P L McCarthy and R Livingstone, 1993. Open Pit to Underground: Making the Transition, *AIG Bulletin* 14, 1993, pages 45-56.

Shaft or Decline? An Economic Comparison

By P L McCarthy [1] and R Livingstone [2]

Abstract

There is a logical progression from open pit mining, through decline-access underground mining to shaft-access underground mining. The economic transition from decline haulage to shaft hoisting is dependent on a number of factors, particularly the depth of mining and production rate.

The transition depth from decline to shaft has historically been taken as about 300m, but current Western Australia practice suggests 500 m or more may be appropriate. By delaying development of a shaft, a "diminishing returns" situation is set up so that, at any time, the remaining ore reserve at depth may be insufficient to justify the capital required for a shaft.

For this paper, the authors studied production rates ranging from 0.25 to 1.5 Mtpa, and orebody depths to 1,000m. Actual costs from mining operations and recent feasibility study estimates were used to model capital and operating costs for shafts and for declines using diesel haulage.

A series of cost curves were produced enabling optimum transition depths to be determined for a range of conditions.

Introduction

There is a logical progression from open pit mining, through decline-access underground mining to shaft-access underground mining. Decline access is attractive for shallow orebodies, particularly where a decline portal can be sited within an existing open pit.

The question of the depth at which shaft hoisting becomes a more economically attractive alternative to decline truck haulage is commonly faced as mining progresses to deeper levels. Because of the capital required, it may be very difficult to justify converting a decline mine to a shaft mine. It is sometimes argued that a shaft mine should be developed from the outset to avoid the need for such a conversion.

There have been considerable advances in diesel truck technology in the twenty years since this problem was addressed for Australian conditions by Northcote and Barnes (1973). It is noteworthy that neither of the two operations cited in that study (Renison Limited and Gunpowder Copper Limited) has subsequently converted from decline haulage to shaft hoisting.

The optimum changeover depth of 350m derived by Northcote and Barnes is still widely quoted, but is not supported by recent operational experience. On the contrary, some deep operations such as Lancefield Gold Mine have been converted from vertical shaft to decline haulage with resulting improvements in costs and productivity.

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This study investigated depths at which shaft hoisting should replace decline truck haulage for a series of production rates and other parameters. It should be noted that every mine has it's own peculiar circumstances which will influence this determination.

Such factors include:

- mining method and ground conditions,
- requirement for service access via a decline,
- requirement for lateral and vertical ramp coverage of the orebody and the lateral extent of the orebody,
- depth from decline portal to the top of the orebody,
- the planned rate of vertical advance and its relation to the ore distribution and hence the production rate,
- the ore reserve and development schedule and thus the planned mine life,
- the existence of exploration shafts suitable for conversion to production hoisting.
- whether the decline can be sufficiently advanced ahead of current mining areas to enable raisebored hoisting shafts,
- the discount rate used in analysis,
- mine life,
- haulage distance to shaft.

Methodology

Two methods were used to investigate the optimum depth of changeover from decline haulage to shaft hoisting. The first was a series of "static" comparisons of haulage and hoisting costs from fixed depths for periods of up to 15 years. The second method is referred to as the "dynamic" approach. This simulated haulage and hoisting costs for mining operations commencing at relatively shallow levels and progressing to deeper levels during the mine's life. The difference between the two ore transport systems is illustrated in Figure 1.

The capital costs for the decline case included trucks, loaders, primary ventilation fans, and ventilation raises. The cost of the decline was not included as it was assumed to be required for servicing the mine in either case. While a service decline might be developed at a lower cost than a haulage decline, the assumption was made that initial mine production would be truck hauled, thus requiring a decline which would accommodate haulage trucks.

For shaft hoisting, capital costs included shaft development & equipping, winder/conveyances, headframe, surface handling system, loading station, underground crusher (less cost of surface crusher), underground loading and haulage equipment, primary ventilation fans, and ventilation raises. Representative mid-range capital costs are shown in Table 1.

Operating Costs for the decline case included truck loading, haulage, decline road maintenance and primary ventilation.

For shaft hoisting, operating costs included underground loading and haulage, underground crushing, hoisting and surface handling. The operating cost for underground crushing was assumed to be the same as for surface crushing.

Production rates ranging from 0.25 Mtpa to 1.5 Mtpa were considered. Representative operating costs are shown in Table 2. Costs were varied for each case according to cost models derived from actual costs from mining operations and recent feasibility study estimates.

"Static" Comparison

For each production rate considered, a series of spreadsheets was created to model the costs over a 15 year period. Each spreadsheet modelled haulage and hoisting from a particular depth and showed:

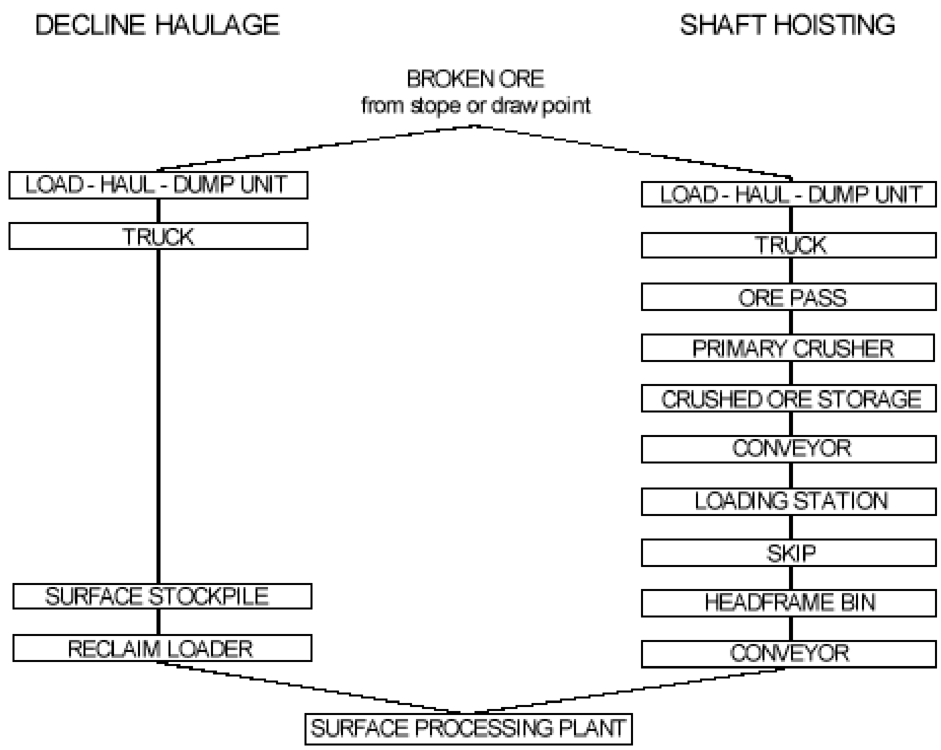
- capital and operating costs for decline truck haulage, shaft hoisting (conventionally sunk shaft) and shaft hoisting (raisebore/strip and line shaft)
- comparison between outlays
- comparison between outlays discounted at 10% p.a.

Dynamic" Comparison

For each production rate considered, a series of spreadsheets was created to model costs over the life of the mine as the average depth of mining increased. Each spreadsheet showed:

- capital and operating costs by year for a mine employing decline haulage for the full mine life
- capital and operating costs for a mining operation commencing production using decline haulage and later changing to shaft hoisting by a conventionally sunk shaft. The changeover date was adjusted in one year increments.
- capital and operating costs for a mining operation commencing production using decline haulage and later changing to shaft hoisting by a "post-hole" type shaft (ie., raisebored and, for larger diameters, stripped and lined). The changeover date was adjusted in one year increments.
- comparison between total outlays for each of the above scenarios over the life of the mine with the outlays discounted at a rate of 10% per annum.

Figure 1 – *Ore transport systems for decline haulage and shaft hoisting*



For a given production rate the spreadsheet which showed the lowest cumulative discounted outlay over the mine life was considered to have the optimum year and depth of changeover from shaft hoisting to decline truck haulage.

A "shaft only" option was not modelled. Such an option would avoid the cost of the surface leg of the decline, 1200 m plan metres of development to the assumed top of the orebody. The remainder of the decline cost would still be required to develop internal ramps. A similar cost saving would apply to the decline case if the portal was located within an open pit.

Parameters

Truck Haulage

Comparisons were made for decline haulage using 50 t and 40 t capacity diesel powered trucks on a gradient of 1 in 8. Truck performance analysis was based on Elphinstone 73B and Toro 40D trucks as representatives of the 50 t and 40 t classes respectively.

The cost model was developed as follows:

1. The theoretical production rate was determined for a single truck in a decline neglecting interference from other vehicles and other operating constraints. This was determined using the VEHSIM truck haulage simulation software package as supplied by Caterpillar Inc.
2. The effect of traffic interference on truck haulage productivity was modelled using General Purpose Simulation System (GPSS) software.
3. Comparison was made with sets of actual truck haulage data from operating mines. Production rates were found to range from 38% to 89% of the estimated mine truck fleet capability. 70% is considered to be a reasonable factor for adjusting theoretical estimates to allow for operating constraints in cases where the truck fleet is well utilised. The results of modelling are shown in Figure 2, Figure 3 and Figure 4.

Table 1 – ***Representative capital costs for a decline, conventional shaft and bored shaft***

Item	Decline	Cost ($M) Conventional Shaft	Bored Shaft
Trucks and loaders	3.45	2.1	2.1
Primary fans	0.75	0.5	0.6
Vent raises (contract)	7.72	2.52	2.52
shaft excavation	-	13.21	3.22
Equipping			
Winder	-	3.09	3.09
Headframe	-	0.94	0.94
Loading station	-	0.62	0.62
Crusher (margin)	-	1.1	1.1
Ore passes	-	0.6	0.6
Surface handling	-	0.5	0.5
Total Cost ($M)	**11.92**	**25.18**	**15.29**

Note: costs are for 500,000 tpa capacity at 600m depth

Table 2 – ***Representative operating costs for shafts and declines at different production rates***

	250 000 TPA		500 000 TPA		750 000 TPA	
	Shaft	**Decline**	**Shaft**	**Decline**	**Shaft**	**Decline**
Load and haul	1.54	2.85	1.54	2.93	1.61	2.97
Hoist	2.21	-	1.68	-	1.36	-
Surface handling	0.89	-	0.15	-	0.15	-
Decline road maintenance	-	1.09	-	1.09	-	1.09
Ventilation	0.9	2.82	0.8	1.22	0.7	1.73
Total cost ($/t)	**5.54**	**6.76**	**4.17**	**5.24**	**3.82**	**5.79**

Note: costs are for haulage from 600m with 50 tonne trucks

4. Hourly operating cost estimates were made including rebuild allowances as shown in Table 3, based on averages of actual site costs. It should be noted that much lower operating costs can be derived for truck replacement periods of 8000-9000 hours, but the total owning and operating cost may not be significantly different.
5. Ventilation requirements were estimated, which resulted in limits of four 50 t or five 40 t trucks in the decline an any time. The resulting limits on decline haulage rates are shown in Figure 5.
6. The estimated haulage cost per tonne was derived for a range of depths and production rates. The estimated truck operating costs are shown in Figure 6, and the resulting haulage system operating costs (for 50 t trucks) in Figure 7.

Shaft Hoisting

Actual cost and performance data from eight mines was used to prepare generalised models. Costs were estimated for a range of shaft depths and production rates.

Capital costs were estimated for shaft excavation and equipping, winders (including conveyances and attachments), headframes, loading stations, underground crusher, ventilation raises and fans, underground haulage equipment, ore passes and a surface ore handling system. The model for the capital cost of vertical development is given in Table 4.

The underground crusher was considered to be substituted for the surface primary crusher, so only the additional cost of underground installation was included.

Figure 2 – ***Plot against depth of mining showing productivity of a single truck***

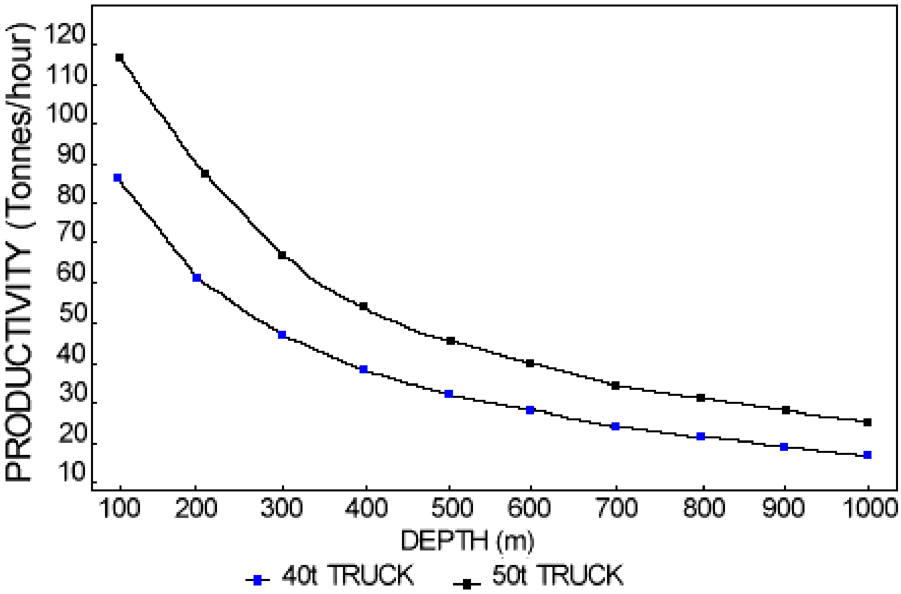


Figure 3 – ***Plot against depth of mining showing decline haulage capacity of one to nine 50 tonne trucks***

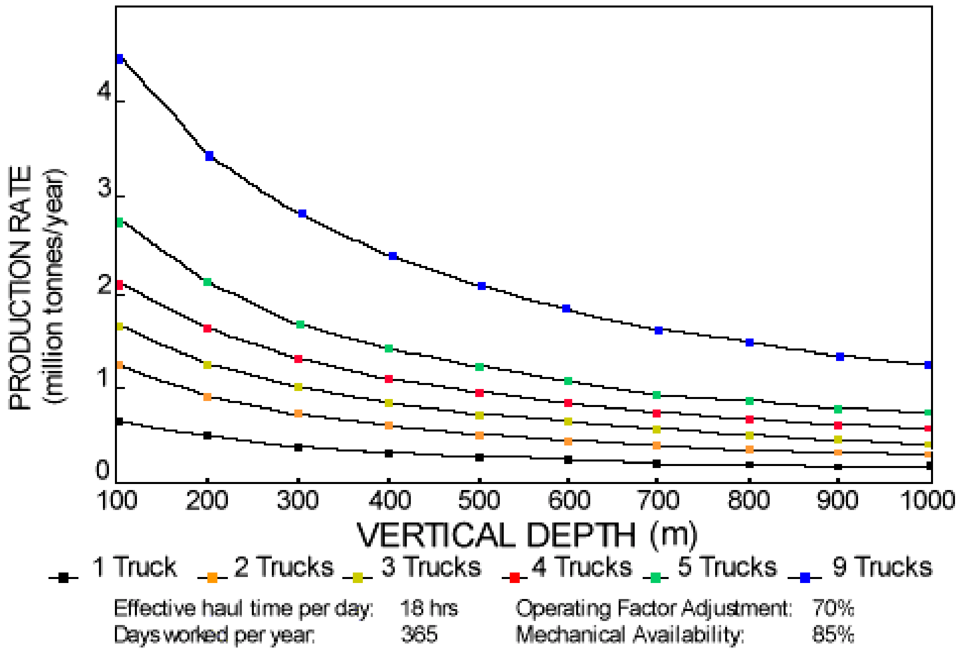


Figure 4 – ***Plot against depth of mining showing decline haulage capacity of one to nine 40 tonne trucks***

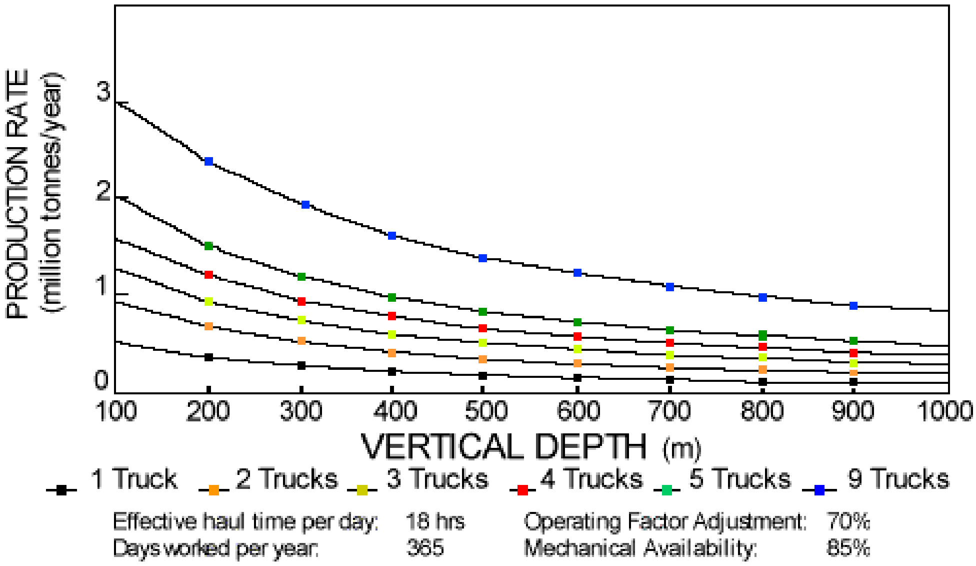


Table 3 – ***Hourly operating costs for 40 and 50 tonne trucks***

	40 tonne class	50 tonne class
Operating labour	42.41	42.41
Operating supplies	21	28.05
Maintenance labour	14.38	14.38
Maintenance supplies	35.74	50.52
Total cost per hour ($)	**113.53**	**135.36**

Figure 5 – ***Plot against depth of mining showing maximum decline haulage rates, based on ventilation limits, for 40 and 50 tonne trucks***

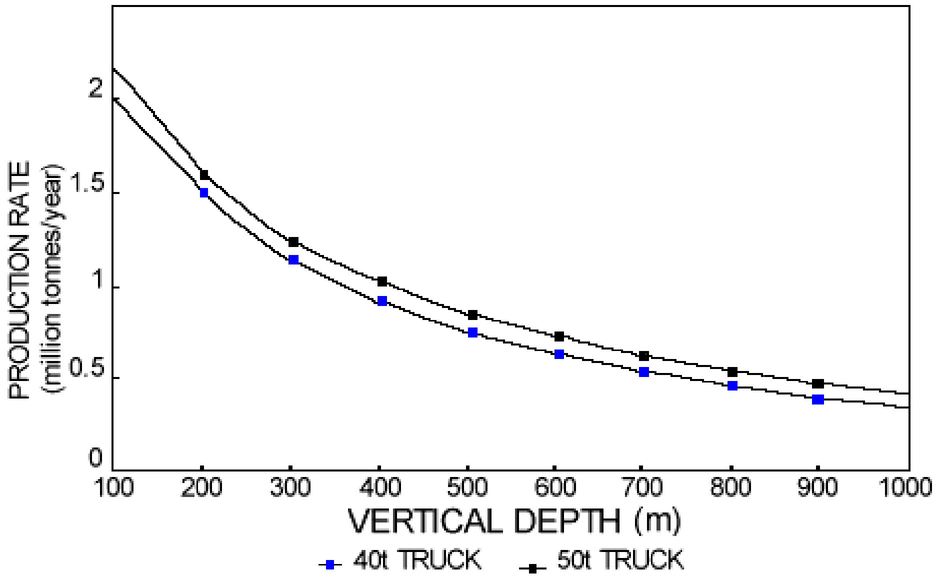


Figure 6 – ***Plot against depth of mining showing truck operating cost per tonne for 40 and 50 tonne trucks***

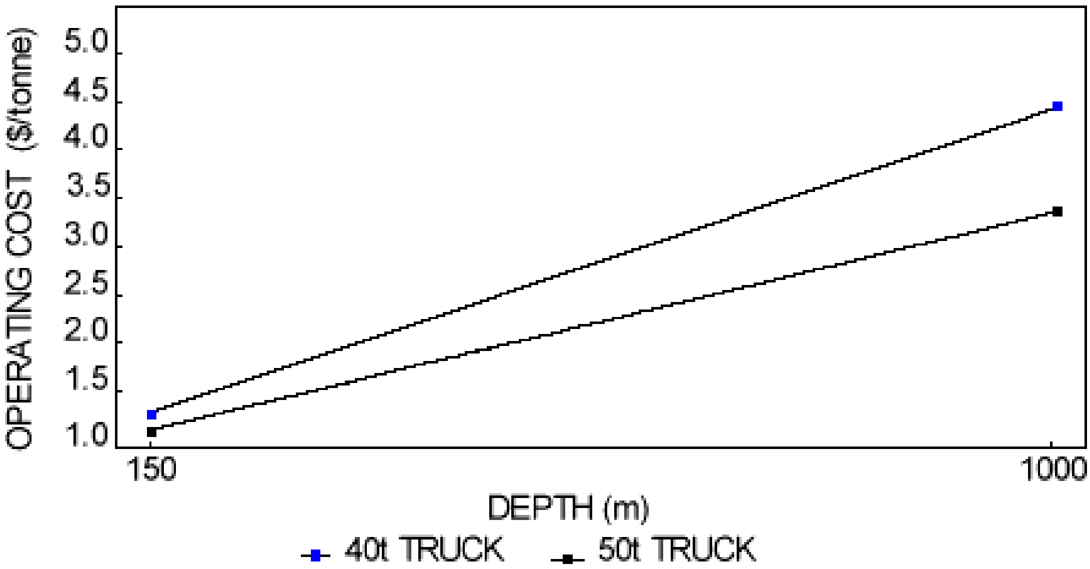
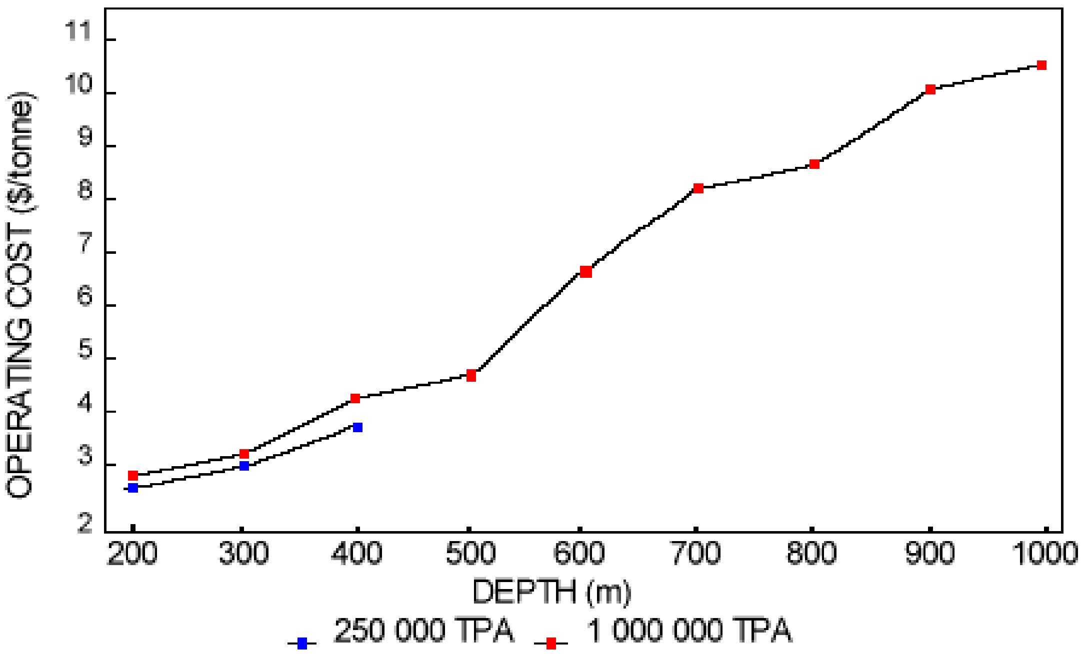


Figure 7 – ***Plot against depth of mining showing decline operating cost model using 50 tonne trucks and for production rates of 250,000 and 1,000,000 tonnes per annum***



The cost of underground haulage is affected by the distribution of ore sources relative to the shaft. In all cases, it was assumed that the cost and productivity of haulage from the ore sources to the orepass is, on average, equivalent to that for truck haulage with a profile of 700m level and 700m at a gradient of 1 in 7. The capital cost allowed was the cost of loaders and trucks to handle this duty.

Two metres of ore pass were allowed for each vertical metre of shaft.

For systems hoisting uncrushed ore it was assumed that ore is trammed from the shaft to the ROM stockpile, an average distance of 150m, by front end loader. The cost allowed was the cost of the loader. For underground crushing cases, crushed ore was assumed to be conveyed directly from the shaft bin to the secondary crusher.

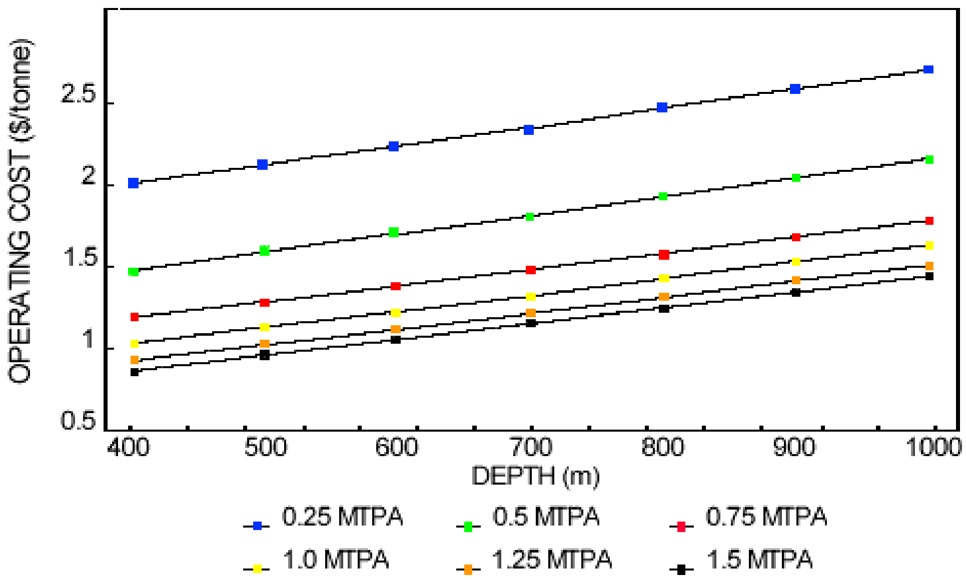
Operating costs included underground haulage, underground crushing, skip loading, hoisting, surface handling and ventilation. The variation of hoisting cost with depth is shown in Figure 8.

Table 4 – ***Cost model for vertical development under variable scenarios***

Method	Size	Fixed costs $'000	Depth variable costs $/m
Conventional sinking	4.0m dia	1,558	12,322
Conventional sinking	6.0m dia	3,624	15,980
Conventional sinking	8.0m dia	3,663	17,979
Raise boring*	1.8m dia	120	1,600
Raise boring*	2.4m dia	120	3,100
Raise boring*	3.0m dia	188	4,061
Raise boring*	4.0m dia	262	6,083
Blind shaft drilling	4.0m dia	1,797	13,448
Shaft strip and line	8.0m dia	2,212	17,681
V-Mole	6.0m dia	5,071	12,613
Longhole raising*	3.0 x 3.0m	0	500
Longhole raising*	1.5 x 1.5m	0	300

*Does not include shaft lining or equipping

Figure 8 – ***Plot against depth of mining showing shaft hoisting operation cost per tonne using automatic winder and for variable production rates***



The additional cost of underground crushing compared to surface primary crushing was considered to be negligible. Costs of skip loading and hoisting were estimated for a range of depths and production rates, for both automated and manual hoisting.

Hoisting system parameters were based on calculations carried out by a hoisting system manufacturer. In all cases the hoisting system was assumed to be devoted entirely to rock hoisting using balanced skips and a double drum winder.

To determine the cost of surface handling of uncrushed ore it was assumed that ore is trammed by front end loader. For surface conveying of crushed ore an operating cost was allowed for maintenance labour and materials.

Ventilation

Decline truck haulage requires a greater ventilation airflow than shaft hoisting. Decline dimensions were assumed to be 6.0 x 5.0m for 50 tonne trucks and 5.0 x 5.0m for 40 tonne trucks. A limiting air velocity of 6m/s was applied, with parallel exhaust airways developed by raiseboring at an appropriate diameter in each case.

The exhaust gas dilution requirement was assumed to be 0.085 m^3/s per KW diesel power in operation. A simplistic mine layout was assumed, with intakes connected to exhausts by four 250m long drives in parallel.

The cost of secondary ventilation was considered to be common to all cases and was thus ignored. For the decline case it was assumed that raise bores were developed in 150 m vertical steps. One velocity pressure of shock loss was assumed at each step.

Figure 9 – ***Plot showing the results of static modelling using a 5-Year mine life and 50 tonne trucks. The optimum changeover depth is calculated for various production rates and mining situations***

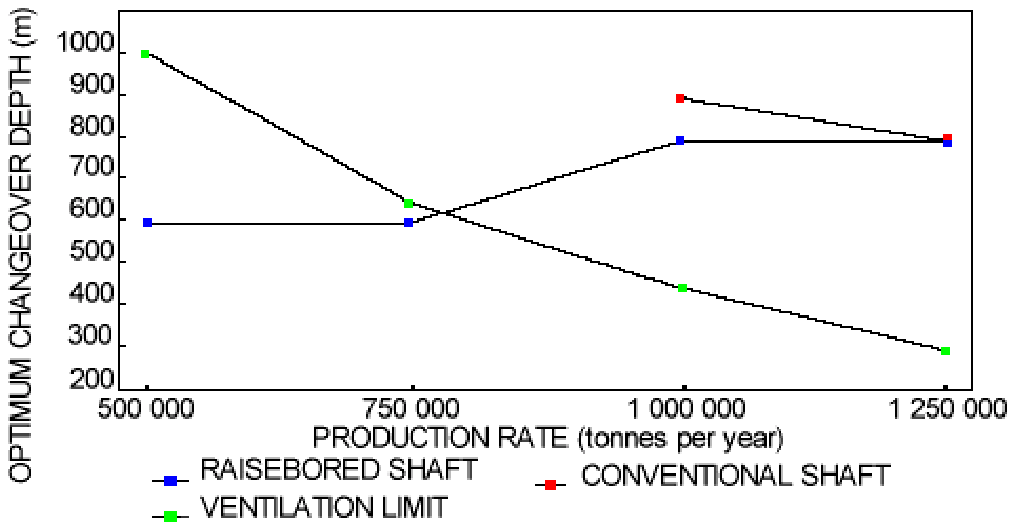


Figure 10 – ***Plot showing the results of static modelling using a 10-year mine life and 50 tonne trucks. The optimum change over depth is calculated for various production rates and mining situations***

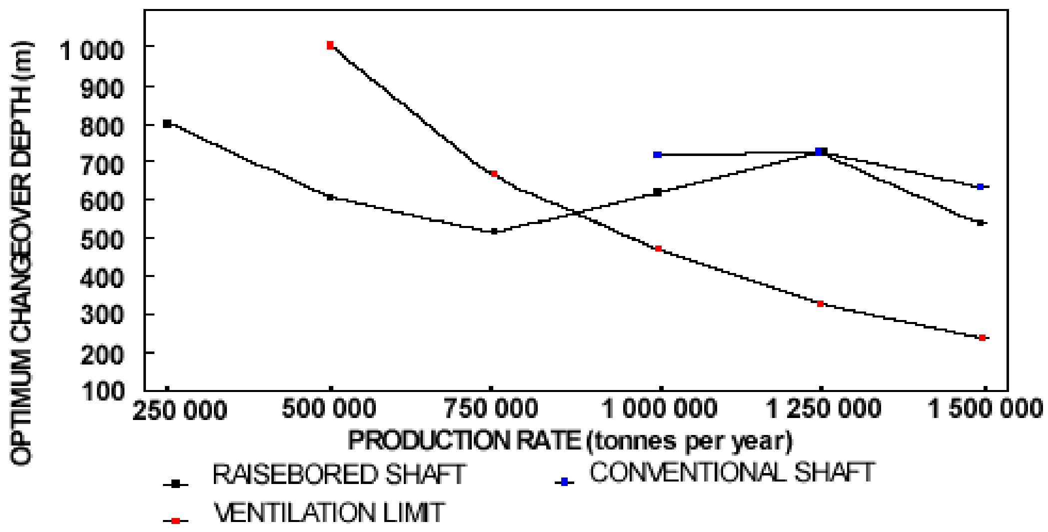
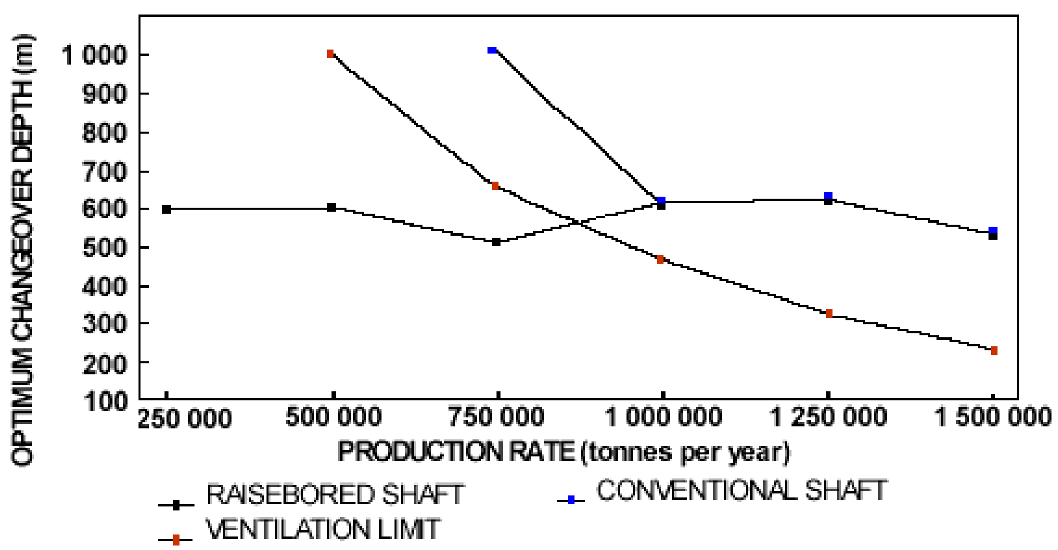


Figure 11 – ***Plot showing the results of static modelling using a 15-year mine life and 50 tonne trucks. The optimum changeover depth is calculated for various production rates and mining situations***



Operations

For "dynamic" comparison it was assumed that operations commence at an average haulage depth of 150 metres and are extended by 50m vertically each year to an ultimate depth of 1,000m.

For conventional sinking it was assumed that, if the current depth of mining is less than 400m, the shaft will be sunk initially to 600m and later deepened to a full depth of 1,000m. If the current depth of mining exceeds 400m it was assumed that the shaft is sunk directly to 1,000m.

Raisebored shafts are generally developed to shorter depths in advance of current mining operations than conventionally sunk shafts due to the requirement for pre-existing underground access. It was assumed that, for mining depths of up to 300m, the initial raisebored shafts were developed to 400m with later extensions to 700m and 1,000m as required.

Where the current mining depth exceeded 300m it was assumed that the raisebored shaft was developed directly to 600m with later extensions to 800m and 1,000m. Where the current mining depth exceeded 500m it was assumed that the raisebored shaft was developed to 800m and later extended to 1000m. Where the current mining depth exceeded 700m the raisebored shaft is developed directly to 1,000m.

Stripped and lined shafts were assumed for larger production rates where pre-existing underground access is available.

For raisebored shafts it was assumed that the shaft and hoisting system were installed within a 12 month period, regardless of depth. For "conventionally sunk" and "stripped and lined" shafts, it was assumed that the shaft was developed and the hoisting system installed over a 2 year period, regardless of depth.
The capital cost of decline development was not considered in the comparison because it was assumed that most mines use trackless mining methods and require internal ramp coverage of the orebody. A decline connection to surface was allowed for in all cases.

The maximum number of trucks considered in operation was 9. The equipment replacement interval was assumed to be 9 years, with provision in operating costs for equipment rebuilds.

Results

The results are presented for production rates beyond the ventilation limits mentioned above. The higher haulage rates may be feasible by using electric trucks, although the effective costs (operating and discounted capital) may not be directly comparable.

The results of the "static" comparison are summarised in Figure 9, Figure 10 and Figure 11 showing the depths at which shaft hoisting becomes a more attractive alternative to decline truck haulage. As the 40 t and 50 t truck results were similar, only the 50 t case is presented here.

The results of the "dynamic" comparison are shown in Figure 12 and Figure 13. In no case did shaft hoisting become a more attractive alternative to decline haulage to a depth of 1,000m. Thus the results are presented as a "cost penalty" incurred for providing a hoisting shaft.

When the results of the "static" comparison are considered it appears that, with a production rate corresponding to a vertical advance rate of 50m per year, there has been insufficient time for the operating cost savings of shaft hoisting to compensate for the capital cost of shaft installation. Shaft hoisting is expected to be more attractive for cases with lower vertical advance rates.

Figure 12 – ***Plot showing the results of dynamic modelling incorporating a cost penalty for a conventional sunk shaft. The additional discounted total cost is shown against the year of commissioning the shaft and for various production rates***

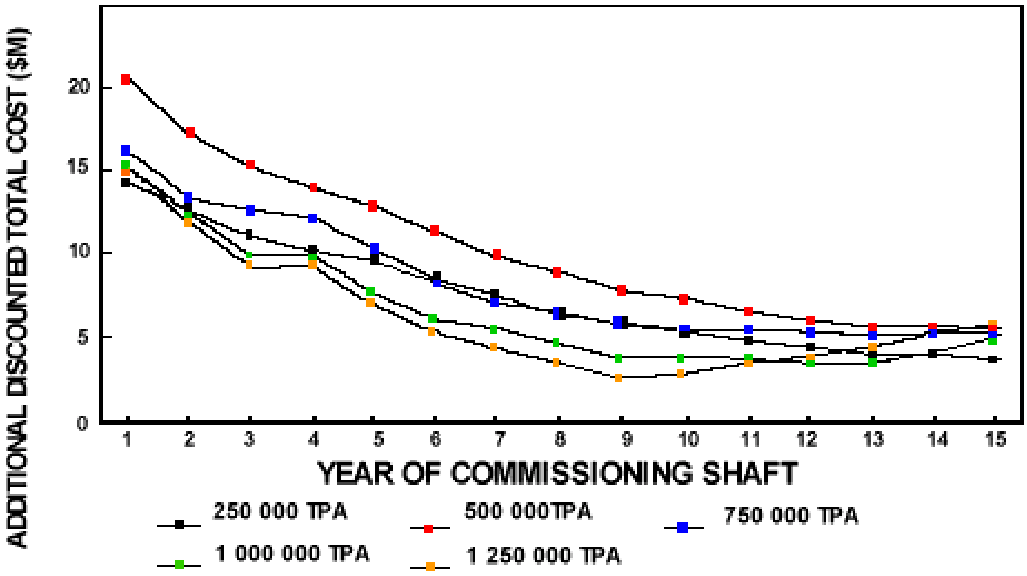
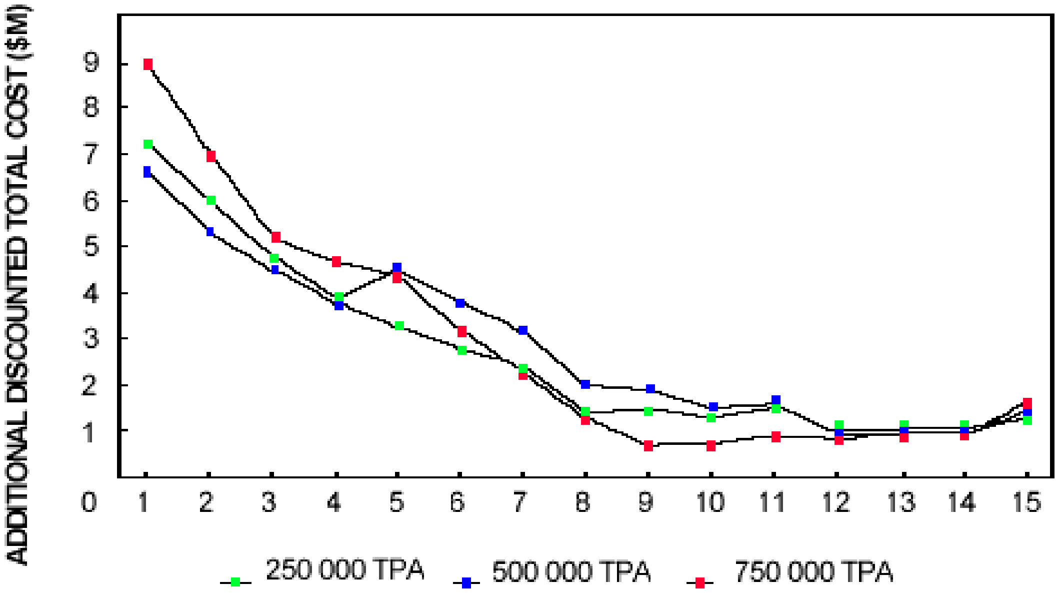


Figure 13 – ***Plot showing the results of dynamic modelling incorporating a cost penalty for a raise bored shaft. The additional discounted cost is shown against the year of commissioning the shaft and for various production rates***



Conclusions

Based on the "static" comparisons the following trends are observed;

1. The optimum changeover depth from decline haulage to shaft hoisting becomes shallower as the mine life increases.
2. The optimum changeover depth from decline haulage to shaft hoisting becomes shallower as production rate increases. The exception to this trend is at the production rate where larger stripped and lined shafts replace "bare" raisebored shafts resulting in a significant capital cost increase.
3. It should be noted that each "static" model compares shaft hoisting with truck haulage from one particular depth over the full mine life. The "dynamic" models are considered to be more realistic simulations of a decline mining operation progressively increasing in depth.
4. The conclusion from the "dynamic" modelling is that, for the operations modelled advancing at a rate of 50m vertically per year, it is not possible to justify expenditure of capital to develop either a conventional or raisebored shaft and install a hoisting system on the basis of savings in haulage costs to a depth of 1,000m. While a lower vertical mining rate is expected to make shaft hoisting a more attractive alternative, this is only likely in massive orebodies.
5. The raisebored type of shaft hoisting system is more economically attractive than conventional shaft systems provided that decline development is sufficiently advanced to enable raisebore development. It is particularly attractive compared to conventional shaft sinking for production rates of 750,000 tpa or less where a bare raisebore hole can be used without stripping or lining.

1. In many cases the decision to develop a shaft will be determined, not by savings in haulage costs, but by the maximum production rate which can be hauled by trucks in the decline. This was determined to be controlled by ventilation airflows required to dilute truck engine exhaust fumes

References

Northcote G.G. and Barnes ELS: Comparison of the Economics of Truck Haulage and Shaft Hoisting of Ore from Mining Operations; The AusIMM Sydney Branch, *Transportation Symposium*, October 1973.M