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.

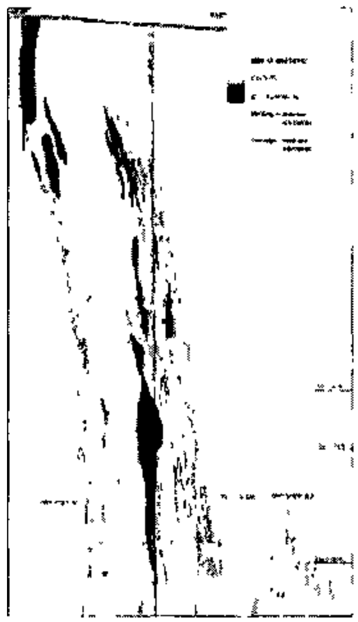
2 MINING AND ORE TRANSPORT

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.

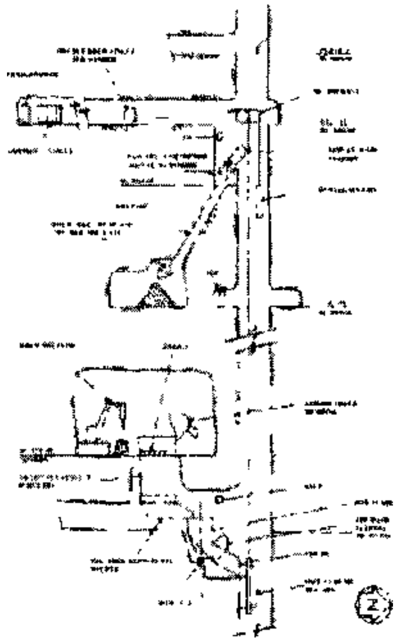


Figure 2. No. 1 shaft internal winder

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

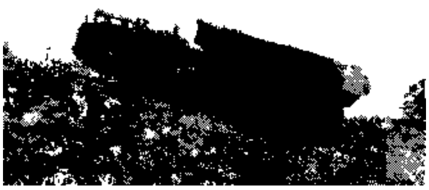


Figure 5. Tamrock Toro 50D

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:

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

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:

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

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	AD 55	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/h)	87.64	126.94	118.[illegible]	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	days
Haul velocity	km/hr	Sched. lost op. days per year	days
Return velocity	km/hr	Machine op. days per year	days
NI	min	Op hrs for each truck per year	hr
SUI	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.

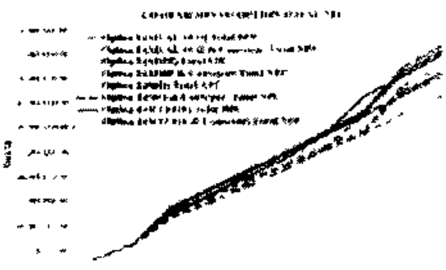


Figure 7. Comaprison of option total NPC

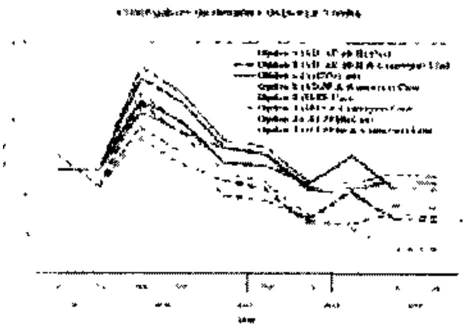


Figure 8 Comparison of option costs per tonne

Table 5 Costs for truck haulage to 10 level

	X® \1. 4U 11	A I) «	W	MT«I n
lOIAJ.NIT tASltf'l	AIXAI 4111	4ÏXS8	45.44	4A5.1
COST ast1	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 K I 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.1 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.

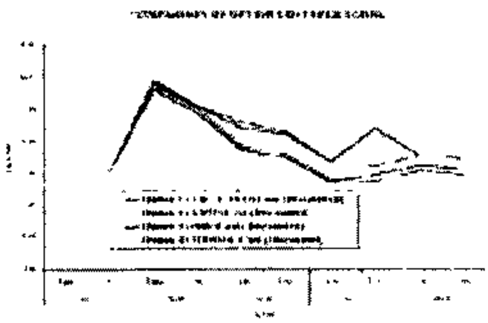


Figure 10. comparison of option costs per tonne

Table 7. Costs per truck haulage to 9 level

	Jia'Ai'4ti it	A D «	SOD	WT50I0
'JOTAL-OTC IAS 10")	41X81	17.98	•5SW	4u.f>S
CÜSI (AS.11	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)	2,000.0
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)	125.0

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

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