Mine Mechanization

JS'" International Mining Congress and Exhibition of Turkey-IMCET2003, i© 2003. ISBN 975-395-605-3

Application of Fuel Cells in Underground Mining and Tunnelling

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ABSTRACT: North America's hard rock underground mining industry faces many challenges. Mines are becoming deeper; health and safety regulations are becoming more stringent; in Canada mining companies must reduce emissions of greenhouse gases. An attractive option is to replace diesel engines by hydrogen fuel cells, which reduces ventilation demands and emissions of harmful particulates. CANMET-MMSL is a member of an international consortium developing fuel-cell-powcred vehicles for use in underground mines. A prototype has been successfully tested at CANMET-MMSL's experimental mine and at Placer Dome's Campbell Red Lake Mine, and is ready for commercial production. No problems are anticipated in certifying the fuel cell locomotive for underground use. Current work shows that the operating costs of a fuel cell locomotive are lower than for a diesel-powered equivalent. Capital costs are currently higher, but costs are falling rapidly. The next phase of the program is to develop a fueled 1-powered load-haul-dump machine. The proponents believe that fuel cells will be the power plant of choice for underground operations - both civil and mining - by the end of this decade.

1 BACKGROUND

Underground mining in many parts of the world, and particularly in Canada, is facing major challenges. The most significant of these is the challenge of continuing to mine competitively. Canada is a "price taker" for most of the commodities it produces - despite being a major exporter of most metals and minerals. Canada nonetheless has relatively little influence on the price of these commodities on the world market. Consequently, Canada's mineral producers face ongoing price pressure from other producers, especially those in developing nations, who often can mine high-grade deposits at low labour costs. Thus it is essential for Canadian mines to continually strive to lower their production costs.

Linked to these economic challenges are a number of major technological issues. For example, although Canada has world-class ore deposits that are as yet unexploited - such as the massive nickel deposit at Voisey's Bay in Newfoundland - and undoubtedly has many such deposits yet to be discovered in her vast landmass, it is much cheaper to mine as long as possible at existing, developed sites, where the infrastructure is already in place and there is an established work force. The Sudbury basin in Ontario is a good example. This area has long been one of the world's major nickel and copper producing regions, but the near-surface deposits there are now largely mined out. However, there are substantial proved reserves at depth, and the producers there - INCO and Falconbridge - are now developing the much deeper mines required to exploit the remaining reserves. These new operations, such as Falconbridge's Onaping Deep mine, will be operating at depths below 3,000 metres.

Mining at these depths presents a number of challenges, including ground control at very high stresses, and the logistical challenges of moving people and materials around efficiently and cheaply. Perhaps most important, temperatures at depths of 3,000 metres in the Sudbury basin are expected to exceed 40 degrees Celsius (Udd, 2002). Both men and equipment face problems in working for extended periods at these temperatures, and ventilation and cooling to remove heat will be essential. Ventilation in a typical Canadian underground mine is not cheap; typically, over 40% of the electrical energy used in underground mines in Canada is used to drive ventilation fans. To minimize these costs in deep mines operators will have to reduce as much as possible heat from extraneous sources, such as internal combustion engines.

The mining industry, in common with other sectors of the economy, is facing a new challenge that arises from Canada's commitment to reduce emissions of greenhouse gases. Canada signed the Kyoto Accord in 1997, and in December 2002 Parliament formally ratified the agreement. Canada is now legally committed to reducing by 2010 emissions of the principal greenhouse gases (GHGs) to 94% of those emitted in 1990. Achieving this goal will not be easy, in part because Canada's economy is growing. By 2010 with expected growth in the economy the target will eflectively be a 40% reduction in emissions. In other words, m the absence of measures to reduce GHG emissions, by 2010 total emissions would be about 140% of 1990 emissions.

Fortunately for the mining industry, there has already been significant progress in reducing GHG emissions. Nonetheless, the industry is required to make additional reductions, which will require it to further reduce energy consumption, because all GHG emissions from the mining industry arc carbon dioxide arising from the combustion of fossil fuels. These fuels - primarily diesel - power stationary and mobile equipment on the surface and underground. Estimates are that underground diesel engines account for about 0.6 million tonnes per year of carbon dioxide emissions, and a further 0.4 million tonnes are emitted in providing the ventilation required to remove the heat and emissions from these engines. Thus use of diesel engines underground results in about 1 million tonnes per year of carbon dioxide emissions, which represents 26% of all undergiound-mining related GHG emissions.

There is a further challenge facing the industry, namely, the increasingly stringent regulations on breathable air in underground operations. Diesel engines are the preferred power source in many underground operations. They are reliable, relatively simple and cheap to maintain, and diesel fuel is not very volatile and therclore poses little flammability or explosion hazard should it leak. However, over the last decade research has clearly shown that diesel emissions are a health hazard. There is especial concern over the very fine particles known as diesel particulate emissions (DPM, Grenier et al 2001). There is strong evidence that these are carcinogenic, and in the last decade allowable emissions of these have progressively been reduced. Typical allowable levels of DPM in Canadian underground mines in the 1990s were 1.5 mg/m . As reported by Grenier et al. 2001, allowable levels in the USA are expected to be set at 0.16 mg/m by 2006 - a reduction by a factor of almost 10.

There has been much effort devoted to meeting these expected levels, by improving fuel quality, improving diesel performance, and by increasing ventilation. But these approaches are costly, especially if increased ventilation is required, and there is increasingly concern that finding an alternative to diesel engines in underground mines may be eventually be the only practical approach to meeting the emission standards.

2. THE FUEL CELL AS A POWER SOURCE FOR UNDERGROUND MINING

A number of the challenges described above have led industry and government to look closely at fuel cells as an alternative to diesel engines in underground mining and related fields, such as tunnelling. The principle of the fuel cell has been understood for many years, but it is only relatively recently that practical fuel cells have been developed. Essentially, a fuel cell is a reverse electrolysis unit. Instead of passing electricity through water to split the water molecule into hydrogen and oxygen, the fuel cell uses a catalysed reaction to combine oxygen and hydrogen non-explosively to yield water and electricity. Proton Exchange Membrane (PEM) fuelcells are best for underground use because of they operate well in the temperature range involved in surface and underground mining (-20EC to +40EC), they are very dependable (as shown in underground tests, Bétournay et al. 2002), and they cost less than other fuelcells.

Compared to internal combustion engines such as diesel motors, fuel cells have several advantages over. First, they are not limited by the Carnot cycle, and can achieve energy conversion efficiencies approaching 100%> (Carnot cycle engines are inherently limited to an efficiency in the order of 30%). Second, they have no moving parts except pumps (if required) to move hydrogen and oxygen (or more typically air, with the nitrogen and other components of air moving unchanged through the fuel cell). Third, the only emissions from a fuel cell are water vapour. Thus the particular advantages of the fuel cell in the context of underground mining are that there are no emissions of GHGs, and no emissions of noxious or carcinogenic particles. Additional advantages are the very high reliability and negligible maintenance required, because of the lack of moving parts, and the very low level of heat emitted. Fuel cells thus promise to:

- provide an underground power unit that eliminates DPM, and hence meet the stringent regulations on allowable particulate matter;
- reduce heat emitted; reduce overall ventilation requirements (as there is no requirement to remove waste engine heat and DPM); and
- reduce equipment maintenance and downtime.

Fuel cells do have disadvantages. One of these is the challenge of how to handle and store hydrogen. Hydrogen is the smallest molecule, and is difficult to confine. There are several storage options. One is liquefaction, which requires expensive facilities and cryogenic storage tanks. Another is as a compressed gas, which is cheaper than liquefaction but requires bulky, high-pressure storage tanks. The third option is to store hydrogen in the form of a metal hydride.

Several metal alloys have the ability to contain very large volumes of hydrogen in the spaces between metal molecules. Typically, the hydrogen is adsorbed under low temperature and pressure, and is released by gently heating the hydride. Hydrogen stored in this fashion is very safe, as damage to the bed does not result in release of any significant level ol hydrogen, nor in any open flame. The energy storage per unit of space is also relatively high. However, the hydride bed is very heavy, and this lorm of storage is therefore not practical for most transportation applications of fuel cells (e.g., cars), because the added weight reduces energy efficiency. Fortunately, in underground mining operations this weight is an advantage, because vehicles such as locomotives and load-haul-dump (LHD) machines must be very rugged, and require weight to ensure good traction.

Hydrogen also has the disadvantage of being very flammable, with a wide flammability range. Us use in underground operations, whether in mining or in cıvıl engineering, which have strict controls on potentially flammable materials, therefore poses problems. Nonetheless, the potential advantages of the fuel cell over the diesel engine mean thai industry and government regulatory agencies in North America have put considerable resources into developing fuel cells and risk reduction measures for underground operations.

3 THE FUELCELL PROPULSION INSTITUTE

The Fuelcell Propulsion Institute (FPI) is the principal force driving the development and adoption of fuelcells as a power source for underground operations. Vehicle Projects LLC. its project management aim, which is based in Denver. Colorado, USA, is supported largely by the US Department of Energy. It also has significant support from the Department of Natural Resources in Canada, and by industry in both countries, as well as support from Mexican companies. FPl's first major project has been the development of an underground production locomotive, typically used for hauling rock and ore in underground mines, and excavated material in tunnelling operations.

An underground locomotive was chosen for the first project because it operates in a relatively controlled environment (i.e., on rails), and must be rugged and heavy in order to survive in underground operations and to generate the traction required to pull loaded rail cars. Most important, such locomotives require relatively little power, and use a simple electric motor and control system. They therefore do not requue a very large fuel cell. Converting a locomotive therefore represented an ideal opportunity to understand how to convert a mine vehicle to fuelcell operation, before moving on to convert a mine loader, which has a more complex power and control system, and requires a much larger fuelcell.

Natural Resources Canada provided the basic locomotive for the project, a typical production unit manufactured by Warren Engineering of Sudbury, and widely used in Canada and in other countries. An Italian Nuvera fuel cell stack was chosen as the power unit, and the modification of the locomotive and development and installation of the fuel cell was done by Sandia National Laboratories in the USA. rigorous safety evaluation of the locomotive was carried out by several organizations in the USA and Canada (HATCH 2002). This included risk management planning, risk identification, qualitative and quantitative risk analysis, risk response planning, and risk monitoring and control. Risk identification was performed using the "what if/checklist" technique. A total of 127 health and safety risks were identified (2 high, 85 moderate and 33 low risks). For each risk, an impact was identified and the likelihood of occurrence, intervention difficulty, impact severity and level of understanding was assessed using scales agreed to during risk management planning. Based on the risk matrix, the project management team was able to develop risk response plans.

Once the regulatory authorities in Quebec and Ontario were satisfied that prototype tests could be safely carried out, the locomotive was taken to CANMET-MMSL's experimental underground mine in Val-d'Or, Quebec, for testing. These tests were followed by a full field trial evaluation at an operating mine, the Campbell Red Lake gold mine of Placer Dome, in Ontario, Canada. The results ot the tests and field trial were excellent.

4 RESULTS OF FIELDS TRIALS

Based on a new locomotive design by R.A. Warren Equipment of Canada, including an improved motor controller, motor and wheel gears, testing was performed under full production conditions at the two minesites. Several parameters were evaluated, the most important being: continuous push/pull effort, power curve definition, hydrogen consumption, vibration, noise, reliability and troubleshooting, safety, and productivity.

Testing of the locomotive indicated that a total of at least 8.5 hours of operation could be achieved from the fuel cell power plant compared to about 6.5 hours for a similar electric battery-powered version. At'the Campbell mine, over 1,000 tonnes were hauled (760 tonnes by the fuel cell locomotive, 240 tonnes by the rechargeable battery locomotive) covering a distance of over 65 km. The fuelcell locomotive proved to be as reliable as the battery version, with no occurrence at all of safety incidents involving hydrogen. Productivity was higher with the fuel cell version.

Permission to use the locomotive underground in Ontario and Quebec under their current mine regulations was received after suitable and exhaustive risk analysis and on site evaluation were performed. The locomotive is now ready for formal certification for use in underground mines by the regulatory authorities in Canada and the USA. These are for Canada: the provincial Chief Inspectors of Mines; and for the USA: the Mining Safety and Health Administration (MSHA). No difficulty is expected in obtaining the required certification and approval for conventional use. The project partners, including these regulatory agencies, are continuing to work to provide the required information for regulations appropriate to use of fuelcells in underground mining..

5 REFUELLING WITH HYDROGEN

The work done to date has focussed on the development and testing of the fuel cell locomotive with recharging of the hydride bed performed in two ways. The first consisted of removal of the hydride bed and refuelling on surface from hydrogen cylinders purchased locally (when the tests were performed at the minesites). This involved the removal of the bed with an overhead crane, and transportation to surface for refuelling. Although only *one* hydride bed was available during these tests, the availability of a second bed would make this refuelling method practical for an operating mine. The second approach required transporting the locomotive to the surface, and then refuelling it with hydrogen from a Canadian-built Stuart Energy System electrolysis plant fed directly to the hydride bed on board the locomotive. In both cases, refuelling look approximately 50 minutes. These two approaches to refuelling were are identified in the risk analysis as practical and safe options for an actual operating mine.

However, in practice both approaches would be neithei efficient nor economical, because each would require frequent movement to surface of a heavy hydride bed or a heavier locomotive. The next phase of this work, supported by CANMET-MMSL and Westinghouse Savannah River, a U.S. government laboratory specializing in hydrogen storage, will be to develop a hydrogen generation and refuelling unit that can be placed underground - in a secure and well-designed area. Ideally a small electrolysis unit would operate underground, and provide hydrogen that can be fed under pressure directly to spaie hydride beds. The concept of a "swappable" hydride bed will certainly be retained, as this would allow the locomotive to remain in use while refuelling takes place, and also would allow the bed to be taken to an area of the underground operations that can safely house the hydrogen geneıatıoıı and refuelling operations.

6 FORECAST OPERATING AND CAPITAL COSTS

Current work by the Fuelcell Propulsion Institute shows that the operating costs of a fuel cell locomotive are lower than for a dicsel-powered equivalent. Currently, an analysis of a fuelcell-powered scoop tram shows annual operating costs of US$70,000 for a fuelcell vehicle, compared with US$100,000 per year for a dicsel-powered unit. Capital costs are currently higher, but these costs are falling rapidly, and within a few years are predicted to be lower than for a diesel-powered unit.

7 THE LOAD-HAUL-DUMP MACHINE

Now thai the locomotive is established as a viable fuel cell powered unit, Vehicle Projects LLC is turning to the next phase of fuel cell development, which is the LHD machine.

A large international consortium of equipment manufacturers, technology developers, mining companies and research organizations is carrying out the modification of a conventional loader, a Caterpillar Elphinslone model 1300 unit. This includes the redesign of the drive train, selection of central and hydraulic motors, machine controller, and design of the operator interface that will provide integrated machine response of the power plant, drive train, hydraulics, and other loads. System testing in house and full field testing at Newmonl and Placer Dome minesites will be carried out by 2005.

8 HYDROGEN SUPPLY

Hydrogen does not occur freely in nature. It has to be manufactured; hydrogen in commercial use today is manufactured either by electrolysis of water, or by steam reforming of methane. (There are other options, including steam reforming of more complex hydrocarbons such as gasoline.) The prototype trials of the fuel cell locomotive, as described above, used commercially-purchased hydrogen delivered under pressure in cylinders or manufactured at site-specific electrolysis plants. In practice, unless and until a large-scale surface hydrogen distribution infrastructure develops - which is unlikely to happen for at least two decades - the preferred alternative is to generate hydrogen locally underground by electrolysis.

As noted above, one of the advantages proposed for fuel cells is the reduction in GHG emissions, which is particularly attractive in Canada. In any real application, of course, it would be necessary to determine the GHG emissions associated with hydrogen production, and offset these against savings from reduced use of diesel fuel. Fortunately, most of Canada's electricity supply is based on hydroelectric or nuclear power, which are both considered not to emit GHGs.

9 HYDROGEN: THE PSYCHOLOGICAL BARRIER

One issue faced by use of hydrogen as a fuel is the attitude of the lay - and even the informed - user. Two questions that frequently arise are the links between hydrogen as a vehicle fuel and, first, the hydrogen bomb, and, second, the Hindenburg disaster. The latter has only slight relevance to the fuel cell; and the former has none at all.

A hydrogen, or fusion, weapon uses intense X-ray emissions from a conventional fission explosion to fuse two isotopes of hydrogen, deuterium and tritium. Molecular hydrogen - the relatively common gas used in fuel cells - requires even more energy to fuse than is liberated by a fission explosion, and it is quite impossible for a fusion reaction to occur in any fuel cell.

The Hindenburg disaster occurred in May 1937. Lighter-than-air transportation was considered the most promising development for high-speed, long-distance travel, and the German-built Zeppelin airships were one of the first commercial attempts at transatlantic airship flight. These craft consisted essentially of a large hydrogen-filled container made of cotton substrate with an aluminized cellulose acetate butyrate dopant. This container provided the buoyancy required, with the passenger and motor compartments slung beneath. Zeppelins were initially very successful, but the maiden flight of the Hindenburg met with disaster shortly after arrival in New Jersey. The airship was moored to a mast, and apparently was hit by lightning. It burned spectacularly, and the disaster effectively ended the development of commercial lighter-than-air transport.

It was initially reported that burning hydrogen caused the fire, and this has been the belief in most of the period since. However, more recent studies (Bain 1997) have shown that the lire was not due to hydrogen combustion, but rather the burning of the cotton impregnated with powdered aluminum that made up the hydrogen-containing structure. In fact, a large mass of hydrogen by itself will neither burn nor explode until it is mixed with air; and hydrogen would also have risen rapidly through the atmosphere as soon as it escaped from the burning airship.

Hydrogen is of course very flammable, and its use as a fuel needs careful engineering and handling procedures. But neither of the common misconceptions constitutes a reason for not using fuel cells, especially as storage and transfer systems like those used in the mining projects have been demonstrated to be safe.

10 CONCLUSIONS

Fuel cells arc widely touted as an important source of power for cm-road vehicles in the future. Indeed, many fuel-cell powered vehicles are running now, such as the transit buses developed by Ballard Power Systems of Vancouver.

However, all the on-road demonstration projects running to date have been heavily subsidized; there is no imminent prospect for the self-sustaining, unsubsidized use of fuelcclls in general transportation. In contrast, the fuel cell in underground mining has unique advantages, especially the requirement to effectively eliminate diesel particulate emissions. For this reason it is likely that fuel cells will be adopted in mining without subsidy.

They will also find a role in related civil engineering work where emissions from combustion engines are a problem. Indeed, the Fuel Cell Propulsion Institute and its backers, including CANMET, believe that fuel-cell-powered equipment will be in commercial use in North American mines within five years and that industrial vehicles, using the same power plant size as developed in the mining projects, will be first to apply fuel cells commercially, ahead of automobiles.

REFERENCES

Bain. A.S. IW.What really happened lo the Hindenhuig.' Cover siory, Smithsonian Institution's Air and Space Magazine. May

Bétomnay. M C. Bunnell. G... 2002. Edwardson. E. Testing a 35 W fuelcell stack in underground conditions: Part I - Exposure to dust and diesel emissions. *Division Repart MMSL 02-24(TR), CANMET.* Natural Resouices Canada. 46p

Grenier. M. et al. 2001. Sampling for Diesel Paniculate Mater in Mines. *CANMET-MMSL report* 01-052 (TR).

HATCH Associates. 2002. Fuelcell locomotive demonstration, pre-developmenl review. *Fuelcell Propulsion Institute.* Toronto.

Udd. .I.E. 2002. Deep Mining in Canada - the Challenges and the Opportunities. *Prm ceilings. 1st International Seminar 'on Deep and High Stress Mining, Australian Centre ftn Ceoinerhanics.* Perth. Australia.

18? International Mining Congress and Exhibition of Turkey-IMCET 2003, <© 2003, ISBN 975-395-605-3

Possibility of Using Mechanical Miners in Underground Chromite Mines' Ore Productions and Two Different Examples

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ABSTRACT: Metallic ore prices have been decreasing recently due to unstable supply and demand relations and developed recycling methods of scrap metals. Therefore, Turkish underground metal mines and production methods should be reviewed especially in terms of economical ore cultability. Rapid and productive technologies should be applied as soon as possible to reduce production costs and increase competitiveness. For this purpose, it has been investigated that whether mechanical miners could be used for production and development purposes in chromite mines. In order to realize full-scale cultability tests in the laboratory, primarily miscellaneous investigations were performed in two different chromite mines: Kayseri-Pulpinar and Eskişehir-Kavak. Some information was obtained about production methods and working conditions and then, ore and country rock samples were collected from. After defining optimum cutting conditions, mechanical miners' production rates were estimated via performance prediction methods. Consequently, it has been found out that a mechanical miner could produce approximately three times more chromite than existing mining methods.

1 INTRODUCTION

The necessity of high investment in mining industry dictates mechanized excavation for efficient mining to reduce operating costs. Due to this fact, exploitation methods of different mines in Turkey such as chromite mines have to be revised for more efficient mining operations.

Recently, metal prices in world slock market decreased considerably (Sullivan and others, 2001). The use mechanical excavators such as roadhcaders, hydraulic hammers, etc have the potential to increase productivity, since ihey have continuous, flexible operation capabilities to adapt to existing methods. Some applications of mechanical excavators for ore excavation were reported to be successful (Atlas Copco-Robbms. 1996, Breitrick, 1998).

In order to investigate the possibility of using mechanized excavators in metallic ore formations, first physical and mechanical properties of the ore have to be determined in the laboratory and in situ such as schimidt hammer rebound value.

Rock mechanics tests, such as uniaxial compressive strength (UCS), tensile strength test (TS), Cerchar ahrasiviiy lest, static and dynamic modulus of elasticity give preliminary assestment for the machinability of the geologic formation. However full scale cutting tests are strictly advised to be carried out for efficient selection of mechanical excavators in optimum conditions.

2 OBJECTIVE OF THE STUDY

In-situ and laboratory tests are performed to investigate the possibility of using the roadheaders and hydraulic hammers for ore excavation. Objectives of the study are given below.

- Investigation into cutting mechanics ol chromite ores and surrounding rocks in selected mines,
- Methods of reducing operational costs and investigating applicability of mechanical excavators such as roadheaders and impact hammers in selected mines.

3 .THE EFFICIENT USE OF MECHANICAL EXCAVATORS IN ORE EXCAVATION

Underground excavation methods are grouped as drill and blast and mechanized excavation. Drill and blast has coarser muck lhan mechanical excavation thus making it more efficient from specific point of view. Low advance rate, vibration and support problems due to overbreaking and safety concerns limit the applicability of this method (Özdemir, 1994) and mechanical excavation become more economical with increasing tunnel length. (Pakes, 1991).

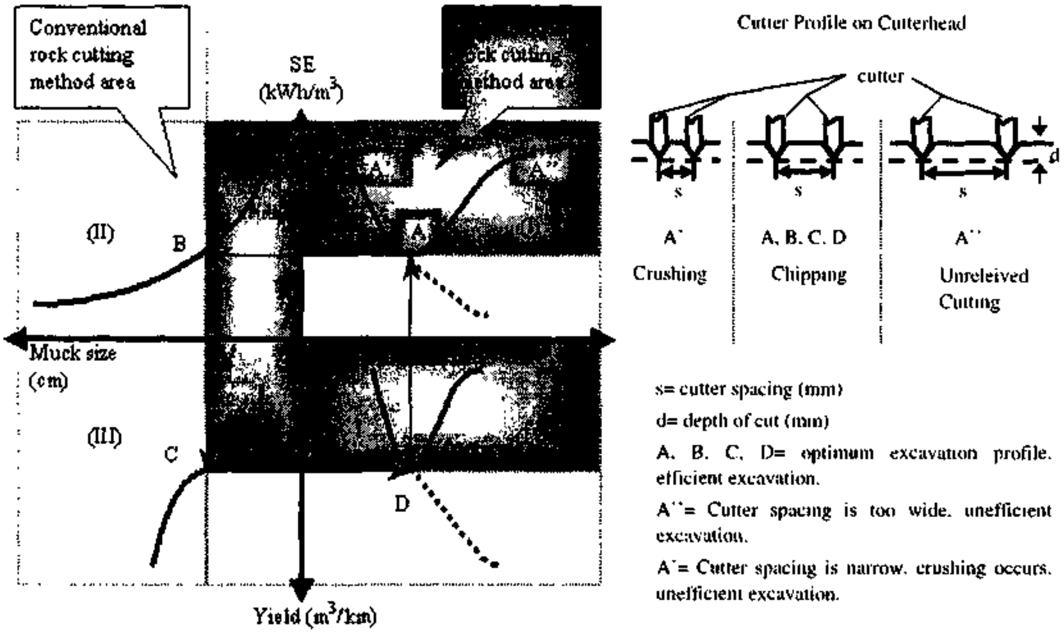


Figure 1. Comparison of conventional excavation to mechanized excavation (Tunçılemır. 2002).

The hypothetical relation between important parameters of mechanical excavation such as specific energy (SE) (the energy needed to excavate unit volume of rock, kWh/m), cutter spacing (s, mm), depth of cut (d, mm), ratio of spacing to depth of cut (s/d) and parameters used in both methods like muck size (cm) and yield volume (nrVkm) are given in Figure 1, which compares the efficiency of mechanical excavation against conventional excavation method.

Point (A) in the region I of Figure 1 shows the optimum s/d for mechanized excavation. An effcient cutterhead design must have appropriate cutter spacing minimizing spesific energy (point A). Therefore the area below point (A) can be defined as conventional excavation area (dotted line in region I.). Although drill and blast method looks more efficient in this area, one should bear in mind that the efficiency of mechanical excavation increases with longer length of tunnel.

Region II in Figure I shows the relation between debris size and spesific energy. Conventional excavation produces coarser debris than mechanical excavation. Debris size in mechanical excavation is controlled through the cutter spacing and depth of cut given optimum spesific energy. In that case debris size in mechanical excavation can not be bigger than point (B). Therefore the left side of the point (B) indicates conventional excavation, ie. drill and blast.

Region III in Figure I shows the relation between yield volume and debris size. There is some factors that limit the yield volume in mechanical excavation. Thereotically yield volume can not exceed the debris volume which is formed by cutter spacing, depth of cut and cut length. For that reason, point (C) shows the end of mechanized excavation capability in terms of yield.

The relation between yield volume and optimum s/d is showed in region IV of Figure I. In mechanized excavation, maximum yield volume and debris size occur at optimum s/d (also the lowest specific energy) which is point D. In drill and blast method "s" is referred to spacing of holes and "d" is depth of holes. Yield volume in drill and blast method will increase by the dotted line and debris size will also increase by the dotted line in region I.

4 PERFORMANCE PREDICTION OF MECHANICAL EXCAVATORS

The general technical requirements of excavation machines, in addition to safety and economy, are selective mining ability, flexibility, mobility, hard and abrasive rock cutting ability.

Geological features (such as joint sets, bedding planes, foliation, hydrogeological conditions, deposit geometry, etc.) and intact rock properties (such as cuttability. abrasiveness, strength, texture, etc.) are the basic input parameters for the efficent selection of mechanical miners and performance prediction.

The predicted cutting performance of a mechanical excavator in the mineral or rock formation is one of the main factors determining the economics of a mechanized mining operation. There arc several methods of prediction and it is advisable to use more than one of lhe.se methods to obtain realistic results. The principal prediction methods are full-scale linear cutting test, small-scale cutting lest (core cutting), an empirical approach, a semi-theoretical approachs and in-situ testing of mechanical excavators.

The full-scale linear cutting test is a reliable approach, since a rock block, 70 cm x 50 cm x 50 m in size, is cut in laboratory with a real life cutter. The cutting force, normal force, sideways force and specific energy values are obtained for different depths of cut and tool spacing values and the production rate of a given mechanical miner is calculated from equation (1 MRostami,1994a).

$$ICR = k \cdot \frac{P}{SE_{opt}} \quad (1)$$

Where ICR is instantaneous production rate, m/h, P is cutting power of the mechanical excavator, kW, and SE_{opt} is optimum specific energy, kWh/m^3.

Small-scale cutting test has been developed from extensive in-situ and laboratory tests and it is widely used. (McFeat Smith, Fowcll, 1977,1979)

Emprical performance prediction models are based mainly on past experince and statistical interpretation of previously recorded case histories. Widely used emprical models depends on many tunneling and mining project datas and prediction models are developed for production rates of axial and transverse type roadheaders and impact hammers (Bilgin 1988, 1990, 1996, 1997, Hartman 1992, Eskikaya, 1998). The Rock Mass Cuttability Index (RMCI) is developed for roadheaders and impact hammers and shows (hat production rate can be predicted by uniaxial compressive strength (UCS) and RQD in equation (2).

$$RMCl = UCSx(RQD/100)^{:/1} \quad (2)$$

where UCS is uniaxial compressive strength, MPa. RQD is rock quality designation, %.

Prediction of roadhcader production rate is estimated using equation (3).

$$ICR=0.28xPx(0.974)^{RMO} \quad (3)$$

Where ICR is instantaneous cutting rate in m'/h, P is cutting power of the roadheader in kW.

Hydraulic hammer performance model was developed using data collected in Istanbul Metro Project. According to this model the performance of a hydraulic hammers can be estimated using equation (4).

$$IBR=4.26P(RMCir"^{w} \quad (4)$$

Semi-emprical performance models utilize computer models. Machine manufacturers, research institutes and consultants have their own computer models(Çopur, 1999, Roslami&Özdemir, 1994b).

For an in-situ machine testing, a new or used machine is hired and tested in-situ (Carlin East Gold Mine; Breitrick, 1998). This method is very expensive and time consuming but gives the most realistic performance prediction results.

5 ROCK MECHANICS AND CUTTING TESTS

Many research works were carried out in the past years to form the fundamental aspects of rock cutting mechanics. Chromite is an important mineral of Turkey mining industry but it has never been subjected to rock cutting tests prior to this study. It is obvious that structural properties of chromite ore will effect cutting mechanism. Mechanical excavators should be consiously used in metallic ores because of their abrasivenes.s. In order to understand better cutting characteristics of chromite ores laboratory cutting tests were performed for this study.

There are two levels of grade in Kayseri Pınarbaşı-Pulpınar chromite mine, these are called high (46-50% Cr_2O_3) (rock 1) and medium (42-46% Cr_2O_3) (rock 2) grade ores. Samples are taken from high, medium grade ores and country rock harsburgite (rock 4) and from Eskişehir Kavak chromite mine which has low grade chromite (20-25% GiOO (rock 3) and surrounding serpantinite (rock 5). Rock mechanics and cutting tests were performed on these samples.

Linear cutting tests were carried out on rock samples using a Sandvik S35-H80 conical cutter. After triming the surface of the samples, depth of cut is adjusted to 5 mm and 10 mm and s/d ratio to 1,2, 3, 4, 5 and releieved and unrelieved cutting tests were carried out. The aim of these tests was to obtain optimum s/d, average cutting and normal force, maximum cutting and normal force, yield and to calculate specific energy using equation 5. Test results give the optimum specific energy which will be used in performance estimation (Roxborough, 1973, Bilgin, 1989, Rostami, 1993).

$$SE_{„,} = \frac{}{Q} \text{ — (MJ/m}^{1} \text{ kWh/m)} \quad (5)$$

wheie SE„ₚ, is optimum specific encıgy, FC ıs mean cutting toıce kN, Q is yield. mVkm in optimum ıatio ot s/d

Rock mechanic test are earned out accoıdıng to ISRM suggested methods and results are given in Table 1

Table 1 Rock mechanics test results of samples tested

Rock	γ	UCS	BTS	E_{st}	E_{dyn}	SHRV	CAI
1	4 03	32 2	3 7	3 5	31 2	28-37	2 12
2	3 39	46 9	4 5	2 3	76 4	43	1 60
3	2 88	46 5	3 7	2 9	35 2	42	2 40
4	2 65	57 7	5 5	2 1	16 1	35 59	0 80
5	2 49	38 1	5 7	2 3	13 9	39 58	1 00

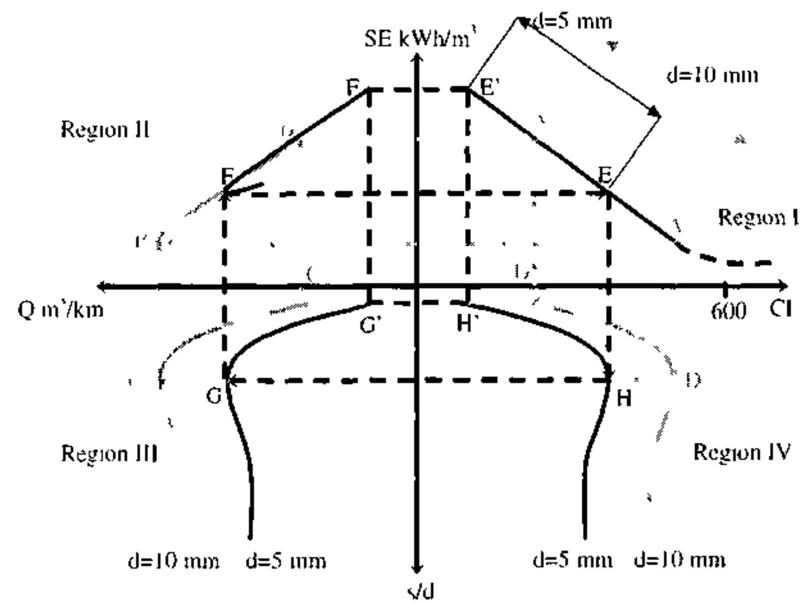
γ = Density (g/cm³) UCS=Uniaxial compressive strength (MPa) BTS=Tensile strength (MPa) E_{st}=Static elasticity modulus (GPa) E_{dyn}=Dynamic elasticity modulus (GPa) SHRV=Schmidt hammer rebound value (N type) CAI=Cerchar abrasivity index

Compiessive and tensile strength tests are earned oui using 55-110 mm (NX) and 55 mm-55 mm core sizes and ELE 3000 hydraulic press Load cell, Ivdt and x-y recorder to measure the static elasticity modulus Pundit equipment is used toi dynamic elasticity N type Schmidt hammei is used both m-situ and laboratory Ceıehaı abrasıvıtıy index test is peiloimed to predict the cuttei consumption

Debus si/e dıstıubıtıon is vciy important loi chıomıte ore +25 mm traction have hıgheı puce on the market Sieve analysis is applied to yield to find the sue dıstıubıtıon on optimum cutting debris Coaiseness index is calculating using sieve analysis results 0 125, 0 5, 2, 8. 25 mm sieves are used m sieve analysis producing 6 ructions thus cumulative sum gives the coaiseness index

6 RESULTS AND DISCUSSION

The relations between coaiseness index (CI) specilic eneigy (SE), spacing to depth ol cut latio (s/d) and yield volume (Q) are gcneiahzed m hypothetical in Figure 2



t-iguie 2 Rtlatıoıis between coaiseness index (CI) and specific eneigy (St) MHO ot spacing to depth ot cut lado (s/d) and yield volume (Q) foi 10 and 5 mm depih ot cut (Tunçdemıı 2002)

In region II ot Figure 2 point (B) has the maximum yield volume and optimum specific eneigy foi 10 mm depth ot cut Point (B') has the minimum yield volume and maximum specific eneigy Point (F) and (F) are same as the point B and B' but they represent 5 mm depth ot cut

Point (C) in region III shows that highest yield volume occuis at optimum s/d foi 10 mm depth ot cut Poınl (C) indicates s/d latio which has the minimum yield volume foi the same depth of cut Point (G) and (C) represent the same phenomenon as point (C) and (C) loi 5 mm depth ol cut

The highest coarseness index for 10 mm depth of cut occurs at optimum s/d which is point (D) in region IV. Point (D') is the minimum coarseness index which is at the minimum s/d tor 10 mm depth of cut. Point (H) and (H") shows the same results as point (D) and (D') but they belong to 5 mm depth of cut.

According to the Figure 2 there is strong relation between specific energy and coarseness index which is inversely proportional. Specific energy for higher depth of cut (10 mm) realizes lower values of specific energy and higher coarseness index values (region I A-A' curve). Coarseness index increases as depth of cut and yield increase. Cutter spacing also effect coarseness index, which has the maximum value at the optimum s/d point. Specific energy has the minimum value at this point.

As a results of these high correlations it may be concluded that specific energy can be estimated by analyzing the particle size distribution of yield for medium to hard rock.

7 EVALUATION OF SELECTED MINES FOR MECHANICAL EXCAVATION

Cutting parameters such as optimum specific energy, maximum and average forces of cutting and normal forces, are determined in laboratory tests for selected mines and results are summarized in Table 2.

Table 2 Summary of cutting test results to selected mines

Rock	s/d_{opt}	d_{opt}	FC	F'C/FC	FN	F'N/FN	CI	SE
1	3	10	395	3.60	272	3.26	395	3.9
2	2	10	516	2.78	379	2.51	431	6.4
.1	3	9	455	3.08	363	2.83	465	5.0
4	S	9	911	2.87	944	241	467	8.4
5	3	9	444	3.17	484	2.65	434	6.2

s = cutter spacing, d=depth of cut. d_{opt} =depth of cut for optimum conditions (mm). FC=mean cutting force (kg). F'C=maximum cutting force (kg). FN=mean normal force (kg). F'N = maximum normal force. CI=coarseness index. SE_{opt}=specific energy for optimum cutting conditions (kWh/m^3)

Results given in Table 2 are used for performance prediction and equations (1), (2), (3), (4) to estimate net cutting rate (m^3/h), production rate (t/h). Results are given Table 3.

Table 3 Performance prediction for selected mines.

Rock	SE_{opt}	ICR1		ICR2		CCR	
		a	b	c	d	e	f
1	3.9	20.7	83	19.7	79	0.53	0.132
2	6.4	12.6	43	15.8	54	0.40	0.118
3	5.0	16.2	47	16.0	46	0.60	0.208
4	8.4	9.5	25	14.1	37	0.20	0.075
5	6.2	12.9	32	17.9	45	0.25	0.100

SE_{opt} = specific energy for optimum cutting conditions, ICR1= instantaneous cutting rate (kWh/m^3) for 100 kW loadheader a(m^3/h)-b(t/h). ICR2= instantaneous cutting rate (kWh/m^3) for 33 kW impact hammer c(m^3/h)-d(t/h). CCR=Cutter Consumption for Roadheaders e(cutter/m^3)-f(cutter/t)

Field studies carried out by Nizamoglu (1978) and Fowell (1993) give die relation between Cerchar abrasivity index and cutter consumption. The prediction equations for cutter consumption are used for selected mines and results are given in Table 3.

According to rock mechanic lest results (Table 1) Pınarbaşı-Pulpınar and Kavak chromite ore and country rocks are classified as medium and hard rock. Thus boom type miners with a power of 100 kW or 33 kW hydraulic hammer can be used for excavation (Bilgin, 1994).

Excavators performance prediction given in Table 3 for high graded chromite, is calculated as below :

High graded chromite ore has the optimum specific energy for 10 mm depth of cut which is 3,9 kWh/m^3 as seen from Table 3. For a boom type miner with a power of 100 kW the instantaneous cutting rate (ICR) may be calculated using equation (1):

$$ICR = 0.8\frac{100kW}{3.9kWh/m^3} = 20.7\ m^3/h$$

Production rate may be calculated using the specific gravity of chromite as :

Production rate = 4,03 x 20,7 = 83 t/h

Hydraulic hammer having a power of 33 kW, will have the following production rate;

For RQD 100% and compressive strength of 32,2 MPa, RMCI may be calculated using equation (2);

$$RMCI = 32{,}2MPa \times (100/100)^{-} = 32{,}2\ MPa$$

Using RMCI in equation (4);

$$IBR = 4.26 \times 33kW \times (32{,}2)^{-0.567} = 19{,}7\ m^3/h.$$

Production rate for hydraulic hammer with a power of 33 kW is 79 t/h.

Pınarbaşı-Pulpınar mine operates 330 days/year. 3 shifts/day and annually output is between 100000 - 120000 tons of ore. Annual output for % 46-50 Cr_2O_3 grade chromite ore with a 100 kW boom type miner and utilization of %50 per shift, may be calculated as :

330 day/year x 3 shift/day x 8 hour/shift x 0.50 utilization factor x 83 t/h = 328 680 t/ycar.

The annuall outputs for above excavation conditions using 100 kW boom type miner and 33 kW hydraulic hammer are calculated and results are aiven in Table 4.

Table 4.Annual output for chromite oie in case die use of mechanical excavators

Rock	CSTO	OT,,,,	OT„
1	100-120.000	328.680	312.840
2		170.280	213.840
3	80-100.000	186.120	182.160

CSTO=Current Status Total Output (1). OT_H»,=Total Output toi l(M) kW roadheader (l/year), OT,,=Total Output for 33 kW impact hammer (t/year).

In this study the mining of chromite ore is investigated only in cutting characteristic point of view. In order to use mechanical excavators for production, underground mine layout, equipments, support and ventilation systems should be reviewed in detail.

8 CONCLUSIONS

Compressive strength of three different chromite ore and country rocks serpantinite and harsburgile changes between 32,2-57,7 MPa and they can be classified as medium to hard rock according to Bieniawski's rock classification (Bieniawski, 1989). Linear cutting tests were conducted on selected ore samples.

Cutting tests and performance prediction estimations based on a boom type miner with a cutting head power of 100 kW for selected mines give a possible increase of annual output from 120 000 l/year to 328 680 t/year (up to 3 times) for chromite ore.

The most important factor that limits excavation method is debris size distribution for chromite ore. The main reason is +25 mm particles have higher price on market. In this case it is necessary to increase annuall output with respect to debris size. Emprical performance models showed that hydraulic hammer will perform coarser debris and capable of high output (312 840t/year from Table 4).

An important selection criteria for mechanical excavator is cutter consumption which occupies maximum share in operational cost. Metallic ore excavation will increase cutter consumption. Thus hydraulic hammer can be more efficient than boom type miners if the tool consumption is considered.

ACKNOWLEDGEMENTS

The authors wish to thank Dedeinan AS. and Turk Maadin A.Ş."s company managers, advisers and employees for permission and help to carry out in-situ investigations.

REFERENCES

Atlas-Copco Robbins. 1996. TBM excavates drift for early ore production. *Project Repon Summary Brochure.*

Bieniawski Z.T.. 1989. Engineering rock mass classifications. *Wiley.* New York. 251 pages.

Bilgin. N„ Seyrek. T. and Shahriar. K., 1988. Roadheader performance in Istanbul. Golden Horn clean-up contributes valuable data. *Tunnels tinil Tunneling.* Vol.20 No:6, p 41-44. June.

Bilgin. N. 1989 İnşaat ve maden mühendisleri için uygulamalı kazı mekaniği. *Binen Yayınevi,* sf.139-140. İstanbul..

Bilgin. N. Seyıek. T.. Erdinç, E. and Shalırıaı. K.. 1990. Roadheaders glean valuable tips for Istanbul Metro. *Tunnels and Tunneling.* Vol 22 No: 10. p 29-32. October.

Bilgin, N., 1994. Yeraltı kazılarında mekanizasyon. *Ulaşımda yeraltı karıları I. sempozyumu,* TMMOB Maden Müh. Od. Yayını, s.53, Aralık, İstanbul.

Bilgin N . Yazıcı S and Eskıkaya S., 1996 A model to predict the peıforıııance of roadheadeıs and impact hammers in tunnel drivages. *Proc. Eurock 96.* p.715-720. Italy-Torino, Balkema.

Bilgin. N.. Kuzu.C . Eskıkaya. Ş and Özdemir, L., 1997. Cutting performance of jack hammers and roadheadeıs in Istanbul metro dnvages. *World Tunnel Congress V7.* Balkema. ISBN 90 5410 8681. pp.455-460

Breitrick. M.E.. 1998. Using a roadheader for underground gold mining. *Mining Engineering,* pp. 43-46. March..

Çopui H.. 1999. Theoretical and expérimental studies of rock culling with drag bits towards the development of a performance prediction model for roadheaders. *Ph.D. Thesis.* Colorado School of Mines, p.362. USA-Colorado-Golden.

Eskikaya Ş , Bilgin N., Dıııçer T. and Özdemir, L.. 1998 A model to predict the cutting perfotmance of rapid excavation systems. *Proc. the Seventh International Svmp. on Mine Planning anil Equipment Seiet lion,* pp.575. Calgary. Balkema.

Fowell. R.J.. 1993. The mechanics of rock cutting. *Compressive rock engineering.* Pergamon press, pp. 155-176

Hartman. H.L.. Breeds. CD. and Conway. J.J.. 1992. Rapid excavation. *SME Mining Engineering Handbook* Chapler:22.1, p. 1876

Mc Feat-Smith. I.. Fowell R.J.. 1977. Rock property testing and culling performance of tunnelling machines. *Tunnels and Tunnelling,* pp.29-32. March.

Mc Feat-Smith. I.. Fowell R.J.. 1979. The selection and aphcation of roadheaders for rock tunnelling. *Proceeding of rapid excavation and tunnelling conference.* Vol.1, pp.261-269. USA.

McFeat-Sniilh I. and Fowell R.J. 1977. Correlation of rock propel ties and the cutting performance of tunnelling machines. *Proc. Conference on Rock Engineering.* The University of Newcastle Upon Tyne. p.581 -602. UK.

Nizamoglu. S, 1978 Contribution a L'étude du fonctionnement ties tunelhers "plain section" et analyse de L'usure de leuis outils de coupe. These pour obtainer Le Grade De Docteur Ing. Ecole Des Mines, p. 139. Nancy.

Özdemir L 1994 Mekanik Kavav Kazısı Prensipleri *Tımıl \e Galen Açına Meslek İçi hçılıııı Seıııııeıı* I I U Maden I akııltesı Editor Prol Dr Şınası TSKIKAYA s 32 bkım
Pakcs G 1991 Selection ot Methods *Win lıl lımıılluif-* No\ p 12
Rost ımı I Ozdemii L and Neil D M 1994a Peitormaiice pıedıctıon a key issue in mechanical haul lock mining *Mining Engmiirimt* 1261 1267 November
Rostami I Ozdemii L 1994b Roadheadeis peiloimancc op timization toi mining and cıvıl constıuctıon *PIMCCIIIIII'\ ni tlu /?" iiiiniitil itLİııuıl mutilant Insliliiu n) Shaft Dillimi, Tulmoh<;\ (ISDT)* 18 21 Las Vegas Nevada Apul
Rostami J Neil DM and Ozdtmu L 1991 Roadheadei aphcation toi llıe Yucca Mountain experimental study facility *CSM* Final lepoit Nevada USA
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fS"1 *International Mining Congress and Exhibition of Turkey-IMCET 2003,* «s *2003, ISBN 975-395-605-3*

Large Mobile Mining Equipment Operating On Soft Ground

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ABSTRACT: The Canadian oil sand deposits of Northern Alberta, operated as large scale bulk handling surface mining operations provide harsh mining conditions in terms of climate and geological characteristics. The ground may be as hard as competent sandstone in winter and as soft as weak clay in summer. The toll on large mobile mining equipment such as > 327 tonne capacity haulers and > 46 m^3 electric and hydraulic shovels is high with structural lives often half that expected. With a move to ever bigger equipment, under the notion that '"bigger is better", the life expectancy of these units is of particular concern. Soft ground conditions cause high rack occurrences that lead to predominantly fatigue failure scenarios, poor stability and incorrect payload evaluations derived from on-board strut pressure information analyzers. This is not solely a concern for mines in the Canadian oil sands, but there exists far reaching implications for surface mining equipment in any soft or weatherable ground operating environments. This paper focuses around the evaluation of payload as affected by soft ground reaction and the effect on frame life for large mobile units, a discussion on the extension of the process to .shovel undercarriage and carbody is made.

I INTRODUCTION

Joseph (2002) introduced the oil sand - equipment interactions program (OsEIP), a joint undertaking by university and consultant researchers, oil sand mining companies and manufacturers of large mobile mining equipment. The focus of the program is an improved understanding of changing oil sand behaviour and associated equipment reactions under static and dynamic conditions. Field tests conducted with > 327 tonne capacity haulers and > 46 m' shovels, including measurement techniques such as passive seismic ground reaction, gave correlations of ground behaviour with equipment duty cycle motion (Joseph, 2002 and Joseph and Hansen, 2002).

In evaluations of truck payload reported by on-board information systems, which report to the mine data management system, which in turn report production Figures to an awaiting plant facility, it was noted that an independent calculation of payload via the sum of forces derived from pressure sensors located at each of the four suspension cylinders (.struts) of the truck were substantially different. Recognizing that the on-board system determines payload at the transition from first to second gear, it was suspected that the effect of soft ground conditions may provide an inadequate stable bearing to allow a consistent measurement of production. Futhermore, on inspection of gear changes, the critical transition was found to commonly occur in the operating pit somewhere between the shovel and pit exit, where little if no road material is added to improve surface characteristics, due to cascading effects resulting in downstream plant interference.

It has been verified by Wohlgemuth (1997), Joseph (2001) and Trombley (2001) that the critical condition for truck frames may be expressed in terms of rack, defined by the difference of the sum of diagonally opposite strut pairs, such that: If L_H, R_F, L_R and R_R denote left front (position 1), right front (position 2), left rear (position 3) and right rear (position 4) struts respectively, then:

$$\text{Rack}= ((L_F + R_R)-(R_F + L_R)j \qquad (1)$$

The life of a truck frame or any rigid mechanical structure such as a shovel carbody or sideframe may be determined in terms of the number of fatigue cycles. Wohlgemuth (1997) and Trombley (2001) expressed an adverse rack cycle as one that exceeded a magnitude of 16 MPa.

2 UNIT CONSIDERATIONS

Although the work conducted by Wohlgemuth (1997) and Trombley (2001) allowed an evaluation of frame life, akin to the procedure adopted by the manufacturer for on-board reporting, it has been recognized that the units of pressure used should be replaced by those of force, due to the front and rear strut pairs being of different active cross sectional area. To enable a similar evaluation of shovel carbody or sideframe performance, force loading at near corner locales is derivable from duty cycle loading; due to the power draw of hoist and crowd systems in the case of cable units (Joseph and Hansen, 2002), or hydraulic cylinder pressure loading In the case of hydraulic shovels. An alternative and more direct approach is to install accelerometers at the corner locales, allowing rack to be determined in acceleration units. Revisiting the suggested units of rack for truck frames, if the load in the truck body is known, then rack may also be determined in acceleration units.

Comparing the use of pressure, force or acceleration units, shows that an adverse rack cycle trigger of 16 MPa is approximately equivalent to an acceleration rack of 1.5g, where g is 9.81 m/s^2. The magnitude of g level will vary, depending on the load in the truck body, however 1.5g is commensurate for an adverse occurrence, with the unit under nominal load. Given values of 16 MPa versus 1.5g, the latter is a more universal descriptor of adverse motion as a performance indicator, recognized by a broader spectrum of people across the industry and may be applied to both truck and shovel structures, and is thus the convention units adopted here.

3 STATIC AND DYNAMIC STRUT RESPONSE

Large tonnage class rear dump haul trucks are typically equipped with four suspension cylinders or struts. The two cylinders at the front of the unit are of a slightly larger diameter than the rear to facilitate greater steering control of the unit, and are designed to accommodate one sixth of a full load each. The rear two cylinders are designed to accommodate one third of a full load each. The front single tire and the rear dual tire arrangements thus allow each tire to be loaded equally under static load on a level bearing surface.

If the bearing surface is level and motion of the unit ensues it is reasonable to expect the unit to pitch (front to back motion depending on whether the unit is accelerating or braking) but not to roll (side to side motion) or rack as described in Equation I.

Where the bearing surface is not level, but undulated as is expected in rough or soft terrain, pitch, roll and rack effects will result.

At rest, any one strut may be overly compressed or extended depending on the ground condition directly below its designated tire set. However as the total load is distributed over all struts regardless of the ground condition and whether one strut is taking its full share of the load, the sum of the load on the struts will remain constant, and be equivalent to the total load. If this sum is determined for the unit at rest, where the truck body is empty, then the tare weight above the struts is determined. Subtracting this from the case where the body is loaded at rest yields the payload.

During motion, where the ground condition is undulated, any one strut may become extended or compressed. In this case the effect of gravity on the load is reduced or enhanced causing a drop or rise in strut pressure. Payload then varies commensurate with this effect and may not give the optimum value triggered by a transition from 1^{M} to 2^{nd} gear.

4 REVISITING NEWTON'S 2^{NU} LAW

When the truck is at rest, i.e. v = 0 and the truck body is empty, then the tare weight of the unit, FTARE. ^{ma}y be determined via the sum of the weight reactions at each of the 4 struts, as gravity is neither enhanced or reduced due to motion:

$$F_{TARE} = g\sum_{i=1}^{4} m_i \tag{2}$$

It should be noted that FTARE only accounts for the weight directly impinging on the struts and does not represent the true tare weight of the unit, as the weight of tires, rims etc. are not included. This exclusion does not affect the determination of payload as they are below the point of reference.

Similarly, when the truck is at rest and the body contains a load, the loaded weight, F_{LO}AD. of the unit may be determined in the same fashion:

$$F_{LOAD} = g\sum_{i=1}^{4} Mi \tag{3}$$

The payload of the unit may then be determined as a difference between the two values:

$$\sum_{i=1}^{4}(M_i - m_i) = \frac{F_{LOAD} - F_{TARE}}{g} \tag{4}$$

In the case where the truck is not at rest, such that v > 0, whether the unit is empty or loaded, it is unlikely that the dynamic weights are the same as the static determinations. In considering Newton's 2^{nc} law applied to the loaded case:

$$F_{DYNAMIC} = \sum_{i=1}^{4} M_i(g + a_i) \tag{5}$$

The mass, M,, contributing to each strut of the unit effectively does not change, therefore we are witnessing a variation in acceleration, a_i, enhancing or reducing the static gravitational constant, g at each strut location. Flexure in the frame of the unit gives rise to different values of a, at each strut on the same unit. The flexure, in turn is also a reflection of the ground conditions giving rise to the phenomenon. FDYNAMIC i* therefore inappropriate for use in determination of the payload.

$$\sum_{i=1}^{4}(M_i - m_i) \neq \frac{F_{DYNAMIC} - F_{TARE}}{g} \qquad (6)$$

Given that the zero velocity condition can be identified and that during this period the strut pressures can be monitored to identify equilibrium values. Equation 4 may then be used to establish the payload of the unit.

For any strut, the impinging mass, m, or M, may be found from the v = 0 condition, as per Equation 2 or 3 respectively, depending on whether the focus is the unloaded or loaded case. Applying this then allows $(g + a_i)$ to be determined under dynamic conditions as described by Equation 5.

At any instance, the dynamic strut pressure data can be used to describe the rack to which the system is subjected, Equation 1, expressing the individual strut response values in number of g's:

$$\# g_{STRUT\,i} = \frac{(g + a_i)}{g} = \frac{F_{DYNAMIC\,i}}{F_{LOAD\,i}} \qquad (7)$$

5 VALUE OF D1MENSIONLESS g UNITS

If the convention of Equation 7 is applied to Equation 1, Equation 8 is defined:

$$Rack = \frac{1}{g}[(a_1 + a_4) - (a_2 + a_3)] \qquad (8)$$

The algebraic difference définition for rack resulting in Equation 8 causes the datum value of g present in Equation 7 to be canceled. This is also the case in determining pitch, roll and bounce, Equations 9, 10 and 11:

$$Pitch = \frac{1}{g}[(a_1 + a_2) - (a_3 + a_4)] \qquad (9)$$

$$Roll = \frac{1}{g}[(a_1 + a_3) - (a_2 + a_4)] \qquad (10)$$

$$Bounce = \frac{1}{g}\sum_{i=1}^{4} a_i \qquad (11)$$

In the latter case, bounce is defined as the sum of the difference between the dynamic and static loading states for all frame suspension elements. Some elements may have a reducing and others an enhancing contribution, such that it is possible for individual struts to be adversely extended or compressed, but overall with a near zero bounce effect.

Thus, the value of the dimcnsionless unit is a measure of the relative g effect in rack, pitch roll or bounce, as may be determined.

6 PAYLOAD

A sample of data was acquired as representative ol typical operation for 44 truck cycles in three sets of operating conditions; firm ground, medium ground and soft ground; judged by the degree of rutting and ground deformation that developed during lhe course of the test periods. For the purposes of this paper, this relative description of ground response is correlated to the stiffness - deformation ground response for oil sand developed by Joseph (2002), reproduced in Figure I.

Utilizing the data acquisition system installed by the manufacturer, the collected data consisted of a time log, strut pressures, unit velocity and payload as determined by the on-board system. The front and rear strut cross sectional areas were also supplied as general information from the manufacturer.

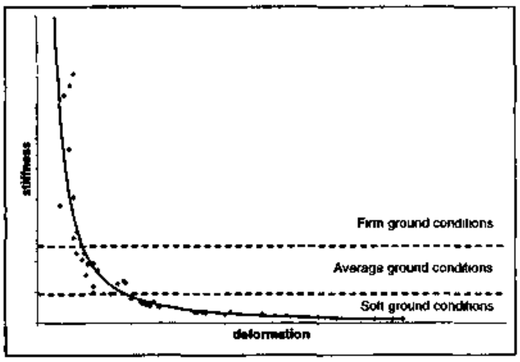

Figure 1 Oil sand ground stiffness - deformation behaviour, after Joseph (2002)

Following the procedure outlined in section 4, the zero velocity condition was identified for both loaded and unloaded states of the unit, allowing the tare weight, F_{TARE}, the loaded weight, F_{LOAD}. and the payload to be determined. This was compared to that reported by the on-board system as illustrated in Figure 2:

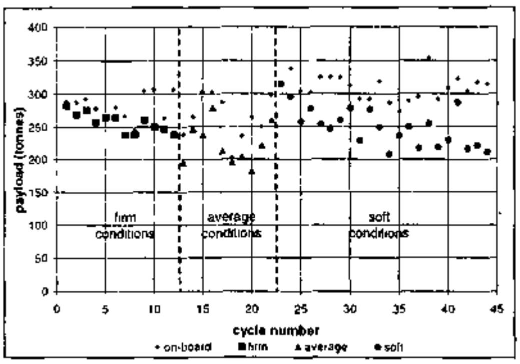

Figure 2 Comparison of on-board to calculated pay load for various operating conditions.

Two trends are immediately apparent from Figure 2. Firstly, that the on-board system consistently over-estimates the payload regardless of the operating conditions and secondly that the softer the ground, the greater the discrepancy between the on-board system and the calculated values. It should be noted that the cycles were recorded consecutively in each of the three operating conditions. This is important when observing the trend of the firm condition data. As the number of cycles advances, the greater the difference between the payload values is observed. This agrees with the earlier work of Joseph (2002). in describing the strain softening behaviour of oil sand with increasing number of cycles. The inability of the truck to record the correct payload on transition from P[1] to 2[ml] gear, due to the undulating nature of the softer ground is clearly visible. Thus in accordance with Figure 1, oil sand may be primarily described as firm, but with increasing cycles may follow the stiffness - deformation trend, through average to soft behaviour as it is worked by loading.

7 FRAME LIFE

About two thirds of the haul trucks within the 44 cycle data set were observed under adverse operating conditions, such that it was suspected that the frame life may become compromised. To facilitate an example, it was suggested that 1 million adverse cycles exceeding 1.5 g in rack may cause structural failure. The design and operating specifications of the units are such that it is expected that the frames should be good for about 10 years under nominal conditions.

For a typical single truck duty cycle. Figure 3 shows the sum of in-line forces at the struts, allowing the empty, loading and loaded portions of the duty cycle whilst operating in-pit to be clearly seen. A typical duty cycle time is about 20 minutes, with two thirds of the time on poor ground conditions in-pit and the remainder on well constructed roads from pit ramp to either dump or crusher locales. Since the in-pit reactions are the most severe, the analysis is restricted to this area of concern, a 13 minute period.

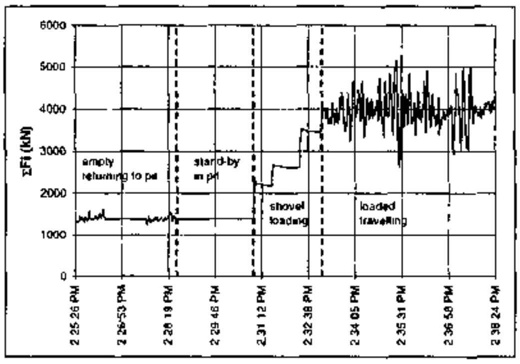

Figure 3 Sum of forces impinging on struts.

The procedure outlined in section 4 was employed to determine strut response in terms of g, with the rack value then determined and illustrated in Figure 4. A comparison of Figures 3 and 4 using the common time base shows that adverse rack is most prevalent when the unit is loaded and in motion.

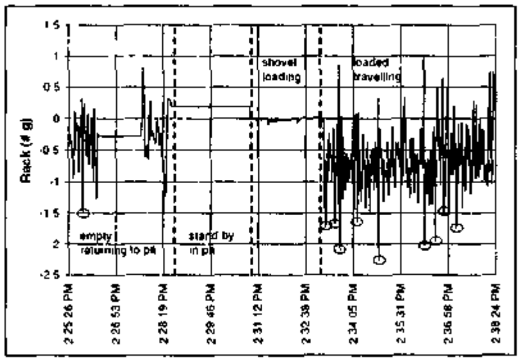

Figure 4 Rack expressed in number of g.

We can define the frame life in simple terms of unit availability (A), utilization (U), rack cycles to failure (F), average number of rack cycles per min (N) and the operating hours per annum (H), given by Equation 12:

$$L = \frac{F}{HAUN} \tag{12}$$

Figure 4 indicates the rack events exceeding 1.5g, totaling 10 events in a period of 13 minutes. If we take typical availability and utilization values of 80% for the unit, for a 350 day operating year, this corresponds to a frame life of:

$$L = \frac{1000000}{350(24)60(0.8)0.8(\frac{10}{(13+7)})} = 6.2\,years$$

If the expected life of the frame is 10 years, then the frame liie estimate is at 62% of nominal.

8 APPLICATION TO SHOVELS

Joseph and Hansen (2002), evaluated the duty loading cycle for a shovel. Further work by the author confirms that duty motions cause the weight distribution on the tracks to adversely rack the carbody when operating at an angle other than perpendicular to (pitch effects) or parallel to (roll effects) the face. The most detrimental rack position was found to occur when the dipper is positioned to dig opposite to, or passes over a sideframe corner in route to or Irom the truck to be loaded.

$$F_{C,H} = \frac{V_{C,H} i_{C,H} \eta_{C,H}}{v_{C,H}} \qquad (13)$$

The extent of loading may be determined from the power draw at the hoist and crowd motors, given that the velocity of motion of the dipper is known, as expressed by Equation 13; where the subscripts C and H refer to the crowd or hoist components, V and i are the voltage and current draws by the hoist or crowd motors, r\ is the efficiency of the system from motor to dipper primarily affected by the reduction gears in the arrangement, and v is the velocity of motion at the dipper. The translation from forces acting at the dipper to the effect at the undercarriage is then merely an exercise in machine geometry relative to snapshots within the duty cycle. However, the short duration of motion for the shovel passing over a corner location, relative to the speed of data acquisition at the hoist and crowd motors, may not permit an adequate evaluation of the load contributions to the undercarriage and a subsequent evaluation of equivalent rack. It is only when the shovel is positioned for some time in the act of corner digging that a clear picture of rack is evident.

Given that the evaluation of rack for trucks, as previously described in this paper was achieved in terms of acceleration units, it seems appropriate that the same units be used to evaluate the rack effect on a shovel undercarriage. As an evaluation of force loading, although simple in principle, may not be practical due to the nature of the machine operation, the least of which is identifying a zero velocity state, a more direct approach for realizing acceleration values is suggested.

The use of accelerometers mounted at machine frame corner locations, such that the rack on the carbody may be evaluated directly for potential damaging occurrences is suggested. Furthermore, if the instruments are located on the sideframes, then an evaluation of the life of these units, which are highly subject to fatigue cracking is made possible.

9 CONCLUSION

It has been shown that firm ground conditions provide a good correlation between on-board payload determination and the evaluation of payload by the method outlined in this paper. However, as ground conditions become softer the difference between the values increases, suggesting that the payloads reported when operating on soft ground may not be sufficiently accurate. In fact, Figure 2 suggests that the difference between reported and actual values may be as much as 100 tonnes, a significant error of up to 45%. It is thus suggested that payload recording while the unit is in motion on soft ground may not be the most reliable reporting of payload.

An approach for estimating frame life, regardless of whether a truck or shovel is being considered has been suggested. An example, using field data from a > 327 tonne unit in operation on soft ground conditions has shown that the life expectancy of a frame can be markedly reduced. In the example, the life of the frame was reduced by 38%.

REFERENCES

Joseph. TG. 2001. OsEIP yeai end progress report. James Piogitlun International Lid. contract with Synciude Canada Ltd.. 16 p.

Joseph. TG. 2002 OsEIP' The oil sands - equipment interactions piogi.im. *Ciinailian Institute <i\ Mining ami Metallurgy Bitlimin.* 95: pp. 58 - 61.

Joseph. T.G. and Hansen, G.W 2002. Oil sands reaction to cable shovel motion. *Cciiuuliiin Institute tif Milling and Metallιιιvv Bulletin.* 95: pp. 62 - 64.

Trombley N. 2001. Vital infoimation management system. Co-Op work term piesenlatton at Synciude Canada Ltd : 24 slides

Wohlgemuth. P. 1997. Snuctuial fatigue clacking failuie trends. *M.Ent; The\is.* Univeisity of Albeita. 42 p

lä" International Mining Congress and Exhibition of Turkey-IMCET 2003, @ 2003, ISBN 975-395-605-3

Dragline Cycle Time Analysis - Case Study

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ABSTRACT: Draglines operate in cyclic nature. Excluding the infrequent walking a dragline spends its time by digging the dirt and paying it out on a spoil pile. Considering a dragline perform tens of thousands of cycles per year, it is obvious that even a small reduction in a single cycle time would result in a significant increase in productivity. Thus it is to the benefit of a mine that dragline cycles are to be critically analyzed and corrective measures taken. Although there exist different opinions on what segments constitute a dragline cycle, in this study it is accepted that a dragline cycle is composed of the following pieces: © loading the bucket by dragging it towards the dragline, © swinging the full bucket along a predetermined arc, ® paying out the dirt onto a spoil pile, © swinging the empty bucket back to excavation face and © positioning the bucket to re-load. The study is based on field investigation conducted on six draglines with different capacities and operating modes. Stopwatch study is performed. Influence of cut dimensions, nature of material excavated, mode of digging, type of bucket employed, swing angle, operator preferences and experience, condition of dragline on cycle time is analyzed. The results are tabulated and presented in tables and graphs.

1 INTRODUCTION

Demand on energy is continuously increasing. Coal, which is the most homogeneously spread raw material throughout the earth's crust, is among the most demanded fossil fuels. A considerable portion of coal is produced by surface mining methods. Regarding the economics of scale extraction methods are highly mechanized and equipment with huge capacity are utilized.

Draglines have been abundantly used in coal mining for decades, either as stripper or stripper and coal extractor. As these equipment possess certain inherent advantages, which their rivals do not, they must be operated in a round-the-clock fashion for high productivity and low costs.

Despite its colossal posture a dragline can be said to have a simple routine of work, which is composed of the following basic procedures: digging and walking. Among them walking is a steady process on which the mine design team has little control. Almost all walking draglines take a step of approximately 2 m within a time period of 0.75-1 min. The design of strip panels, equipping a specific unit with one operator's room on the desired side or with two on both sides and the management's strategy in coal loading operation largely affect the frequency and the length of long deadheading periods, during which the unit is unproductive.

Digging, on the other hand, is a controllable item. It is a repetitive process which mainly consists of scooping, swinging the full bucket along a circular arc of predetermined length, dumping, swinging back and repositioning the bucket for the next bite. It must be immediately noted that transition between these successive components can not be sharply distinguished and therefore there is no common agreement as to what components constitute a dragline cycle. In its simplest form a cycle is a function of scooping, swinging+dumping and retuming+positioning. A cycle however, is described as composing of dragging to fill the bucket, swinging, dumping and returning back to the cut face (Anonymous, 1977; Szymanski et al. 1989; Parlak, 1993; Rai et al. 2000). A last approach adds one more component, which is termed as positioning of the bucket or preparation to drag (Bandopadhyay and Ramani, 1979; Anonymous, 1984; Anonymous, 2001).

2 DATA GATHERING METHODOLOGY

2.1 *Objectives of the study*

This paper presents the results ot a field study carried out to analyze dragline cycle time. The main objectives of the study are as follows:

a. Determining the components of a dragline cycle
b. Analyzing scooping and time spent during this process
c. Exposing the correlation between full swing angle and swing time
d. Analyzing dumping and time spent during this process
e. Exposing the .correlation between back swing angle and swing time
f. Analyzing the differences between full swing and back swing processes
g. Analyzing repositioning and time spent during this process
h. Introducing conditions that determine "hoist-dependent" or "drag-dependent" cycles.

All of the objectives listed above were analyzed for various excavation modes, such as key cutting, main cutting and chopping.

2.2 *Methodology followed*

Out of nine units operating at various surface mines in Turkey six were visited. Basic characteristics of the systems at the time of visits are given in Table 1.

Neither of the draglines was equipped with a duty-cycle recording and data acquisition systems. For this reason a precision stopwatch with 10 lap functions was used for recording dragline cycle components.

Dragline swing angles were measured at a sensitivity level of 5 degrees. The circle along which the dragline can make a full turn was divided into intervals of 5 degree central angles. Recording the starting and finishing intervals, swing angles were determined. The methodology is depicted in Fi«ure 1.

In order lo analyze the influence of location of a particular digging point on the time spent for filling the bucket another method was adapted. The cut was divided into regions on the basis of nearness lo the point on which the dragline is located. Thus, the horizontal and vertical planes were divided into three regions: near, medium and *far;* shallow, medium and deep, respectively (Figure 2). It must be noted here that this method is qualitative and relative in nature, which does not take into consideration real dimensions. When it is considered that dragline cuts are designed in the same manner, it can be safe lo assume (hat the dimensions can be eliminated. For instance, under normal operating circumstances the farthest point a dragline can reach is the vertical projection of the boom sheave on the cut. Likewise the deepest point for a dragline is the one on which the limit of the hoist rope is reached.

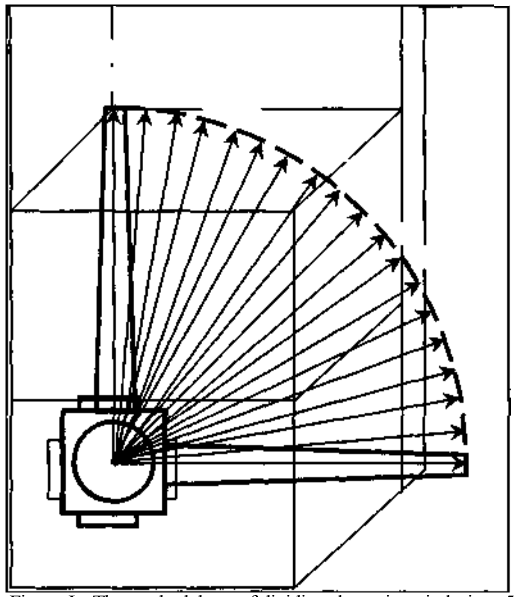


Figure I. The methodology of dividing the swing circle into 5 degree intervals.

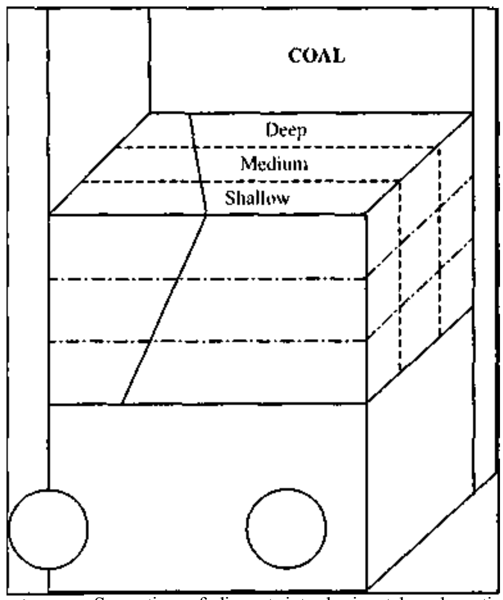


Figure '. Separation of die cut into horizontal and vertical renions.

Table 1 Charactetıstıcs ot'draglines» and panels on which they aie deployed.

Di agîmes	Bucket capacity (in^3)	Pit width (ill)	Bench height (111)	Mode of operation at ihe tune of visit	Overburden Chaiactenstics
Diag#1	22 9	75	25	Bencil prepaıatıon	Blasted marl & conglomeiale
Diag#2	24 S	50	30	Budge prepaıatıon	Rehandled mai l
Dug #3	24 5	65	25	Budge preparation	Unblasted limestone
Diag#4	25 2	60	12	Duect excavation	Blasted marl
Diag#5	49 7	88	32	Woikmg on budge	Rehandled mai l
Diaj!#6	53 5	50	.30	Direct casting	Unblasted clayey marl

2.3 *Discrimination of cycle time components*

As mentioned earlier one of the main difficulties in recording a dragline's cycle time is discriminating successive components from each other. They really seem interconnected. For instance swinging full and dumping appeal as two successive parts of a single operation as do swinging back and repositioning. It is likely this reason that a dragline cycle is accepted to compose of different components among researchers. For the authors of this paper, the solution to decide on what components a dragline cycle could have appeared as watching the operator closely. At the boundary of any two phases all the operators used the drag and hoist levers and swing pedals more clearly and more sharply. Knowing that a dragline's response to an operator's moves take some seconds the method developed during data gathering phase was the continual observation of the bucket.

Accoiding to the methodology given above the digging phase commenced when the bucket was started to drag in towards the dragline. The phase of lull swing started when the bucket cleared off the ground. Dumping began when the mouth of the bucket inclined down and material flown. Backward swing started when the bucket is saved from momentary tixed suspension and accelerated towards cut face. Finally transition to repositioning is discriminated by the conscious effort of the operator in finding a suitable location tor the bucket to position tor the start ot the next dig cycle. It must be noted that in this particular phase, the boom can still be swinging back slowly.

3 CASE STUDY

The ultimate aim of such a study would actually be discovering ways to reduce cycle time and thus increase the productivity of stripping systems by some significant percentages. In this study field observations were conducted at six dragline panels to analyze the components of cycle time on various bases.

3.1 *Digging*

Two criteria were employed in classifying draglines: depth and proximity of the point on which digging started. Though the cuts were divided into blocks of two dimensional pairs such as shallow-near or far-deep, etc. only the results of one-dimensional analyses are presented here.

When key cutting practices are concerned, data gathered from three operations reveal that digging time is positively correlated to the depth and proximity of the digging point. While draglines #2 and #5 worked on easy-to-excavate material, dragline #4 dug blasted bench that was harder. This situation can be observed in Figures 3 and 4.

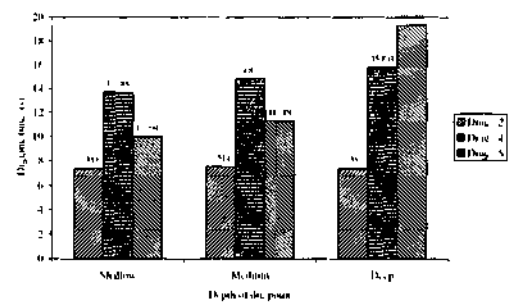


Figure 3. Digging time as a function ot depth ot dig point on key cutting

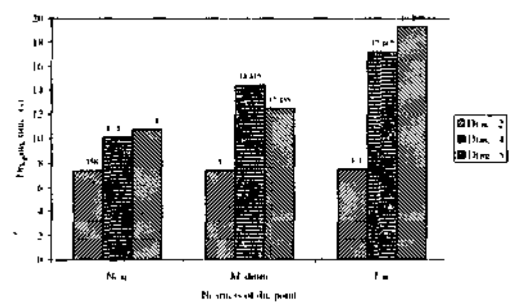
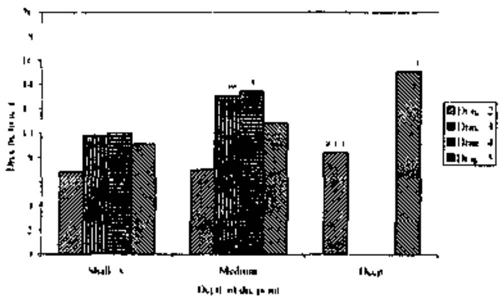
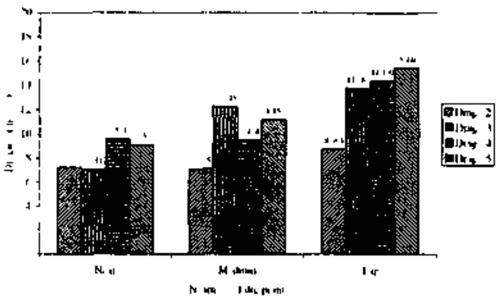


Figure 4. Digging lime as a lunciion ot pioximily of dig point on key cutting

A similar behavior was obseived in main cutting practices Digging tune is positively correlated to the depth (Figure 5) and proximity (Figure 6) ot" the digging point. Time spent at this operation increases with going away from the dragline It should be noted that, like #3, dragline #4 operated on hard material, as well Another obvious point to mention is that digging main cut matenal took 2-3 seconds less than key cut material This difference is attiibutablc to the working spaces



Figuie S Digging time as a function ot depth ot dig point on ni.un cutting



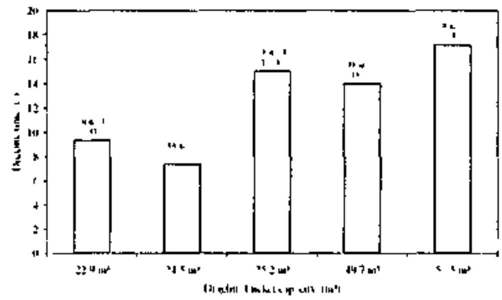
Figuie 6 Digging time as a function ot proximity of dig point on main cutting

There exists a positive correlation between the digging time and the bucket capacity ot units, other parameters being equal Figures 7 and 8 illustrate the cases on key and main cutting, lespecuvely

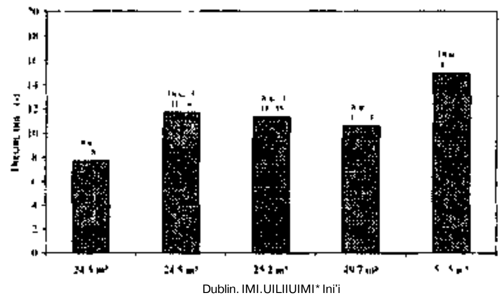
3 2 *Swinging (full & back)*

The time spent during full and back swinging phases weie observed, recorded, averaged and grouped at 5 degree swing angle intervals for various modes of excavation

On all modes ot excavation there exists a strong positive correlation between swing angle and swing time The relations are presented in Figures 9, 10 and 11 toi key cutting, main cutting and chop cutting, lespectively



Figuie 7 Digging time .is a function of diagline bucket capacity on key cutting



Figuie 8 Digging time .is a function of dragline bucket capacity on main cutting

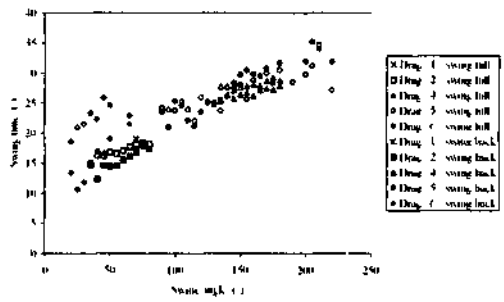
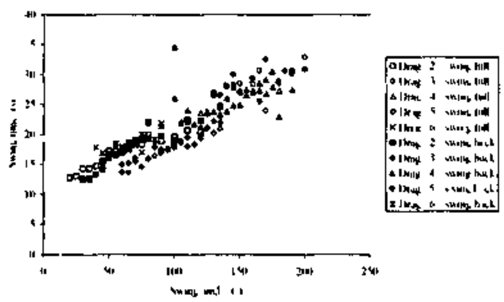


Figure 9 Swing time as a function of swing angle on key cutting



Figuie 10 Swing time as a function of swing angle on main cutting

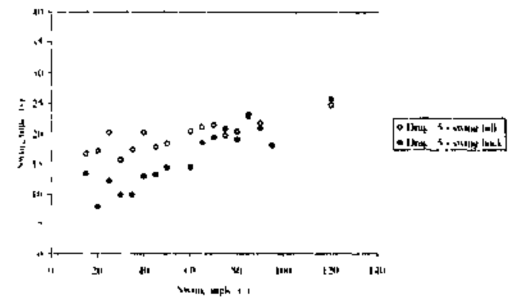


Figure 11. Swing time as a function of swing angle on chop cutting

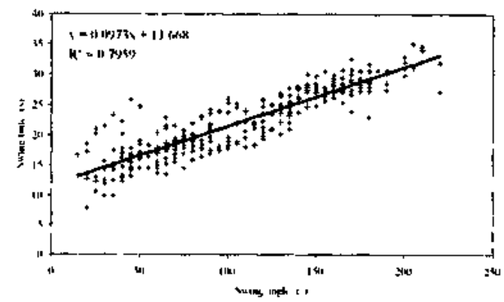


Figure 13. Swing time as a function of swing angle for all modes of excavation

It can be observed from Figures 9, 10 and 11 that swing back times are slightly less than (-1-2 s) those of swing full. This can be attributed to the fact that lesser load is carried by the boom when swinging back. To better visualize the case, all the data gathered in full swing and back swing phases were re-handled and statistically evaluated. The results, which are supportive, are shown in Figure 12.

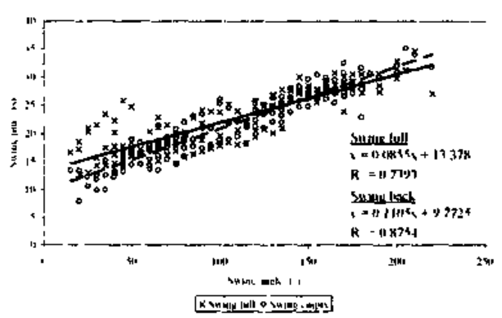


Figure 12 Swing time as a function of swing angle on full and hack swing cycles

A final analysis, which covered all the data from swing cycles, is presented in Figure 13. The data show an irrefutable relation between swing angle and swing time. A regression equation is fitted with an acceptable degree of fit. Table 2 presents the relation between swing angle and swing time in statistical terms for various modes of operation.

Table 2. Results of regression analyses on swing nine.

Operating mode	Regression equation y = swing time, (s) x = swing angle. (°)	Degree of fil (R*. ◁f)
Chop cutting - swing full	y = 0.0708x + 15.727	7.3.01
Chop cutting - swing back	y = 0 1546x + 7 005	S3.18
Key cutting - swing full	y = 0.()742x + 16 003	74.46
Key culling - swing back	y = 0.1125x + 10.223	91.98
Mam cutting - swing full	y = O.I()57x+ 10.039	85.30
Main cutting - swing back	y = 0.1()31x+ 10.170	82.58

3.3 *Dumping*

Bucket dumping is a straightforward procedure, on which very few operational parameters is believed to be significant. Larger buckets may require longer time to dump or the operator may speed up or retard the process. The data observed in this study is presented in Figure 14. Dumping time seems to be within 3-5 seconds for all modes of excavation, which is slightly less than those published previously (Szymanski et al. 1989; Rai et al. 2000).

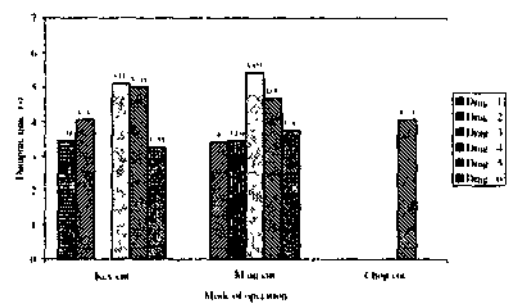


Figure 14. Dumping time for all modes of excavation

3.4 *Bucket repositioning*

Analyzed data for bucket repositioning is presented in Figure 15. Owing to the fact that the working space is obstructed, repositioning lime for key cutting is longer than other modes of excavation in the order of ~ I -2 seconds.

4 CONCLUSIONS

Results of a field study on dragline cycle time analysis were presented. Six dragline operations were visited. The detailed analysis of data gathered indicated the following:

a. Digging time is greatly influenced with the fragmentation of the material excavated. Bucket fiil factors and digging times could well be improved by better blasting practises.

b. Digging time can also be improved by maintaining the bucket in good condition. A proper angle of attack between the teeth of the bucket and the ground and sharp teeth are thought to be essential.

c. Swing times (both full and empty) are positively correlated to swing angles. Since time passed-for swinging cannot be reduced then dragline panel design must be so optimized that dragline swing angles are kept at a minimum.

d. Almost all of the cycles were swing-dependent. In the case of narrow and deep key cuts, cycles tended to be hoist-dependent. Where the swing angles were smaller than 30 degrees, cycles became drag-dependent, which took longer than larger-swing-angle cycles due to longer pay-out processes. They must be avoided.

e. Dumping time and repositioning time are fluctuating within a narrow time interval. They could be taken constants for all modes of operation.

f. Operator's experience is thought to play role on the following phases: digging, dumping and repositioning.

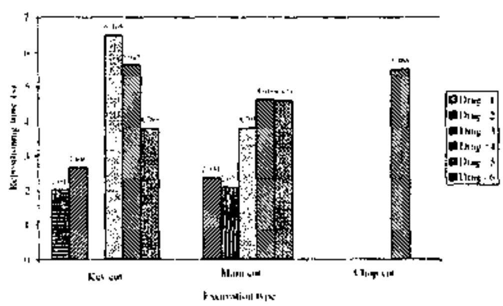


Figure 15. Repositioning time for all modes of excavation

ACKNOWLEDGMENTS

The Research Foundation of Cumhuriyet University is gratefully acknowledged for providing financial support on field studies.

REFERENCES

Anonymous, 1977. *Surface Mining Supervisory Training Program.* Bucyrus-Erie Company.

Anonymous, 1984. *The Fundamentals of the Dragline.* Marion Power Shovel Division. Dresser Industries. Inc.

Anonymous. 2001. *Dragline Production Monitoring.* Aquila Mining Systems.

Bandopadhyay. S. and Ramani. R.V.. 1979. Digital simulation of dragline deployment schemes. *Proc. 16" Int. Symp. on the Application of Computer Methods in the Mineral Industries.* O'Neil. T. (Ed). Society of Mining Engineers. Tucson. USA. 431-448.

Parlak. T.. 1993. *Uygulamalı Kömür Acık işletmeciliği.* Bursa. Turkish Coal Enterprises (TKİ) (In Turkish).

Szymanski. J.K., Borysuk. W. and Williams. C. 1989. Statistical analysis mode! of collected dragline time cycle data. *Division Report CRLNV-I2(TR) CANMET,* 12 p.

Rai. P.. Trivedi. R. and Nath, R. 2000. Cycle time and idle time analysis of draglines for increased productivity - A case study. *Indian Journal of Engineering & Materials Sciences.* Vol. 7: 77-81.

/S"' International Mining Congress and Exhibition of Turkey-IMCET 2003, ® 2003, ISBN 975-395-605-3

Draglines - The Same "Old" Stuff?

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ABSTRACT: Many people have labeled draglines the "Dinosaurs of the Industry". The perception is that they have evolved at a far slower rate than the industry in which they arc deployed, and "is it fair to expect a machine that is already working in excess of it's initially anticipated life of 30 years to adjust to the demands of today's market?"

Much research has been performed on the application of, as well as the technical aspect of draglines, with papers being presented at most Dragline Users forums and Technical Conferences hosted throughout the world. These papers have covered subjects ranging from bench stability, to use of modified motors, to determining the excavatability of draglines. But what is being done now to enhance the productivity of these machines?

This paper takes a brief look at productivity enhancing modifications that can be applied to draglines today. Scheduling packages and other computer generated aids, digital upgrades utilizing the latest digital control technology, and new boom geometry configurations with associated motion control technology are just some of the enhancements that are currently being developed to bring these machines into the 2 l^{sl} century. Some of these technologies, at time of writing the paper, are still probably considered to be in the development phase, but the magnitude of the initial results justifies sharing with world dragline users.

1 INTRODUCTION

Our perceptions of the pre-historic era are generally of large reptiles foraging in the forests and plains of the world. Life was slow - except when under threat by a T-Rex - and Steven Spielberg's Jurassic Park further emphasized that, and the entire world became more aware of the earth's early inhabitants. These predominantly gentle giants just "plodded along" from day to day.......

Draglines have generally been the equipment of choice when it comes to stripping of coal overburden. Yes, there are shovel/truck applications exposing coal around the world, as there are draglines in other applications such as phosphates, bauxite, diamonds and oil sands, but the majority of coal mines have the dragline as the primary mover of overburden. Although they have a considerable capital outlay, they generally are the cheapest method of moving overburden. These monsters of the mining industry just seem to "plod along" from day to day, moving overburden and exposing coal, moving overburden and exposing coal, moving......Little wonder they get the name of "Dinosaurs of the Industry". The one problem is however, that the reference to "dinosaur" is also used in another connotation - that of being "prehistoric" and therefore "out of date". Is this a justified comment? Is the coal mining industry hanging on to an archaic technology? Or is it really archaic? What's been done to make dragline operations more efficient, productive and cost effective?

2 A QUESTION OF POPULATION?

Probably the most visible area the Mining Industry sees improvement is that of Shovels and Trucks. The last 15 years in particular has seen huge growth in sizes of both Shovels and Trucks, and the population of equipment out there certainly has attracted the attention of developers of technology. We see lots of advertisements on Shovels as well as Trucks and the technology to make them more productive and cost effective. Some of the developments the industry has seen are:

- Larger dippers to load the large trucks in 3 passes
- Digital control technology to lower cycle time
- "Throw away" truck bodies to increase payload

- Remote monitoring of vital signs to better identify imminent problems
- Predictive Diagnostics offers proactive rather than reactive maintenance
- Current developments to minimize the repetitive tasks the operator has to perform
- Some truck manufacturers have been looking at "autonomous" trucks

The above is merely a small sample of what has been going on in that area, and we all know about it - why? There is a huge population of shovels out there, and an even greater population of trucks, so they are "getting all of the attention", we see the adverts, we go to mining shows and see these trucks in "real life" - proudly on display for everybody to "kick the tires". So everybody has a heightened awareness of the technological advancements in that area.

There are far fewer draglines than shovels in operation around the world. They are large and therefore we cannot take them to shows. They lack the "action" and excitement" of a highly productive truck and shovel operation. There are not many units sold - probably averaging 1 every 18 months recently, therefore we see fewer press releases and overall they just have a lower profile.

3 USER GROUPS

The smaller population has created a smaller support community, but that's about where the use of "small" ends. The regional dragline user groups are extremely active, addressing issues such as:

- Maintenance practices
- Productivity improvements
- System upgrades
- Application concepts
- Parts longevity
- Parts Pools

Most of the development work on Draglines has been initiated through interaction with these user groups, and there has been a lot of development. We work very closely with the user groups around the world to keep pace with their requirements. This close knit community of users has been responsible for initiating many of our R&D initiatives.

4 ARE DRAGLINES OBSOLETE?

Operations are still buying them, they are still investing in development programs, and many projects that are still "on the table" have draglines as the equipment of choice for overburden removal. Dragline user groups still meet with OEM's with a view to improving reliability and productivity and OEM's still invest R&D dollars. Many existing draglines still have many years of operating life ahead and need to be productive and cost effective. So in essence, no, they arc not obsolete.

5 ARE DRAGLINES OLD TECHNOLOGY?

The concept of the wheel goes back some time, millennia in fact, and although designs and materials may vary, the concept has remained and has definitely not become outdated. The dragline is probably no different, other than being somewhat "younger", the concept goes back a long way relative to our generation, and this basic concept is still what remains to date. Having said that, does that mean they are "old technology"? On the contrary, the latest technology is applied to new draglines and the developments are conducted so that they can be applied to older machines. What are these developments and technologies?

6 DESIGN DEVELOPMENTS

Sophisticated computerized design technology enables designers to build complete machines before s single steel plate is cut for machine production. This ensures that designs are optimized for the payload and geometrical requirements of the specific operation. It is worth mentioning at this stage that although draglines are really individually tailored to the operation into which they are going, designers consider the applications of their work to the existing fleet already in operation as well. The one issue with these design developments is that they manifest themselves in long term benefits, in other words, they don't necessarily display immediate benefits. Long term issues such as reliability and availability are definitely sought after characteristics, but somehow lack the "pizzazz" of something like a reduction in cycle time which can be measured immediately.

7 TECHNOLOGICAL DEVELOPMENTS

What has had the most impact in recent years is probably Digital Drive Technology. This has replaced the old analogue with digital technology and has allowed us to stretch the performance of motor characteristics without compromising the reliability of the equipment. We are now able to have motors perform over a wide range of peak power, optimizing the drag, hoist and swing motions, resulting in shorter cycle times, translating to higher productivity. New draglines have the digital control technology as standard, but as mentioned, we develop technology with a view to retrofit existing models in the field. Where we have installed these retrofits, productivity improvements in excess of 10% have been regularly recorded.

As can be seen in Figure 1 below, this example shows the drag motion of a 1570 W dragline prior to a digital drive control upgrade. As can be clearly seen, the drag motion is limited to 700 RPM (at the motor shaft). By utilizing a constantly variable digital control technology, this speed limit can be automatically adjusted to increase overall line speed to 1050 RPM or 150% of 700 RPM (see Figure 2) and keep within the motor commutation limits. Obviously this means a performance increase, and it can be obtained in all dragline motions.

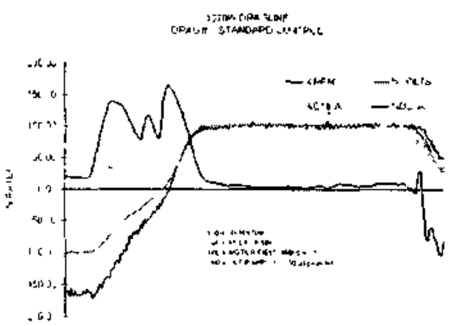


Figure 1 Drag motion of a 1570 W dragline

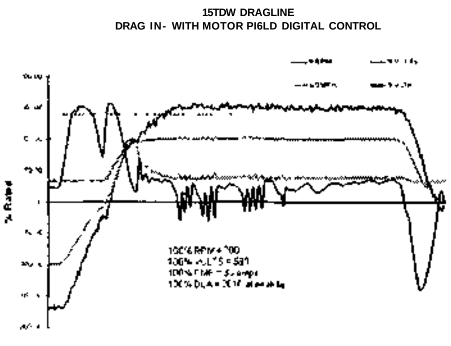


Figure 2 1570 W Dragline with a constantly digital control

There are other technology advancements through the addition of PLC (Programmable Logic Controllers) that are also becoming very popular with dragline owners, they include:

- Boom profiling (anti lightline)
- Propel shoe synchronization

Boom Profiling - this feature simply monitors the position of the bucket relative to the boom based upon the amount of hoist and drag rope paid out and limits operator joystick reference when the bucket approaches the boom. The operator control is limited based upon the speed that the bucket is approaching the boom as well. This simple innovation saves through technology saves many hours of lost production and repairs should the bucket come into contact and inevitably damage the boom structure.

Propel Shoe Synchronization - this feature saves much potential walking structure damage. The theory of operation has propel shoe synchronization monitoring the position of both walk shoes via positional input devices that are connected to a PLC. The PLC watches for discrepancies during the rotation of each shoe. Should a mismatch in the position of any shoe relative to the position of the other occur, then the PLC send a reference signal to the drive control to either speed it up or slow it down until both shoes are synchronized. Simple, but very effective, and no doubt a great maintenance enhancement.

8 PLANNING AND SCHEDULING PACKAGES

There are companies that have made significant investment in developing packages for use in dragline operations, cut diagram generators and scheduling packages enhance the effectiveness of the mine planner. Many more scenarios can be tested "on paper" before making a final decision on a mining technique. The designers of these packages continually improve their offerings to the industry as the available technology allows them to do so. These packages, although they have nothing to do with the design of the dragline, enable the operations to optimize the use of their equipment.

9 WHAT'S GOING ON NOW?

We mentioned earlier that the concept of the dragline had really not changed since the "early days". The concept of drag, hoist/swing, dump, lower/swing is still the same. Design technology has enabled us to increase payloads, or change the geometry of the machines, and digital drives have enabled us to speed up that process, but we have been strictly limited by geometrical constraints and the machine and the cycle elements - especially the digging and dumping cycle.

Each dragline has a "digging envelope" in which it operates, and "digging" generally contains the following elements:

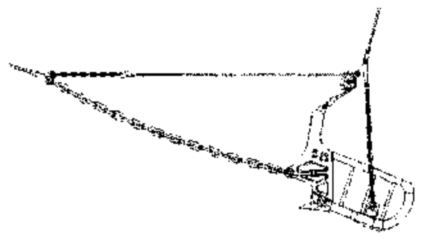
- The bucket has been lowered to the point at which you wish to start digging
- Drag the bucket until -
    - it is full, and
    - it is at a stage in the digging envelope that it can be picked up without spilling much of the load as it "nods" on pick up
- Pick up and start swinging

The '"dumping" portion of the cycle is limited geometrically since material can only be dumped at the boom point position of the operating radius. It is possible to dump closer in, but utilizing techniques that are unproductive and inefficient. Draglines have therefore been limited in their ability to choose the dumping position. The result is that much unnecessary time is spent in the dig cycle, and material is generally placed at a point with no alternative option, other than inefficiency.

There has been an alternative to this, a concept that is not necessarily new, but the availability of digital control technology has really given us the ability to implement it - UDS, or Universal Dragline System has arrived!

10 UDS (UNIVERSAL DRAGLINE SYSTEM)

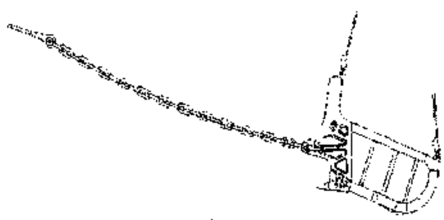
What is UDS? - The conventional rigging system of a dragline is represented in the diagram below:



(a)

Figure ? Conventional rigging system of a dragline

UDS takes a new approach. The rigging is significantly reduced. Instead of the two hoist ropes attaching to an equalizer - which attaches to the dump block assembly - which in turn is connected via chains to the hoist trunions, the hoist ropes attach directly to the bucket - one on the arch and one at the back of the bucket. The dump block, ropes and chains are eliminated since the hoist ropes operate independent of each other, bucket carry angles can therefore be controlled and set by the operator.



(b)

Due to the changes in the hoist rope configuration, the boom of the dragline requires modification to have the sheaves one behind the other versus side-by-side, see diagram below.

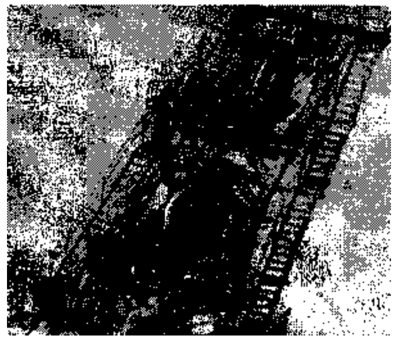


Figure 4 Modified dragline boom to have sheaves one behind the other vs side-by-side

The diagram below gives an indication of the new configuration on a dragline.

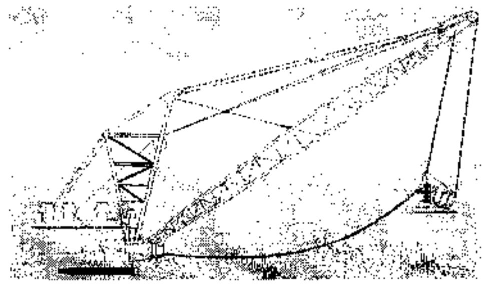
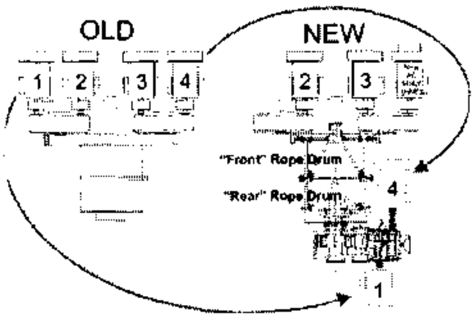
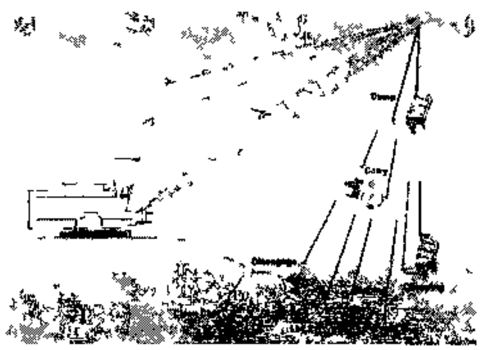


Figure 5 New configuration on a dragline

The controls of the hoist have to be changed in order to go over to the new configuration, in this instance the hoist drum is also split giving the ability to control the ropes independently. There is some deck layout modification required for this.



Since this paper is directed more at the application ot the UDS rather than exactly the setting up or it, we will leave the technical description tor another torum The result ot the above modifications is that the dragline operator is now able to dig and dump in positions that are most convenient



UDS has a number ot benefits

- It ıeduces the number (and therefore mass) of rigging components - the weight savings can be converted to payload
- Reduction in cycle time - less time spent digging
- Absence ot dump block and spreadei bar gives extra dump height - which equates to less rehandle due to lower bench height
- Less down time due to rigging maintenance - especially dump ıope change out
- Bucket can be picked up from digging mode as soon as it is full
- Dumping can take place up to 40m inside of boom point - without use of "special" techniques - much quicker and more efficient pad prepaıatıon and bridge building
- Faster hoisting speeds due to improved geometry - due to the hoist controlling the angle ot cany, the use ot drag in no longei required to keep bucket at carry angle
- Cany angle can be automated by use ot PLC

The bottom line is that trom the results ot the pilot pioject and field installation, production benefits ot well in excess ot 20% seem to be attainable It is perhaps a little early to fully justify the benefits, but numbers ot this magnitude justily a little more attention, and we will be seeing moie measuiable benefits in the not too distant future

11 TRAINING

All ot the productivity improvement tools carry a price, but there is one tool that has not been mentioned thus far, which also carnes a puce but which really has an influence ovei the effectiveness ot any ot the other tools - TRAINING - of opeiators as well as maintenance crew This is an element that has historically been the most neglected by many operations around the globe, but really has the most impact Training is not an event - it is a process, and we should be striving to ensuie that bad habits do not creep into opeıatıons These habits impact not only productivity, but also more importantly - SAFETY'

One of the moie recent developments in the training field is that ot "virtual reality training Here the opeiator is been put behind the controls ot a shovel or behind the wheel of a truck and exposed to just about every possible scenario they could encountei in the field - WITHOUT LEAVING THE TRAINING FACILITY' At this stage, true to the trend outlined eaılıer about shovels and trucks getting the high profile, this has indeed happened Diaglines are definitely on the radar screen foi these companies and the Industry should see something soon

12 CONCLUSION

So, are draglines "Dinosaurs ot the Industry"' My contention is that their size may cause them to be viewed as such, but that's where the analogy should end Much has been invested in productivity improvement tools and it's really up to us to capitalize on them and put them to use

ACKNOWLEDGEMENTS

I wish to extend my appreciation to the following

- P&H Mining Equipment lor giving me the op poitunity to present this paper
- P&H MinePio Sei vices - Australasia who have heen involved in the development ot the UDS and provided me with the information pie sented on the UDS
- Leigh McKenzie - Director of PicH Model mzation and Upgiades Division who pro vided intoimation on digital duves

lä^h International Mining Congress and Exhibition of Turkey-IMCET 2003, <ö 2003, ISBN 975-395-605-3

Haulage Optimisation in An Australian Underground Metal Mine

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ABSTRACT: One of the most important factors in the economical operation of an underground metalliferous mine is the cost associated with ore haulage. Four haulage operating scenarios were examined to optimise the haulage .system in an Australian underground metal mine based on the economic evaluation of mining at increased depth. The conclusions drawn suggested that, for the short term, the implementation of the Elphinstone AD55 mining trucks combined with the implementation of a transfer conveyor on 9 Level would be the most economically viable solution to the continuation of ore production from the mine. With respect to the long-term future of the mine, a detailed investigation into the possibility of the establishment of a new shaft and hoisting system was recommended into the possibility of the establishment of a new shaft and hoisting system was recommended.

I INTRODUCTION

Most underground mining operations that utilise ore transportation in the form of shaft hoisting, truck haulage or conveyor systems will reach a stage in which increasing distances between production areas underground and the surface, or crushing/hoisting system start to have an adverse effect on operational costs. If this problem were not properly addressed, the economic viability of mining, particularly from deeper deposits would be questionable. A case study involving an underground metal mine in western New South Wales, Australia, was conducted with respect to the various options available for the expansion of the ore handling facilities. The aim of this case study was to determine the most economical and effective proposal that will allow the continuation of mining at depth.

The mine, known as MINE A, is located approximately 750 km North-West of Sydney, NSW. The production capacity of the Mine at the time of study was set at approximately 500,000 tonnes per annum with future plans to expand the production to 747,600 tonnes per annum.

Mineralisation of the ore deposit occurs in several copper and copper-lead-zinc systems in parallel zones in sequence of thinly inter-bedded siltstones and fine-grained greywacke. Each system consisted of veins, veinlets, stockworks, and disseminations of base metal sulphides. Economic mineralisation occurred over a strike length of 400 m and at a depth *of* possibly 2000 m. The geology *of* the deposit is shown in Figure I.

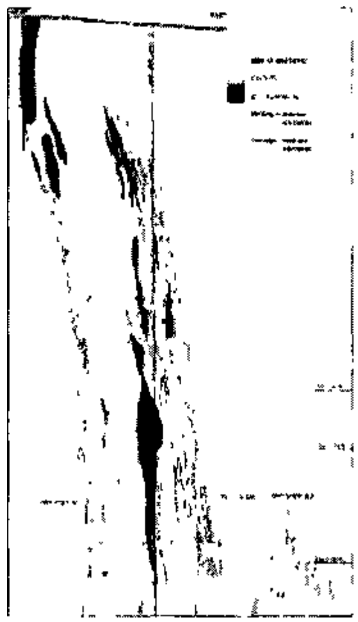
2 MINING AND ORE TRANSPORT

The main levels in the mine are numbered 1 to 11. 9 Level (5909 RL) housed the crushing facilities and 11 Level (5715 RL) was the existing main level, with production ore being sourced from beneath this level. The mining method used was longhole open sloping.

The ore handling facilities at the mine was based on diesel Load-Haul-Dump vehicles, which loaded broken ore from the draw points into low profile diesel trucks (Elphinstone AD/AE 40 II) that hauled the ore to the crushing/hoisting facilities. The ore handling facilities include:

- No.l Shaft hoisting system
- No.2 Shaft hoisting system
- Crushing station (9 Level)
- Loading station

The No.l Shaft was used as an upcast ventilation shaft and emergency escape way for underground personnel until. 1991. Increasing truck haulage costs, when mining commenced below 11 Level, necessitated the installation of the internal winding system in the 4.2 m diameter shaft. Ore was trucked from the production levels to the 10 Level tipping station, where it was dumped on a 700 mm 700 mm apeiture grizzly Larger rocks, greater than 700 mm size, were broken on the grizzly with an impact rock breaker The broken ore was then weighed and loaded into an 8 5 tonne skip where it was then hoisted to 8 Level. As the skip approaches 8 Level, it deceleiated until it was stationary in an overturned position. The ore from the skip gravitated down a 3 m diameter, steel lined ore pass to 9 Level, where it was then transteired to the 9 Level crushing facilities via LHD, which is not shown in the Figure 1



Fıguıe 1 Deposit geology

Schematic ot the internal hoisting system is shown in Fıguıe2.

The No.2 Shaft (5.5 m diameter) hoisting system consists ot three sections that are the crushing system, loading station and the shaft hoist. The ciushing system was located at 9 Level and facilitated the loading of uncrushed ore at the tipping grizzly. The gnzzly is positioned above a storage bin and ore was fed into a Jaques single oscillating jaw crushei that crushed the material to -150 mm. The crushedmatenal was stored in two 1300 t capacity storage bins located directly below the crusher.

The loading station was located directly below the ciushed ore storage bins and adjacent to the No.2 Shaft Ore is fed onto the loading station belt where the ore is weighed before being transferred into the skip foi hoisting to the surface.The shaft hoist consisted of a tower mounted ASEA fnction winder hoisting a single 14 t payload skip which was counterbalanced by a 6 t payload skip/cage. Ore was hoisted from the loading station to the surface skip dump station that fed an inclined conveyor, which fed four surface storage bins to the mill. The capacity of the hoisting system was 300 t/hr from a depth of 860 m and at rope speed of 12 m/s.

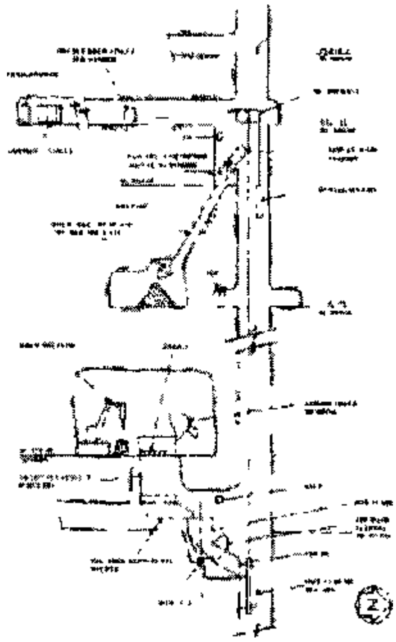


Figure 2. No. 1 shaft internal winder

3 PRODUCTION SCHEDULE

Production rates were based upon 500,000 tonnes/year, increasing to a maximum 747,600 tonnes/year, most of which was obtained initially from Ore System 3 and later on, in 2003, the production will come from Ore System 1. Production form Ore System 1 would prolong the working life of the mine for a maximum five years, based upon current proven and probable reserves. Deep drilling conducted in Ore System 3 showed a strong continuity of the orebody to a depth 550 meters below the current workings. This provided the potential to extending the life of the mine beyond five years. The opportunity also existed to mine ore below the planned cut-off grade at marginal cost once the operation was profitable and stabilised.

4. DIESEL TRUCK HAULAGE

The transportation of ore, men and materials in an underground mining environment contributes significantly towards the mine's running costs, every underground mining environment the ideal objective of transport is to provide a means whereby transportation is achieved at the lowest unit cost. In terms of ore transportation, the unit cost is often expressed in dollars per tonne ($/t). Recent activity in Australia with open pit gold mines looking to develop into underground has seen a great deal of comparative work being undertaken on the question ot shafts or declines. As a result, tonnages of 1.5 Mt/yr were achievable with truck haulage from a depth of 1000 m, via decline alone (Chadwick, 2000). As mine A was equipped with shaft hoisting facilities, the renewed application of truck haulage was timely and essential.

The trucks that this paper have examined were the Elphinstone AD/AE 40 II, the recently released Elphinstone AD55, the Tamrock Toro 50D and the Atlas Copco Wagner MT5010. The reason behind this selection was that all were low profile articulated trucks, with dimensions very similar the existing AD/AE 40 II trucks. Any trucks that were bigger could not fit down the decline and the cost of dismantling for transportation underground via the No.2 Shaft proved prohibitive. The trucks included in this study are shown in Figure 3 to 6. The capital costs of these trucks were as follows:

Elphinstone AD/AE 40 II: A$950,000
Elphinstone AD55: A$1,250,000
Tamrock Toro 50D: A$ 1,188,000
Atlas Copco Wagner MT5010: A$l ,300,000

Figure 3. Elphistone AD/AE 40 II

5 OPERATIONAL PARAMETERS

Operational parameters include costs and production rates.

Figure 4. Elphistone AD55

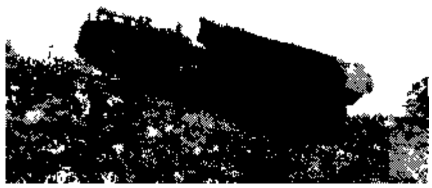


Figure 5. Tamrock Toro 50D

Figure 6. Atlas Copco Wagner MT5010

5. / *Costs*

Apart from capital costs, two other categories of cost were considered. The first category was the owning costs, where factors such as depreciation, investment or interest cost, insurance cost and taxes are borne by the company, regardless whether the machine is utilised or not. The second category of costs, the operating costs include fuel, lubrication, filters, repairs, tyre replacement, operator wages and miscellaneous items. As expected, these factors are widely influenced by the nature of work, local prices of fuel and lubricants, shipping costs from the factory, interest rates and local labour rates. The investigation of various operating costs of mining operations proved to be very difficult as many companies regard the operational cost figures as confidential, in order to remain competitive. Hence, the need or reasonable cost estimation procedures become nevitable, as any attempt to compare operational ata would be fruitless.

5.2 Production Rates

Production rates are also affected by a range of anables such as rolling resistance, acceleration nd deceleration, variations in haulage grade, oading and queuing time. However, it is ecessary to generalise and make assumptions or the variables involved in production rates, ften this may be based upon personal experiences or when access to a computer simulation package does not exist. The following formula for the calculation of cycle time was used:

$$T_t = SL + SUL + \frac{d}{60}\left(\frac{1}{v_u} + \frac{1}{v_d}\right) \text{..... (Northcote \& Barnes, 1973)}$$

Where,

Ti. = Cycle time (min.)
SL = Spot and load time (min.)
SUL = Spot and unload time (min.)
d = Haulage distance (m)
Vu= Uphill travel speed (m/s)
Vd= Downhill travel speed (m/s)

To calculate the productivity of each truck, factors such as machine utilisation and availability were taken into account. Assuming a machine utilisation lactor of 75 % (available time per shift after cribs, shift meetings and travel time to jobs) and an availability factor of 80 % (available time per shift after mechanical breakdowns, repairs and maintenance), productivity can thus be calculated usine the following formula:

$$Productivity = \frac{60}{T_t}(C_t \times \alpha \times \beta) \text{ (tonnes per hour)}$$

Where,

d = Truck capacity (tonnes)
a = Machine utilisation factor (%)
b = Machine availability factor *(%)*

The production characteristics for the four trucks arc as shown in Table 1.

Table I. Production characteristics

	AD/AE40 II	AD55	Toro 50D	MT 5010
C_t (t)	40.00	55.00	50.00	50.00
SL (min)	4.20	4.20	4.20	4.20
SUL (min)	1.20	1.20	1.20	1.20
α (%)	75.00	75.00	75.00	75.00
β (%)	80.00	80.00	80.00	80.00
v_u (m/s)	2.22	2.5	2.64	2.50
v_d (m/s)	4.72	4.72	4.72	4.72
d (m)	1000.00	1000.00	1000.00	1000.00
Productivity (t/hr)	87.64	126.94	118.66	115.40

6 CONTINUED TRUCK HAULAGE TO 10 LEVEL

The evaluation of diesel trucking requirements was based on the production schedule. The average haul distances were used to simulate trucking requirements and a number of variables were also incorporated into this simulation. The operating costs of the existing system was also examined, including the unit costs of road maintenance, hoisting, transfer, crushing, and the implementation of the four different mining trucks and a conveyor performing transfer duties on 9 Level. The ultimate objective was to determine the lowest operating cost per tonne. Table 2. shows the average haul lengths. A list of parameters that was used in the simulation is shown in Table 3.

The costs of continued truck haulage to 10 Level consisted of the following:

- Road maintenance = A$0.26/t
- Internal hoisting = A$0.78/t
- 9 Level transfer LHD = A$l .39/t
- 9 Level crushing = A$1.50/t
- Hoisting 9 Level to surface = A$2.00/t

The operational costs for the mining trucks (excluding driver costs) are:

- AD/AE 40 II = A$65/hr (from mine cost reports)
- AD55 = A$79/hr (Six Tenths Rule)
- 50D = A$75/hr
- MT5010 = A$80/hr (sourced from Atlas Copco)

The costs associated with truck drivers/operators were made up with basic wages. However, secondary costs such as employee insurance, payroll tax and superannuation contributed to the total cost carried by the company. A common method to approximate total employee costs was to use multiplication factor, which, in this case, was 2.0. Thus a typical wage for a truck driver/operator with an annual salary of A$65,000 a year, would amount to about AS 130,000. Since the mine operates on two

12-hour shifts per day, two truck operators are required per truck, which equates to A$260,000 per truck per year. The capital cost of purchasing and replacing vehicles must also include the capital cost ot upgrading the mine ventilation system, as the increasing fleet size, would contribute to higher diesel exhaust output.

Table 2. Average Haul Lengths

	Month	Average distance to 10 L (m)	Average distance to 9 L (m)
2001	H1 (June)	2847.14	4244.14
	H2 (Dec)	2900.75	4297.75
2002	H1 (June)	3362.35	4759.35
	H2 (Dec)	3355.00	4752.00
2003	H1 (June)	3464.00	4861.00
	H2 (Dec)	3572.50	4969.50
2004	H1 (June)	3413.75	4810.75
	H2 (Dec)	3302.25	4699.25
2005	H1 (June)	3950.00	5347.00

Table 3. Parameters used in simulation

Op. Parameter	Unit	Op. Parameter	Unit
Truck cap	t	β	%
Truck fill factor	%	Shift length	hr
Actual capacity	t	Number of shifts/day	-
Req. daily production	t	Tonnes hauled by each truck	t
Total haul distance	m	Sched. Op. days per year	day
Haul velocity	km/hr	Sched. lost op. days per year	day
Return velocity	km/hr	Machine op. days per year	day
M	min	Op hrs for each truck per year	hr
SUT	min	Truck life	hr
Haul time	min	Potential Production	t/hr
Return time	min	Potential Production	t/day
Cycle time	min	Trucks required (theoretical)	-
α	%	Trucks required (actual)	-
		Potential prod. per year	t

7 TRANSFER CONVEYOR IMPLEMENTATION

The option of installing a Transfer conveyor system was under consideration for some time and it was anticipated that it will become a reality in near future The conveyor drive was already completed and all that required was an investigation into the design and operation of such a system. The operating cost of such a system was expected to be low, as it is largely dependant on the conveyor design and operation of the ore pass discharge arrangements. The design of such a system would be to ensure that the ore pass will not be empty, ore pass hang-ups are minimised and the feed to the conveyor would be controlled to allow minimum impact and wear. It was also considered that the implementation of the transfer conveyor would eliminate the cost incurred from a LHD performing transfer duties (A$l.39/t). The capital cost per component of the transfer conveyor system is shown in Table 4.

Table 4. Transfer conveyor capital cost

ITEM	COST (A$)
Conveyor Drive	N/A
Bunker Discharge System	$180,000.00
Conveyor	$258,000.00
Motor	$ 8,000.00
Head and Tail End Assemblies	$199,500.00
TOTAL COST	$645,500.00

The following annual costs were estimated:
- Rollers = A$ 10,000
- Feeder/discharge maintenance =A $ 15,000
- Misc. parts/lube = A$5,000
- Maintenance labour = A$30,000

The cost comparison between the resulting Net Present Cost (NPC) and cost in dollars per tonne for the implementation of the various trucks for the continued truck haulage to 10 Level is shown in Figures 7 and 8 and Table.

The cost comparison between the resulting NPC and cost in dollars per tonne for the implementation of the various trucks for the continued truck haulage to 10 Level, including the implementation of the transfer conveyor on 9 Level is shown in Table 6.

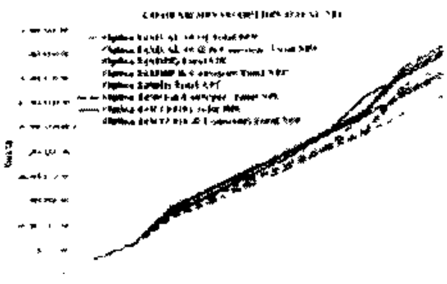
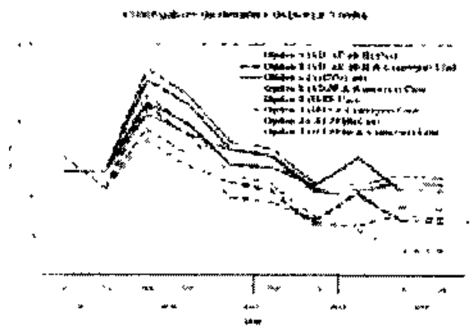


Figure 7. Comaprison of option total NPC



Figuie 8 Comparison of option costs per tonne

Table 5 Costs for truck haulage to 10 level

	X® \1. 4U 11	AI)«	W	MT«In
lOIAJ.NIT tASltf'l	AIXAI 4111	4ÏXS8	45.44	4A5.1
COST astı	S.I7	7.47	« 0	S.»

Table 6. Costs for truck haulage to 10 level, including transfer eonveyoi

	Ati.«i 4011	SD55	«015	MT5H10
10 KI NIT <ASI0"j	40.1«	Ï6M	40.«)	41.*)
COSTrA&tj	7.54	(\(A	7,470	7.67

It can be seen from Table 6 that the combined implementation of both the Elphinstone AD55, and the transfer conveyor results in the lowest Net Present Cost at the end of 2005. This represented a saving of 27.03 % in operating costs compared to the existing system discussed previously (i.e., haul to 10 Level with AD/AE 40 II, June 2001).

8 CONTINUED TRUCK HAULAGE TO 9 LEVEL

There are several points to note regarding this haulage option:

- Increased simplicity as the conveyor 8 Level to 9 Level ore pass and internal hoist infrastructure system would become obsolete,
- The longer haul distances to 9 Level will result in greater trucking requirements,
- The increased trucking requirement would result in a poorer mine environment in tenus of heat, contaminants and dust.

The cost comparison between the resulting Net Present Cost and cost in dollars per tonne for the implementation of the various trucks for truck haulage to 9 Level is shown in Figures 9 and 10 and Table 7. It can be seen from Table 7 that the implementation of the Elphinstone AD55 mining trucks results in the lowest Net Present Cost by the end of 2005. Although this is markedly lower than the current operating costs, the implementation of the Elphinstone AD55 and transfer conveyor remain the least expensive.

9 ALTERNATIVES FOR THE LONG TERM FUTURE OF THE MINE

If the operation of the mine was to continue for the next fifteen years, it is evident that a shaft extension of either the No. 1 or No.2 Shafts, or the establishment of a new shaft hoisting system in a separate new shaft would be required. It has been suggested that the construction of new shaft, known as No.3 shaft, from the surface down to 2000m deep would be desirable alternative to extending the existing shafts for the following reasons:

- The No.1 Shaft was primarily designed as a return ventilation shaft. It has been decided to keep the internal winder in place as part of the solution for the short-term truck haulage problem. Once production reaches the point where truck haulage to 10 Level becomes uneconomical, it is planned to remove the internal winder and restore the No.l Shaft to its primary purpose as a return ventilation shaft, resulting in a substantial decrease in ventilation costs due to the reduced resistance in the shaft.

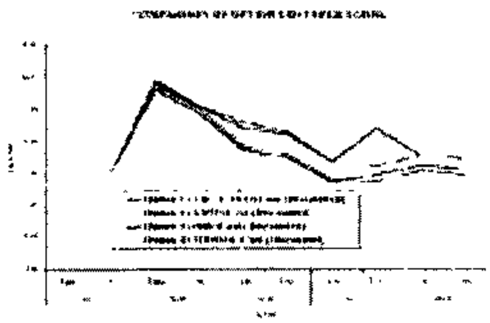


Figure 10. comparison of option costs per tonne

Table 7. Costs per truck haulage to 9 level

	Jia'Ai'4ti it	AD«	SOD	WT50I0
'JOTAL-OTC IAS 10")	41X81	17.98	•5SW	4u.f>S
CÜSI (AS.ll	7.46	(>.9i	7.12	7.41

• The No.2 Shaft is the main ore-hoisting shaft to the surface. Any extension of this shaft would render it inoperative for months. If this shaft was extended, the winding gear and headframe would have to be replaced or upgraded and it is anticipated when the capital costs and profit losses from the inoperable shaft are compared to the reduced haulage costs, the project will break even, therefore not achieving any benefit.

It has been determined that the hoisting capacity of the new shaft would be in the region of one million tonnes per annum. For shaft construction, the most likely method of development would be by raise boring. As a result of the required shaft depth of 2000 metres, it is suggested that the raising of the No.3 Shaft be completed in 3 stages, with the first two stages consisting of raising a distance of approximately 670 metres and the third stage raising the remaining metres to the surface, i.e.:

• Shaft bottom at 4715 RL (2000 m below surface)
• Stage 1 raise from 4715 RL to 5385 RL (1330 m below surface)
• Stage 2 raise from 5385 RL to 6055 RL (660 m below surface)
• Stage 3 raise from 6055 RL to surface In March 2000, the lmpala Platinum Mine in South Africa successfully collared a 27 1112, 770 in long ventilation shaft using a Sandvik CRH 12E reaming head (Bartlett, 2001). Using this type of reaming technology, there is no reason why this type of technology cannot be applied in the construction of the new No.3 Shaft. A summary of shaft raise boring capital costs is shown in Table 8.

Table 8. Shaft raise boring capital costs

ITEM	CAPITAL (A$)
Access tunnel site	$ 150,000.00
Excavate site	$ 15,000.00
Develop decline to 4715 RL ($4000/m)	$20,800,000.00
Access shaft bottom	N/A
Drill pilot hole ($440/m)	$ 880,000.00
STAGE 1	
Excavate winding chamber	$ 600,000.00
Raise to 5385 RL ($21,700/m)	$14,539,000.00
Line shaft ($800/m)	$ 536,000.00
STAGE 2	
Excavate winding chamber	$ 600,000.00
Raise to 6055 RL ($21,700/m)	$14,539,000.00
Line shaft ($800/m)	$ 536,000.00
STAGE 3	
Excavate winding chamber	$ 600,000.00
Raise to surface ($21,700/m)	$14,322,000.00
Line shaft	$ 528,000.00
TOTAL	$68,645,000.00

It is expected that shaft raise boring and hoisting system capital costs will total A$82.37 million. It is envisaged that when shaft construction has been completed, a main level will be developed at or slightly above the 4715 RL. This will allow the necessary installation of a crushing station, loading station and transfer system similar to the current arrangements on 9 Level.

It has been assumed that the production rate will remain constant at 747,600 tonnes per year. However, the specified requirement of a shaft hoisting system capable of handling 1,000,000 tonnes per year will be used in the determination of the hoisting system requirements in case production is boosted to 1,000,000 tonnes per year. A summary of the hoisting system requirements is shown in Table 9.

Table 9. Hoisting system requirements

Output (tpy)	1,000,000
Hoisting depth (m)	2000.0
Available operating hours per year	6,400.0
Shaft diameter (m)	5.8
Operating rate (tph)	156.0
Skip velocity (m/s)	14.0
Skip payload (t)	8.0
Skip tare (t)	6.4
Number of skips	2.0
Rope diameter (mm)	38.0
Rope unit mass (kg/m)	6.0
Breaking force (kN)	1,000.0
Drum diameter (mm)	3,400.0
Skip 1 winder power (kW)	2,200.0
Skip 2 winder power (kW)	2,200.0
Headframe height (m)	46.0
Headframe weight (t)	935.0

10 CONCLUSIONS

For the short term, it was recommended that the mine management proceed with the installation of the transfer conveyor system on 9 Level and implement the Elphinstone AD55 mining trucks at the end of June 2002. For the long-term future, the option of establishing a new shaft to 2000 m depth would be a desirable option, provided that the results from the diamond drilling prove the existence of sufficient reserves for the continuation of mining well into the next decade.

REFERENCES

Bartlett, L. 2001. Biggest CRH 12E head to date collars 786 metre raise m Soulh Africa

Rock World. January. Sandvik Tairuock Corp.. Helsinki Media Business Publications. Finland, p. 15

Chadwick. J. 2000. Underground haulage. *Miiiinn MuRazine*. September, p. 103

Noıhcote G G and Bdines F L S 1971 Comparison of the economics tit truck, haulage and shall hoisting ot ore from mining opeıatıons *Pie Au\IMMTnm\p<nlulitm Symposium* AuslMM Sydney Blanch pp 1 'S — "lO