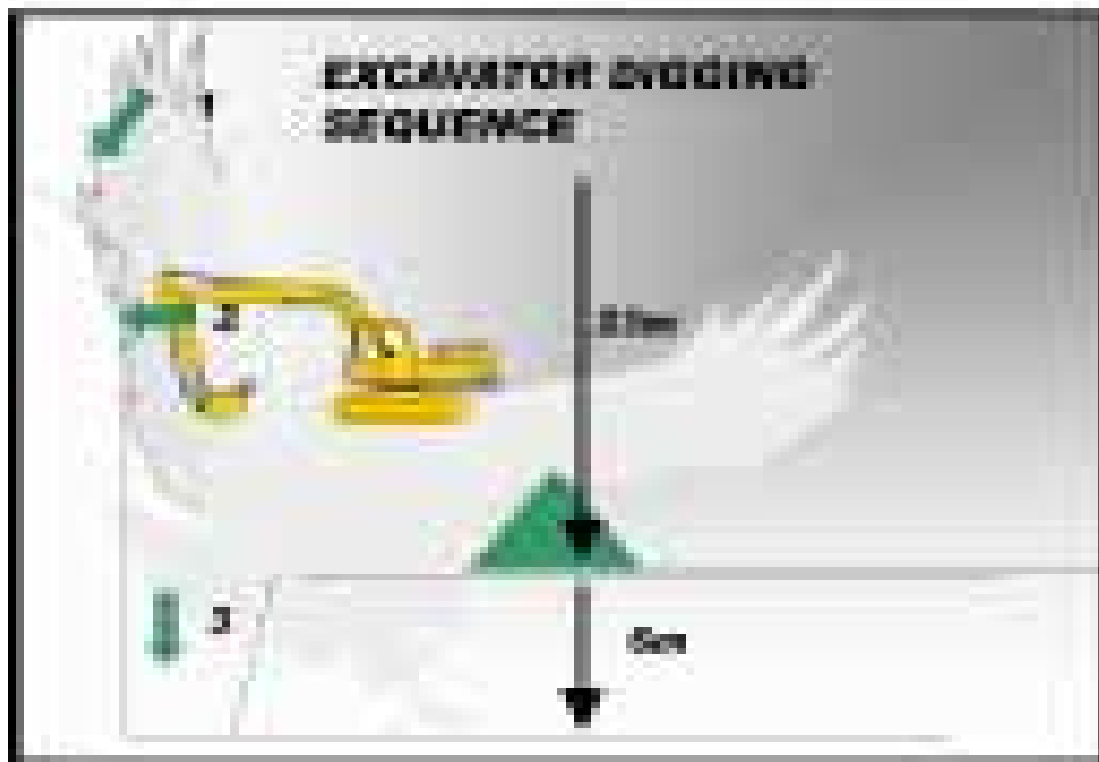


FIGURE 20

After the overburden blast, the machine stands on the highwall and removes soft material to a safe 1:2 batter angle.

STEP 2

The machine moves in-pit onto the blasted overburden and "brushes" any loose material from the highwall.

STEP 3

The excavator digs the highwall slot and places the material on the spoil side of the bench. Note that the machine is placed at a right angle to the highwall. This is a safety measure to ensure that the operator and his machine are as far from the highwall as possible.

Operators

The Contel system was used to monitor operator performance on a shift by shift basis over a 3-month period. This exercise, based on swings and cycle times, exposed a problem with the performance of one operator and also showed up shortcomings in the performance of the others. The system is now being used to include other performance related issues such as swing angles, bucket fill times and double digs. The following two graphs, Figures 21 and 22, illustrate the improvement obtained after three months of monitoring. (One operator was re-located to training).

Figure 22 shows that the average swings increased from 300 per shift before monitoring to 400 per shift after 3 months of monitoring.

Systems

Video Monitoring

A video camera was installed on the dragline in May 1999 to monitor the digging cycle. Initially this action was not well received by the operators because it was perceived to be a method of finding fault with their performance. This attitude changed when a set of drag ropes broke. The video playback clearly showed that the operator was not at fault and subsequent investigation proved

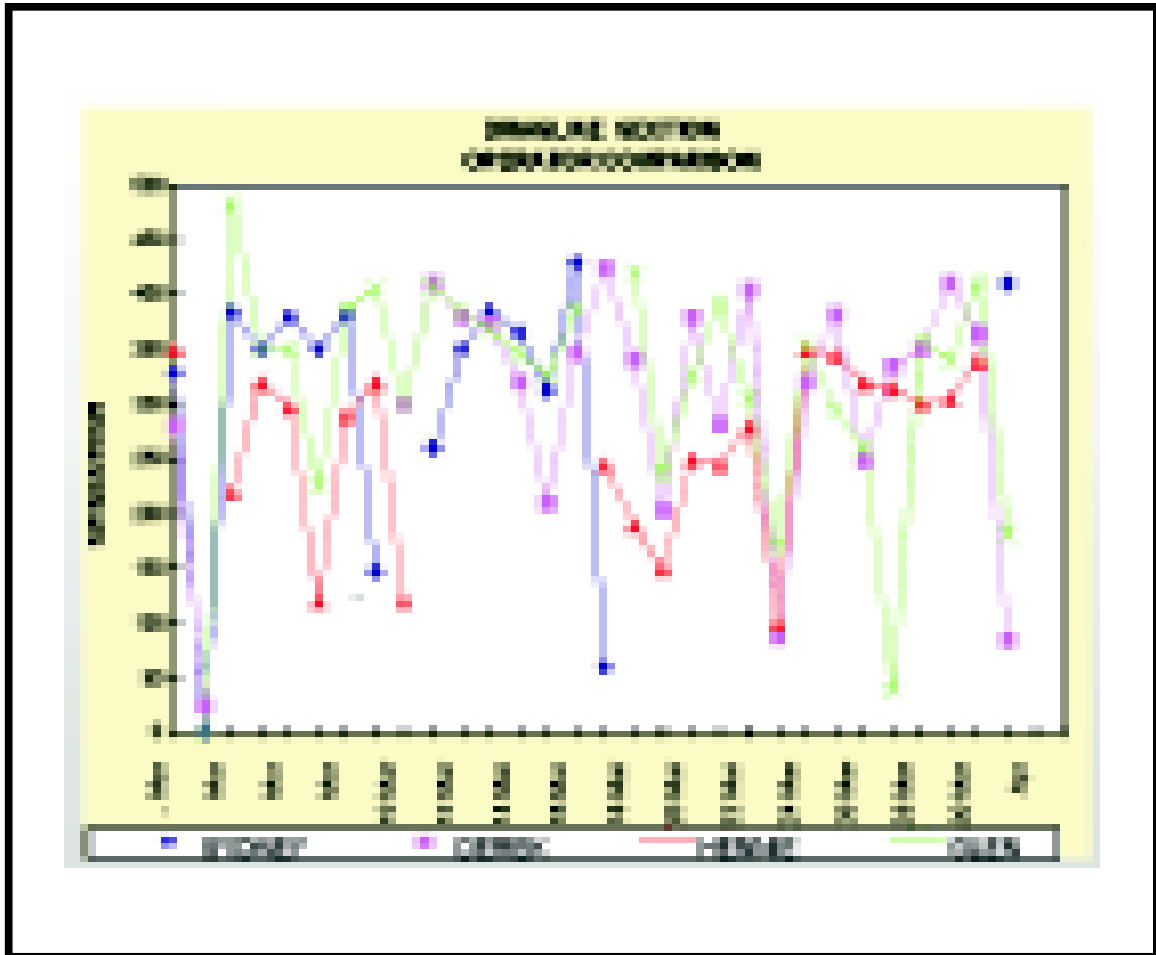


FIGURE 21
OPERATOR COMPARISON BEFORE MONITORING

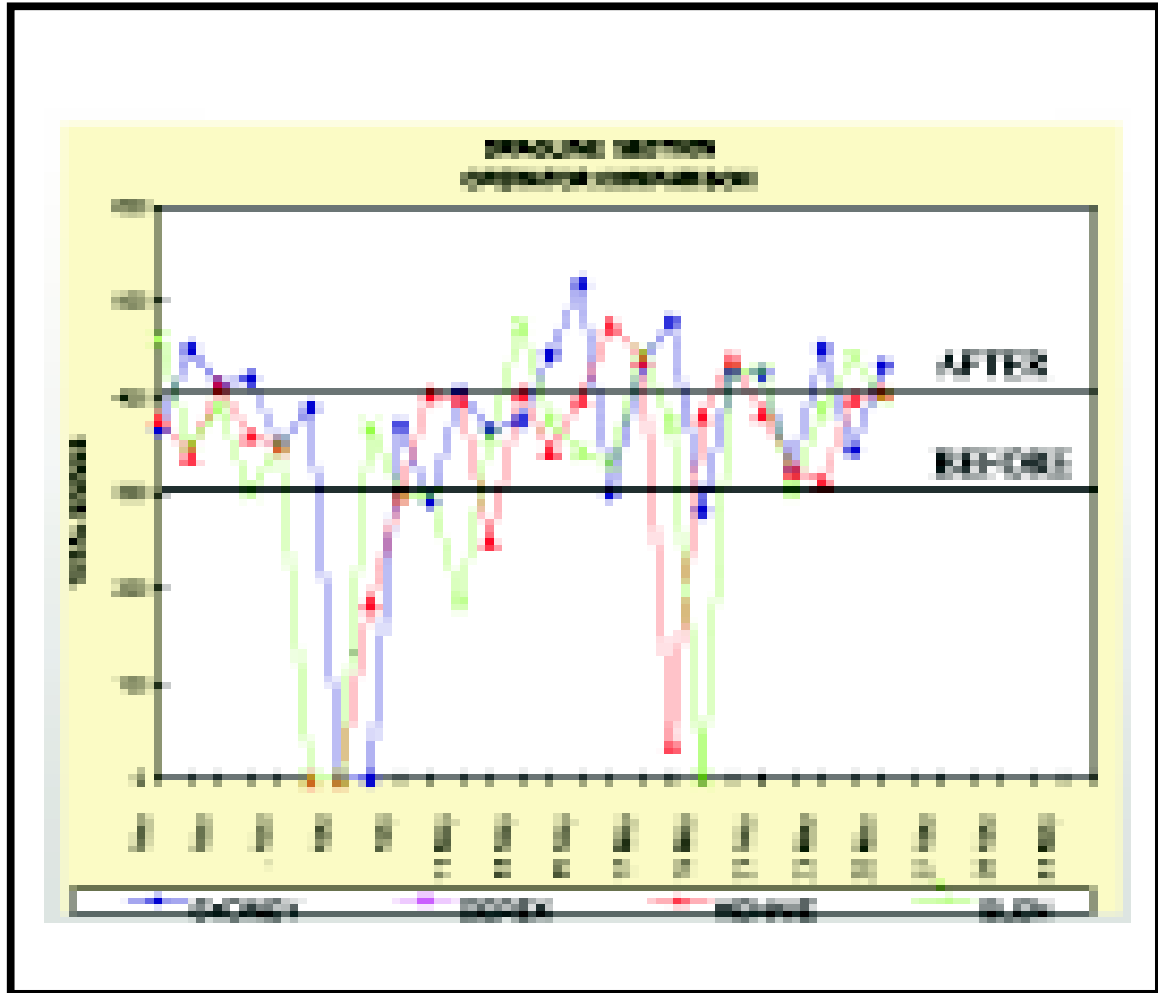


FIGURE 22
OPERATOR COMPARISON AFTER MONITORING

that the ropes were from a defective batch. Without this evidence the operator may well have been suspected of causing the damage through poor operating procedure. The operators now recognise the benefit of the camera and realise that by working correctly they will not be blamed for poor performance if something goes awry. The following advantages have been identified since installation:

- The dragline superintendent can view a 24-hour playback to assess operator performance and technique
- Remedial action can be taken within 24 hours to rectify shortcomings in technique
- Digging conditions are continuously monitored therefore blasting efficiency can be assessed
- Climatic conditions are clearly shown and reduced productivity during dusty periods or thick mist can be instantly recognised
- The Time and Date recording facility allow the monitoring of engineering responses to breakdowns and therefore steps can be taken to reduce downtime
- Analyses of the recordings assist the engineers in determining causes of wear or breakdown, therefore guesswork is greatly reduced.
- Suppliers of drag ropes now realise that they have to improve their quality control because historically they were quick to blame poor rope performance on operating technique.
- The handling of the trailing cable during moves is monitored and follow-up training to rectify faulty handling is quickly undertaken.

Soon after the system was installed, many poor operating techniques were recorded. These were:

- Hitting the spoils with the bucket
- Hitting the highwall during chop down
- Placing the bucket down too hard on the coal seam
- Pulling the drag ropes through the blasted rock
- Swinging the boom before lifting the bucket
- Excess slack on the hoist ropes
- Excessive drag distances to fill the bucket

All of these poor operating methods contribute to reduced productivity and cause damage to the bucket and ropes. Re-training has reduced this and the graphs below indicate that bucket and rope life have

improved since installation of the video recording system. More data is needed to reach conclusive results. The video-recording unit has a 24-hour recording facility on a 3-hour tape and tapes are changed at the start of each shift.

The duration between major repairs is shown to have increased after installation of the camera. Although blasting fragmentation can influence the trend (and this will have to be taken into account where a change is recorded), management's perception is that the video monitoring is paying dividends.

The performance of the drag ropes in terms of volume of material moved does not appear to be very different before or after installation of the camera. The trend after video monitoring has a consistent upward movement and it will be interesting to see if this continues over time. Although the results to date are not conclusive, they are encouraging.

The camera is mounted on the machine house above the fairlead sheaves. The recorder is kept in the power control room, which is locked and therefore tampering with the recording unit is avoided. The camera mounting is shown below.

Benchmarking

A benchmarking exercise started in February 1997 as part of the strategic objectives detailed in the Ingwe business plan for 1996-1997. The strategy to increase coal to the export market prompted a re-evaluation of the stripping methods and application of new technology in the dragline operations. A team of 4 Ingwe personnel took 12 months to benchmark the Group's 17 draglines. The objective was to increase coal exposure and reduce operating delays by identifying the best practices currently in use. The physical collection of data was initiated through a comprehensive questionnaire.

The Douglas team increased dragline productivity by 6.7% for the period July 1997- June 1998. (From 1800Bcm/hr to 1922 Bcm/hr). At this point of the benchmarking program there were two dragline teams that reflected best practices during the 1998 year, one of these being the Douglas BE 1570 machine. The Dig Index for the best performing dragline on each of the four opencast mines averages 40.11. The worldwide benchmark index is recorded in the World Mining magazine as 38. The exercise highlighted

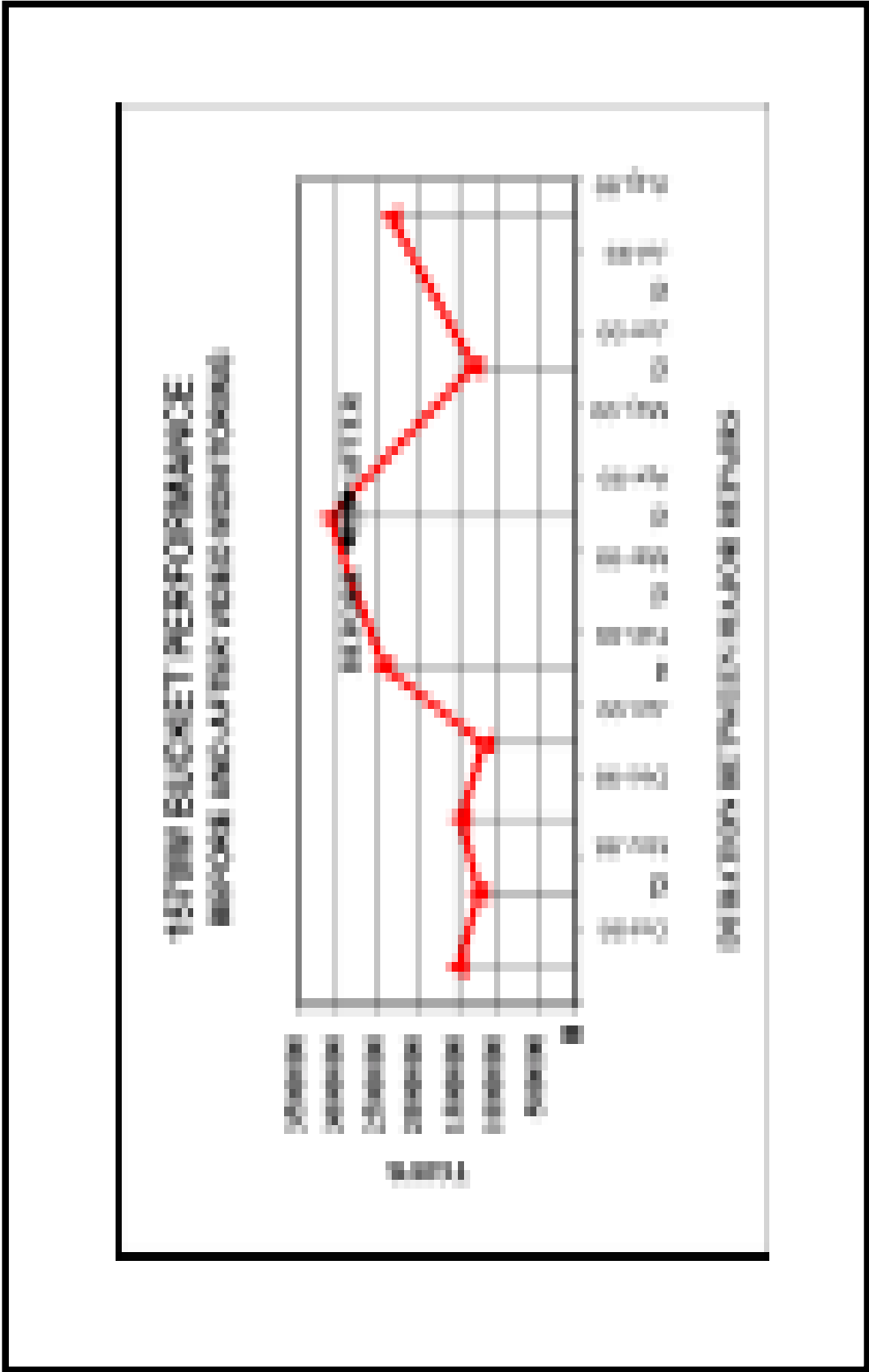


FIGURE 23

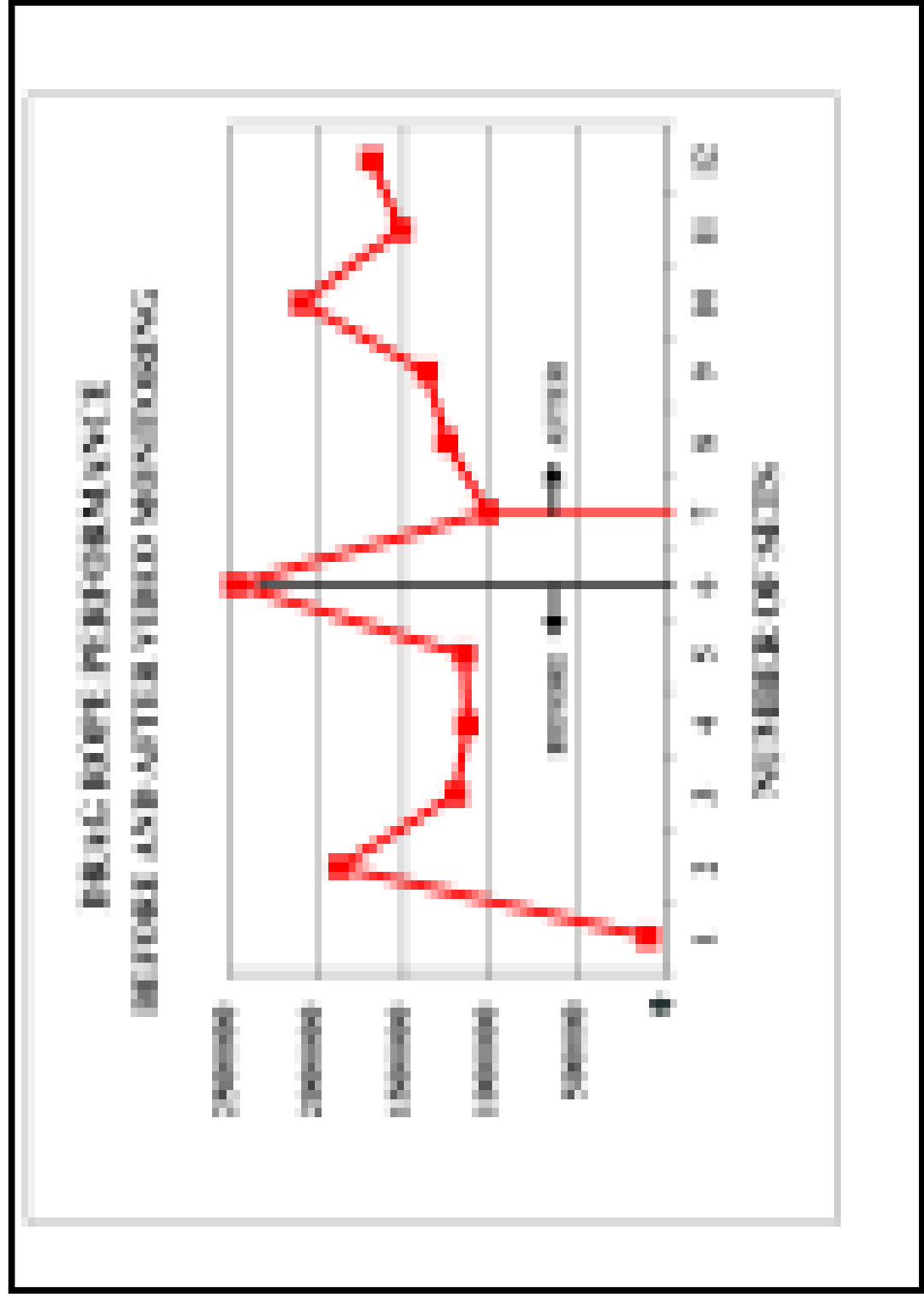


FIGURE 24



FIGURE 25

the fact that no individual dragline can display all the best practices but that benchmarking does make all data transparent to end users. The results of the benchmarking exercise show how the productivity improvement study at Douglas was bearing fruit, as indicated in the following graphs.

Dragline Productivity (TCMs/Hr)

Mining convention prefers a ranking of the best productivity rates in TCM's per hour for each mine. The top performers in this area are ranked as follows (figure26);

Dragline Productivity (BCM/ Hr)

Argument has been led that true effective production is realized when the bank volume is moved as opposed to the total cubic metres. The best performers for a BCM's per hour rate are as follows (figure 27):

From July 1998 to June 1999 the Dig Index has improved to 47.54. The production rate has increased by 23.2 %, from 1922 bcm's per hour to 2368 bcm's per hour.

Contel Monitoring system

The Contel system monitors all activities related to the dragline. Both engineering and production data is recorded and the output is given on a Windows based report format. Reports are customised and reflect issues such as swings, cycle times, swing angles, volumes, invalid swings, drag and hoist power used, split operational and engineering downtimes and fault finding. The system is linked by radio to a dedicated PC and is accessible 24 hours a day. The application of radio-pads enables an on-line system, which is networked to several PC's. Information is down loaded at the end of each shift for reporting.

'Pit Boss' (2-dimensional dragline cut simulation)

The Pit Boss software was re-introduced as a production tool whereby the foremen and overseer were trained to use the system on a weekly basis. The rationale behind this was to give ownership of production planning and control to the production personnel.

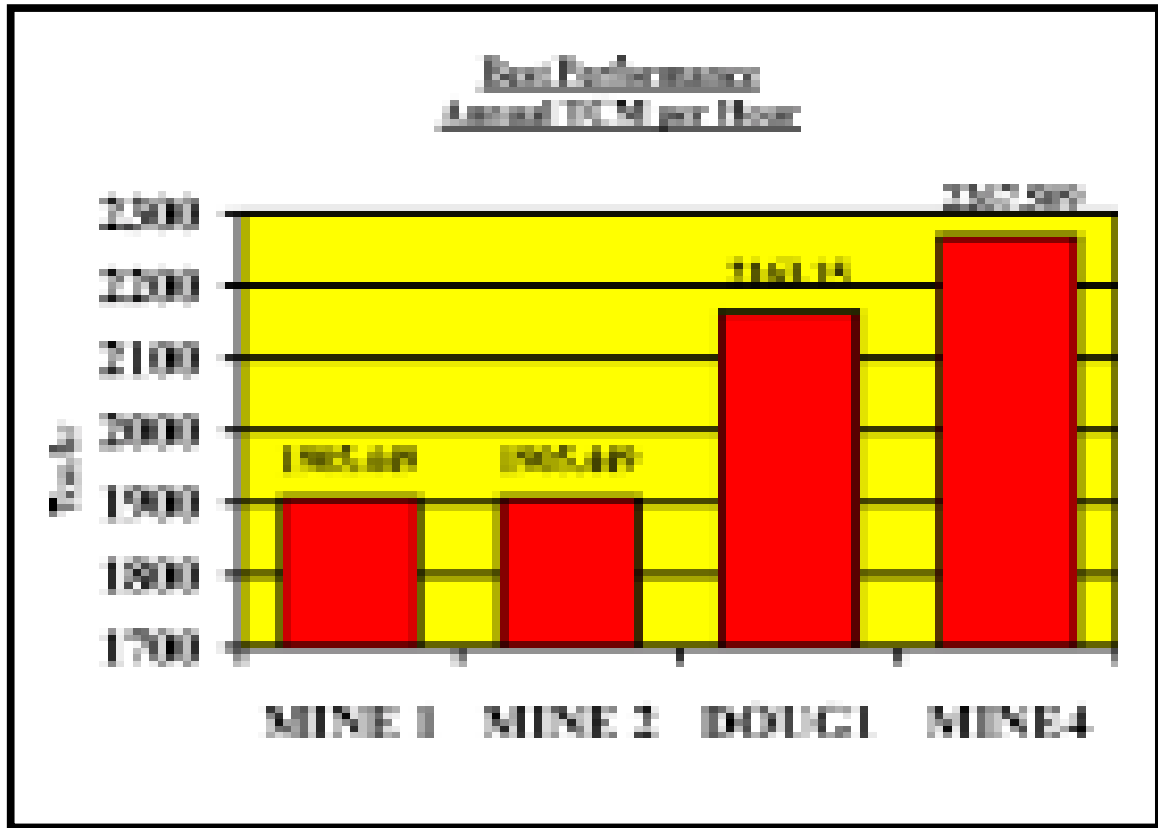


FIGURE 26

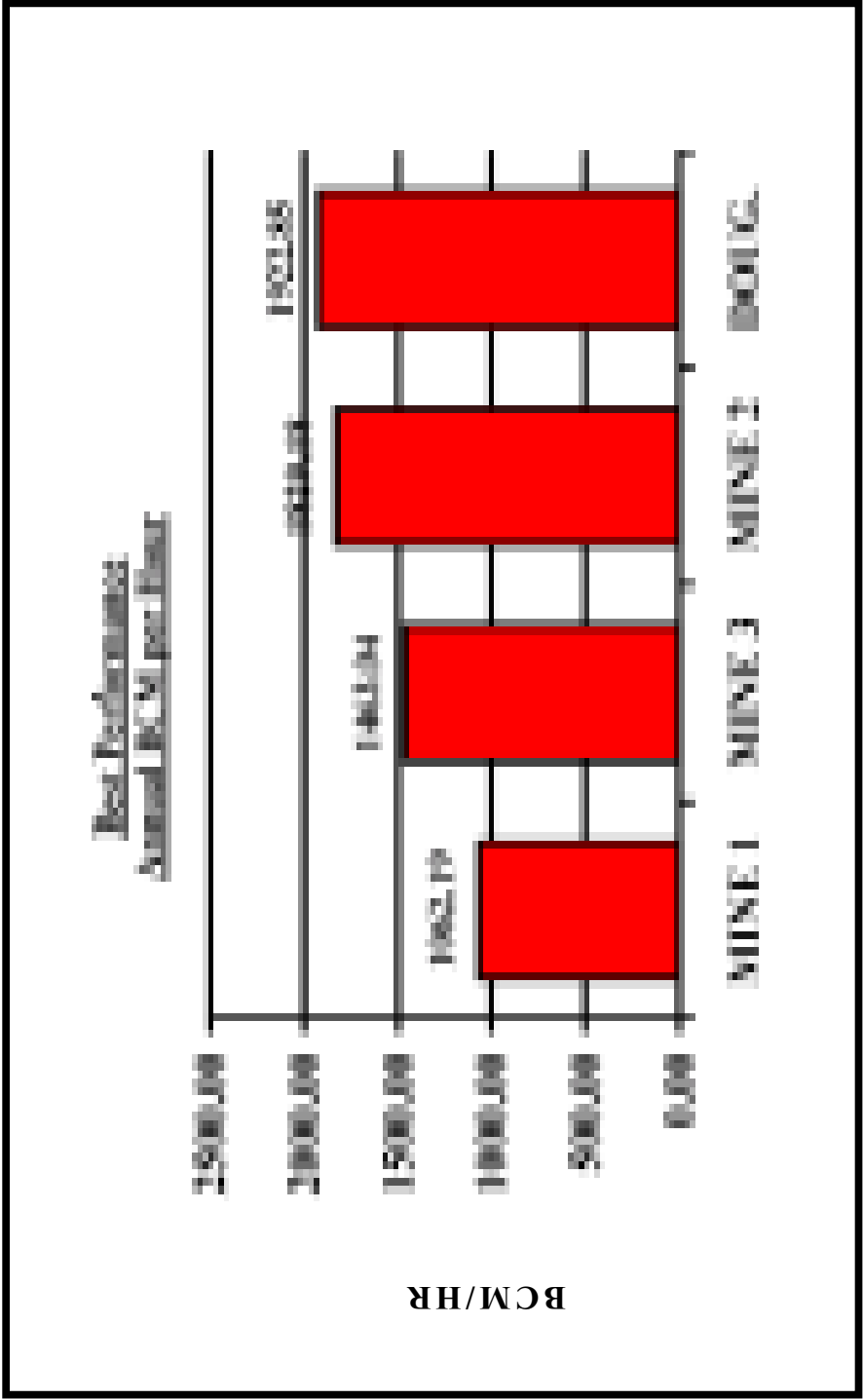


FIGURE 27

Together with the Planning department, a weekly target is set and then reconciled against a Survey measurement of actual production. Any necessary changes to the plan are then made for the following week. Figure 28 is an example of a range diagram created in "Pit Boss".

DC 2000 System

The DC 2000 System is extensively used overseas, but has had limited application in South Africa. The system was fitted to the Douglas machine when it was on a two-month major overhaul. The cost of the system was R 4.5 million Rands. This is the first retrofit DC 2000 in the industry. The operators were sent to the USA for simulation training for one month, to familiarise with the new operating mode. However, in practice the application proved to be more challenging than anticipated, with a decline in productivity recorded for January and February of 1998.

The DC 2000 System consists of 9 core drives, each drive being silicon controlled rectifiers (SCR). These drives are controlled by microprocessors. Through programmed logic control (PLC) the processors interface with the drives and genius blocks. This in turn controls all rotating equipment via MG sets, generators and motors. The acquired end result is an electronic optimisation of dragline motions i.e. drag, hoist, swing etc. A detailed description of the system is given in annexure 2. Experience with the DC 2000 at Douglas has highlighted the following advantages and disadvantages:

- Advantages
 - speed of motors is electronically controlled
 - motor speed is maximised from 750 rpm to 1150 rpm
 - the increased speed results in faster motions
 - motor and generator fields are controlled by core drives

- Disadvantages.
 - availability of spares
 - very expensive due to Rand / Dollar exchange rate
 - teething problems due to being the first retro-fit machine
 - long operator learning curve (> 5 months)

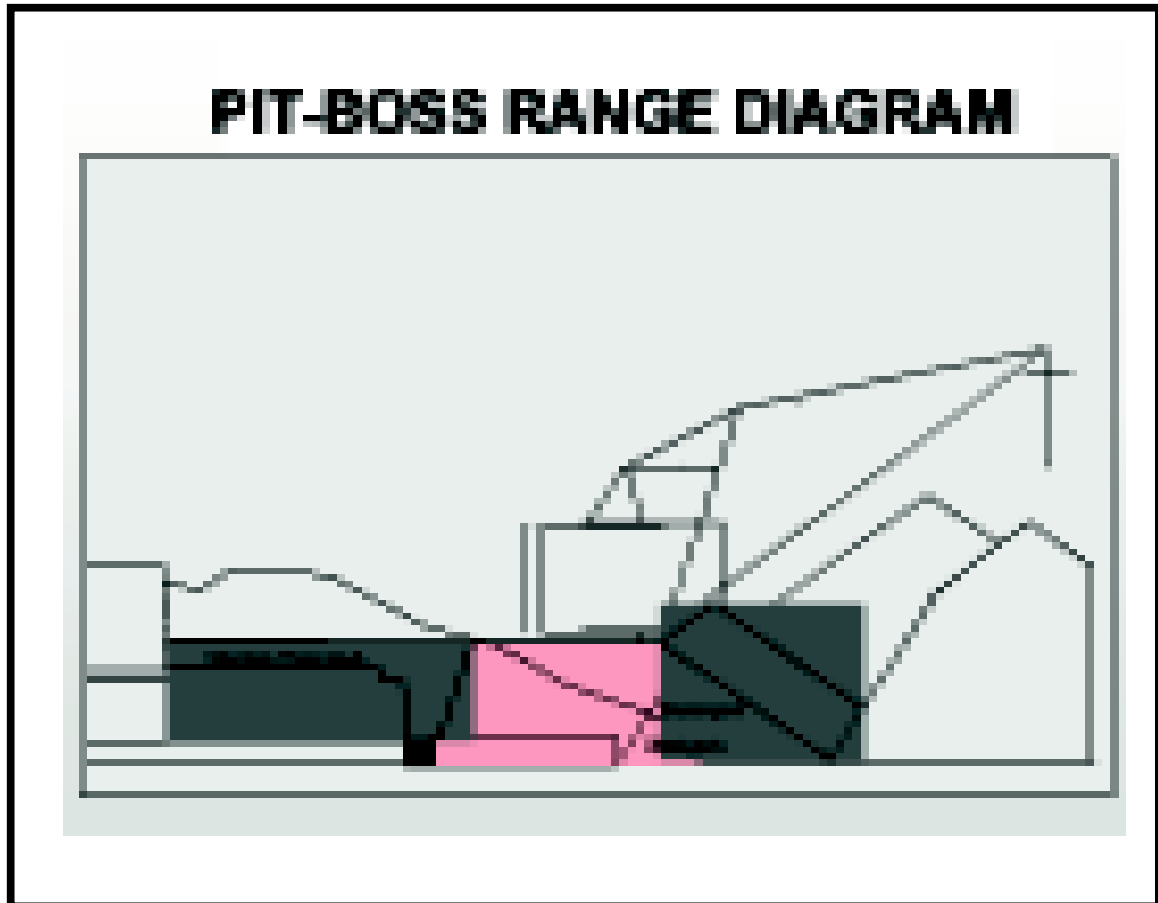


FIGURE 28

- .fault finding and repair skills in short supply

PIT MANAGEMENT

Douglas management's vision is that a high standard of housekeeping and safe working practices will enhance productivity. The following areas concern the dragline section:

Routing of highwall roads

The highwall road is constructed alongside the outer edge of the topsoil stripping area. This allows feeder cables to be placed where equipment cannot be driven over them. This road position also minimises the inter-action of service vehicles and production equipment. The road is built parallel to the strip and is therefore in a straight line for the full pit length. This eliminates hazards related to winding roads, with the result that there is safer access to the operations.

Demarcation of feeder cables

Unnecessary damage to feeder cables is caused by poor cable demarcation, especially at night. To eliminate this, cables are now demarcated with bright red or yellow reflective cones spaced at 20m intervals. This reduces the amount of downtime on equipment related to cable damage. Figure 29 shows cable demarcation.

Cable support

Cable Bridges

The correct placing of overhead cable bridges shortens equipment routes. For example, the dragline dozer has to travel from the back of the dragline pad to the top of coal. Previously, this dozer had to tram around the feeder cable, resulting in longer travel times. A specially constructed cable support bridge ensures that the dragline feeder cable is suspended clear of the highwall and allows the dozer to pass in-pit to reduce it's tram distance. Cable damage has also been reduced.



FIGURE 29
CABLE NEXT TO HIGHWALL ROAD



FIGURE 30
SKID MOUNTED CABLE SUPPORT BRIDGE - DRAGLINE
BENCH

Previously, the dragline feed cable was placed directly on the highwall, resulting in frequent damage. Moving of the cable was also very difficult. The support bridges alleviate the problem because the cable is suspended well clear of the highwall. The bridges are towed along the bench (above) and highwall (below) to keep the cable in position with the advance of the dragline.

Cable Towers on Access Roads

Cable towers are placed so that all service vehicles such as graders, cable reeler, Diesel tankers, water tankers, dozers and drills, can pass underneath the cables without causing damage. Previously, the cables were buried under topsoil.

Signs

Highly visible signs are placed at hazardous areas such as the dragline operating area, heavy vehicle crossings, road construction points, highwalls, entrances to mining area and haul road intersections.

Hazard identification

All personnel in the operational area are required to identify hazards, and take any steps necessary to eliminate such hazards. For instance, any person driving on a haul road is required to stop and remove large stones or objects from the road surface, in order to reduce tyre or vehicle damage.

CONCLUSION

The graph in figure 34 shows a steady increase in dragline output, which results from the unflagging effort of all concerned to achieve and maintain a World class performance. Also of significance is Douglas Management's view that there is always scope for improvement, even a little at a time, hence the installation of the video monitoring system at time when further improvement was needed. During the period July 1996 to June 1999, the coal exposure rate has increased from 270 000 tons per month to 420 000 tons per month and the dragline digging rate has improved from 1500 bcm per hour to 2368 bcm per hour. This represents a 57.9 % increase in digging rate and this machine is now in the forefront with other



FIGURE 31
SKID MOUNTED CABLE SUPPORT BRIDGE ON HIGHWALL



FIGURE 32
SKID MOUNTED CABLE SUPPORT TOWERS - ACCESS
ROADS



FIGURE 33
EXAMPLE OF HAZARD WARNING AND RESTRICTED
ACCESS SIGN

top performing machines in the country. This has happened because the critical issues of drill and blast control, operator technique and performance, teamwork and pit management, were correctly and practicably addressed by mine management. At a time when the depressed coal price is placing the local industry under extreme threat from global competitors, it is important to produce coal safely, efficiently and cost effectively to remain competitive. Douglas will continue to look at means to improve productivity.

FUTURE IMPROVEMENTS

Having achieved considerable success with the current exercise, the next step is to identify further areas for improvement. The following areas are under consideration:

Contel Monitoring System

Greater use must be made of the Contel performance monitoring system. The output reports allow for detailed analysis of performance criteria and in particular such important cycle elements as bucket fill time, which in some cases can be related directly to operator performance. For instance, the operators have a habit, developed over 10 or more years, that does not allow for optimal drag distance to fill the bucket. Observation shows (and confirmed by the video system) that the operators accept drag distances of 24 metres as being normal. With the improved digging conditions this distance should not exceed 13 metres. To rectify this, the operators must undergo thorough on-the-job training to improve their techniques.

Global Positioning Systems - Drill Rigs

There are already GPS driven systems in use on drill rigs in South Africa. The system in place at Optimum Colliery has shown improvements in both spotting time and accuracy of hole positioning. Douglas has budgeted for a GPS system within the next two years

Computerised Blast Initiation

The system is extensively used in underground mines but has not yet found favour with opencast mining. At Douglas the geographical location of the opencast areas is such that mining takes place in

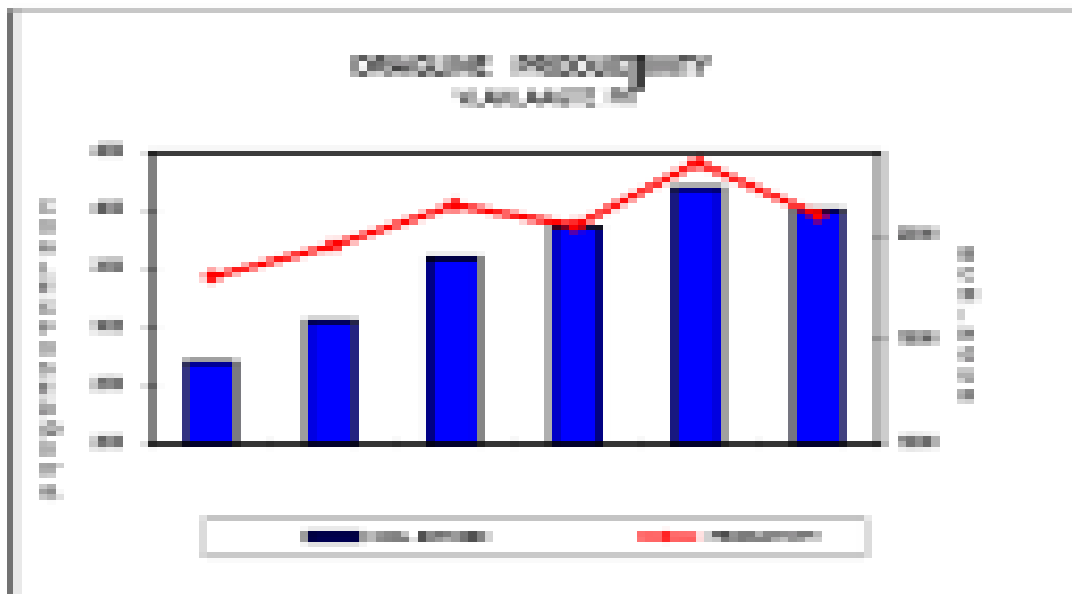


FIGURE 34
PRODUCTION PROFILE OVER 3.5 YEARS

close proximity to sensitive infrastructure. This requires careful timing of blast holes so that ground and air blast vibrations remain within accepted threshold limits. The system enables precise hole by hole sequencing and also checks on the correctness of the blast tie-up.

Pit Control

A radio-based control system has been implemented and operates from a centrally located control room. The benefits are as follows;

- Earlier reporting of machine breakdowns reduces the turn around time for repairs and therefore more production time is available.
- Allocation of machines to specific areas is simplified and efficient, resulting in greatly reduced idle time.
- Where machines are shared, such as graders or tyre dozers, job priority is established through the controller, therefore the most important tasks are carried out first.
- The pit controller records production, idle and breakdown data as well as production statistics. This results in accurate recording of essential equipment and production data.
- The production data for draglines and coaling operations is updated hourly and is on a computer network. This enables management to view production performance at any time.

For the system to be efficient it is necessary that all major equipment be fitted with two-way radios. Operators will only be able to communicate with the pit controller or supervisory staff. In 1999 a more sophisticated, electronically controlled system will be introduced. This method uses on board screens to show operators e.g. truck drivers, which loader or dump they must go to. The system will be satellite driven and will use GPS technology.

NOTES:

1. Blast cast - the amount of material moved by explosives energy and not by machines.
2. Drill chipping - the material cut by a drill bit is continuously ejected from the blast hole by compressed air and forms a

conical heap around the perimeter of the hole

3. Easer holes - these holes are placed in addition to the normal pattern and are essential for reducing front row burdens where back break is excessive
4. Wire line logging - this process uses a radiometric isotope to scan strata emissions, which are correlated to rock types
5. Minescape model - Commercially available software (Gemcom -Australia) which uses gridded data to create mine designs/ mine schedules / blast plans / 3-D expressions
6. Stratalogger - an on-board condition and performance monitoring system. Data is transferred to a lap top computer for further processing. The system provides detailed reports
7. Tie-up configuration - stated simply, this is the shot firing sequence for the blast and consists of non-electric shock lines with inter-row millisecond delay detonators.
8. Double digs - under ideal conditions the bucket is filled when dragged through the rock for twice the bucket length (13 meters). In hard digging it often requires taking a second pass to fill the bucket with an attendant increase in cycle time.
9. Width to height ratio - when the width to height ratio exceeds 1:1, then cast blasting becomes less attainable. At Douglas the midburden ratio is in the order of 3:1
10. Freezing - when material against the pre-split line is not fragmented and forms a bulge of hard rock on the highwall
11. Stand-off distance - the distance between the last row of holes in the main blast and the pre-split line
12. Pre-split - a crack created in the rock mass by explosives. A straight line of closely spaced blast holes is lightly charged and all holes are detonated simultaneously.

The explosives charge mass is calculated so that sufficient gas pressure is generated to overcome the tensile strength of the rock between adjacent holes, thus creating the crack. The pre-split is set off before the main pattern is drilled.

13. T-min - the time, in milliseconds, that it takes for the front row burden to move before the next row detonates. The measurement is done by high-speed photography and is used to determine ideal inter row delay periods.
14. Powder factor - the amount of explosives used to fragment rock and is expressed as kilogrammes per cubic meter
15. Column rise - the charge length of the explosives placed in a blast hole and expressed as a percentage of the total hole depth.
16. Blast over - the amount of material moved across the pit by explosives energy and not by machine. It is expressed as a percentage of the pre- blast volume.
17. Dozer push over - the blasted material is pushed into the void of the previous strip by means of D11 dozers to create a bench for the dragline
18. DC 2000 system - a system which electronically controls all dragline motions.
19. Lightweight bucket - this type of bucket has an 8.9 % payload increase over a standard bucket due to lighter material and thinner lining.
20. Double bench stripping - The stripping method used at Douglas exposes both the upper and lower seams from a common bench elevation over a 90 meter pit width.
21. Chop down - cutting into the material with the bucket against the highwall and with limited free face for the bucket teeth to bite into.
22. Key cut - a slot cut against the midburden, extending 30 meters back and reaching down to the top of the 2 seam,

to create a free face for the main strip.

23. Contel system: an on-board performance and condition monitoring system, radio-linked to a PC for data storage and report generation.
24. Range diagrams: sections drawn through the pit at right angles to the strike length and showing the stripping cycle of the dragline. The “Pit-Boss” software used enables planners to optimise bench height reach and dump height.

High Extraction of Coal in Horizontally Stressed Zones

C. J. Joubert

Shaft Manager - Brandspruit Colliery, SASOL Mining

SYNOPSIS

SASOL Mining adopted a strategy of optimal reserve utilisation through practising high extraction mining methods. Making use of continuous miners in the pillar extraction activities, lead to machines and sometimes operators in cabins, being trapped underneath tonnes of rock from an unexpected or premature goaf formation. In the search for answers why such incidents occur, it was realised that because of the extent of high extraction areas in the Secunda Mines complex and the movement in the earth's crust because of plate tectonics, or continental drift, horizontal stress was a reality.

Research proved at the Bosjesspruit Colliery of SASOL Mining, that horizontal stress, exceeds vertical stress by a factor of 2,2. Working at coal seam depths of 220m requiring large pillar centers, necessitated a review of the pillar removal strategy. After numerous discussions, visits to other high extraction operations, reading articles on pillar extraction and consultation with Rock Mechanic practitioners, the NEVID method of pillar extraction was designed and implemented. The essence of this mining method, was to control the speed of goaf formation and keeping the goaf away from the pillar being extracted.

This method is suitable for certain conditions and is implemented in approximately 80 % of the high extraction operations at the Secunda Mines complex. Dramatic reduction in incidents where machines have been trapped underneath a goaf was attained, with a corresponding reduction in damages on machines, less production losses and most important, a reduction in potential injuries.

BACKGROUND

Through the years of coal mining, various methods of maximising extraction of coal were designed and modified, to suit various conditions and situations. Some of the methods were suitable for drill and blasting operations and had to be adapted when using continuous miners. Of the more commonly known, and in instances, forgotten high extraction continuous mining operations were known as the Usutu-, Kriel-, Wongawilli-, Munmorrah-, Checker board-, Sigma Ben-, Rib Pillar-, Fish bone- and Pocket and fender pillar extraction methods.

In optimising their coal reserves, SASOL Mining adopted the strategy of high extraction mining. Because of various reasons, the longwall operations were stopped and focus was on board and pillar, as well as pillar extraction continuous mining.

Pillar extraction was widely accepted as a mining art form, rather than a science. It is well known that in pillar extraction you need to know when to stand and when to run. People working in pillar extraction sections had to be experienced in letting down the roof and this is also a legal requirement. During high extraction operations, numerous incidents occurred where a continuous miner, (or during the times when onboard machines were still used, sometime with the operator still in the machine) was trapped underneath tonnes of rock from an unexpected or premature goaf formation. These unfortunate situations were sometimes associated with injury, but every time with major damages to the equipment and production losses.

The magnitude of these incidents, lead to a re-evaluation in the overall mining strategy, especially at the Bosjesspruit operations of SASOL Mining at Secunda, producing approximately 7,5 million tonnes per annum, utilising 11 continuous miners. (1998/99)

GEOLOGY / ROCK MECHANICS

At the Secunda operations of SASOL Mining, all underground mining operations are taking place in the no. 4 coal seam of the Witbank coalfield. The specific situation at Bosjesspruit Colliery is as follows:-

No. 4 Coal seam thickness - 1,8 to 3,5m
Depth below surface - 170 to 220m
Overlain by a dolerite sill - 5 to 80m
Minimum pillar centres - 28 x 28m
Laminated to competent sandstone roof requiring systematic support.

Cracks in the immediate roof structures, dominantly running North / South, indicated some stresses in the rock formation. Making use of the technology of a U.K. based company, tests were conducted and it was found that horizontal stress exceeded vertical stress in this area by a factor of 2,2. Because of the vast area at Secunda mines where high extraction mining has taken place and movement in the earth's crust due to plate tectonics, the presence and reality of vertical stress was realised.

Vertical stress or pillar load is shown diagrammatically in figure 1 and the effect of horizontal stress, in figure 2.

CONSIDERATIONS

Pillars were mostly designed to be smaller than 24m, hence the maximum cutting distance of not more than 24m. Standard pillar centers in pillar extraction panels, would be 28 x 28m. Such splitting of pillars would also require the face to be ventilated through some auxiliary means. Statistics from the past indicated that it was during this support process, when a pillar was already weakened, that the most accidents occurred. Increased depth below surface, lead to bigger pillar center being mined. Having pillar centers in excess of 30 x 30m, would require supporting the splitting of the pillars in the center, after having cut 24m.

In the event of any roof failure whilst mining such big pillars, it was also possible that a machine could be trapped deep into a pillar, making reclamation operations extremely difficult and most probably impossible. The number of incidents where machines have been trapped under an un-expected roof failure was a matter of great concern and taking the big pillars into consideration, a decision had to be taken regarding the continuation of high extraction under certain conditions or even discontinue the practice.

To stop pillar extraction, would, in the light of diminishing reserves, be out of the question, therefore alternatives had to be generated. Visiting various high extraction operations in the coal mining industry, reading articles and talking to experts in the field of pillar extraction and various Rock Mechanic practitioners, numerous alternatives were considered. It must therefore be stressed that this article will not address the rights and wrongs of other high extraction methods, but focus on the specific situation at Bosjesspruit colliery and the best solution, applicable to those conditions.

The normal Pocket and fender method (see figure 3) that was widely used at all of the Secunda mines operations, delivered an average in-pillar extraction rate of 65%. To cease this practice would have had a major impact on the life of mine. Consideration was given to splitting the pillars, but supporting the center cut when mining these big pillars and the possibility of a pillar run in this weakened safety factor area, made this a non viable proposition.

Evaluating the split and quarter approach proved negative, regarding the large unsupported roadways, creating an unsupported intersection and having the continuous miner deep into the pillar, at it's weakest position. In the event of a fall of ground, the machine would be at its most vulnerable and the operator would be in a dangerous position. It was however considered from a pure theoretical point of view, that splitting and quartering from all four sides of a pillar, (see figure 4) would eliminate the aforementioned constraints.

Knowing that this was not practically possible and should there be a way around this problem, cuts could be made into the pillar at angles as indicated in figure 5. This was the basis of the new approach in mining pillars, now known as the NEVID method of pillar extraction.

THE NEVID METHOD OF PILLAR EXTRACTION

In utilising the theoretical model as shown in figure 5, the individual diagonal cuts, when connected, forms the standard pattern of cuts removed from a pillar (figure 6). Every cut into a standard pillar is less than 15m before a holing is obtained, which compliments ventilation flow and cutting distance. Each of these cuts provides for a double lift being taken. The cutting sequence (figure 7)

also provides for the ease of cable handling.

Leaving the triangles on the 3 corners (figure 6) stabilises the roof and leaving the center rib between cuts 2 and 4 (figure 6) ensures that the goaf is kept away from the pillar that is being extracted. Mining pillars according to this sequence and method, keeps the goaf formation normally more than two rows of pillars back from the active goaf line.

The presence of the center rib left in each pillar, allows for gradual loading of the goaf onto these ribs, resulting in the goaf formation being slow and less violent than traditionally known.

It is crucial to measure and mark the directions of each cut to be taken against the roof, prior to extraction, enabling the supervisor to evaluate his environment for any rock movement, discontinuities and as a control measure. After having extracted on average two rows of pillars, a stopper pillar needs to be left in the center of the panel, (figure 7) to control the goaf even further, to reduce the effect of an airblast and assist in the gradual formation of the goaf.

Full column resin bolts are used for normal systematic support and the same type of roof bolt is also used in a single row as breaker lines. Policeman sticks are placed strategically as indicators of any roof movement.

It must also be stressed that when considering pillar extraction, panels have to be designed for this purpose and that the evaluation of Rock Engineering practitioners of any board and pillar workings considered for pillar extraction is essential. Extraction from this method was found to vary between 55 and 60 %.

LESSONS LEARNT

- The shorter the cutting distance into a pillar, the smaller the risk of trapping a continuous miner,
- Auxiliary ventilation would normally not be required,
- Ventilation is directed over the continuous miner into the goaf,
- Marking off of cuts and direction lines at 45° is of utmost

- importance in this method,
- Evaluation of individual cuts by supervisors creates a better sense of awareness,
- Evaluating every panel to be extracted by Rock Engineering and production personnel, proved to be a basic requirement,
- Designing panels and pillars for pillar extraction is essential,
- Stopper pillars are required to assist in controlling the goaf formation,
- The systematic approach and the control measures, assists the training of inexperienced pillar extraction crews,
- Horizontal stress is a reality in coal mining,
- A width to height ratio of >5:1 is required.

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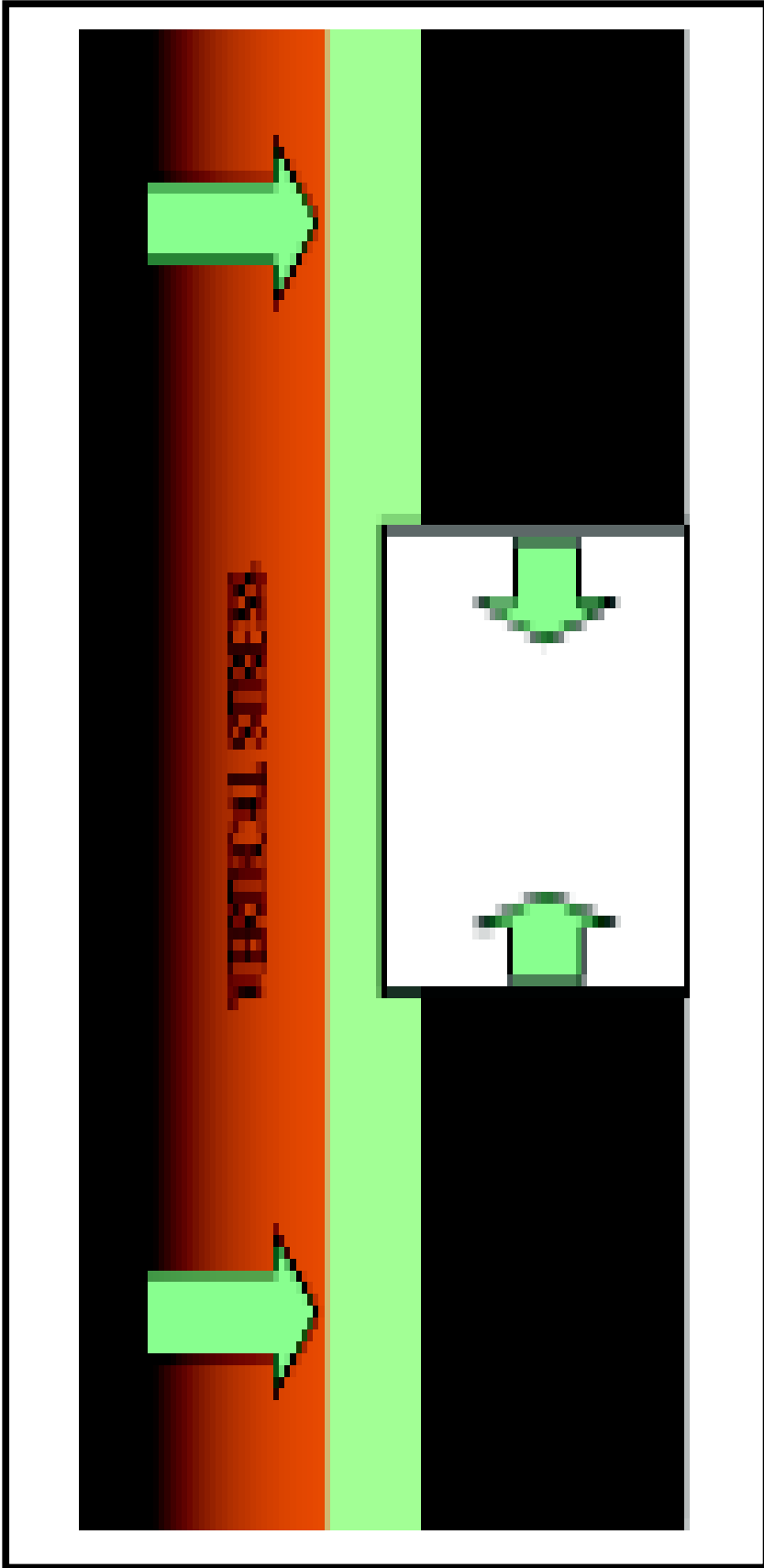


Figure 1

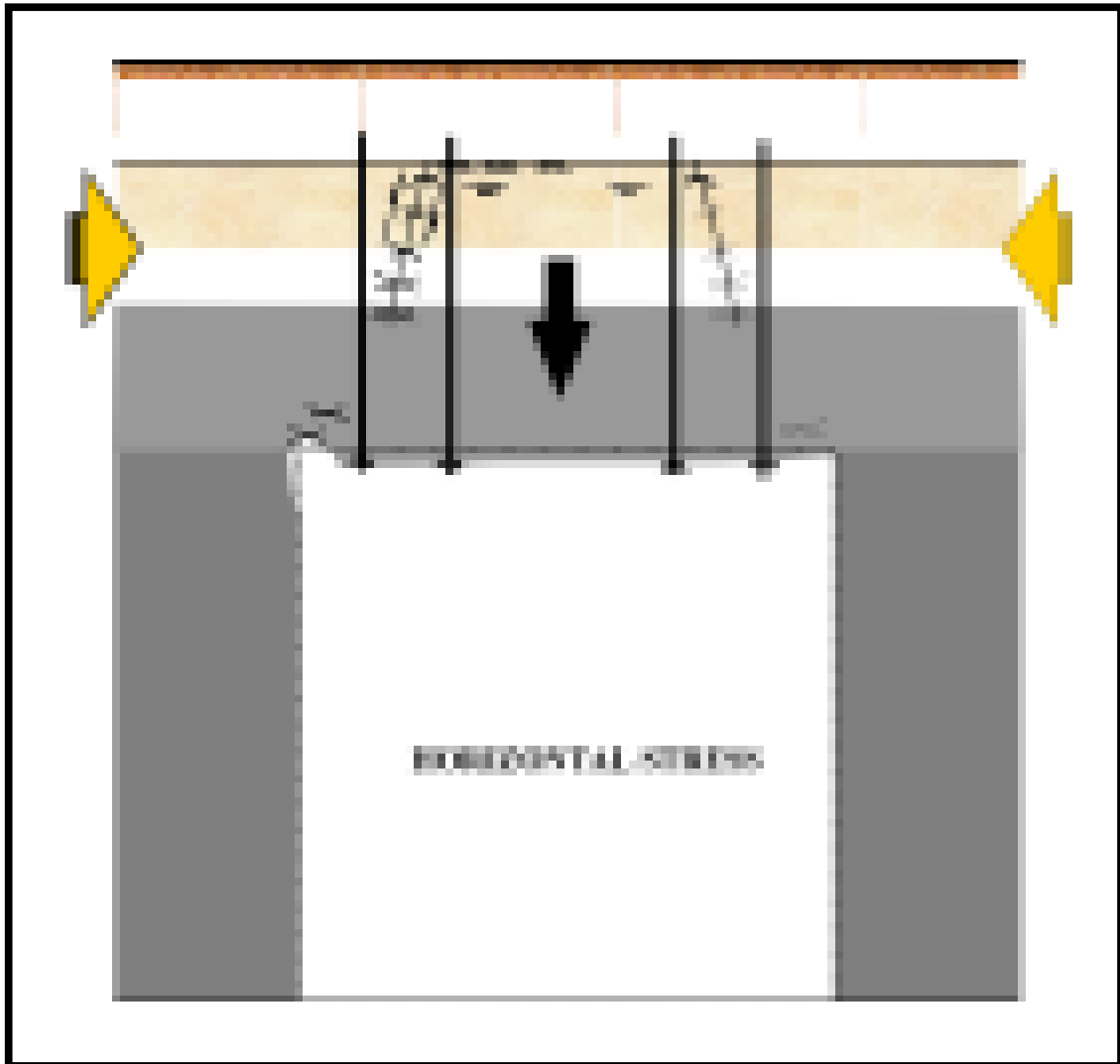


Figure 2

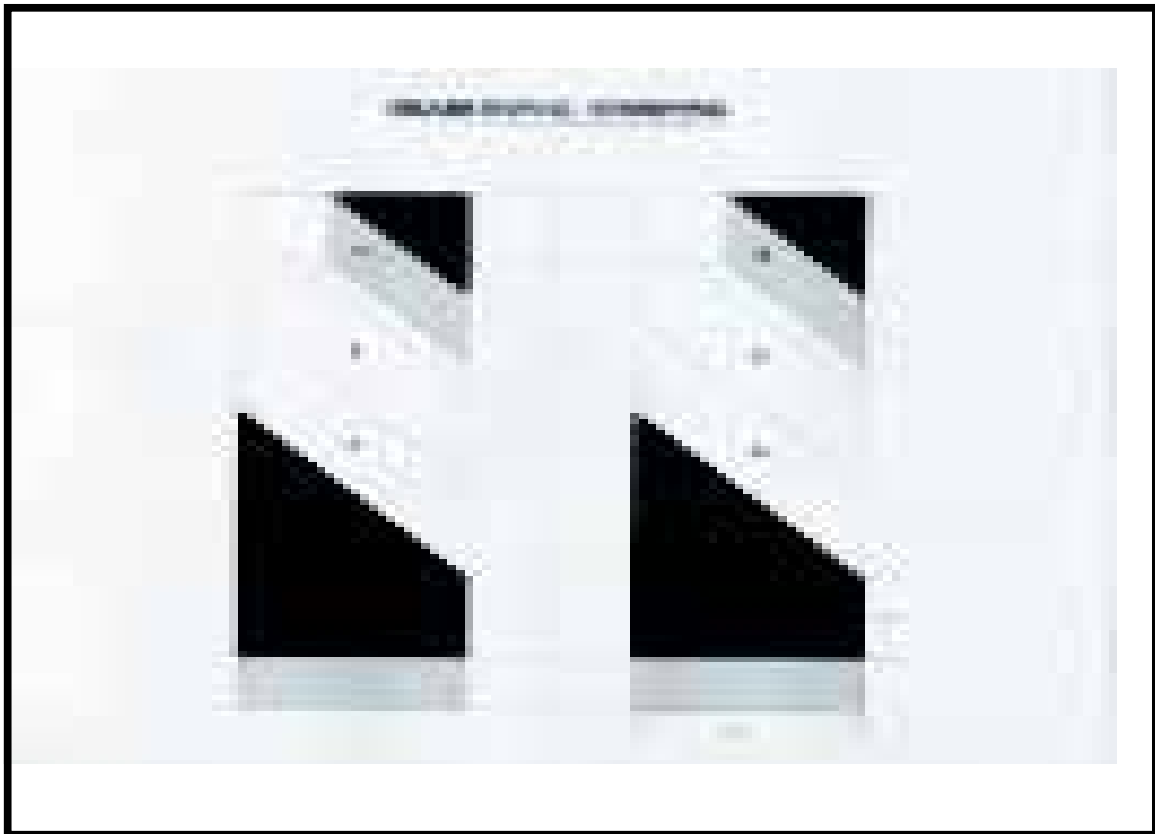


Figure 3

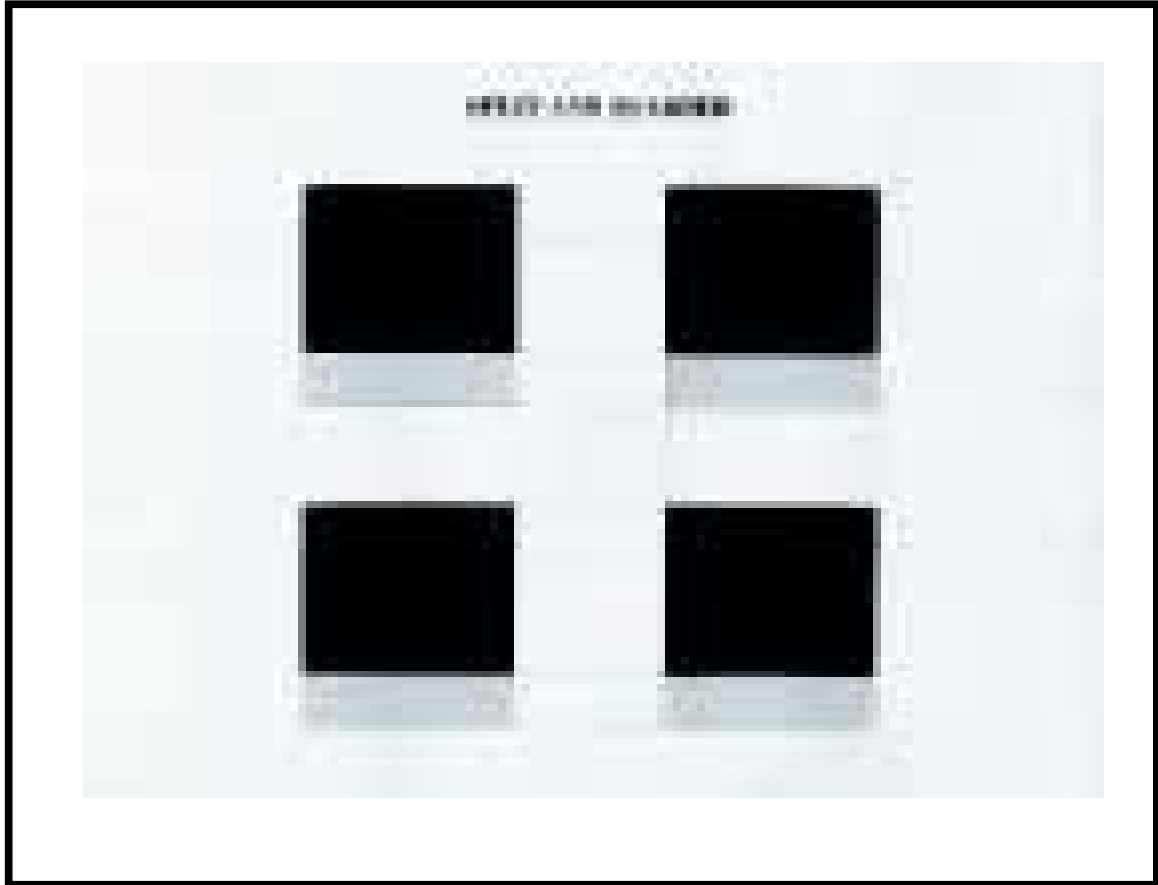


Figure 4



Figure 5

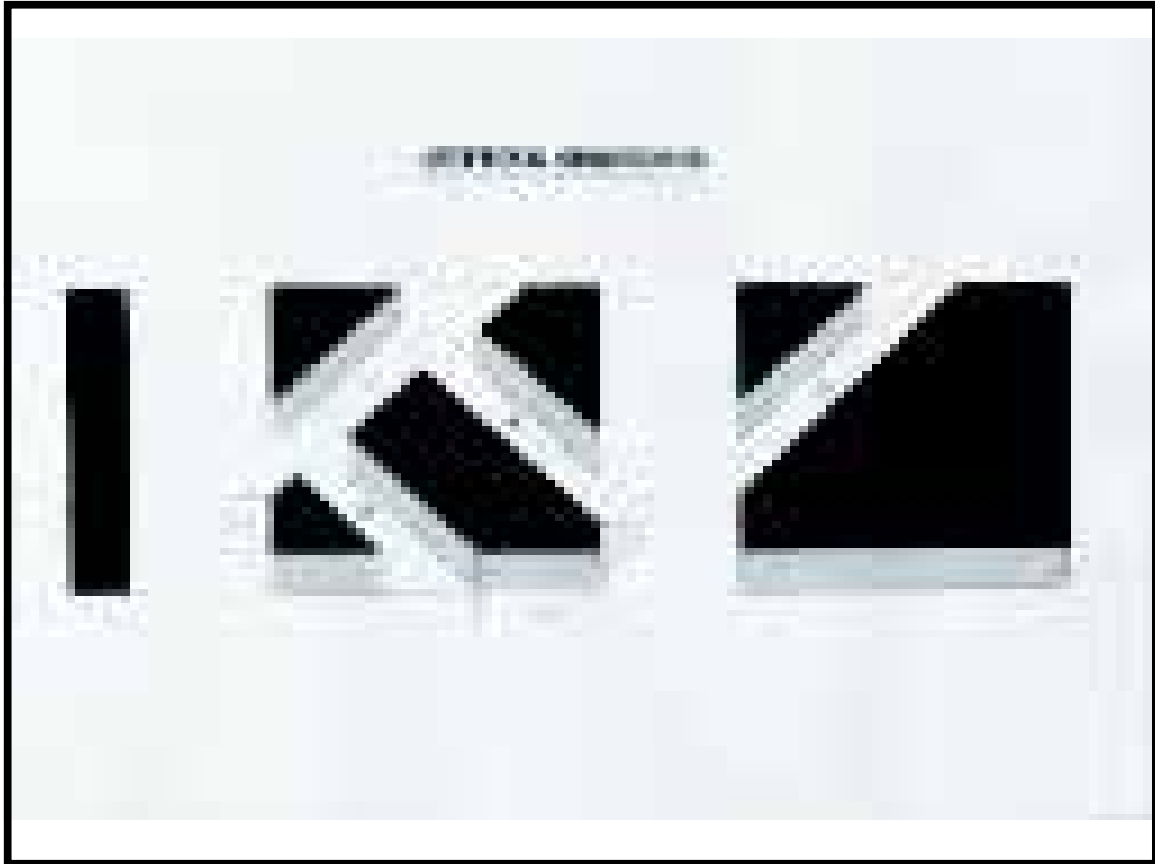


Figure 6

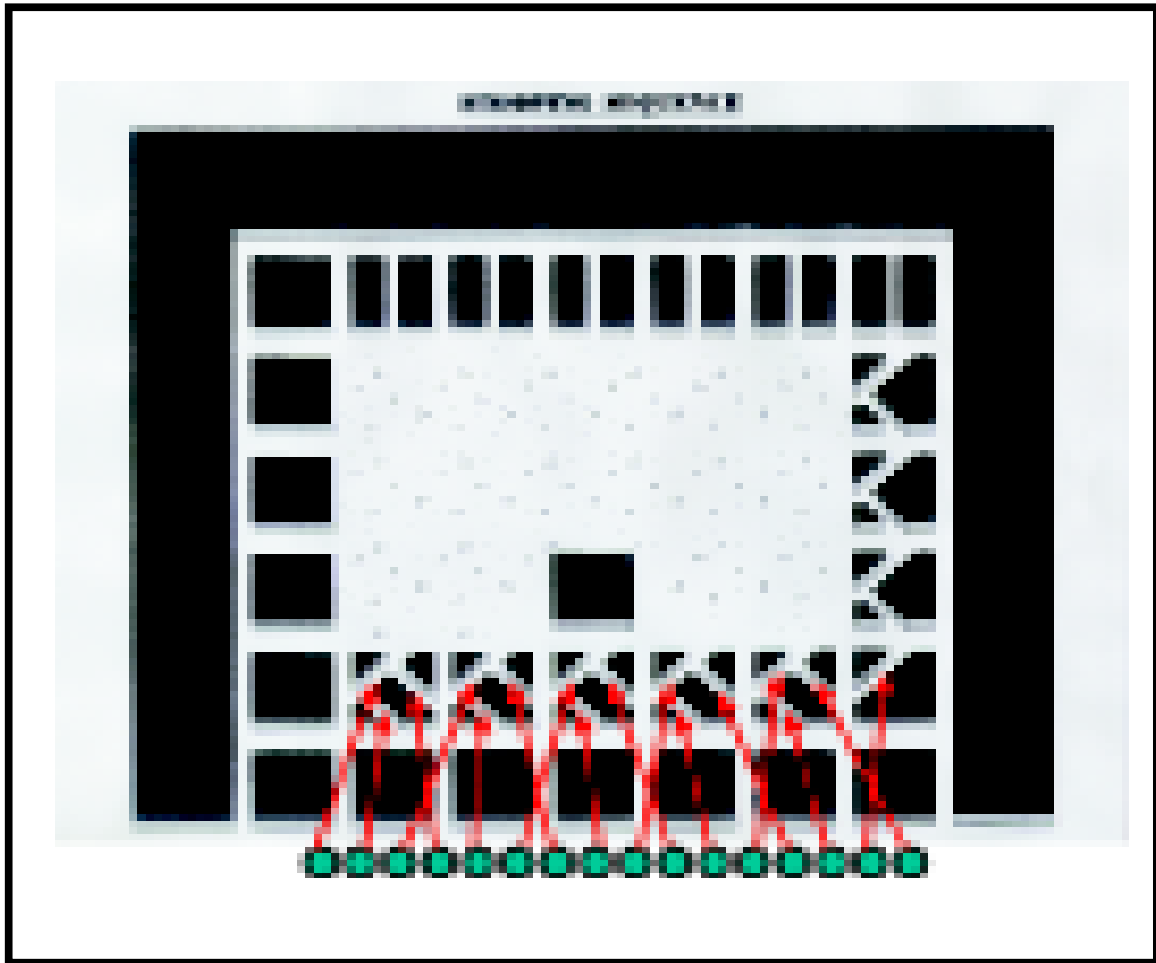


Figure 7

