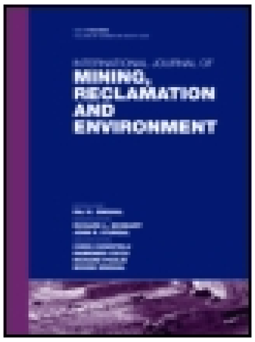

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The equipment utilisation versus mining rate trade-off in open pit mining

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ABSTRACT

This paper investigates the potential value adding role that 'schemes of exploitation' may have as part of the open pit mine planning process. The deployment of loading equipment within the push-back of an open-pit mine ultimately determines the 'mining rate'. Traditional mine planning processes seek to adopt schemes of exploitation that maximise the utilisation of the loading equipment as this will typically minimise mining cost. This paper argues that this does not always lead to the creation of value. A case-study demonstrates that alternative schemes of exploitation, with higher mining costs and lower shovel productivity can actually generate greater value. The results show an increase in Net Present Value from US$920M to US$966M when a less productive configuration of four shovels is set instead of a configuration of two shovels. A sensitivity analysis is presented to show the economic and technical conditions that can favour this new proposal.

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Open pit mining; mine planning; schemes of exploitation; mining rate; equipment utilisation

1. Introduction

The main asset in mining is the mineral deposit. This is finite and non-renewable, which imposes a very unique constraint on mining in comparison to other businesses. The projected mining life thus becomes a decision variable that needs to be carefully determined when planning the development of the mine [1]. The exploitation phase of each mineral deposit has a beginning and an end, which is largely determined by the characteristics of the mineral deposit (size, shape and grade) and by its exploitation strategy. Additional considerations should be taken in account to estimate the life of the mine such as environmental approvals and infrastructure development [1]. The primary objective of mine planning is to select the best alternative that will create the highest value for the defined level of risk. This value can be quantified by the Net Present Value (NPV) of the project which remains the most widely used metric by the majority of mining companies [2].

The aim of this paper is to demonstrate that current industry practise in regard to the way mining costs are treated as part of the mine planning process may actually be destroying value. This will be demonstrated within this paper for the particular case of determining the mining rate. The mining rate refers to the rate at which exploitation depletes the mineral resource. Even though numerous other variables play a very important role as part of the strategic mine planning process (especially cut-off grade policy), this analysis will focus solely on mining rate estimation within the open pit metalliferous mining context.

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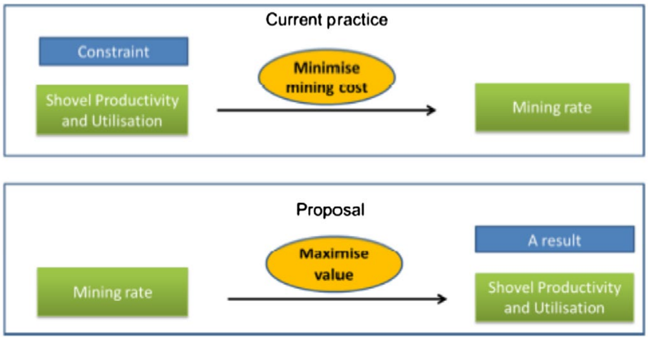

Figure 1. The minimisation of costs and the maximisation of the value.

The mining rate considers the extraction of ore and waste from the mine. The extraction of ore and waste through the life of mine is planned to give continuity the processing plant operation. Large copper deposits in northern Chile are examples of operations where the mineral is spread over a large and deep area where ore and waste are extracted through the whole life of the mine.

The determination of the mining rate is closely related to the selection of mining equipment. The selection of the loading equipment and the estimation of its operating costs are generally made using the premise of optimising the performance of the equipment. This means that there is a need to ensure that the loading equipment will have ample space in which to operate by avoiding configurations that involve having several pieces of loading equipment operating within the same pushback. There appears to be an ingrained belief that an optimum mine plan must consider high levels of utilisation and productivity for each unit of the loading fleet with the aim of minimising the mining cost to guarantee an appropriate level of investment in the mining equipment. This reasoning is aligned with the idea that has previously been presented by Camus [3] and Whittle [4], which highlight a current practice in mine planning that privileges the minimisation of costs over the maximisation of value during the planning stage.

The problem with this approach is that it limits the maximum mining rate to a certain level of equipment performance and so excludes the 'more aggressive' mining rates which could be useful for analysis as part of an optimisation process. The result is usually a fixed sinking-rate (that is the number of benches depleted per year within a pushback), which is given by the size of the pushback and by the expected equipment performance. This sinking rate is typically included as a fixed parameter in the strategic open pit planning process [5].

This paper will investigate whether more aggressive mining rates, including those associated with poorer equipment performances can add value and whether this can challenge the widely held view that 'the lower the cost – the better the business'.

An increase in the mining rate as a result of an increase in the number of shovels in a pushback will generally increase the productivity of the pushback. If space is limited however this may decrease the productivity and the utilisation of the loading-fleet that is operating within the pushback.

There thus seems to be an opportunity for analysing whether or not the current strategy for determining the mining rate does always generate a real economic benefit for the mining business. Figure 1 illustrates current industry practise in deriving a mining rate and the method that this paper proposes. Both methods will be used within a conceptual case study and the results compared. Shovel productivity and utilisation are considered to be a constraint in current industry practice during the estimation of a mining rate. The importance of equipment utilisation is well recognised by Runge [2]. An alternative view considers the calculation of the mining rate without this constraint, which is based on the equipment performance. It may also be considered that these performance indices are a result of the optimisation process rather than being a constraint. In the experience of the authors within large scale open pit copper mining operations in South America, the use of mining rates without the

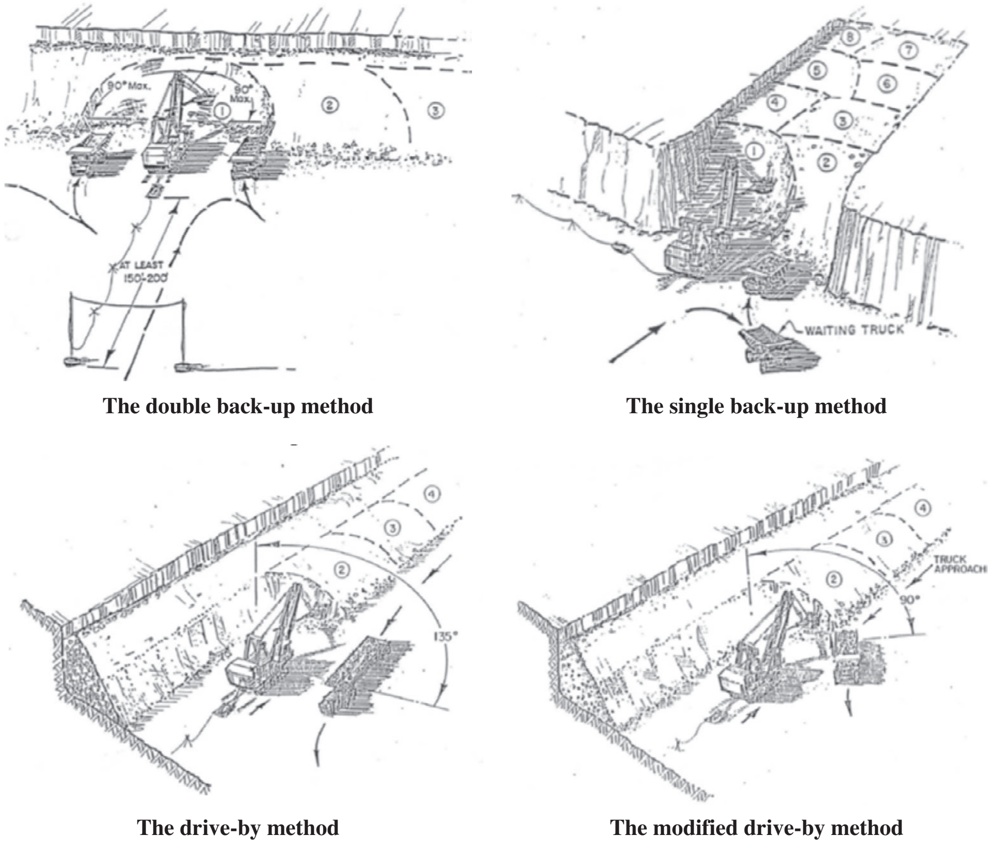

Figure 2. Shovel mining – truck loading methods [7].

equipment performance constraint may be used in special circumstances but is not typical. An example of this may be when the mine has to accelerate the extraction of material to reach a specific type of ore while keeping the processing plant continuously operational. Pinochet [6] recognised that mining rates that do not consider the utilisation of the loading equipment in their calculation correspond to a possible option of flexibility to the planner during the mine planning process. It appears that this proposal could create value for the mining business if it were to be used from the outset of the project, when the strategies for exploiting the deposit are created, rather than in emergency situations during the mining operation.

2. Background

There are four main, shovel mining-methods on the basis of shovel setup relative to the bench face, positioning of the truck when being loaded and truck travel routes to and from the shovel as identified by Calder et al. [7].

These shovel mining methods are illustrated in Figure 2 and offer numerous advantages and disadvantages, which have to be evaluated by considering the shape of the pushback, the space available for loading, the characteristic of the loading and hauling equipment and operator competence as discussed by Calder et al. [7]:

- *The double back-up method:* the shovel faces the material allowing a double side loading activity. Generally, it maximise the shovel performance because of the lower swing angles and less delay times associated with truck manoeuvres.

- *The single back-up method:* is similar to the previous method but with restricted loading space that only allows one side loading.
- *The drive-by method:* the shovel travels parallel to the pit wall. This configuration has higher safety risks especially when large pit walls present stability issues. The trucks are also driving along the blasted material and the performance can be affected by larger swing angles compared to other two methods.
- *The modified drive-by method:* similar to the previous method, but with a smaller swing angles related to a change in the truck position. This improves the performance of the system but remains a high risk configuration.

It is important to recognise that each of these methods will generate different productivities depending on various circumstances. Each method may well be used at various stages in the exploitation of pushback depending largely on the scheme of exploitation employed on a particular bench.

Figure 3 illustrates a typical half-moon bench in an open pit pushback. The ramp in this case enters the bench in the central area. This will change depending on ramp location and depth of the bench. The ramp could also enter close to the end of the bench. In some cases an auxiliary ramp will facilitate access to a bench. The number of shovels required to exploit a pushback is variable and depends upon the strategy that has been designed by the mine planner.

It is also important to note that there are different bench shapes depending on the design of the pushback to be extracted. Pinochet [6] proposes four main bench types where a classification is made on the basis of their shape. These are the expansion of the sunken cut, the hillside expansion, the deep hillside expansion and the top cut. One of the most important processes in bench design is the presence and extension of a free face.

For the typical half-moon shape bench depicted in Figure 3, it is generally comprised of four distinct regions:

- The ramp is developed to connect two or more levels. There are different kinds of ramps. The final ramp or the design ramp is the one that will allow access to all of the benches of the pushback. It will thus remain until the exploitation of the next pushback. The auxiliary ramp can be developed as a temporary access to an inferior level. It can be designed for access by trucks and by auxiliary equipment like drill rigs and bulldozers.
- The control area extends along the pit wall. The drill and blast design of this area is developed to maintain the stability of the latter. The loading of material in this area is a challenging activity, because the cut line must be precisely achieved to allow the lower benches to maintain their design shape and size.

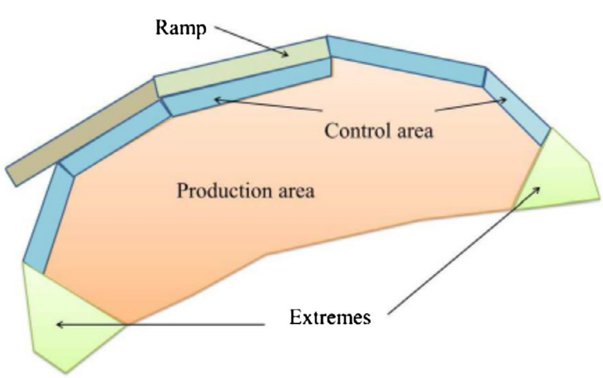

Figure 3. A typical half-moon bench in an open pit mine.

- The end areas are smaller than the others and are thus considered to be a restrictive space for the loading activity. In general, the swing angles of shovels increase considerably in this area and the productivity of the loading equipment is thus reduced.
- The central production area is the location that presents no restrictions on the loading equipment from a geometric point of view. The loading equipment can achieve its highest level of productivity in this area.

A previous study by Arteaga [8] which overlapped the drilling and blasting pattern, the rock type and the geometry of the bench, demonstrated the existence of significant differences in the productivity of the shovel fleet within each macro zone region. Operator competence was also found to be a key variable.

The scheme of exploitation corresponds to the deployment of loading equipment on a bench within a pushback. If only one shovel is positioned to extract the bench, it may follow a sequence illustrated in Figure 4, where the numbers represent the sequence in which each region is to be extracted. The exploitation of the bench begins with the ramp that is associated with number 1. This is followed by control region 2 before opening up the main production region 3, and so on.

In another situation, where more than one piece of loading equipment per bench are available this may result in a scheme of exploitation represented in Figure 5. In this case, both shovels will follow the sequence illustrated by the arrows which indicate that both shovels will extract the bench in opposite directions from the commencement of region 3. This is ideal as it minimises interactions between the two loading units and thus maintains a high productivity. An alternative scheme of exploitation could consider having both shovels working in the same direction. This configuration would give rise to a different scheme of exploitation. More complex schemes of exploitation involving more shovels and shovel movements are presented by Arteaga et al. [9].

Once the ultimate pit has been determined, the next step is to develop a strategy for extracting each part of the ore body. The mineral resource is divided into sections called pushbacks where the geometry of the ore body dictates the geometry of the mining activity [10]. The order in which each portion of the mineral resource or pushback is mined is known as sequencing and the objective is to achieve the highest NPV. Generally, the first part to be extracted will be the part that provides the highest cash flow, with the subsequent sequencing of pushbacks following the same criterion.

The pushback design, the selection of the mining-equipment (size, type and quantity) and the design of the schemes of exploitation are interrelated activities, as illustrated in Figure 6. Ahumada [11] points out that different configuration of these three variables have to be evaluated in order to determine the material requirements of the deployment strategy.

Ahumada [11] proposes two new schemes of exploitation for the Chuquicamata mine in Chile. The authors study provides an example of the interdependence of these variables. The objective of

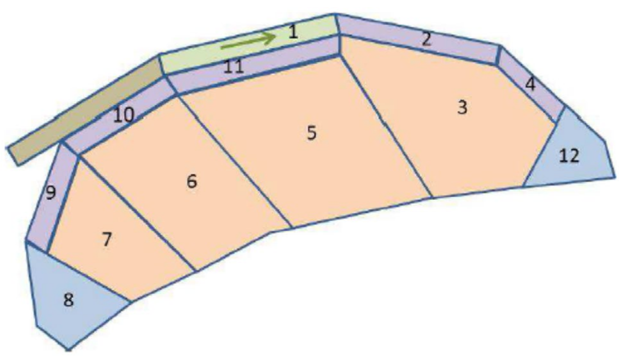

Figure 4. A representation of a sequence for extracting a bench in an open-pit mine.

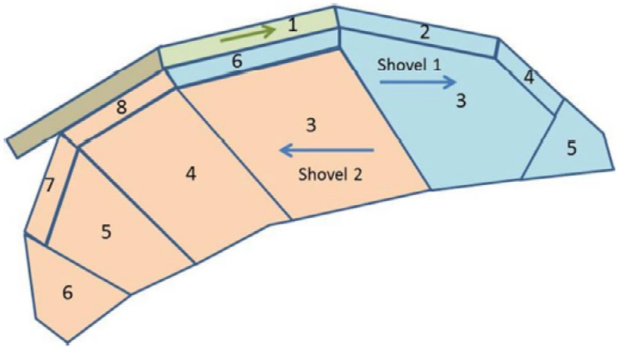

Figure 5. A sequence for extracting a bench with two shovels.

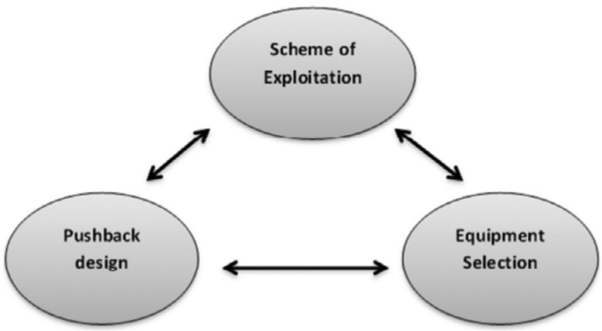

Figure 6. The variables in the design of the schemes of exploitation [11].

the research was to solve the problem of the introduction of new larger trucks. The original design considered schemes of exploitation with only one shovel per bench and a route that had been designed in accordance with the size of the trucks in operation. The authors' analysis provides a technical and economic evaluation of two alternative schemes of exploitation which considered a modification of the pushback design at the same time. The first proposal was to change the size of the pushback and the number of shovels per bench. The second proposal considered maintaining the size of the benches but changing the pit-angle to adjust to the change in the haul route dimensions.

In general, when a scheme of exploitation is designed, the focus is to try to maximise the use and productivity of the loading equipment. Marek and Welhener [12] have noted that a mine plan has to allow an appropriate space for the operation of the loading equipment. The design of the scheme of exploitation must thus consider the dimensions of the loading and hauling fleet (size and quantity) and the mechanical behaviour of the rock [11].

With this information, it is possible to determine the minimum working width for the loading equipment in order for it to carry out its activities without restriction as illustrated in Figure 7.

Although based within an underground copper mining context, a recent study by Salama et al. [13] showed that higher and more expensive mining rates at elevated commodity prices, achieved a better NPV than a lower and less expensive mining rate. Even though pushing mining rates beyond traditional limits may increase mining costs, the higher mining rate at elevated commodity prices was more beneficial.

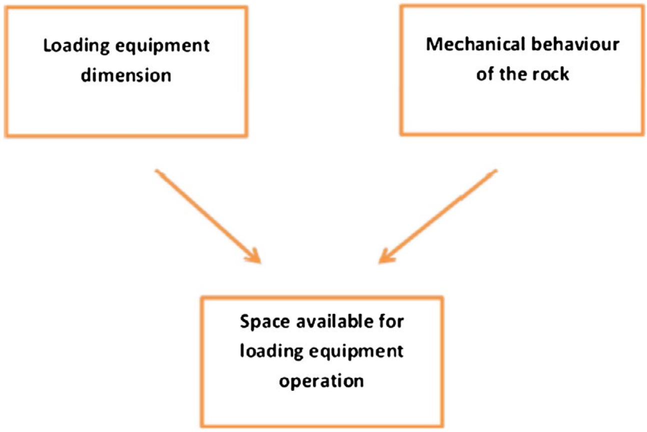

Figure 7. The variables for the determination of the minimum space for loading [11].

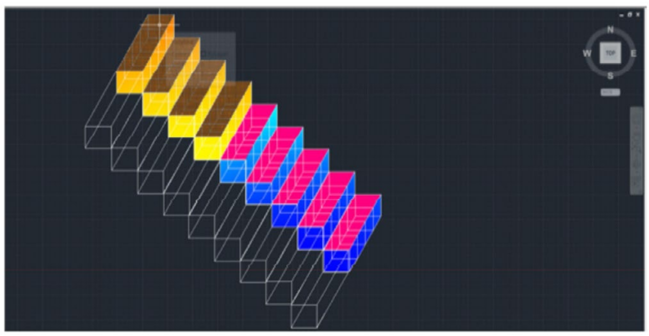

Figure 8. The pushback for the design of the schemes of exploitation.

3. Methodology

A case study for the purpose of evaluating the influence of mining rates via various schemes of exploitation on NPV is presented in the next section. It is firstly important to grasp the methodology behind determining many of the metrics that form the basis for the following case study. Consider firstly the hypothetical pushback illustrated in Figure 8. A constant density of 2.50 t/m^3 is assumed so as to emphasise the effect of the selected scheme and the lack of space in the productivity of the pushback rather than the influence of other variables.

Figure 8 shows a 9 bench pushback whereby each bench has a total area of 360,000 m^2 (bench width 300 m) and is 10 m high containing a total of 9 Mt. The scheme of exploitation design will incorporate P&H 4100 XPC shovels and Caterpillar 793F trucks and considers their typical dimensions. It is assumed that each shovel has a short-term productivity of 4200 tonnes/h.

The shovel mining method that forms the basis for this investigation is the double back-up method as this is typically used wherever possible in highly productive open pit mining of massive mineral deposits, as illustrated in Figure 2.

It was earlier established that the productivity of the shovels in different areas of the bench can be affected by the geometry of the loading area. For the purposes of this case study, three macro-zones were identified. These were: the ramp, the control area and the production area. The productivity of the shovel in the ramp macro-zone is set as being 18% less than the productivity of the shovel in the production area. Shovels operating in the production area do not have geometric restrictions and are therefore able to reach full productivity levels. Full productivities in this case are also assumed for the control area.

For safety reasons, it is assumed that two shovels could not be located for loading in the adjacent bench areas of different levels as this would allow rocks from the upper level to cause significant potential harm if they are to come into contact with personnel and equipment on lower levels.

With respect to the sequence of loading activities within the pit, it is assumed that once the loading activity of the upper level has been completed to expose the material on the lower level, it can only be loaded until an appropriate amount of time assumed for drilling and blasting has past. Such time can vary and depends heavily on the drill rigs used in the operation as well as the quality of rock. In this example, the time for drilling and blasting was considered to be equal to the time required to mine the area.

The definitions of the key typical performance indices that are used in this investigation are as follows:

$$\text{Shovel Utilisation } U(\%) = \frac{\text{Production time}}{\text{Available time}}$$

$$\text{Shovel Availability } Av(\%) = \frac{\text{Available time}}{\text{Calendar time} - \text{Stand by time}}$$

$$\text{Long Term Shovel Productivity (LTSP)} = \frac{\text{Tonnes loaded}}{\text{Calendar year}}$$

$$\text{Short Term Shovel Productivity (STSP)} = \frac{\text{Real capacity of the truck}}{\text{Loading time} + \text{Parking time}}$$

$$\text{Long Term Shovel Productivity} = \text{Short Term Shovel Productivity } \times U(\%) \times Av(\%)$$

The *Production time* corresponds to the time when the shovel is carrying out its main function, that is, when loading the trucks. The *Available time* corresponded to the time when the shovel is, mechanically in operational condition. The Stand-by time corresponds to the time while a shovel is shut down for external reasons to the operation or because this is planned as part of the long-term mine plan.

Figure 9 provides an example of a typical, production-per-hour graph for an open-pit mine that is also used in this case study. It illustrates the main assumed operational delays and the assumed changes in productivity of the equipment as a result of these delays.

The *x*-axis in both graphs represents a 12-h day and night shift. As shown, the productivity decreases to 0 in the middle of the each shift as a result of a break in operations due to meal times. In a similar way, the reduction in the productivity at the beginning and at the end of each shift results from the changeover in the roster. The main delays and the reasons for these are indicated in Table 1.

The times indicated in Table 1 are given on the basis of an utilisation rate of 86% of the available shovel time. For the scheme design however an utilisation rate of 85% is assumed which includes other possible delays such as the need for movements before blasting or for cleaning of a loading area.

Figure 10 illustrates an intermediate bench within the pushback which is being extracted using the scheme of exploitation intended for the use of four shovels. This bench is divided into coloured regions, which is based on the geometry of the different areas inside the bench. These regions represent the drill and blast progression across the bench. The cells in red represent the access ramps across the bench in operation and the dark blue cells correspond to the final ramp of the pushback as part of the pit-wall.

The production area is further divided into sub-regions of different shapes and size. In this case, the smaller squares represent the minimum space available for loading. Some blocks, such as those in orange and blue labelled 7, are smaller but still have faces that correspond to the minimum space required for loading. The depletion of these blocks is designed so that the loading equipment would

Day shift production in tons			
8:00	9:00	1600	
9:00	10:00	2000	
10:00	11:00	2000	
11:00	12:00	2000	
12:00	13:00	2000	
13:00	14:00	0	
14:00	15:00	1600	
15:00	16:00	2000	
16:00	17:00	2000	
17:00	18:00	2000	
18:00	19:00	2000	
19:00	20:00	1800	
		21000	1750

Night shift productions in tons			
20:00	21:00	1600	
21:00	22:00	2000	
22:00	23:00	2000	
23:00	0:00	2000	
0:00	1:00	2000	
1:00	2:00	0	
2:00	3:00	1600	
3:00	4:00	2000	
4:00	5:00	2000	
5:00	6:00	1600	
6:00	7:00	2000	
7:00	8:00	1800	
		20600	1717

Figure 9. The production profiles for shovels in open-pit mining.

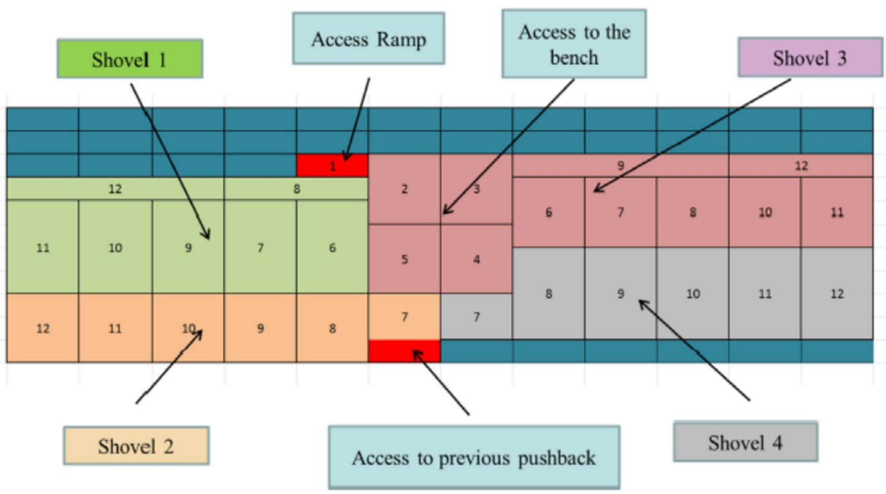

Figure 10. The scheme of exploitation with utilising 4 shovels.

Table 1. The reasons for major delays in the calculation of shovel utilisation.

Day shift (min)	Night shift (min)	Delay name
10	10	Shift crew change and inspection
60	60	Lunch
15	15	Ramp up after lunch
10	10	Shift crew change
0	15	Safety stop
95	110	Total

face the larger face in order to retain the selected shovel mining method (double-sided loading). The number inside the blocks represents the sequence that each shovel must follow in the depletion of the bench. The scheme of exploitation will change with each bench depending on the current location of the access ramp and the number of shovels deployed. In this case, the initial access to the bench was developed by shovel 3 beginning with the extraction of the ramp that is located at the red block labelled 1.

The extraction of blocks 3–5 create the access to the bench. In this case, the access is driven through the middle of the bench. This is favourable for adding more loading equipment, which can then load in different directions. When shovel 3 begins to load the pink block number 6 – shovel 1 begins to extract green block number 6. In this case, the scheme of exploitation is able to utilise a second shovel on the same bench. The other two shovels are meanwhile still used in the loading of previous benches or are on stand-by mode due to a lack of loading space. This scheme utilises four shovels on the same bench commencing with the extraction blocks labelled 7. The incorporation of shovels 2 and 4 have to wait until there is enough safe space for loading. In this case, the separation between shovel 3 and 1, are ideal from the outset in terms of space available for loading. Shovel 2 and 4 do not have this separation initially during the extraction of blocks labelled 7 as they are placed in the same direction next to each other. An appropriate scheme of exploitation was also developed for a 2 shovel operation and a 3 shovel operation.

4. Case study

The evaluation of different mining rates via different schemes of exploitation for exploiting a mineral deposit is illustrated by way of a case study. Figure 11 illustrates a hypothetical two dimensional copper deposit amenable to open pit exploitation. This typically involves the removal of the waste that overlies the deposit in order to gain access to the valuable ore [14]. The deposit in this case is placed in a deep zone with the aim of illustrating the current challenges for the industry in dealing with deeper mineral deposits and the implications of various schemes of exploitation.

Figure 12 provides a two dimensional block model representation together with it respective sequence of the mineral deposit depicted in Figure 11. The extraction of the yellow blocks corresponds to pre-stripping activity. The blocks in red correspond to the ore. The first pushback corresponds to the blocks in purple and the blocks labelled 1 correspond to the two ore blocks contained within pushback 1. The other pushbacks are represented by different colours as shown.

A comparison of three, different mining rates for exploiting the deposit is undertaken. These alternatives analyse a mining rate that optimise the utilisation of the shovels with the two other mining rates aiming to optimise the productivity of the pushback. The former strategy represents current industry practise and the later strategies represent an alternative view in selecting an optimal mining

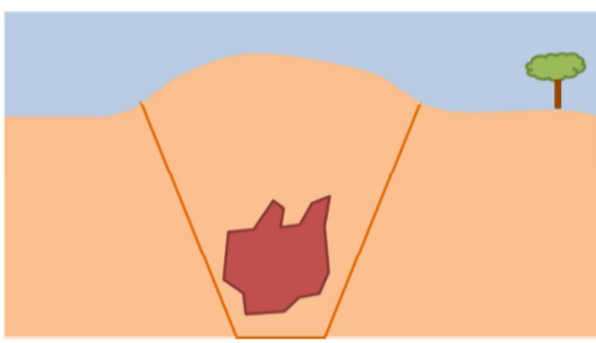

Figure 11. The mineral deposit.

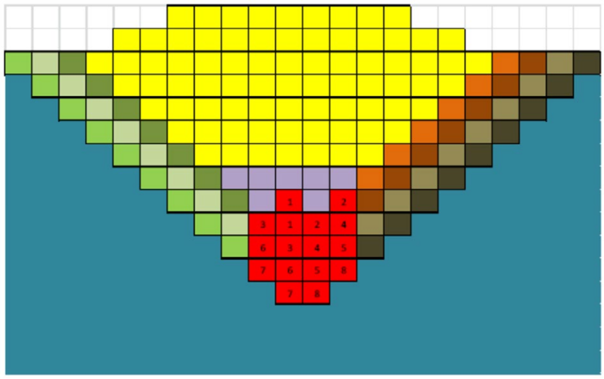

Figure 12. Block model representing the mineral deposit of Figure 11.

rate. Having two alternatives to the current strategy it will show how the value changes through the reduction in the productivity of the equipment.

The three strategies are based on the same pushback design with different numbers of loading equipment (2, 3 and 4 shovels respectively). Keeping the same pushback design was relevant for analysing the different shovel productivities that resulted from a change in the working space and thus a change in the scheme of exploitation, as previously discussed. It is a typical part of the mine planning optimisation process whereby the pushback design is set.

Each block in this case contains 100,000 tonnes of material regardless of whether it is an ore block or waste block. Ore blocks contain a grade of 1.20%Cu with a recovery of 73% using a standard leaching SX EW circuit. A mining cost of $1.5/t, a processing cost of $10.0/t, a copper price of $3.6/lb and a discount of 10.0% is applied.

It is assumed that the mining activity for the strategy with two shovels is carried out by shovels (PH 4100 XPC) with a productivity of 4,200t/hr each. The operation also utilises: 16 trucks (793F), 3 bulldozers (D10), 2 drill rigs (Atlas Copco Pit Vipers (270)), 2 wheel dozers and 2 water trucks. The strategy with 3 and 4 shovels include the capital that is proportional to the capital expenditure of the strategy with two shovels. Therefore, the strategy with four shovels considers a doubling of the capital expenditure considered in the strategy with two shovels.

The annual cash flow depends on the annual production profile which is given by the different schemes of exploitation for each strategy to define the mining capacity. It is assumed that the mining operation has a plant facility that can operate at a maximum throughput rate of 20 million tonnes per year. This capacity is otherwise filled with material from low grade stockpiles with an average grade of 0.4% copper and a recovery of 58%. The exploitation of the copper deposit shown in Figure 12 has an opportunity cost given by the operation of the plant with the material from the stockpiles. A major overhaul is considered in year 8 (costing $200.0 M) to keep the plant facilities operational until year 16.

5. Results

The results that correspond to the design of the schemes of exploitation are divided into four sections. The first section lists the key performance indices for the shovels across each case when these are analysed in terms of their productivity and utilisation. The second section corresponds to the productivity of the pushbacks as a result of each mining rate. The third section presents the sinking rate or the number of benches, which are depleted per year for each case with the aim of comparing the different mining rates and the extraction speeds. The financial metrics of each strategy are then computed and compared.

Table 2. Key performance indices.

Strategy	Average shovel productivity (Mtpa)	Shovel utilisation (%)
Exploitation with 1 shovel	26.2	85.0
Exploitation with 2 shovels	26.0	85.0
Exploitation with 3 shovels	24.2	77.0
Exploitation with 4 shovels	19.6	59.0

Table 3. Material extraction per year (Mpta).

		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Strategy 1	SHE1	25.9	26.1	26.4	26.1	26.2	26.3	26.0	26.3	20.0
Total		25.9	26.1	26.4	26.1	26.2	26.3	26.0	26.3	20.0
Strategy 2	SHE1	26.1	26.1	26.1	26.3	10.3				
	SHE2	25.1	26.3	25.9	26.4	10.8				
Total		51.3	52.4	52.0	52.6	21.0				
Strategy 3	SHE1	25.1	25.1	23.6	3.9					
	SHE2	24.4	24.3	23.8	4.0					
	SHE3	23.1	23.8	24.2	3.9					
Total		72.6	73.3	71.6	11.8					
Strategy 4	SHE1	22.3	20.3	20.0						
	SHE2	19.8	20.8	16.5						
	SHE3	20.1	20.2	17.9						
	SHE4	16.2	17.4	17.4						
Total		78.4	78.7	71.8						

5.1. Key performance indices

Table 2 shows the key performance indices for the schemes of exploitation with 1–4 shovels.

As shown, there is an inverse relationship between the number of shovels included in the mining strategy and percentage utilisation. The highest utilisation rates (85%) are associated with the schemes of exploitation that uses only one and two shovels. The lowest utilisation rate (59%) is associated with the scheme of exploitation that uses four shovels. This is a similar pattern found for the average shovel productivity rate where the highest number of shovels in the pushback corresponds to the lowest average shovel productivity.

5.2. Production profile across life of mine

Table 3 shows the production per shovel on a yearly basis for each of the strategies that are being investigated to deplete each pushback. As shown, each shovel is labelled as 'SHE' with an associated number to distinguish the equipment count. The life of the mine for the case with one shovel is nine years with an average extraction rate in the first eight years of approximately 26.0 Mt. The strategy with four shovels has a mine life of only three years. In this case, the production of each shovel is different because their pattern of utilisation is different. Shovel 4 also only operates when it has sufficient space for loading.

5.3. Sinking rate

Table 4 shows the sinking rate or the number of benches which are depleted per year for the four strategies. The sinking rate increases when additional shovels are added to the schemes of exploitation. The sinking rate of strategy 2 is exactly double the sinking rate for strategy 1 because those shovels operate without space restriction so the yearly production rate for each shovel remains the same. This represents a 100% incremental increase in the sinking rate.

Strategies 3 and 4 are however subject to restricted loading space to thus reduce the production of the associated shovels. This results in an increase in the sinking rate at different proportions. The

Table 4. The pushback productivity and the sinking rate.

	Material (Mtpa)	Benches/year
Strategy 1	31.2	3.4
Strategy 2	62.4	6.8
Strategy 3	86.6	9.6
Strategy 4	93.6	10.3

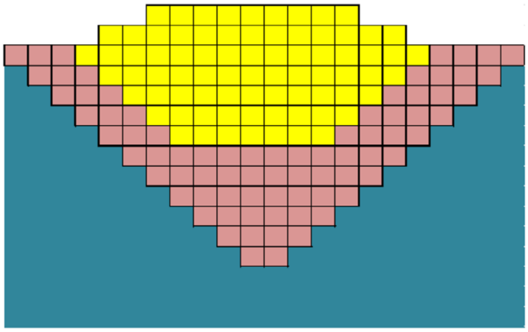

Figure 13. The pre-stripping activity.

Table 5. The pre-stripping phase.

	Strategy with 2 shovels		Strategy with 3 shovels		Strategy with 4 shovels	
Year	Mtpa	Mt	Mtpa	Mt	Mtpa	Mt
0	0	770	0	770	0	770
1	100	670	140	630	150	620
2	100	570	140	490	150	470
3	100	470	140	350	150	320
4	100	370	140	210	150	170
5	100	270	140	70	150	20
6	100	170	70	0	20	0
7	100	70				
8	70	0				

incremental increase in the sinking rate from the strategy with two shovels to the strategy with three shovels is 41.1%. The incremental increase in the sinking rate from the strategy with three shovels to the strategy with four shovels is 7.3%.

The productivity of the pushback follows the same trend as that of the sinking rate. An increase in the number of shovels needed to deplete the pushback increases the productivity of the pushback from 31.2 Mt in the case with one shovel to 93.6 Mt in the case with four shovels. From Strategy 1 to Strategy 2, the incremental increase in productivity is 100%. From Strategy 2 to Strategy 3 it is 38.8%, and from Strategy 3 to Strategy 4 it is 8.1%.

It is clear that adding more shovels to the system accelerate the extraction of material from the pushback, albeit, less effectively with each addition.

5.4. Valuation of strategies

The pre-stripping phase involved a total of 770 million tonnes. The blocks in yellow in Figure 13 correspond to the waste that was removed in the pre-stripping activity.

Table 5 shows the pre-stripping activity for each of the exploitation strategies, which are analysed in this case study. The scheme of exploitation with two shovels considers a mining rate of 100 Mtpa.

This strategy maximises the utilisation of the shovels and minimises the mining costs. The waste, which is considered in the pre-stripping phase, is depleted in eight years using this mining strategy.

As shown, the strategies with three and four shovels have faster designated mining rates of 140 Mpta and 150 Mpta, respectively. The time for depleting the associated waste of the pre-stripping phase was six years for both these cases. However, the remaining waste of the pre-stripping in the sixth year is different. This consists of 70 Mt for the strategy with three shovels and 20 Mt for the strategy with four shovels.

Figures 14–16 represent the annual extraction amounts of ore and waste for each mining strategy. The saw graphs are built manually to illustrate how each pushback is depleted. They consider the equipment available with its respective productivities and the amount of ore and waste of each phase. The sequence of each saw graph is the same for the three cases; however, the timing to start a new pushback is different and depends on the mining rate of each case.

In Figure 14, pushback 1 (corresponding to the green line) shows waste is removed in years 8 and 9 followed by its ore removal in year 10. In each of these cases it is assumed that ore must be uncovered in the year prior to its extraction.

Figures 14–16 illustrate the production profile for each loading strategy. The first case corresponding to Figure 14, is a strategy with two shovels shows waste extraction begins with pushback 1 in year 7 and finishes with pushback 8 in year 17.

Figures 15 and 16 show the profiles of the strategies that have higher mining rates. In these cases, the extraction of waste is increased with the aim of exposing the ore more quickly to begin the processing

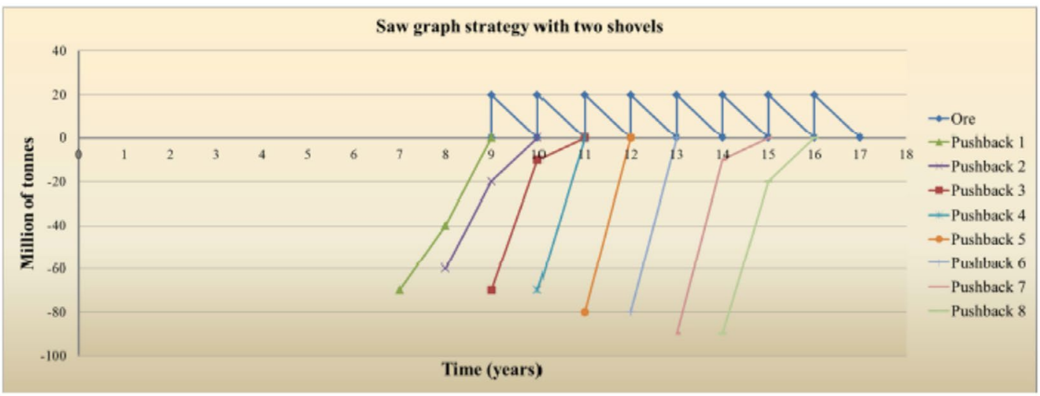

Figure 14. The saw graph for two shovels.

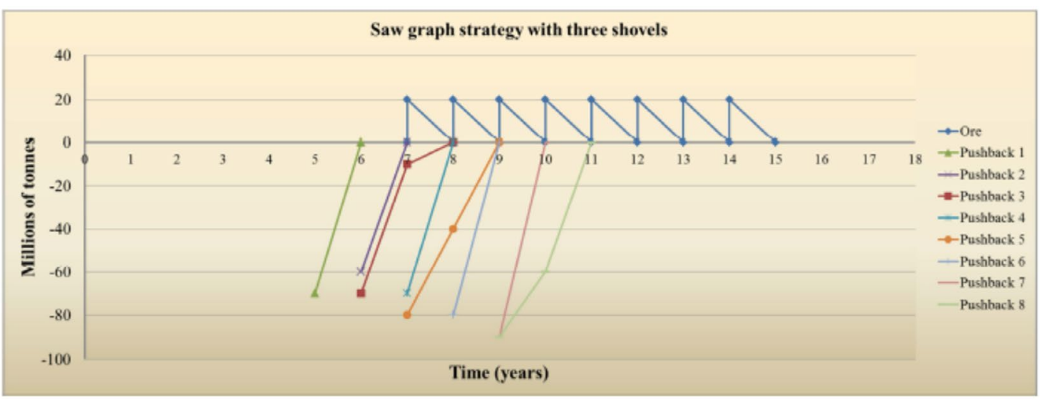

Figure 15. The saw graph for three shovels.

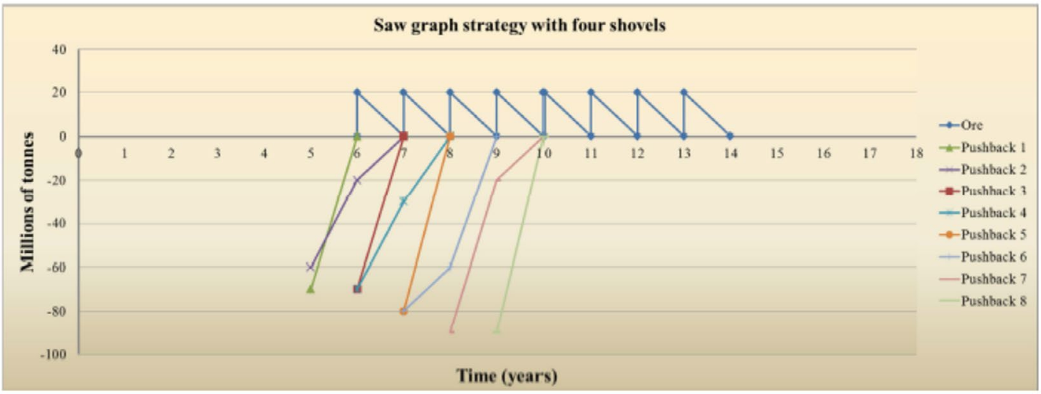

Figure 16. The saw graph for four shovels.

of the material in a shorter time. The waste extraction lasts for six years for the scheme with three shovels and for five years for the case of the scheme with four shovels.

In these three scenarios, the production of ore lasts nine years and is equal to 20 Mpta. The difference in the three saw graphs occurs in the first year of ore production. The strategy with four shovels is the first to begin the processing of the ore in the sixth year. This is due to an acceleration in the extraction of waste from pushback 1 and of waste from the pre-stripping phase.

Table 6 shows the cash flows for the mining strategy which considers the scheme of exploitation using two shovels. This represents the optimised base case. The life of the mine for this strategy is sixteen years and positive cash flow begins in year 9. The PV for this case is $920 M.

The operation produces negative cash flows for the first eight years as a result of pre-stripping activity and waste extracted from pushback 1. In this case, ore and waste are extracted until the end of the life of mine. From year 9 onwards positive cash flows are achieved. As the discount factor decreases with time the cash flows towards the end of mine life have a lower impact on present value.

Table 7 shows the cash flows of the mining strategy with three shovels. The deposit for this strategy is extracted in 14 years with positive cash flows beginning in year 7. The PV for this case is $939 M.

The extraction of waste for the strategy with three shovels finishes in year 11 with negative cash flows finishing in year 6 due to more rapid access to ore. The positive discounted cash flow begins in year 7, decreasing until year 14. This strategy reduces the years of negative cash flow to only six but results in higher costs as a result of a higher production of waste in these years as well as the decrease in the key performance indices of the shovels.

Table 8 shows the cash flows of the mining strategy with four shovels. The life of the mine in this case is thirteen years with a positive cash flow beginning in year 7. The PV for this case is $966 M.

The strategy with four shovels has negative cash flows for only the first five years. It also has a declining, positive cash flow from year 6 until year 16 as a result of the effect of the time value of money.

These three cases all have negative cash flows at the beginning of the operation as a result of the pre-stripping activity. The strategies with three and four shovels however have higher negative cash flows because they have accelerated the extraction of waste and have concentrated it in the early years. These strategies also have higher mining costs as a result of a higher material movement per year.

6. Discussion

The design of the schemes of exploitation is an activity that considers the technical elements of mine planning and is related to the dimension of the pushback and to the characteristic of the loading fleet. Current industry practice is represented by the strategies that use just one and two shovels. The main consideration in these strategies is the need to have enough space for loading to maximise the

Table 6. Cash flow for the two-shovels strategy.

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ore (Mt)	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Waste (Mt)	100	100	100	100	100	100	100	100	80	80	80	80	80	80	80	20
Metal (kt)	44	44	44	44	44	44	44	44	175	175	175	175	175	175	175	175
Revenue (M$)	349	349	349	349	349	349	349	349	1390	1390	1390	1390	1390	1390	1390	1390
CAPEX (−M$)	309	15	15	15	15	15	15	215	15	309	15	15	15	15	15	115
Mine cost waste (−$M)	150	150	150	150	150	150	150	150	120	120	120	120	120	120	120	30
Mine cost ore (−$M)	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Process cost (−$M)	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200
Tax (−M$)	0	0	0	0	0	0	0	0	363	363	363	363	363	363	363	399
Cash flow ($M)	−340	−46	−46	−46	−46	−46	−46	−246	662	368	662	662	662	662	662	616
Dis. cash flow ($M)	−309	−38	−34	−31	−28	−26	−23	−115	281	142	232	211	192	174	159	134

Grey shading represent years with negative cash flows due to pre-stripping activities.

Table 7. Cash flow for the three-shovel strategy.

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ore (Mt)	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Waste (Mt)	140	140	140	140	140	140	120	120	120	120	60	0	0	0	0	0
Metal (kt)	44	44	44	44	44	44	175	175	175	175	175	175	175	175	44	44
Revenue (M$)	349	349	349	349	349	349	1391	1391	1390	1390	1390	1390	1390	1390	349	349
CAPEX (–M$)	456	23	23	23	23	23	23	223	23	309	23	23	23	23	23	123
Mine cost waste (–$M)	210	210	210	210	210	210	180	180	180	180	90	0	0	0	0	0
Mine cost ore (–$M)	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Process cost (–$M)	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200
Tax (–M$)	0	0	0	0	0	0	333	324	324	330	366	402	402	402	0	0
Cash flow ($M)	–547	–113	–113	–113	–113	–113	625	434	634	341	682	736	736	736	97	–3
Dis. cash flow ($M)	–497	–94	–85	–77	–70	–64	321	203	269	132	239	243	213	194	23	–1

Grey shading represent years with negative cash flows due to pre-stripping activities.

Table 8. Cash flow for the four-shovel strategy.

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ore (Mt)	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Waste (Mt)	150	150	150	150	150	130	130	130	130	110	0	0	0	0	0	0
Metal (kt)	44	44	44	44	44	175	175	175	175	175	175	175	175	44	44	44
Revenue (M$)	349	349	349	349	349	1391	1391	1391	1390	1390	1390	1390	1390	349	349	349
CAPEX (−M$)	603	30	30	30	30	30	30	230	30	309	30	30	30	30	30	130
Mine cost waste (−$M)	225	225	225	225	225	195	195	195	195	165	0	0	0	0	0	0
Mine cost ore (−$M)	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Process cost (−$M)	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200	200
Tax (−M$)	0	0	0	0	0	320	320	311	311	336	402	402	402	0	0	0
Cash flow ($M)	−709	−136	−136	−136	−136	615	615	424	624	350	728	728	728	89	89	−11
Dis. cash flow ($M)	−644	−112	−102	−93	−84	347	316	198	265	135	255	232	211	23	21	−2

Grey shading represent years with negative cash flows due to pre-stripping activities.

utilisation of shovels. The incorporation of the second shovel is considered to increase the productivity of the pushback and, also, to increase the sinking-rate without adversely affecting key performance indices of the shovels.

The design of more aggressive schemes of exploitation, as presented in this paper, follows the same principles that were considered for the other cases. The activity that was associated with these more aggressive schemes was however, more complex because it was not possible to have the shovels loading at all times. In this case study, the reduction in the shovel productivity was the result of a lack of space for loading. This reduction in productivity as a consequence of a lack of space was considered in this paper but a complex analysis could also have included an additional reduction in productivity as a result of having major traffic in the area and also, as a result of interactions between shovels. The latter refers to situations where it is necessary to move cables and delay times related to the drilling and blasting process.

The economic valuation process considers both the productivities of the pushbacks and of the shovels and also the economic parameters, which are related to the cost of each strategy. The cash flow for each mining alternative shows the difference in the cost of each strategy. In the first years of operation, these costs are mainly associated with waste mining corresponding to pre-stripping and the waste of pushback 1. The more aggressive strategies for mining the mineral deposit considered higher mining costs. These reflected the double effect of not only an increase in the number of shovels and mining equipment but also, a decrease in the productivity of this equipment.

The benefits of an aggressive mining rate as represented for the last two strategies are based on the premise that these alternatives, by bringing forward the extraction of the ore, would create positive cash flows in advance of those associated with the first case. These strategies would also reduce the life of the mine so that the positive cash flow would then be less affected by the effect of the time value of money. The Present Value for the three investigated strategies is presented in Figure 17. This diagram clearly illustrates how the exploitation strategy that employs four shovels has a higher present value ($966 M).

The maximum NPV corresponds to the schemes of exploitation with four shovels that had a shovel utilisation of only 59% and thus higher mining costs per unit of material than the strategy with two shovels. These results demonstrate that an increase in the mining cost can result in a higher NPV when the benefits of bringing forward positive cash flows and of reducing the life of the mine outweigh the increase in costs.

The design of the schemes of exploitation, which changes the number of shovels in the same pushback, thus modifies the mining and the sinking rate. A higher number of shovels in the pushback will increase the sinking rate and the mining cost. The addition of more loading equipment may not always be feasible due to major decreases in equipment performance. The benefits of accelerating the extraction may thus not be enough to pay for the extra mining costs of an additional shovel. The

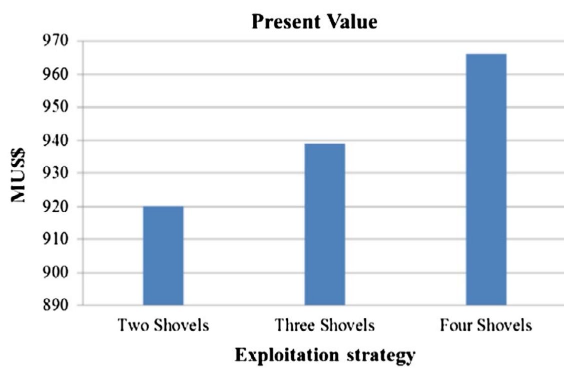

Figure 17. The PVs for each exploitation alternative.

higher sinking rate, with higher mining costs and lower equipment performance therefore do not always create value.

The results presented also illustrate the cash flow of two completely different strategies; the strategy with two shovels corresponding to current industry practice which optimises the equipment performance and the strategies with three and four shovels which instead optimised the performance of the pushback. In this case the mining rate that best optimised the associated value was the one that focused on pushback performance. This result thus demonstrates that a focus on the pushback performance can generally offer a different result (in terms of value) than does a methodology that focuses solely on equipment performance. The value that is associated with mining rates with a focus on the pushback productivity may be higher than the value generated by current industry practice whereby the mining rate is constrained by mining costs.

The saw graphs presented also illustrate the different production profiles for each of the mining alternatives being analysed. The focus on value for finding the optimum mining rate led to the development of alternative production profiles represented by the 3 and 4 shovel saw graphs. These were associated with higher mining costs as a result of the decrease in the performance of the equipment in comparison to the 2 shovel strategy. The result of the economic valuation of each strategy is evidence that the mining rate that optimises the value can differ from the mining rate that minimises mining cost.

In this paper, the mining rate was modified with different schemes of exploitation as part of the mine planning activity. The idea was to demonstrate that certain preconceived ideas about a high equipment performance could lead to the exclusion of a group of possible configurations that has higher costs but also could have a higher value. It is relevant to notice, however, that the mining rate is not exclusive to the deployment of the loading and hauling on the pushbacks. Other variables such as rock hardness, material quality, slope stability, presence of water, and material mixing requirement at the plant could affect the selection of an optimum mining rate. These variables could represent other constraints during the mine design but also represent variables inherent to the mineral deposit. The focus of this paper is in a variable that affect the mining rate but it is under the control of the mine planner. The deployment of the loading equipment is designed when the mine planner put an 'operational' plan in place. As the problem becomes more complex adding additional constraint other mathematical tools can be used to mining rate estimation problem.

7. Sensitivity analysis

The proposal presented in this paper challenges the current practice of minimising cost as a constraint during the planning process. This practice excludes some possible schemes of exploitation with a low equipment performance and misses some opportunities to obtain higher returns through the exploitation of a mineral resource. The sensitivity analysis presented in this section aims to show when this new proposal could be a source of value for the mining company.

Table 9 shows the breakeven price and the IRR for the strategies with three and four shovels in comparison to the case with two shovels. The strategy with 4 shovels has a lower breakeven price and a higher IRR that is aligned with the higher PV of this strategy presented in Figure 17. Scenarios with prices below 3.19 US$/lb indicate that the strategy with 2 shovels is more attractive and should be the preferred option to exploit the pushbacks of this deposit. Therefore, more aggressive scheme of exploitation can be a source of value in scenarios of high price.

Table 10 shows the present value for different copper prices and discount rate scenarios. In the low price scenarios the strategy with two shovels adds more value. On the contrary, the high price case scenarios show a result more aligned with the mid case where three and four shovels in a more complex scheme of exploitation adds more value. The three discount rates in Table 10 show that the more intense mining rate adds a higher value. This is aligned with the IRR presented in Table 9 that are higher of these three scenarios for the discount rate.

Table 11 show the Present Value from a sensitivity analysis for different ore head grade. The new proposal of accelerate the mining rate has higher benefits in a high grade scenario where the strategy

Table 9. Breakeven price and IRR.

Strategy	Breakeven price (US$/lb)	IRR (%)
3 Shovels	3.33	13
4 Shovels	3.19	16

Table 10. Sensitivity analysis with copper price and discount rate.

	Copper price US$/lb			Discount rate (%)		
	Low	Mid	High	Low	Mid	High
Strategy	3.0	3.6	4.0	8.0	10.0	12.0
2 Shovels	264	920	1358	1263	920	656
3 Shovels	240	939	1405	1285	939	665
4 Shovels	243	966	1447	1303	966	694

Table 11. Sensitivity analysis with copper head grade.

	Cu Grade (%)		
	Low	Mid	High
Strategy	0.8	1.2	1.5
2 Shovels	228	920	1439
3 Shovels	102	939	1567
4 Shovels	45	966	1657

Table 12. Sensitivity analysis with unitary mining cost and copper head grade.

Strategy	Mining cost US$/tm	PV (US$M) mid case HG	PV (US$M) high case HG
2 Shovels	1.5	920	1439
3 Shovels	1.6	851	1479
4 Shovels	1.7	791	1481

with 3 shovels has an NPV of US$128M (US$1567–US$1439) with respect to the 2 shovel case. The NPV became more than US$200M when the comparison is between the strategy with 4 shovels and the 2 shovels.

In this paper higher capital cost where included accounting for the additional mining equipment of each strategy. Higher operational cost where considered in each strategy for the mine (including additional drilling & blasting cost and an additional maintenance related to the different fleet). However, the unitary cost at the mine was considered equal for the three strategies. An increase in the unitary cost decreases the value from the strategies with more complex schemes of exploitation. The Table 12 shows the present value of a sensitivity analysis that includes a higher unitary mining cost for the cases presented in Table 11. The outcomes show that increasing the unitary mining cost is detrimental for the value of the more aggressive schemes of exploitation for the mid case head grade. However, the outcome shows that the new proposal is still attractive from a present value point of view for the high grade scenario. In this last case, the revenues from a higher ore grade overweigh the additional costs from this new strategy.

The new proposal presented in this paper shows that new mining rates as a result of more complex and aggressive scheme of exploitation could raise to higher value solutions. This increases the options for the mine planner when the decisions about the equipment deployment in the benches are made. The sensitivity analysis, however, shows that only in certain conditions the minimisation of cost and optimisation of value go in different directions. It seems to be that when the revenues are considerable higher that the costs the benefits are more evident. This could be the case of a high price scenario or a high grade scenario where the revenues are more significant and the time value of money go in favour of this new proposal. However, in mines with higher operational cost and lower margins the benefits could not be enough to pay the additional costs. Therefore, in low price scenarios the minimising the cost through a high equipment performance could be the best solution that bring the higher value to the company.

8. Conclusion

Mine planning is a complex activity that has the objective of creating value via the exploitation of the mineral resource. Different variables and assumptions are considered during the planning stage. The use and productivity of the equipment in the mine is considered important for maintaining lower exploitation costs and for increasing the margin with respect to revenues. The minimisation of cost is thus one of the principal operational objectives and numerous activities are aligned towards achieving this.

From a strategic point of view, mine planning with the objective of minimising costs can lead to sub-optimal solutions which can destroy value. This paper has however, presented an example where the design of the schemes of exploitation can actually provide an opportunity for improving the revenues of the mining company.

The potentially beneficial schemes of exploitation which are presented correspond to the effective deployment of the equipment in each pushback. Such schemes will determine both the mining rate and the sinking rate of the pushback. The actual design of these schemes of exploitation considers the utilisation and the productivity of the loading equipment as a constraint. The mining rate is thus the result of a deliberate strategy that optimises the performance of the equipment and reduces mining costs.

This paper has presented a case where higher sinking rates combined with higher mining costs and lower equipment performances can create higher value in comparison to the current strategy which optimises the key performance indices of the loading equipment and seeks to reduce mining costs. It has been demonstrated that the productivity of the pushback and the productivity of the shovel are relevant to the estimation of the optimal mining rate of the mineral deposit. However, a focus solely on equipment productivity can also lead to sub-optimal cases. It was shown that a strategy that privileges equipment performance could lead to accepting the strategy with two shovels as the best alternative for exploiting the deposit. However, in this case it was also shown that the optimum mining rate was actually associated with the strategy with four shovels; this strategy had been created with a focus on the productivity of the pushback. Therefore, according to the results which have been presented in this paper, the productivity of the pushback may be more important in the estimation of the mining rate than is the productivity of the loading equipment.

This paper has further demonstrated that the mining rate that optimises value can be different to the mining rate that minimises cost. From the analysis of the cases that were presented, it can be concluded that this proposal would be most likely to generate major value for the business under certain commodity price environments.

Table 10 reinforce this idea showing that in high prices scenarios the NPV from this new proposal generate greater value in comparison to the strategy that optimise the equipment performance. The same is seen in Table 11 where in scenarios with higher grade the NPV of the most aggressive scheme of exploitation can be greater than US$200M.

9. Recommendations

The design of the schemes of exploitation was performed manually for this project by following mine planning principles as discussed throughout. A more realistic case scenario might include the use of more equipment or the imposition of other technical constraints on the loading activity; these could include such constraints as the type of rock, and the presence of fractures and water among others. In such cases, it may often be difficult to manually calculate the rate of productivity and to determine the best utilisation of shovels. It is recommended that these future scenarios should include optimisation tools with the aim of devising more intelligent designs that will better optimise the resource and better accelerate the extraction of the mineral.

The conclusions presented in this paper are not exclusive or limited to a certain type of mineral deposit. In fact, in deposits with special scatter grade or different ore qualities a variable sinking rate as a part of more or less aggressive extraction rates could benefits the mine operation reducing the risk related to grade variability through the deposit. However, it is type of deposit could be another variable that can be included in the optimisation problem of estimate the mining rate through the life of the mine. It is recommended include this variable in further research in the mining rate estimation.

Slope stability is out the scope of the analysis presented in this paper but could be included in future research. More intense mining activity could be a challenge in conditions with low slope stability from a highly fractured rock. Changes in the slope angle could be a solution in these cases but it could also modify the ore/waste ratio and as a consequence the sequence. The optimisation problem in this scenario becomes more complex and additional mathematical tools are recommended to address this topic.

Drilling, blasting, loading and hauling work as a system in the mine and therefore, the performance of each fleet affect the performance of the others. Delays in drilling can cause delays in loading if the blasted material is not enough to assure a continuous loading activity. Similarly, hauling performance is directly related to the loading performance. In this paper, the decrease in the loading performance is considered the result of aggressive scheme of exploitation when the available space for loading is restricted. These configurations impact the utilisation and productivity of the shovels. However, other variables, such as the hauling performance, could also be included in the analysis. Increasing the mining rate with more aggressive scheme of exploitation will increase the traffic in the benches in exploitation. This is an additional source of delays in the operation than can decrease even more the productivity of the loading and hauling fleets. Further analysis is recommended in this aspect.

The incorporation of the mining rate as a variable into the optimisation of cut-off grade gives rise to a complex, optimisation problem with several constraints and parameters. Modelling and solving this complex dynamic problem is not easy. Network flow or mixed-integer programming could be usefully employed in any analysis. It is recommended that research in this area should continue to endeavour to focus on the need to obtain the best solution to the optimisation problem for these three variables.

This paper has proposed the adoption of more aggressive schemes of exploitation to accelerate the extraction rate at the mine and to bring forward access to ore. This strategy could also be incorporated as an operational response to positive changes in the price of the commodity. Such an analysis of these kinds of flexibilities could be included in any evaluation with real options. This analysis would seek to evaluate, in any calculation of the value of a mining project, the possible responses of management to changes in the inputs of parameters such as commodity price. It is recommended that any future research should endeavour to explore the benefits in value for such an evaluation strategy.

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Disclosure statement

No potential conflict of interest was reported by the authors.

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