The Use of Extractive Blending Optimisation for Improved Profitability

C Wharton[1]

ABSTRACT

The last decade has seen major changes in mining operations. Metal price reductions and increased competition have driven operators to cut costs and improve efficiencies.

There are many situations where improving the quality of the input into a process, through the use of blend optimisation or other linear programming techniques, can lead to better throughputs, reduced costs and increased recoveries. This paper will look at the ways in which extractive blending techniques can be used to improve process recoveries, generate better pitshells and even provide solutions for better utilisation of multiple plant capacities.

INTRODUCTION

The last decade has seen major changes in mining operations. Metal price reductions and increased competition have driven operators to cut costs and improve efficiencies.

Imrie (2001) cites the need to look over the entire operation, not only individual components in the chain, and shows how changes in blasting can lead to improvements in recovery in the mill. King (1999) looked at scheduling rock types through the process based on their $/tonne contributions rather than just their economic grade. Indeed there are many situations where improving the quality of the input into a process, through the use of blending or other linear programming (LP) techniques, can lead to better throughputs, reduced costs and increased recoveries. Examples of these include:

- Mill throughput: can be sensitive to average rock hardness, work index or the ratio of materials such as clays. Control of these can lead to improved throughputs.
- Process improvement: control over sulfur content into autoclaves can result in reduced costs and faster reaction times. If the concentration is too low, there is not enough energy liberated for the process to continue and if the concentration is too high, there may be insufficient oxygen to continue the process.
- Process recovery: deleterious materials can reduce recoveries and increase processing costs.

The application of extra control may come at additional cost and complexity and may lead to reduced reserves but hopefully it will lead to increased Net Present Value (NPV). Typically, the control factors can be included in the block model as additional elements and used in the extractive blend as constraints.

This paper will look at the way extractive blending techniques can be used to control input to the process, generate better pitshells and even provide solutions for better utilisation of multiple plant capacities.

All the techniques described in this paper can be implemented in the Whittle Strategic Mine Planning package using the Blending module, which incorporates the necessary LP capabilities.

EXTRACTIVE BLENDING

Extractive blending is the term given to the use of blending, to meet constraints, prior to input into a mineral liberation and separation process. Figure 1 shows a typical extractive blend schematic. Material taken directly from the pit may be combined with material from one or more optional stockpiles, to create one or more desired blends, each of which is suitable as an input to a process. From a modelling point of view, the in-pit material is represented by a number of bench/phase panels, each of which has defined tonnage and grade characteristics. These panels have a defined availability depending on their position in the mine and the defined mining sequence. Material in stockpiles is likewise represented by tonnage and grades. The combination of panels and stockpiles forms the basis for a Linear Programming (LP) Optimisation. The blend optimiser seeks to maximise cash flow from the processing of the available panels while taking user constraints into account. The constraints for the optimisation are:

1. the available mining, processing or product limits;
2. the blend minimum or maximum element grade limits;
3. stockpile quantity limits; and
4. block usage cannot exceed block tonnage.

The formulation can be expressed mathematically as:

$$
\begin{array}{c}
\begin{array}{cc}\text{Panel/bin sources} & \text{stockpiles}\end{array} \\
\begin{bmatrix}
v1 & v2 & . & . & . & vn & vm & . & . & vs \\
p1 & p2 & . & . & . & pn & pm & . & . & ps \\
s1 & s2 & . & . & . & sn & sm & . & . & ss \\
ma1 & ma2 & & . & & man & mam & . & . & mas \\
mi1 & mi2 & & . & & min & mim & . & . & mis \\
t1 & t2 & . & . & . & tn & tm & . & . & ts
\end{bmatrix}
\begin{bmatrix} x1 \\ . \\ . \\ . \\ . \\ xn \end{bmatrix}
\begin{array}{c} \\ \le \\ \le \\ \le \\ \ge \\ = \end{array}
\begin{bmatrix} va1 \\ P1 \\ S1 \\ MA1 \\ MI1 \\ Tn \end{bmatrix}
\end{array}
$$

where:

v1..vs	are value contributions and va1 is derived objective function
p1..ps	are processing tonnages and P1 is process capacity constraint
s1..ss	are selling (production units) and S1 is selling capacity constraint
ma1..mas	maximum grade contributions and MA1 is maximum grade constraint
mi1..mis	minimum grade contributions and MI1 is minimum grade constraint
t1..ts	are usage constraints to ensure that each bin cannot exceed its own capacity
x1..xn	are the amounts of each item to take to maximise the objective function

There can be multiple P, S, MA, MI and T constraints.

The mining constraint is applied externally to the formulation and controls the amount of material supplied. The user can control whether a fixed amount of mining is used or the minimum of the available capacity and the amount required to meet production requirements. The blending LP solves the equations in a manner that maximises the objective function. The result defines, for each period, what proportion of each panel is blended, stockpiled or rejected and what proportion of each stockpile is used.

You must define one blend for each process and for each blend you must specify its allowable source rock types, stockpile material, constraints and blending costs.

1. 66 Rathmullen Quadrant, Doncaster Vic 3108. E-mail: chrisw@gemcomsoftware.com

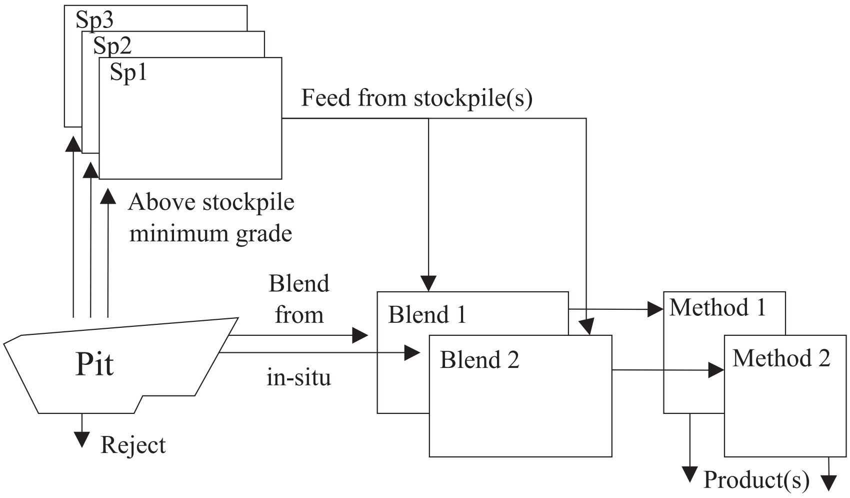

FIG 1 - Extractive blend schematic.

A key consideration is the definition of blend material. You can leave it up to the blend optimiser and only define element constraints or you can limit what is to be considered as blend material either by only providing parcels for material that you wish to consider for the blend in the block model, specifying a minimum or maximum grade in the process definitions, or by defining your own additional blend bin constraints.

BLEND BINS

The Whittle blend optimisation does not consider each individual block in the model, but accumulates material into a number of bench phase panels, then schedules from these accumulators. In the simplest case, the bench phase panel will have one average grade for each element for each rock type. In general, this does not lead to good blending; however, a user can create their own 'blend bins' and define up to 50 more accumulators on each panel. These blend bins can categorise the data into a series of grade ranges, which represent a degree of selectivity that is reflective of the actual mining conditions, to improve the chance of blending.

The 'bins' are defined in a similar fashion to stockpiles. They are defined for each rock type and have a minimum and maximum grade per element. If you add bins then the analysis program accumulates data for each bin in each panel and makes this extra information available to the LP optimisation. This results in better blending. You can control whether entry to a bin is based on grade minima and maxima or on equivalent metal grade. See example one and three for further discussion.

EXAMPLE ONE – MINIMISE CONTAMINANTS

The example detailed below uses a constraint of soluble copper; however, the same technique would apply to controlling rock hardness, work indexes, rock-type ratios or any other element limits. These techniques could work equally well with leach or flotation processes.

Lahtinen (2004) showed that a specific geological ore type can have a wide range of metallurgical recoveries depending on input copper concentrations. Tran *et al (1997)* state that each one per cent of soluble copper in an ore increases cyanide consumption by up to 23 kg/t so there is an obvious potential processing cost benefit to limiting the soluble copper. Furthermore, soluble copper inhibits the floatation of copper sulfide ores leading to potentially lower gold recoveries. If your metallurgist suggested that if the soluble copper content can be limited below 25 per cent then the extraction process could achieve better recoveries, then it would be worth investigating the feasibility of this action. In this example, we will examine the modelling of this behaviour and the application of blending strategies to deal with it.

This example is based on the Marvin porphyry copper mine model included in the Whittle demonstration data set. It contains gold (AU) and copper hosted in three ore types; oxide (OX), mixed (MX) and primary (PM). The model includes data for both copper sulfide (CuS) and soluble copper (CSol) concentrations.

Base case

Table 1 summarises the base case settings. The base case has non-linear recoveries for copper and has different recoveries for each of the rock types with maximum recoveries of 62 per cent, 82 per cent and 94 per cent for OX, MX and PM respectively. The base case life of mine generates an NPV of $273 M. Mine life is 12.7 years. As can be seen in Figure 2, the soluble copper grades are all elevated in the first five years (44 per cent, 42 per cent, 36 per cent, 32 per cent and 28 per cent respectively). This may possibly make blending a difficult proposition in the early years.

TABLE 1

Base case settings.

Item		Copper	Gold
Recoveries	OX	CROX	0.85
	MX	CRMX	0.5
	PM	CRPM	0.6
Income		$20.00/%M	$12.00/gram
Selling costs		$ 7.20/%M	$ 0.20/gram
Mining costs		$0.90/tonne	
Processing costs	OX	$4.00/tonne	
	MX	$4.00/tonne	
	PM	$3.85/tonne	
Throughput	Mine	60 M	
Limits/annum	Mill	20 M	
Investment capex		$250 M	
Discount rate		10%	

Copper recoveries vary with grade:

CROX max(0,(62-13.30/max(0.20,CuS.G)))/100

CRMX max(0,(82- 6.05/max(0.05,CuS.G)))/100

CRPM max(0,(94- 3.20/max(0.02,CuS.G)))/100

Maintain soluble copper grade under 25 per cent (with blend bins)

Let us assume, in this example, that when the soluble copper grade is kept below 25 per cent that the recoveries shown in Table 2 can be achieved by the process. Figure 3 shows the grade tonnage distribution for soluble copper in the Marvin data set. While the weighted average for the entire body might be below the target, the contribution from each rock type is quite different. The oxide, mixed and primary ore types have average copper soluble grades of 40 per cent, 35 per cent and 11 per cent respective and represent six per cent, 42 per cent and 52 per cent of the available ore. Blending the mixed ore is going to be the major issue. There will have to be some user-defined blend bins, for each rock type, to provide additional sources. The bin minimums for gold and copper (CuS) can be calculated from marginal cut-offs based on the revised recoveries. These are shown in Table 2. The bins will want to have a range of CSol grades to provide better blending capabilities. A possible spread of soluble copper grades might be: <15 per cent, 15-20 per cent, 20-25 per cent, 25-30 per cent, 30-35 per cent, 35-40 per cent and >40 per cent. These are shown on Figure 3 to illustrate the tonnages involved. There is only one blend required, which has a maximum constraint on soluble copper of 25 per cent and a blending cost of $0.05/tonne to allow for additional grade control costs.

The results of the optimisation are shown in Figure 4. As can be seen, the copper soluble grade is kept below 25 per cent in all periods. This, however, causes a major problem for supply to the mill in the first period. Despite this, NPV is increased to $385 M, which represents over $100 M improvement premium over the base case. Indicating that, despite the additional cost of blending ($6 M) and the stripping problem in the first period, blending to control soluble copper is (in this case) warranted. An aspect of this schedule, which is also of interest, compared to the base case, is that an additional 26 M tonnes of material is now rejected because of high soluble copper grade. By comparison, a blend without blend bins yields a solution of $334 M and clearly shows the advantages of using user-defined blend bins to assist with the blending.

This result warrants further investigation, and the next example examines the feasibility of prestripping and stockpiling material to provide a better feed to the plant in the first year of operation. It also demonstrates the potential of using stockpiles to smooth mining production and provide plant feed during periods of low ROM supply.

TABLE 2

Revised recoveries and blend bin settings.

Item		Copper	Gold
Recoveries	OX	0.62	0.85
	MX	0.82	0.7
	PM	0.92	0.7
Blend bin	OX	0.50	0.40
Minimum	MX	0.38	0.68
Cut-offs	PM	0.32	0.54

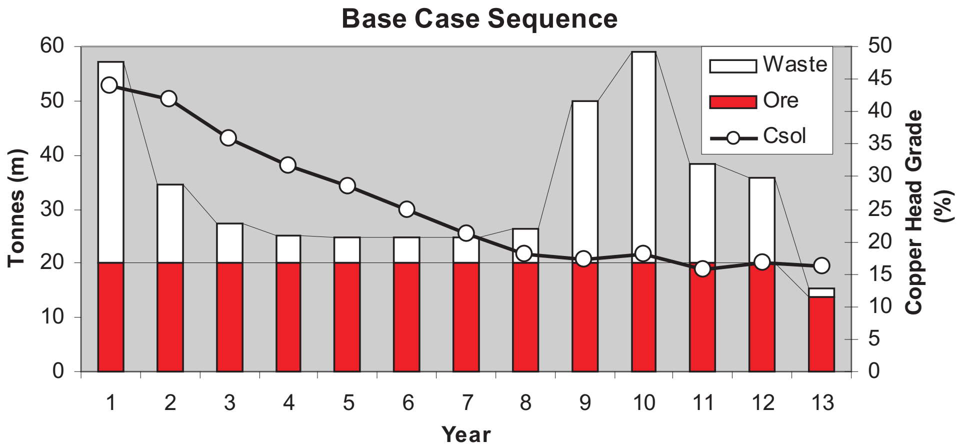

FIG 2 - Base case sequence.

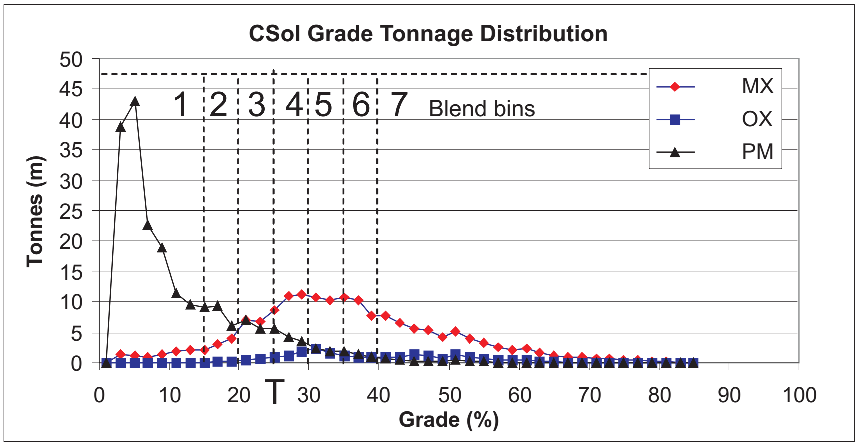

FIG 3 - CSol grade tonnage distribution.

Prestrip for one year and stockpile material

Gold and copper minimum cut-off grades need to be calculated for the stockpile by adding the stockpile rehandling costs ($0.20/tonne) to the processing costs (Table 3). Two stockpiles were set up for each rock type, with maximum soluble copper grades of 35 per cent and 40 per cent respectively. The mining rate was increased to 80 M tonnes to simulate a contract mining situation and then reduced to 35 M from year two onwards.

The prestrip life of mine sequence is shown in Figure 5. NPV, allowing for part delayed capital expenditure, is approximately the same as the previous case; however, the plant is 85 per cent full in the first year of operation. Furthermore, the reduced mining rate will keep the plant full throughout the rest of the mine life by feeding from stockpile during periods of low ROM supply. The stockpile reaches a maximum size of 20 M tonnes in year six and is predominantly used in periods seven, eight and nine. If stockpiling is not an issue then this might be attractive, especially with the better utilisation of the mining fleet.

TABLE 3

Stockpile settings.

Item		Copper	Gold
Stockpile	OX	0.53	0.42
Minimum	MX	0.40	0.71
Cut-offs	PM	0.34	0.57

EXAMPLE TWO – IMPROVED PITSHELL GENERATION

From the previous studies we have seen that some of the high CSol material is not required and is always hard to use in a blend. Could the pitshell generation be modified to try and avoid some of this material?

In this example, we will examine the use of two further tools: user-defined block value expressions and 'Pushback Chooser' a process whereby a large number of pushback alternatives are evaluated to find the best NPV.

Traditional pit optimisation does not allow a user to do this easily. The latest release of Whittle (V3.2) allows a user to derive their own block values based on user expressions and these can therefore contain value conditions. A simple heuristic approach would be to exclude all material with CSol >35 per cent from the Lerchs-Grossmann pitshell optimisation calculation. Some blocks may be included in the pit because other blocks pay for their inclusion, but we want to avoid any pit expansion that is based on these blocks alone. Table 4 shows the user-defined expressions used to derive block value calculations. This is simply a sequence of calculations that calculates the recovered copper and gold for each parcel in a block and then determines parcel revenue and selling and processing costs. If net revenue is positive then the parcel processing values are used, otherwise the parcel is treated as waste and only has mining costs associated with it.

Running another pit optimisation based on these user-defined block values does indeed reduce the pit tonnage. A comparison of the pit volumes is shown in Figure 6. A and B mark the same

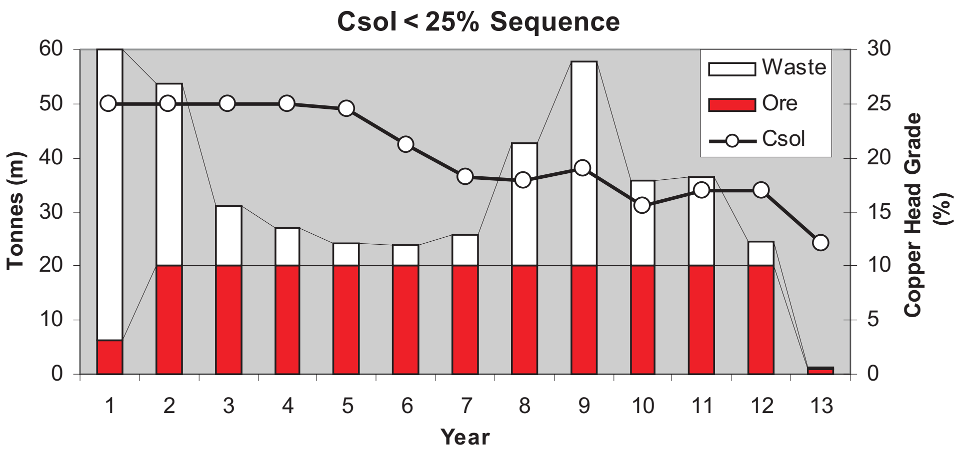

FIG 4 - CSol <25 per cent sequence.

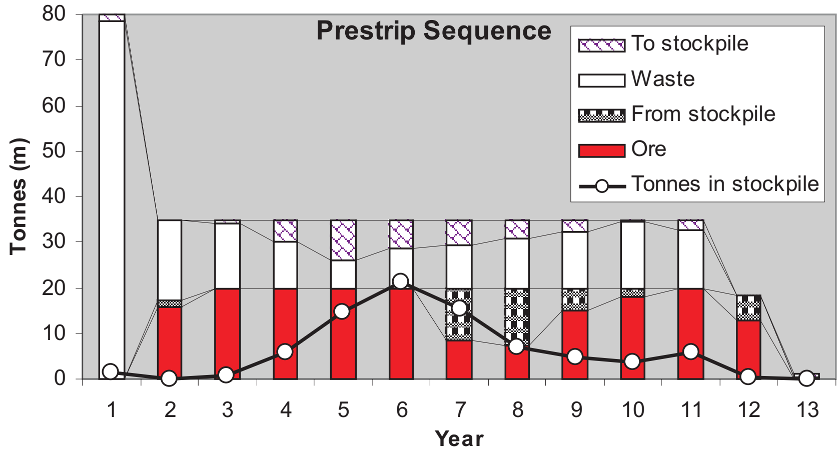

FIG 5 - Prestrip sequence.

TABLE 4

User-defined expressions for block calculation.

Code	Description	Expression
CUOR	OX copper recovery	if(CSol.g>35,0,OX.Q * MAX(0, CuS.g - 0.214516) * .62)
CUMR	MX copper recovery	if(CSol.g>35,0,MX.Q * MAX(0, CuS.g - 0.07378) * .82)
CUPR	PM copper recovery	if(CSol.g>35,0,PM.Q * MAX(0, CuS.g - 0.034043) * .94)
AUR	Gold recovery	if(CSol.g>35,0,OX.Q*au.g*.85 +mx.q*au.g*.5+PM.Q*AU.g * .60)
REVI	Element revenue	(CUOR + CUMR + CUPR)*20 * REVFAC + AUR*12 * REVFAC
SELL	Selling costs	(CUOR+CUMR+CUPR)* 7.20 + AUR * 0.20
PROC	Processing cost	(OX.Q*4+MX.Q*4.00+PM.Q*3.85)*BLOCKP
ROCA	Rock adjustment	-OX.Q*0.1-MX.Q*0.05
REVN	Parcel revenue	REVI -SELL -PROC
CSTM	Cost of mining	0.9
BLKV	Block revenue	SUMPARCEL(IF(REVN>0, REVN,0)) - (BLOCKT+SUMPARCEL(ROCA))*CSTM*BLOCKM

where:

.G	defines a grade
.Q	defines a quantity
BLOCKM, BLOCKP	are block mining and processing costs adjustment factors
BLOCKT	is block tonnage
REVFAC	is a revenue factor used to generate pitshells
SUMPARCEL	sums over all parcels in the block

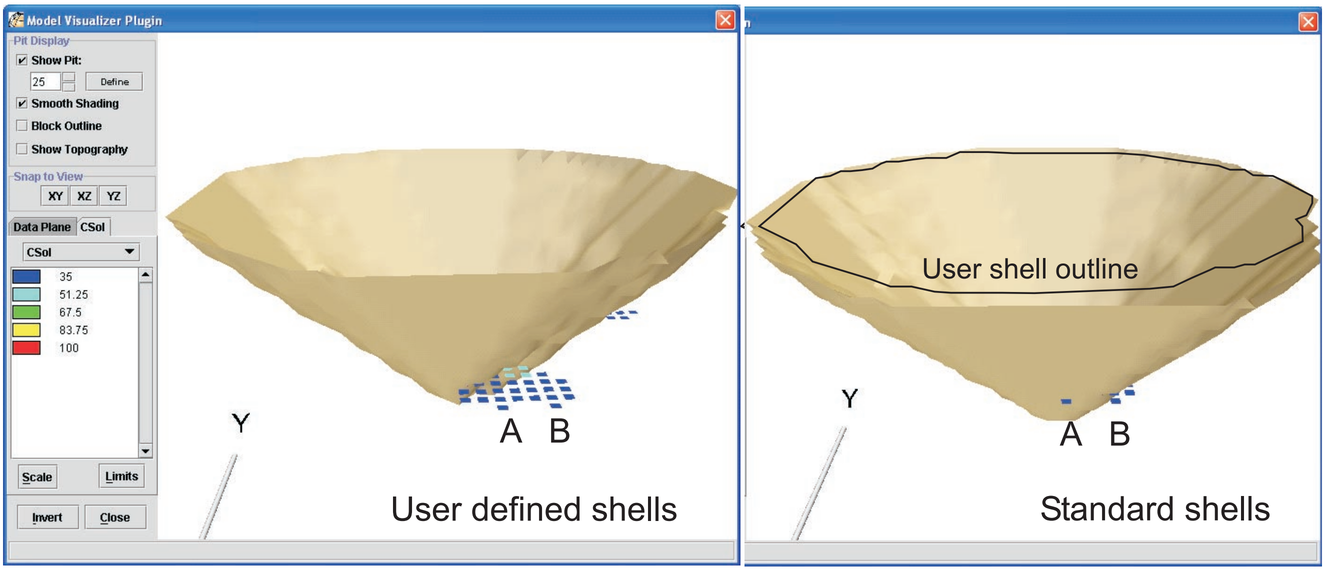

FIG 6 - Comparison of pitshells.

blocks in both the user-defined shells and the normal shells. The highlighted blocks have CSol grades above 35 per cent. As can be seen, some high CSol MX blocks are excluded from the bottom of the pit in the vicinity of A and this leads to a general reduction in pit size as shown by the overlay of the user-defined pitshell in the normal pit view. This is easier to see with a size versus value graph and Figure 7 shows NPV versus pit size for both normal pitshells and pitshells derived from the user-defined block values. What this shows is potentially a $30 M increase in NPV for the same pit size (A) or an 80 M tonne reduced pit size for the same NPV (B).

The above comparisons were done with the use of the benchmark scheduling method called 'Best Case', whereby each pushback is mined out sequentially. The basis for this is a set of nested pitshells, generated by a pit parameterisation technique. This scheduling approach leads to schedules with very high NPVs, but the very large number of pushbacks involved generally makes the schedule un-minable. The details of the approach and the importance of long-term scheduling on mine NPV, are described in various papers including Wharton (2000).

In order to generate a feasible schedule, it is necessary to further constrain the problem by limiting the number of pushbacks that can be mined during the life of the mine. Generally speaking, fewer pushbacks will lead to greater ease of mining, and lower NPV. It is an engineering/economic decision as to the level at which this compromise should be struck. Various heuristic methods for choosing pushbacks have been developed over the years, but none have been found which reliably lead to good pushback choices for a range of different orebody characteristics. The only certain way to determine the best selection of *n* pitshells, is to try every possible combination. The Pushback Chooser module in Whittle does exactly that – it tries every possible combination of *n* pushback choices and returns the selection that maximises NPV, subject to all other constraints in the simulation.

Figure 8 shows a life of mine schedule based on a Pushback Chooser sequence using the revised pitshells for the smaller pit option. While the mine life has been reduced, the NPV is marginally greater ($394 M). Further analysis of this pit might lead to a reduced fleet size and reduced capital expenditure.

Figure 9 shows a life of mine schedule based on keeping the pit size the same and choosing a new set of pushbacks for the larger mine. The mine life increases to 12.3 years and the NPV is $450 M. This really shows the power of using the revised pitshells, and also demonstrates the value of iterative analysis.

Table 6 summarises the results from all of these investigations and as can be seen, there has been a big impact on NPV and IRR through the use of blending and user-defined pitshell generation. There is more reject material but the quality going through the mill leads to improved profitability.

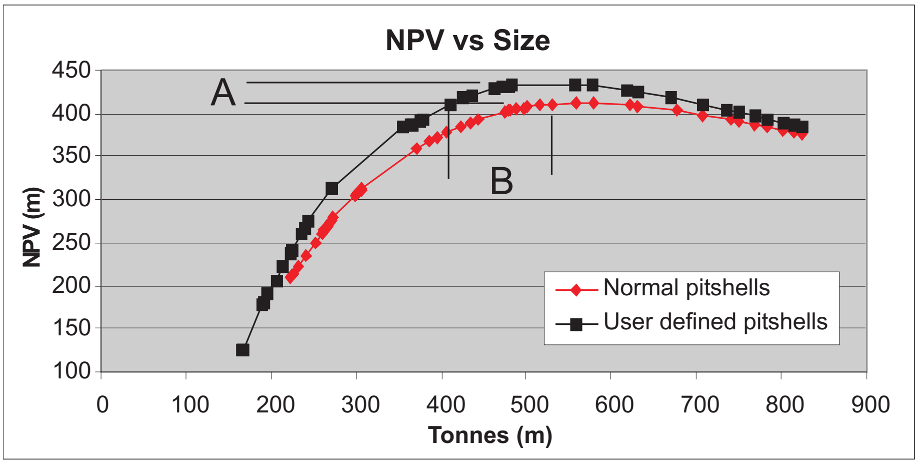

FIG 7 - NPV versus size.

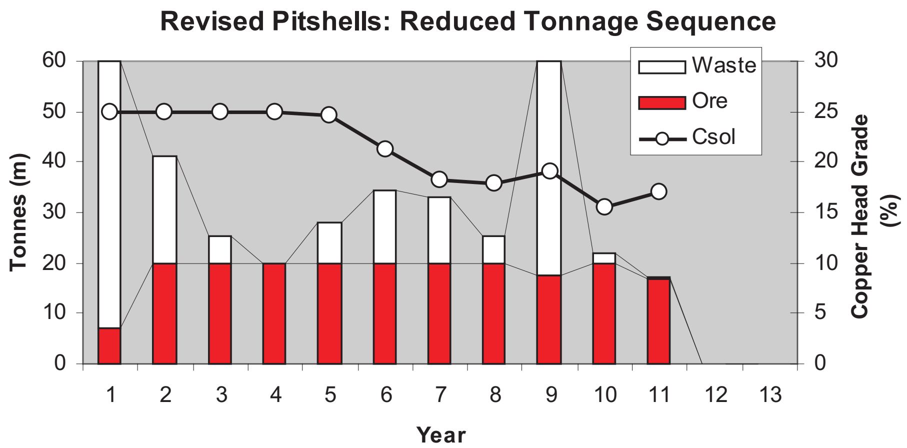

FIG 8 - Revised pitshells: reduced tonnage sequence.

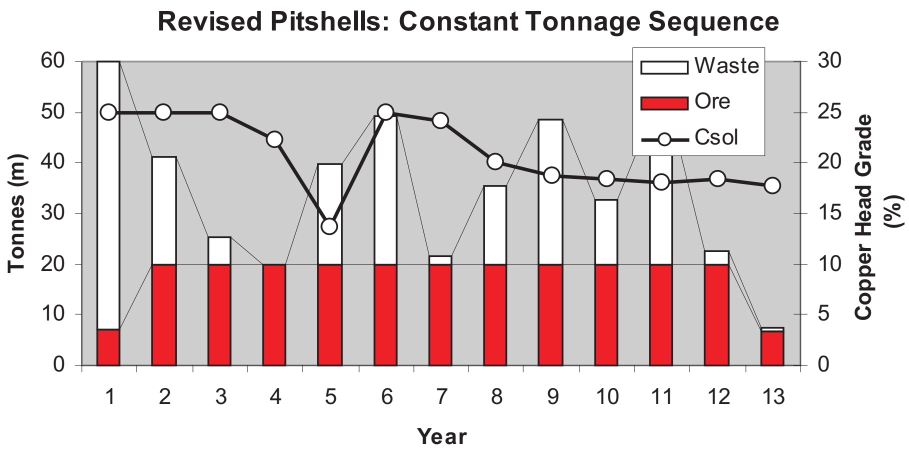

FIG 9 - Revised pitshells: constant tonnage sequence.

EXAMPLE THREE – PROCESS BALANCING

This example is included to show how a blending LP optimisation can be used to solve other problems on a mine site. For example, in many cases, companies have access to two or more process plants, which often have different operating costs and recoveries.

If you just set up the two processes, each with their own capacity, costs and recoveries, you will find that they are not both used to full capacity, since material will be delivered preferentially to the lowest cost process. It is common when simulating these cases, to simplify the problem by averaging the costs and recoveries, on a weighted basis, to achieve a single plant model. However, when using this approach you will, in fact, be assigning some material that could be processed through the better plant, to waste, and putting some marginal material through the poorer process that should be assigned to waste.

A way around this dilemma is to set the problem up as an extractive blend. The key issue is how to control what material is sent to each of the processes as there are no overriding grade constraints to apply. This can be achieved by setting up the blend bins so that they only contain viable material. To illustrate the concept you can take the base case example and split the plant capacity up into 15 M through the new plant and 5 M through the old plant with the old plant having, say, five per cent increased costs. This will mean that the old plant cut-offs will be five per cent higher than the new plant.

The first step is to calculate the required cut-offs for each element, for each rock type, in the best process and apply these to the first blend bin(s). This exercise needs to be repeated for each process/rock-type combination. Table 5 shows the cut-offs required for this example. The next step is to create a blend for each process, but do not assign any other constraints or costs. The mining rate has been reduced to 50 M tonnes per annum to illustrate what happens when the processes are starved of material.

TABLE 5

Blend bin settings for old and new process.

Item		New mill		Old mill	
		CuS	Au	CuS	Au
Blend bin	OX	0.72	0.40	0.76	0.42
Minimum	MX	0.46	0.68	0.48	0.71
Cut-offs	PM	0.36	0.54	0.38	0.57

Figure 10 shows a typical schedule. Note that in period ten the mining rate is not sufficient to fill both mills. In this case the optimisation has reduced the input into the old mill, as it has higher costs and hence lower revenue per tonne.

CONCLUSION

The preceding examples are not intended to be final designs; they have been used to illustrate the various techniques that are available to current mining engineers. There are many practical situations on existing mine sites where improving the quality of the input into a process, through the use of extractive blending, can improve the bottom line for companies. This improvement can be based on better throughputs, reduced costs and/or increased recoveries.

The use of user-defined blend bins provides superior solutions and means that users can take existing models without having to re-categorise the data to provide required selectivity for blending.

The examples have also attempted to show how user-defined block values can be used to produce more robust pitshells in cases where you wish to avoid deleterious material.

The techniques described in this paper can all be executed using the Whittle blending software and can lead to a better understanding of the mine dynamics and may lead to new ways of treating old problems.

TABLE 6

Life of mine comparisons.

Case description	Total tonnes (M)	Reject tonnes (M)	NPV ($M)	IRR (%)	Mine life
Base case	443.7	38.1	273	22.7	12.7
Constrain CSol < 25%	443.7	64.7	385	27.2	12.0
Prestrip and stockpile	443.7	76.5	385	30.6	11.9
Revised pitshells (A)	451.7	68.1	450	31.2	12.3
(B)	366.0	54.0	394	30.4	10.8

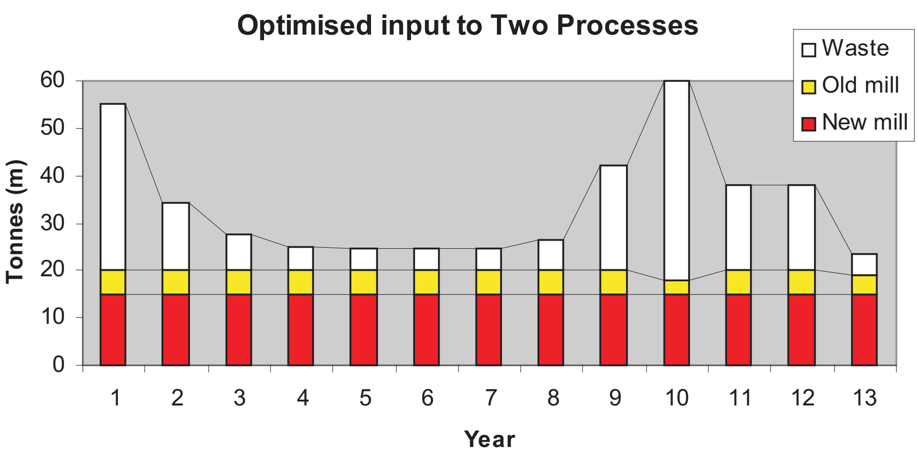

FIG 10 - Optimised input to two processes.

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