A Back Analysis of Dilution and Recovery in Longitudinal Sublevel Caving

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Abstract

The Big Bell Gold Mine utilised longitudinal sublevel caving for a significant period of its mine life. Draw and blasting databases were compiled for daily reporting, and analysis of mining performance and for future studies of the draw mechanisms during the sublevel caving period.

The writers' objective was to gain a better understanding of the factors that influence dilution and ore recovery in longitudinal sublevel caving, mainly as the knowledge base for this particular mining method is limited when compared to other bulk mining methods. The analyses involved 3,358,301 fired tonnes and 3,520,322 drawn tonnes spread over 106 "stopes" and 5 years of mine operation. A study by Perera (2005) determined blasting parameters and draw geometry were the most significant influences on the longitudinal sublevel cave draw. Blasting parameters include tonnes per drill metre, powder factor, burden to spacing ratio and number of rings fired. Draw geometry included number of drawing points and location, drawpoint width, footwall angle, orebody width and draw height. The average and variance for each parameter was discussed in the study. Hangingwall rock mass strength variation was empirically determined to be a significant influence.

According to the principles of gravity flow by Kvapil (1992), the Big Bell cave can be evaluated as having achieved 'good' draw. Several theories of sublevel cave draw were examined; with back analysis showing good correlation. It is clear from the review of theoretical literature and the analysis of the Big Bell draw data that sublevel cave draw is a balance of several different competing variables.

The data presented may be used to benchmark new or existing operations which exercise a similar technique or mine geometry.

1 Introduction

Sub Level Caving (SLC) is a multivariable optimisation problem. The primary objective in this study is to quantify dilution and recovery in longitudinal sublevel caving and identify the variability that occurs in the draw. Draw and grade data from the Big Bell Gold mine was manipulated to analyse recovery, dilution and grade on a stope by stope basis. Several blasting parameters and mine geometries were examined for their effect on recovery. Several graphs will be presented, in which North is on the left and South on the right hand side. Levels increase in depth from top to bottom, they correspond to the depth below surface. The analysis comprised of the following quantities:

- Total Mass Fired = 3,358,301t

- Total Mass Drawn = 3,520,322t

- Ore Drawn = 2,738,428t

- Waste Drawn = 748,379t

- Mineralised Stope Fill (MSTP) Drawn = 33,515t

2   Big Bell Gold Mine

The Big Bell mine was located approximately 550 km north- north east of Perth, in the Wiluna Greenstone Belt of Western Australia. Mining of the area by prospectors commenced in 1904; since then Big Bell mine experienced changes in mining methods from alluvial to underground, to open pit and back to underground.

The datasets used in the study were compiled during the Longitudinal SLC phase of operations. The draw and blasting statistics are contained in two MS Access databases. Prior to interpretation, the raw data was formatted and filtered so as to obtain a comparable set of stopes. A stope is defined as a block of ore at a mining horizon of a particular length. Stopes with the same number occupy the same northings but are on different levels.

The study was restricted to the following two periods:

- September 1998 to August 2000,

- January 2002 to July 2003.

A history of the operational and mining method changes implemented at Big Bell due to the unusually high stress environment is referred to in the bibliography; these changes define the two selected time periods. The mine loaded ore from each firing to a cut-off grade rather than percentage of tonnes fired during both the SLC periods under review.

Stopes excluded from the study, but are within the time periods, served as slots for the beginning of a level, had short strike lengths or was a massively bridged stope. Figure 1 is a schematic of the stopes used in the analysis. North is on the left of stope 70 and South on the right hand side of stope 70. There are 106 'stopes' in total.
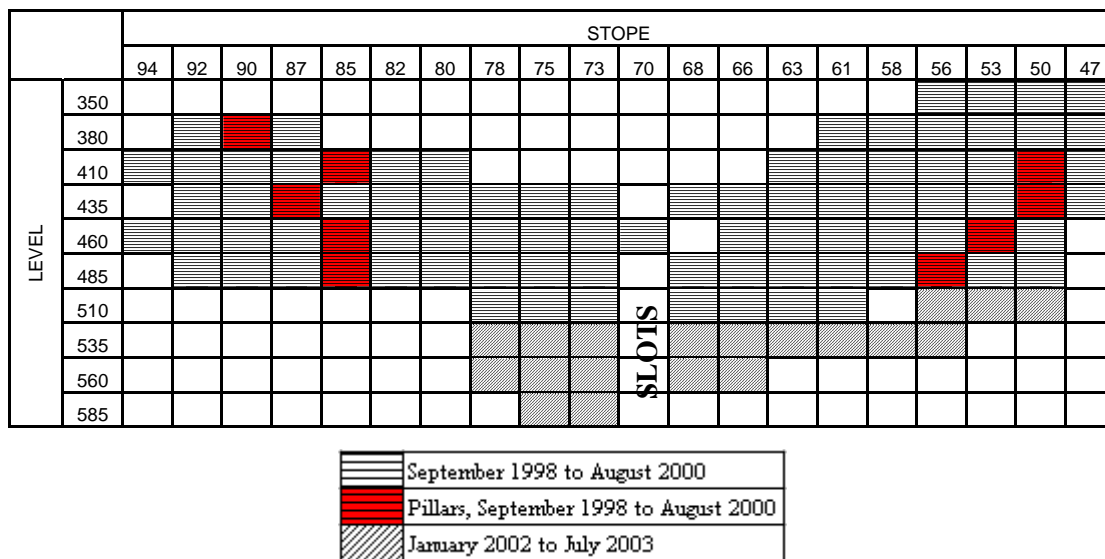
	STOPE	94	92	90	87	85	82	80	78	75	73	70	68	66	63	61	58	56	53	50	47
LEVEL	350																				
	380																				
	410																				
	435																				
	460																				
	485																				
	510																				
	535																				
	560																				
	585																				

▤	September 1998 to August 2000
▅	Pillars, September 1998 to August 2000
▨	January 2002 to July 2003

Figure 1   Stope Selection

3   Gravity Flow and Dilution

There have been several projects undertaken to understand the flow of material in a mine environment. Kvapil 1965a and Kvapil 1965b are the foundation upon which modern day SLC theories are based. These studies involved the gravity flow of granular materials in hoppers and bins, and have come to be known as the 'classical' theories of SLC. Susaeta (2004, p. 167) highlights the following ideas on gravity flow as important considerations to optimise recovery in sub level caving.

- An isolated draw column is generated when a drawpoint is extracted independently.

- Even draw of neighbouring drawpoints minimises dilution.

- Isolated draw diameter is independent of the drawpoint width.

- Closer drawpoints lead to better ore recovery.

- Interaction between two drawpoints occurs if the distance between them is less than 1.5 times the isolated draw diameter.

- Higher draw columns decrease total dilution.

- The diameter of a draw column is a function of the fragmentation of the material.

- Fineness of fragmentation is directly related to ore extraction.

When material inside a cave starts to flow, the movement area takes on an ellipsoidal, described by its eccentricity. Kvapil (1992, p.1795) states that eccentricity is a function of particle; size, shape, form, surface roughness, angle of friction, density, moisture content and the extraction or discharge rate of the material. These factors dictate certain behavioural patterns which influence particle mobility. Greater mobility directly translates to a smoother gravity flow, and a high eccentricity. Section 4 evaluates Big Bell recovery data against Kvapil's dilution curves (Kvapil, 1992) and Section 6 tests the last two items of this theory using powder factor as a proxy for fragmentation. It was not possible to test remaining criteria from the available data.

4   Recovery of Tonnes Fired

Recovery can be calculated by tracking actual draw from a stope to that planned in the stope design, i.e. tonnes drawn divided by tonnes fired. By changing the type of material in the numerator it is possible to calculate total, ore, waste or MSTP recovery. Graph 1 shows the average recovery of tonnes fired from each stope.
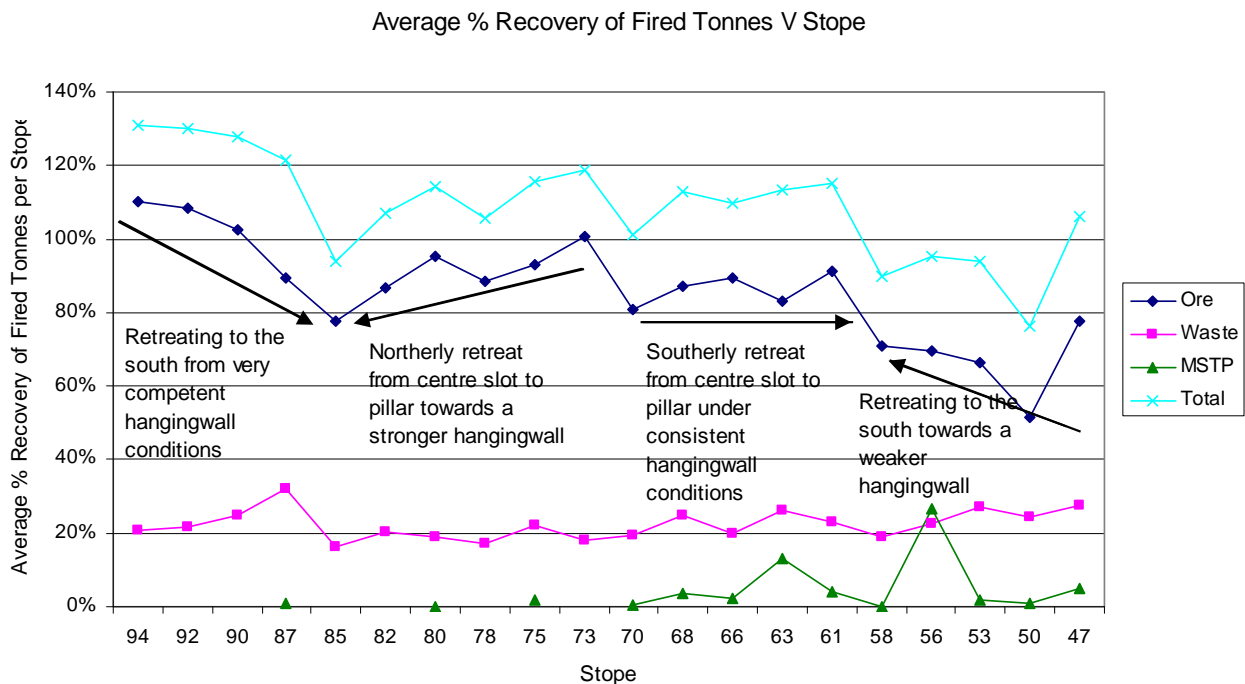
**Graph 1   Average Percent Recovery**

Graph 1 shows that very high total recoveries were achieved at the northern end of the mine. There was a significant decrease in ore recovery from north to south accompanied by fairly consistent waste levels. Player (2004) explains that high seismicity experienced in the northern end of the mine was in part influenced by competent hangingwall conditions increasing the loading stiffness on the country rock. The

idea of strong hangingwall conditions was supported with observation of block sizing in the surface expression of the cave (significantly larger at the northern end), the location of cable bolting within the pit (dominantly at the southern end) and the size of the waste that reported to the draw points underground (finer towards the southern end of the mine). The theory of differential fragmentation states that finer ore particles flow in preference to the coarser waste (also consistent with Bull and Page, 2000).

Stope 70 was generally used to commence each new level. This appears as a dip in the total recovery and ore recovery in Graph 1. A similar dip in recovery occurs at stopes 85 and 50; which represent repeated locations for pillar positions. Stopes that have a pillar located in once across the multiple levels do not appear to affect the overall recovery graph.

4.1 Ore Recovery

Graph 2 is a histogram for the ore recovered from the fired tonnes. The population is skewed to the right with a mean of 84% and standard deviation (σ) of 24%. A recovery of more than 100% of ore tonnes fired implies that ore remaining from previous shots was recovered, dilution material consisted of unfired ore, or a sampling error may have occurred.
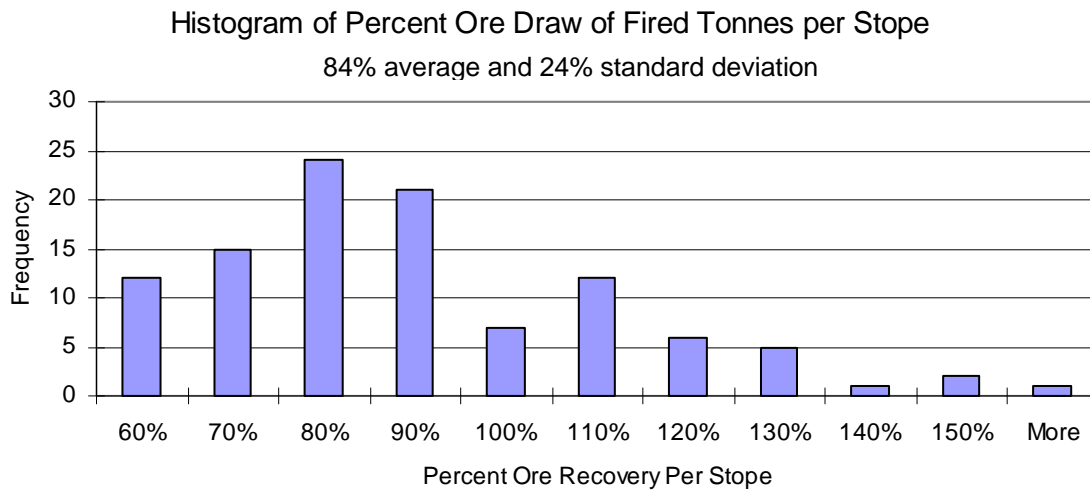
**Graph 2   Overall Ore Recovery of Fired Tonnes**

Summary of performance across the mine:

- Total tonnes drawn of fired tonnes, 108% and σ 28%.
- Ore tonnes drawn of fired tonnes, 84% and σ 24%.
- Waste tonnes drawn of fired tonnes, 23% and σ 9%.

MSTP recovery accounts for less than 1% of tonnes fired. Hence 84% +23% +1% = 108%. [N.B. values are rounded].

To investigate performance due to hangingwall conditions the recovery histogram in Graph 2 was split into Northern (stope 94 to 78), Central (75 to 66) and Southern (stope 63 to 47) sections. The recovery profiles are generally normal distributions with a progressive increasing skewness to lower recovery from the north to south.

The lower recovery at the south end of the mine is evident in Graph 3 and is inferred that the weaker hangingwall rockmass had a significant effect on the draw by allowing early waste entry by fragmenting into smaller blocks and starting to cave earlier.

Histogram of Ore Recovery Along the Mine
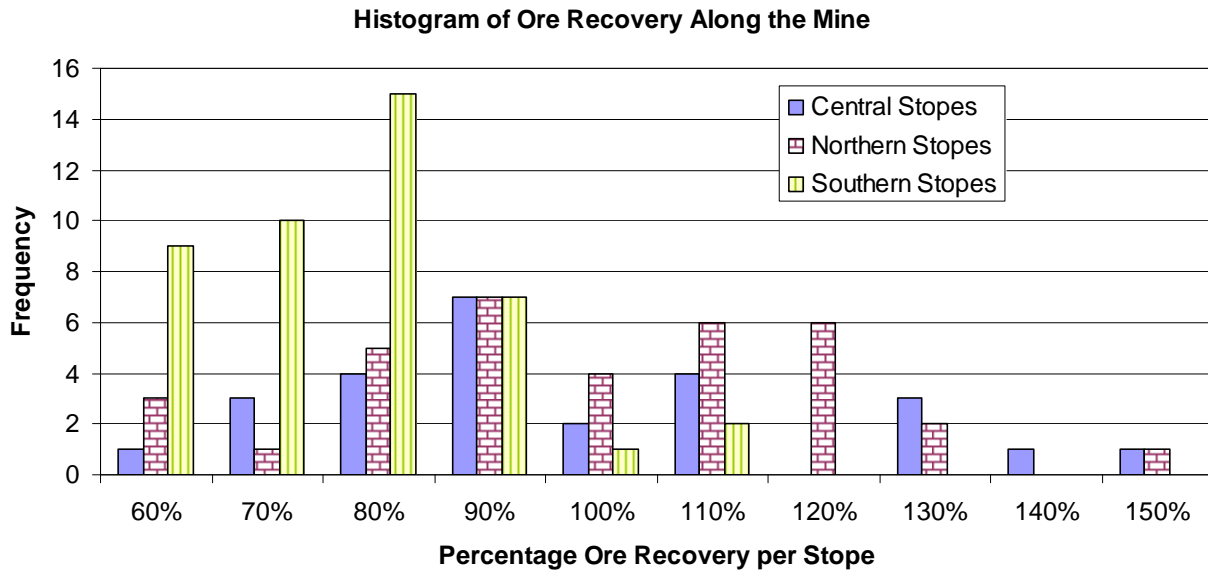
Graph 3   Ore Recovery by Area of the Mine

4.2   Recovery Summary Statistics

Table 1 presents the recovery statistics for the data set, simple arithmetic averages and the standard deviation (σ). The table shows that the Northern and Central sections of the mine had the same overall performance, and across the mine each recovery category had a similar standard deviation. The variance ratio test (the F test) showed that the variances of the data for the total, ore and waste recovery were equal at the 5% confidence level. This allowed the "t-test" with equal variances to be used to assess the mean recovery. The t-test showed that average total and ore recoveries are the same for the north and central blocks and different to the south block. Waste recovery is consistent across all blocks.

Table 1   Recovery Statistics

	North Stopes (94-78)			Central Stopes (75-66)			South Stopes (63-47)		
	Total	Ore	Waste	Total	Ore	Waste	Total	Ore	Waste
Average	116%	94%	22%	114%	93%	21%	98%	72%	24%
σ	26%	21%	9%	28%	23%	8%	28%	20%	10%

These recovery values are the product of many mining related parameters. The scatter represents the variability in draw, which is an inherent risk of longitudinal SLC. Understanding the factors which control these variations could contribute significantly to the understanding of the draw mechanism.
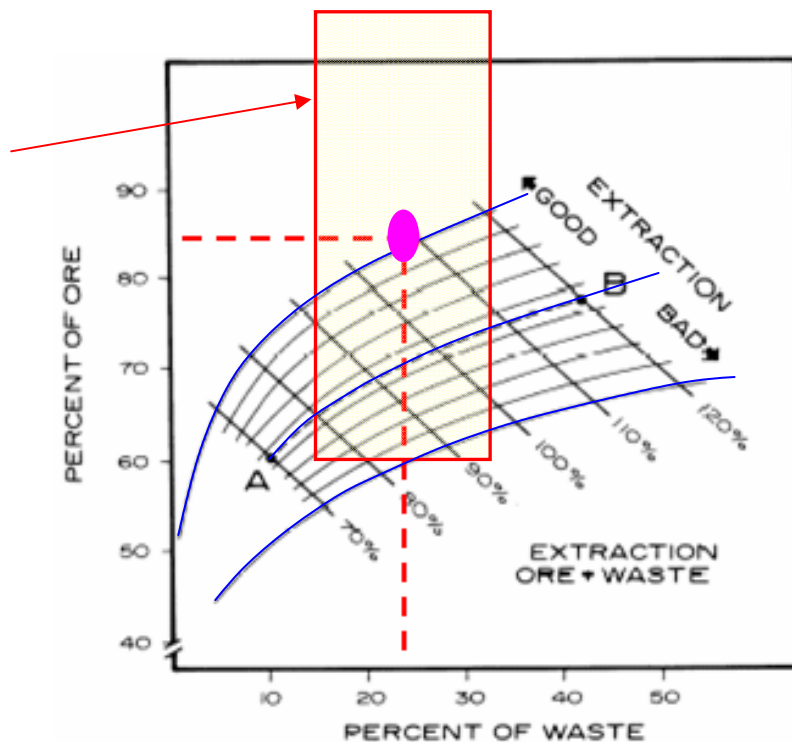
4.3   Big Bell and Kvapil

The Big Bell recovery statistics compare well to dilution curves found in Kvapil (1992, pp.1806-1807). Kvapil assumed extraction to stop at 110%, Big Bell averaged 108% recovery of fired tonnes. Given this, it is possible to make the comparison shown in Table 2. If Kvapil's dilution values are taken as an initial benchmark then it can be concluded that overall longitudinal SLC at Big Bell had good recovery. Graph 4 shows the average draw by the marker and one standard deviation of the data set has been defined by the box.

Table 2   Big Bell and Kvapil

	Big Bell	Kvapil (1992)		
		Good	Intermediate	Bad
Total %	108	110	110	110
Ore %	84	83	75	65
Waste %	23	27	35	45

One population standard deviation about the average ore recovery and waste.

The size of point represents one standard error of the mean values of ore recovery and waste.

Graph 4   Kvapil Dilution Development Curve

Big Bell's averaged stope by stope recovery data is overlain on Kvapil performance curves in Graph 5. The data has been subdivided into northern, central and southern stopes and Kvapil's good, intermediate and poor curves. The graph provides a sense of the variability in draw across the mine.

Big Bell's Dilution Development Plot
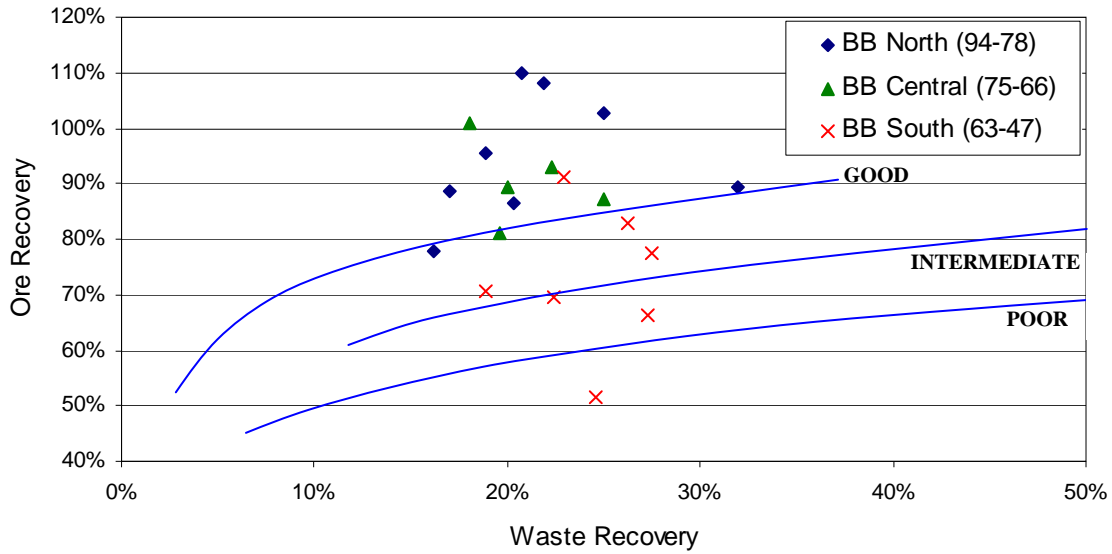
**Graph 5   Big Bell Dilution Development**

5   Grade

Two different types of grade values were calculated in the data analysis, these being recovered and in situ grade, depending on whether the denominator used was total tonnes drawn or ore tonnes drawn respectively.

- Recovered Grade - This gives the mine head grade or the grams of metal recovered per tonne of rock (ore, waste and MSTP). This is the typical grade quoted in any mining analyses.

- Calculated In Situ Grade - Yields the grams of metal recovered per tonne of ore drawn. It is important to note that the grams of metal used in the numerator have come from ore, waste and MSTP. In reality dilution was not without grade but below a cutoff grade as the orezone had a disseminated boundary. In the process of dividing the value of grams by ore tonnes it is assumed that dilution runs at zero g/t, i.e. all the grams come from the fired ore.

5.1   Grade Dilution

The difference between 'recovered grade' and 'calculated in-situ grade' is 'grade dilution'. Graph 6 depicts the arithmetic average 'grade dilution' per stope. The 'recovered grade' is lower in the north, peaks around Stope 78 and declines towards the south. The 'calculated in-situ ore grade' starts similarly but plateaus towards the south. The 'grade dilution' per stope is represented by the vertical bars on Graph 6 and increased towards the south. Hangingwall differences between the north and south sectors of the mine are considered to have contributed to this trend. Longitudinal SLC resulted in a 'grade dilution' of between 0.8g/t and 1g/t assuming the dilution material had no grade.

Average Head Grade and Average Calculated In Situ Grade per Stope
V
Average Grade Dilution (g/t)
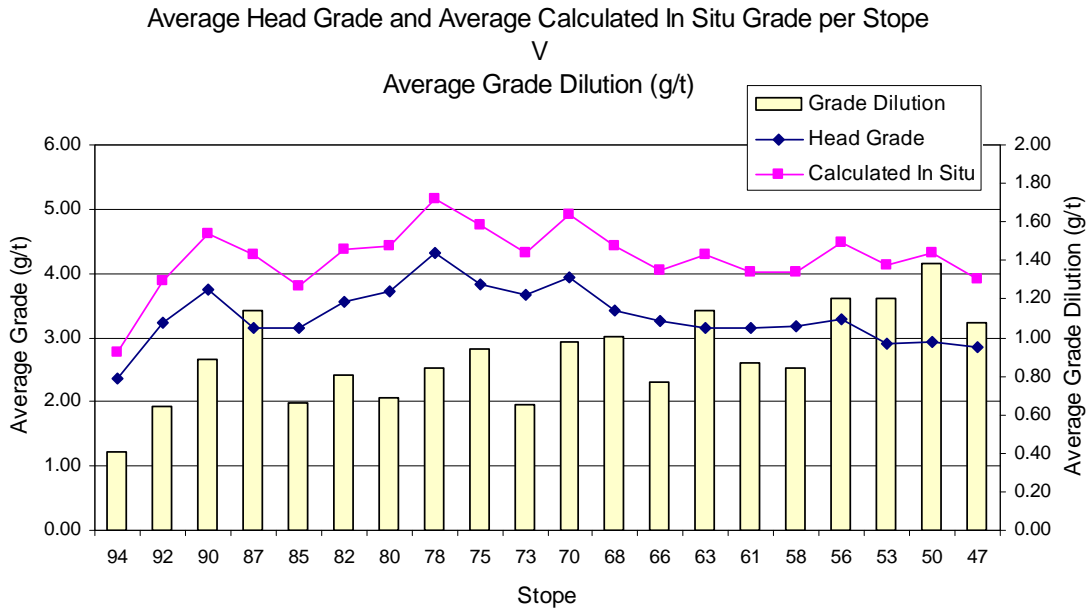
Graph 6   Grade Dilution

Graph 7 shows average grade per stope and average recovery per stope on the primary and secondary axes. Increase in 'grade dilution' occurs with higher of waste recovery and reduced ore recovery.

The stronger hangingwall in the north may have resulted in less waste entry which subsequently allowed the cave to run at a lower grade prior to cut off; hence the greater ore recovery values and lower grade dilution.

Conversely the higher grade dilution experienced towards the south may be explained by the weaker hangingwall conditions which resulted in greater inclusion of waste within the draw.


Average Head Grade and Average Calculated In Situ Grade per Stope
V
Average % Recovery of Fired Tonnes
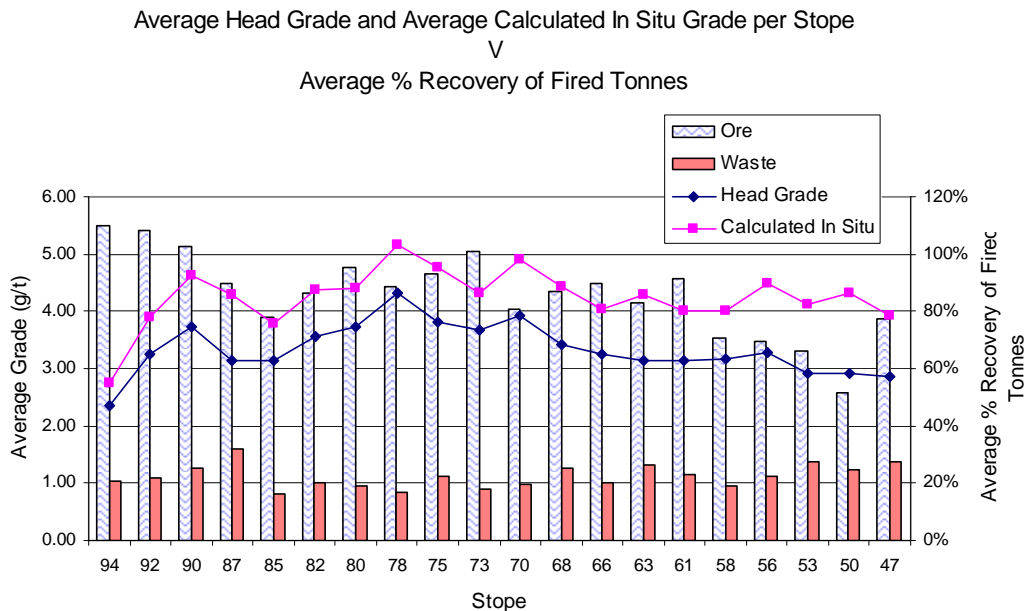
Graph 7   Grade Dilution and Recovery

5.2   Grade Dilution Adjustment

'Calculated in-situ ore grade' was initially based on the premise that dilution was drawn at zero grams per tonne. The assumption that no ounces came from the waste material, provides a slightly elevated figure for the 'calculated in situ grade', and also the 'grade dilution'.

To improve the representation of the 'calculated in-situ grade' of the actual ore tonnes drawn it was necessary to assume a grade for the dilution. Intuitively and as seen in Graph 8, as the grade of the dilution material increases, 'calculated in-situ grade' (shown by the lines on Graph 8) will approach the recovered grade. However, even at waste grades of 1 g/t there is still a significant cap between in-situ and recovered grades.

Average Recovered Grade and Average Calculated In Situ Ore Grade with a Waste Grade per Stope
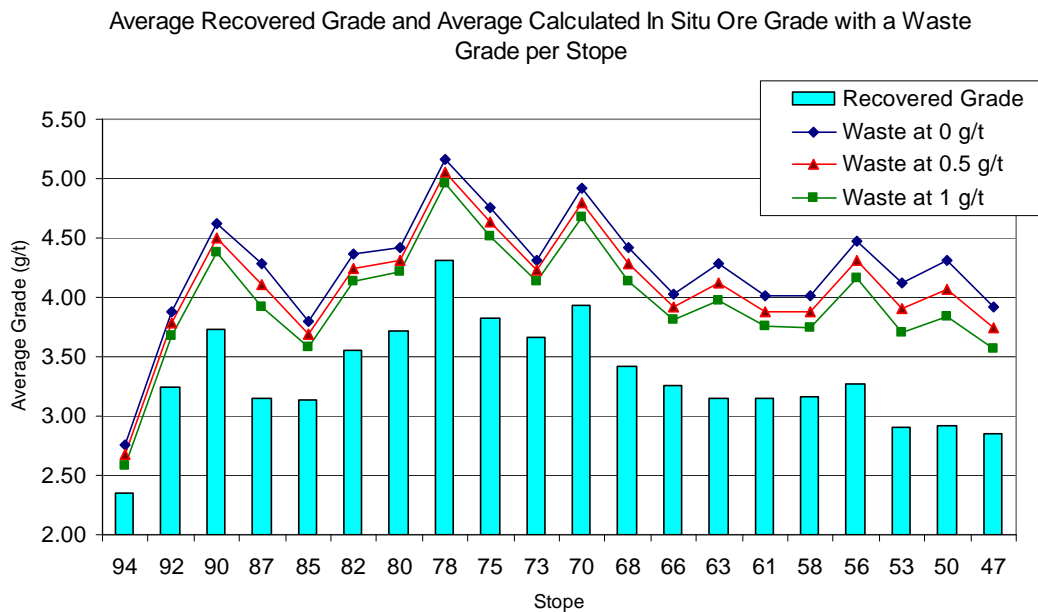
Graph 8   Recovered and Calculated Grade

Big Bell typically assigned a dilution value of 0.24 g/t. The summary statistics, calculated using this dilution grade, show the arithmetic mean head grade reduction is 0.84 g/t.

6   Blasting Parameters

The shot by shot blasting database used by the mine made it possible to track the performance of the cave with changes in blasting parameters on a stope by stope basis. Several blasting parameters were studied for their effect on recovery, with two strong relationships identified. These were the powder factor and tonnes per drill metre (TPDM). It is important to note that recovery is simultaneously influenced by many different variables.
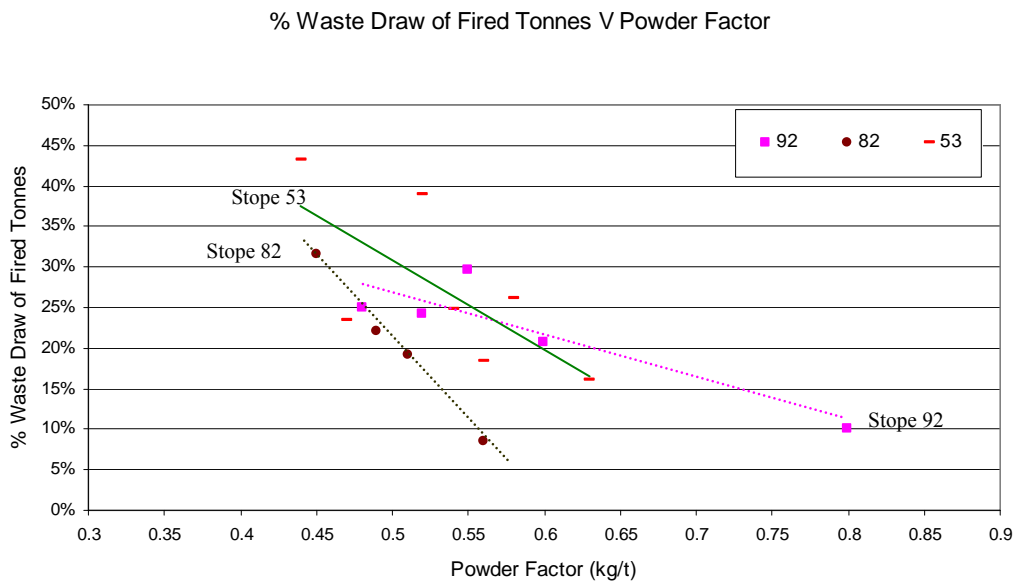
6.1   Powder Factor

Powder factor can be used as a proxy for fragmentation. Graph 9 plots this with respect to ore recovery of fired tonnes versus powder factor. Using the stope by stope data set, positive relationships are generally observed - with higher powder factors resulting in better fragmentation which leads to greater ore recovery. The typical Big Bell powder factors ranged from 0.4 to 0.65 kg/t.
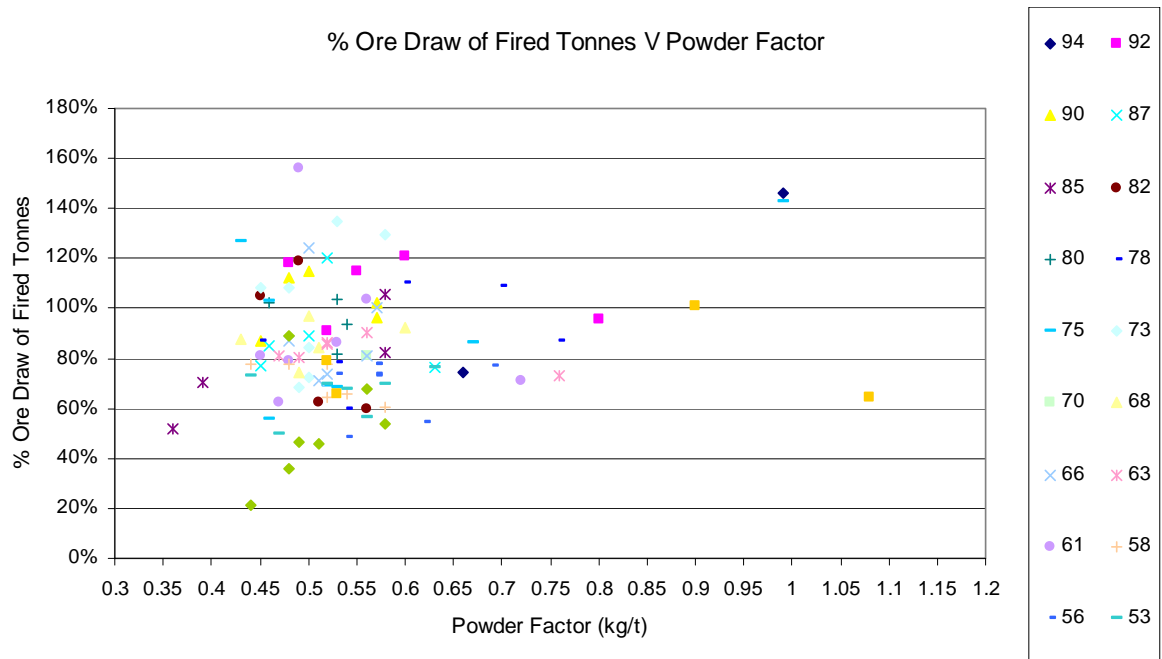
Graph 9   Ore Draw and Powder Factor

The higher blast energies associated with high powder factor shots have the ability to compact the waste / caved material which should assist in cleaner ore recovery. A similar graph showing percentage dilution had significant scatter, as would be expected from the high variance reported. But selected stopes do show the expected relationship that waste drawn decreased with increasing powder factor, as illustrated by Graph 10.

Graph 10   Waste Draw and Powder Factor

7   Geometry Factors

A number of geometry factors were examined as part of the project; the strongest relationship identified was an increase in recovery from using one rather than two parallel ore drives across a wide range of orebody cross sectional areas.

7.1 Number of Drives and Average Cross Sectional Area

The effect of one and two ore drives, and stope size, on recovery were examined by plotting average cross sectional area on the horizontal axis and percent recovery on the vertical axis. Recovery is expressed as a percentage of the designed tonnes fired. The mine generally implemented two ore drives when the orebody exceeded 22m in width. This was in line with Kvapil (1992, p.1811), which states that two ore drives should be used in order to increase recovery in areas where orebody thickness is greater than 20m.

The dataset of stope recovery was split into discrete groups of one and two ore drives. The following calculation yields a simplified average cross sectional area per stope.

$$Average\ Cross\ Sectional\ Area\ per\ Stope = \frac{Total\ Stope\ Tonnes}{Density \times Strike\ Length}$$

The calculation was necessary as the fired cross sectional area was not documented for each blast, neither was the average stope height or width. For the purpose of the study the review was undertaken on a stope by stope basis rather than individual blasts. The calculation assumed an ore density of $2.7t/m^3$ and strike lengths of :

- 30m (Stope 47 – Stope 63)

- 24m (Stope 66 – Stope 99)

It is inferred from Graph 11 and Graph 12 that given the same stope cross sectional area, a two ore drive strategy yields an improved recovery despite the overlap between the datasets. Total recovery increases by 25% and ore recovery by 20%. This trend seems to apply to all available cross sectional areas. The negative gradients of the linear regression curves from Graph 11 and Graph 12 indicate that recovery and cross sectional area are inversely related. That is, smaller stope dimensions yield better ore recovery. The issues associated with the number of ore drives and the fired cross sectional areas are,

- Reduction in recovery by using only one drive,

- Extra development cost associated with implementing two drives,

- Whether the cave advance rate is reduced to an unacceptable level by the requirement for additional development drive.

- An additional ore drive increases the time for the development cycle.

- Two ore drives reduce the maximum blast hole length for the same orebody width. This effectively allows an increase in the level interval, and control of drill hole deviation.

- Reduction in drill hole deviation which reduces the probability of hang-ups/ bridging and should allow more uniform blast hole patterns.

- One ore drive reduces mining flexibility, particularly if brow loss occurs.

The cross sectional area is a dependent variable because it was calculated from the tonnes fired per stope, and more appropriate calculations require the physical geometry of each fired stope to assess the parameters listed above and in section 3.

Percent Total Recovery and Average Stope Cross Sectional Area
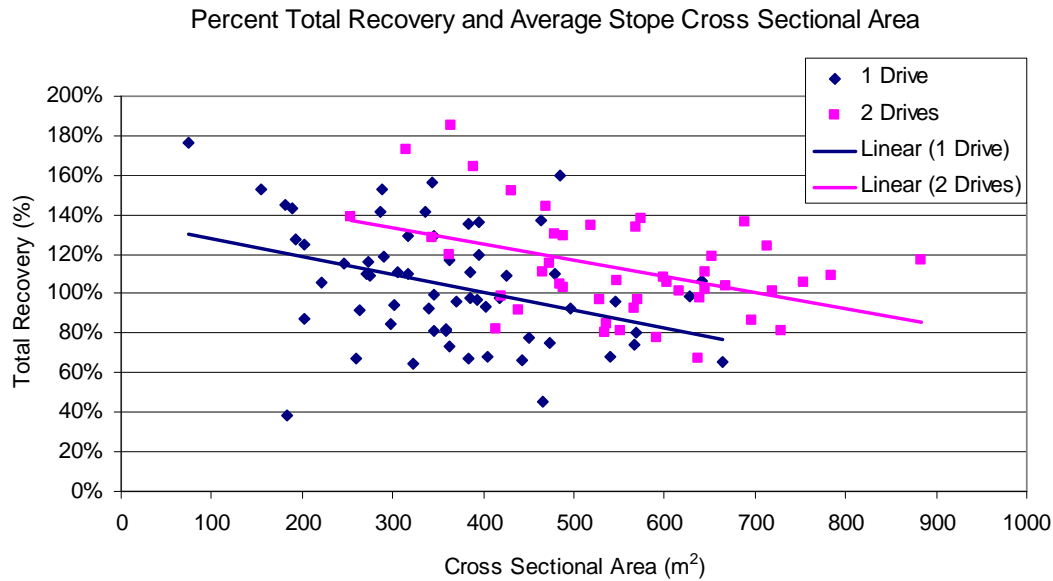
Graph 11   Total Recovery and Stope Cross Sectional Area

Percent Ore Recovery and Average Stope Cross Sectional Area
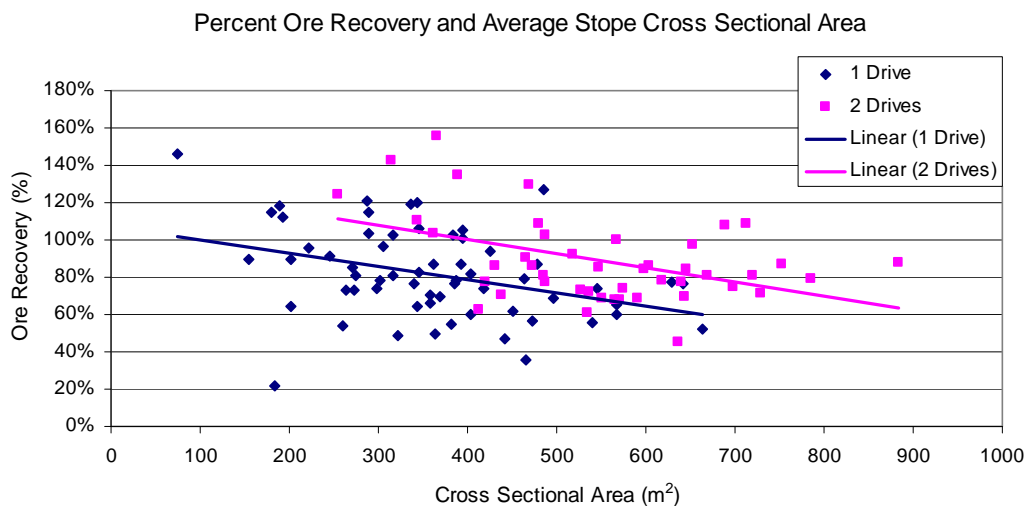
Graph 12   Ore Recovery and Stope Cross Sectional Area

7.2   Multiple Regression Linear Analysis

A multiple linear regression was undertaken using Microsoft Excel for the parameters that might be expected to influence recovery.

It is important to select independent parameters for analysis. The parameters were ranked as Type 1, Type 2 or dependent parameters. Type 1 parameters are fully independent and can be set at will by the mine, Type 2 parameters are chosen randomly from the matrix of the data. Dependent parameters in a linear regression analysis Type 1 parameters are solved first, followed by type 2 parameters and generally dependent parameters are not included:

Type 1 parameters -   stope number, mining level, number of drives,

Type 2 parameters -   reserve grade, tonnes fired per drill meter or tonnes fired per total drilled metres, powder factor (kg/t), number of rings fired, number of stopes open at a time, number of days a stope is open, and cut off grade,

Dependent parameter -   stope cross sectional area. This is dependent because it was calculated on the tonnes fired in the stope rather than the true cross sectional fired area.

The key parameters identified by the linear regression were stope number, number of drives, powder factor and tonnes fired per total metres drilled.

Percentage Ore Recovery = -68+ 0.874*(Stope No.) + 11.8*(No. of Drives) + 81.3*Powder Factor +27.1*Tonnes/(drill+redrill) metres

r^2 =0.38, i.e. 38% of the variation in the percentage ore recovery is accounted for by these four factors.

Table 3 gives the "t statistic" and P values for the significant parameters.

Table 3   Multiple Linear Regression Statistical Analysis

Parameter	Coefficient	"t-statistic"	P-value
Intercept	-68	2.98	0.0036
Stope Number	8.74	6.22	<0.0001
Number of extraction drives (1 or 2)	011.8	3.11	0.0024
Powder Factor (kg/t)	81.2	3.83	0.0002
Tonnes per (drilled + redrilled) metres (t/m)*	2.71	2.71	0.008

Factors that were not statistically significant in the ore recovery included:

- Mining level (Reduced level below surface)
- Reserve grade
- Cut off grade for the small range of cut off grades used
- Average number of rings in a shot during the extraction of the stope
- Number of Days a stope was active for
- Number of stopes open during extraction of this stope

Undocumented factors of actual fired height and width, footwall angle, distance from the footwall drive to footwall of the orebody, as well as random variation and random error and bias, may be expected to contribute to the remainder of the variation.

8   Conclusion

The objective of sublevel caving should be to maximise ore recovery and minimise dilution. The unique challenge to this mining method is to cleanly draw a volume of ore surrounded by a caving hangingwall.

A future or existing longitudinal SLC operation similar to Big Bell could expect to experience on average the following recovery statistics.

- Total tonnes drawn, 108% of fired tonnes with a standard deviation of 28%.

- Ore tonnes drawn, 84% of fired tonnes with a standard deviation of 24%.

- Waste tonnes drawn, 23% of fired tonnes with a standard deviation of 9%.

- Average Head grade reduction of 0.8 g/t to 1 g/t given a 'dilution material grade' of zero g/t.

- Head grade reduction of 0.84 g/t given a 'dilution material grade' of 0.24 g/t.

At Big Bell the northern and southern sectors of the mine have different draw mechanisms due to a change in hangingwall strength. A stronger northern hangingwall resulted in reduced waste entry which allowed the greater recovery of ore. The mine wide average for ore recovery and dilution correlate well with Kvapil's evaluation criteria.

Multiple linear regression analysis was used to determine the statistically significant parameters that influenced recovery.

When powder factor was used as a proxy for fragmentation, it was found that ore recovery increased with finer material, while waste recovery decreased in certain stopes, that is stopes 92, 82 and 53 showed clear trends, but each stope had its own trend line. Plots of recovery and tonnes per total drill meters showed that ore recovery increased with more drill metres and was accompanied by a decrease in waste entry.

In terms of stope geometry, two ore drives delivered better ore recovery (increase by 20%) with similar amounts of dilution (increase by 5%) when compared to one ore drive drawing the same cross sectional area. Ore recovery was greater in stopes with a smaller cross sectional area. Similarly, a smaller stope height (or width) was observed to yield higher ore recoveries. An increase in required draw size by 60% resulted in an approximate 20% reduction in ore recovery.

It is possible to conclude that the scatter in the data indicates the variability inherent in the mining method. From the results of this study it is evident that there are several variables that must be simultaneously satisfied for the successful implementation of a sublevel cave, these include establishing databases that allow a comprehensive documentation of draw performance during each shot, and the blasting process with shot and drive geometry.

Acknowledgements

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