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IMPROVEMENTS IN TRUCK REQUIREMENT ESTIMATIONS USING DETAILED HAULAGE ANALYSIS

Patrick Doig and Mehmet S Kizil

ABSTRACT: Accuracy in haulage analysis is fundamental for reliable cost and productivity estimation. The level of detail to which haulage analysis is conducted can significantly influence these estimations. The recent advancement of computer processing has enabled a range of software to manage large datasets and run multiple and complex haulage scenarios, thus increasing the level of detail. Substantial evidence is available to affirm the benefits of detail in haulage analysis through the scope of truck cycle time and truck prediction methods. However, due to the novelty of advanced software, no literature that documents the level of detail and frequency of haul roads required for haulage analysis was found. It was therefore, the objective of ongoing work for this research project to quantify the value added through additional detail in haulage analysis, specifically, the benefit of frequently changing haul roads. To facilitate this process, nineteen haulage scenarios were analysed with varying detail. In addition, a geological model and topography was created. From the analysis conducted, a clear relationship was identified between decreasing haul road calculation frequency and inverse variance error from the mean cycle time. The research showed that performing two as opposed to a single haulage analyses for a strip can affect the calculated truck cycle times from 6% to 14%. Additionally, it was found that changes in horizontal distance from the endwall were more significant than the vertical change for the analysed strip.

INTRODUCTION

The capital and operating costs associated with truck haulage contribute heavily to the overall expenditure of a surface mine (Blackwell, 1999). The costs of truck and shovel operations alone comprise 50 to 60 per cent of the total surface mining operation costs (Nel, *et al.*, 2011). A truck can be considered as the lifeline to a loading unit. The capital cost of a truck ranges from A$1.8 to A$4.7 million for size ranges between 100 to 240 t respectively (R2Mining, 2012). Without the required number of trucks, production can be significantly impacted; a surplus of trucks however, comes at a high cost and reduces profit. Thus, truck haulage modelling is an important and complex problem for mining operations. The impacts of over or under trucked fleets can significantly influence costs, capital intensity and productivity of mining operations.

Traditionally, trucking numbers are calculated using Talpac© or other similar programs from the truck manufacturers. These methods require intense manual input and thus substantially limit the number of haul roads analysed which leads to a subsequent reduction in detail for the analysis. Increased processing capacity of computers and software development in recent years has enabled haulage models to become more detailed. Software programs such as Deswik® have automated the calculation of trucking routes. This has enabled the potential for mining and dump locations to be cut into smaller portions i.e., block and bench. From this, multiple options for haulage routes and destinations can be calculated to best achieve a predefined objective.

The benefit of additional detail in a haulage model has not been quantified. With a greater understanding of the downstream effects of detail, mining engineers can quantify the value added through detailed dump and haulage designs and make adjustments as required. This has the potential to significantly change the outputs of many mine designs and the time and cost of haulage projects. Although detail is beneficial to most projects, engineers are restricted with time and costs. Current mining practices therefore, use simple models in order to make them more transparent and auditable. Mining companies however are focusing to better align actual and maximum potential truck and shovel productive capacity (Nel, *et al.*, 2011).

This paper aims to determine the value added through more detailed haulage analysis. The primary metric was the truck cycle time and difference in excavation rate. Varying detail was assessed with mining locations, dump destinations and haul roads. The desired outcome was that additional detail in

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haulage analysis, specifically frequently changing haul roads, significantly adds value to a mining operation and thus greatly impacts on accuracy of budgets and productivity.

TRUCK CYCLE TIME PREDICTION METHODS

There are many products and methods currently available for the prediction of truck cycle times which utilise a combination of empirical or calculated data to formulate results. All of these methods appear to provide results that are acceptable within industry standards. The methods presented can be divided into three subgroups dependant on the level of manual input required, historical data and degree of automation. The three subgroups include:

- Talpac©, Arena® and FPC®;
- Multiple Regression and Artificial Neural Networks; and
- Deswik and MineSchedTM.

Chanda and Gardiner (2010) presented a comparative study of truck cycle time prediction methods for open-pit mining. This study evaluated three methods which included computer simulation, in this case Talpac©, artificial neural networks and multiple regressions with an aim to determine the best method for predicting truck requirements. Through this study, Chanda and Gardiner (2010) found that computer simulation methods both under and overestimate the truck cycle times for short and long hauls respectively. Evidence suggests that artificial neural networks and regression models are superior to computer simulations in their predictive ability (Chanda and Gardiner, 2010; Blackwell, 1999). Artificial neural networks and multiple regression methods however require detailed historical data for the calculation of their formulas. These methods therefore cannot be transferred between mining operations due to the nature and complexity of creating these systems.

The Talpac©, FPC®, Arena®, artificial neural networks and multiple regression methods typically utilise static haul roads in a single point in time. In these methods, judgment is usually made for the centroid of the dig and dump locations for the haulage analysis. Although the mining schedule usually does not significantly change in sequence, the dump schedule can vary considerably. Often mining engineers have detailed dump designs with simple spread sheets and running balances. This can lead to a large margin of error when creating a haulage analysis.

Although it can be critical in knowing the precise performance of a haulage fleet and the possible variations, longer term data is sometimes required. Most mines are much more complex with progressively changing dig and dump locations. Within these changes, large variations can be seen in the gradients and total distance travelled significantly affecting the truck performance. It is therefore beneficial that simulations or time studies are used for the calculation of truck cycle times using predetermined dig and dump strategies such as Gemcom's MineSchedTM and Deswik's Dump Scheduler software. Many mining companies have adopted Deswik software to assist in creating more accurate mine plans and cost estimations (Johnson, 2012).

The Deswik Dump Scheduler utilises a detailed method, for the calculation of truck requirements and cycle times, through the use of dynamic haul roads. The dump scheduler utilises mining solids and a mining schedule which can be created in either the Deswik Scheduler or through another scheduling package. A series of dump designs and solids are required whereby they are connected with the mining solids using haul strings. The dump schedule can then be created using a variety of different haulage objectives such as shortest haul, minimum relative level and minimum cycle time.

The truck cycle times and truck requirements are calculated using individual haulage calculating the uphill and downhill components through the use of rimpull and retard curves, stop signs and speed limiting factors. In addition to this, average utilisation, availability factors and load and dump times are used. This analysis considers hundreds of thousands of options for haulage routes and destinations within minutes. On completion of the dump analysis, 3D animations and reports can be produced. The Deswik Dump Scheduler method can help the mine planner identify errors within the analysis easily allowing for adjustments to be made and reruns of the analysis to be completed within a matter of minutes. This can result in significant time and costs savings for haulage analysis.

METHODOLOGY

Model creation

In order to uphold confidentiality agreements, actual geological and topographic mine data was not utilised in this project. Instead, a training dataset was created for the research project. The created topography and geological features replicate typical open-cut dragline and truck and shovel mining operations in the Bowen Basin coal fields. They were however created to simulate as many mines as possible.

A detailed mine design was created for four pits, R02N, R02S, R01N and R01S. These designs were completed to a short term planning level of detail which incorporated ramps to the coal floor for ten progressive strips. A total of seven coal seams of varying seam and interburden thickness were created in Excel® dipping North-East between 6% to 8%.

Five main haul roads were designed for the pits. Dragline spoil piles were designed for each of the four pits and ten strips equating to a total of 40 progressive pit designs. Each of these designs incorporated a low wall ramp that toed out at the coal floor. Upon completion of the dragline spoil designs, a spoil balance was conducted. From this spoil balance, the entire Seam I overburden material was allocated to the dragline. In addition to this, approximately half of the R01S and R01N, Seam W overburden was allocated to the dragline material due to the Seam I overburden being reduced dramatically by a roll in the coal seam.

Prestrip dump designs were completed for each progressive strip of the four ramps. The prestrip designs incorporated dump heights of 15 m with variable bench widths ranging from 15 to 50 m. Each prestrip dump design also included an offset of two dragline spoil piles for geotechnical stability reasons. Figure 1 shows an isometric view of R02N prestrip design.

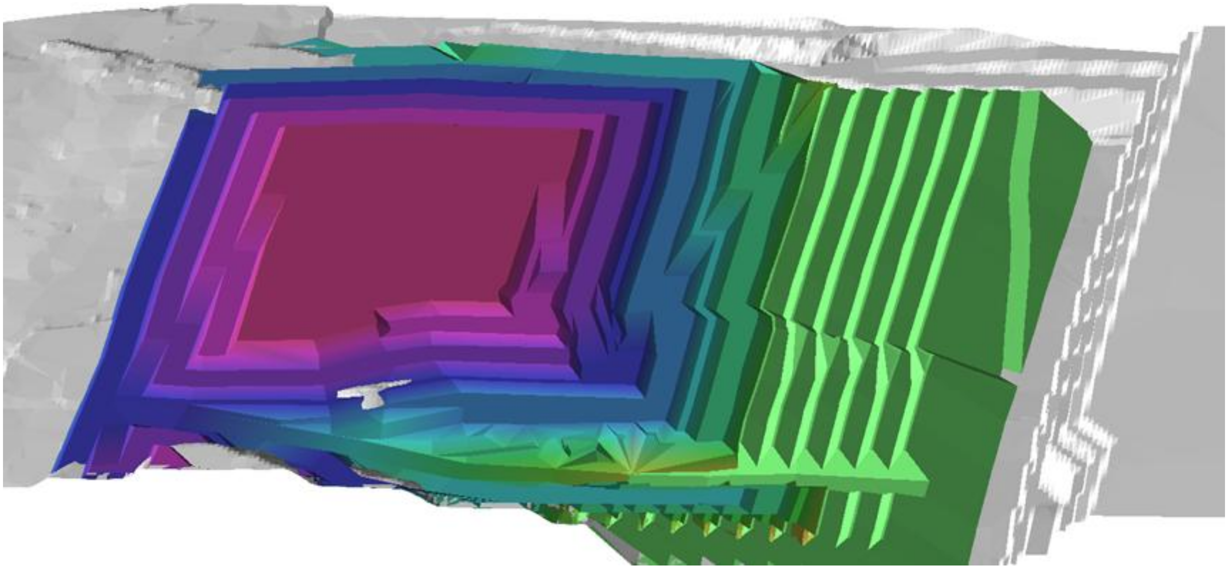

Figure 1 - Ramp RO2N isometric view

Two Caterpillar 8200-124 draglines were selected for the mine designs and schedule and two excavator fleet sizes were used in order to replicate the common practice for coal mines in the Bowen Basin. The primary overburden excavators used were RH340 Caterpillar backhoe configuration excavators with an average rate of 1750 BCM/h. The partings and coal were removed using a smaller RH120 Caterpillar backhoe configuration excavator with an average production rate of 900 t/h. It was presumed that the truck size had been optimised for this operation and Caterpillar 785B coal body trucks were selected for transport of coal. Caterpillar 785B class trucks were selected for overburden removal. The required truck numbers and cycle times were calculated in the haulage analysis.

A detailed mining schedule was created for the four ramps. Resources were then divided and allocated to the northern and southern ramps and included, coal excavators, Caterpillar 8200 draglines, and two prestrip excavators. A resource levelling algorithm was then completed using strip ascending and seam number ascending priorities. Within this process, start and finish dates were allocated to each of the mining blocks using the assigned mining resources.

Data acquisition

A haulage analysis was conducted using a ".trux" file for a Caterpillar 785B in which a minimum cycle time scenario was run. The haulage analysis provided a variety of routes and cycle times for the dig and dump locations of the individual mining blocks. In order to remove the variable of multiple cycle times per individual mining block, a weighted average cycle time was assigned to haulage from each mining block. The resultant weighted average cycle times were subsequently imported onto the mining solid attributes which were then exported to Excel® for data analysis.

Following inspection and categorization of the exported cycle time data, it was apparent that too many variables existed between individual seam horizons to elicit trends. Consequently, a single seam horizon with the largest quantity of overburden, the S horizon, was chosen for data collection. The original haulage cycle time calculations were performed on a 25 m block size with 5 m vertical bench heights. This scenario was completed to the greatest level of detail and formed the base case. To compare the effect of the level of detail on cycle times, analysis was also completed on the following block sizes: 50 m, 100 m, 200 m, 400 m, 700 m blocks and total strip with 5 m benches, 400 m and 700 m blocks with 10 m, 15 m and 20 m benches and total vertical horizon and finally total strip with 10 m, 15 m and 20 m benches and with total vertical horizon.

RESULTS

The cycle times from the haulage scenarios were calculated to three decimal places. However, to facilitate comparison of the data results, haul cycles were pooled into 0.1 min intervals. A simplified plot of the cumulative dump volume against the pooled cycle time can be seen in Figure 2 for three scenarios. The graph clearly demonstrates that with progressively less detail, the distribution narrows, drawing closer to the mean cycle time. The weighted average cycle time of each scenario was consistent as the mining and dump block locations and their respective destinations are unchanged. There was however significant variation within the weighted average of each scenario. To ascertain the effect that increased horizontal distance from the endwall had upon the cycle time, a graph was plotted for each strip as shown in Figure 3. As expected, the cycle time progressively increased as the horizontal distance from the endwall increased. It should however be noted that there were a few exceptions to this trend. These occurred when extra ramping was required for the first few blocks mined from the endwall and, where new dumps were progressively becoming available. A similar trend was also found when analysing the weighted average effect of increased depth on cycle time.

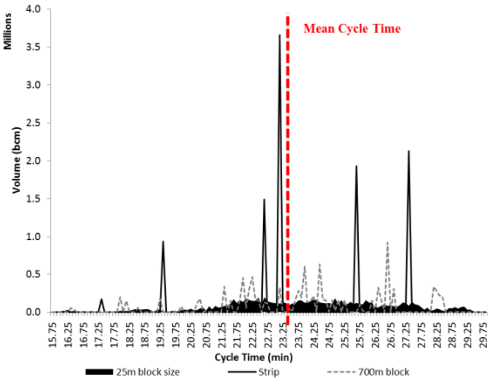

Figure 2 - RO2N S seam cycle time distribution

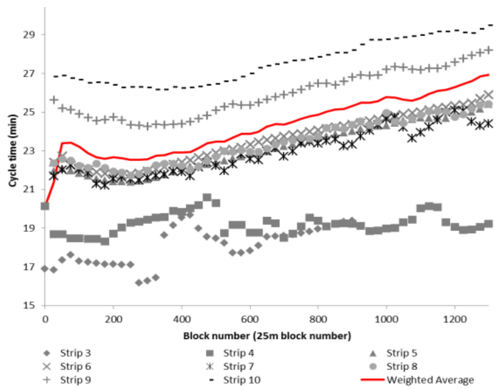

Figure 3 - Cycle time and increasing block number correlation

Figure 4, shows the increase in cycle time against the horizontal distance from the endwall for strips seven and ten. The shaded area around the weighted average cycle times for strip seven and ten shows the minimum and maximum cycle times for varying depths at a particular horizontal position. This represents the variance or error caused by changes in elevation. Figure 5, shows the weighted average cycle time for strips seven and ten against the vertical bench height. The shaded areas represent the error created by minimum and maximum horizontal distance from the endwall at a particular vertical bench height. As can be seen in this graph, a much larger variation is evident compared to that shown in

Figure 4. This shows that the horizontal distance from the endwall effects the haul cycle time to a greater extent than the vertical distance. This however could vary depending on ramping structure.

DISCUSSIONS

To assess the differences between the data, the standard deviation was calculated for each set of block ID's. To account for the fact that each cycle time pool had variable volume, the standard deviation and the variance was volume weighted. A standard deviation curve of the total S horizon overburden for horizontal block size increases can be seen in Figure 6. There was a noticeable difference in the standard deviations between the detailed haulage analysis to 700 m, and the whole strip horizon.

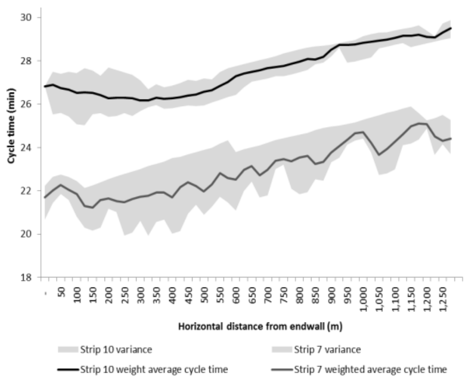

Figure 4 - Strips seven and ten distance against cycle time

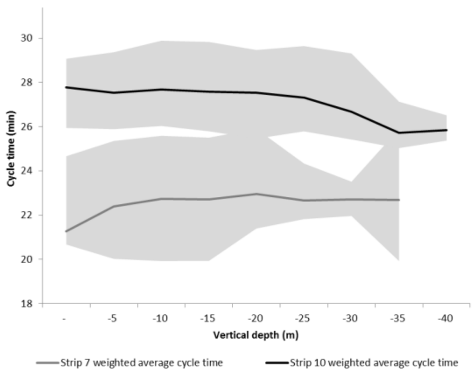

Figure 5 - Strips seven and ten bench depths against cycle time

In order to further quantify the differences, the total error was calculated for each of the detailed scenarios. This total error was calculated as two times the variance. To make the error more transparent within the scenarios, the error was translated to a percentage of the mean cycle time. Then, in order to present the 25 m block size as the base line, the error percentage was inversed as seen in Figure 7. As the detail decreased, the inverse error increased from the initial base case as the individual data points come closer to the mean cycle time.

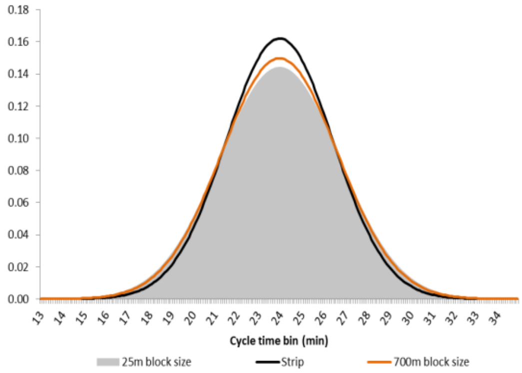

Figure 6 - S horizon overburden standard deviation for horizontal block size increase

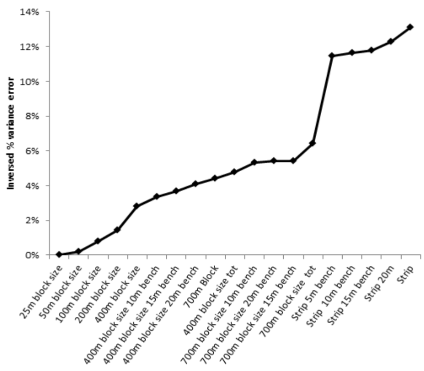

Figure 7 - Inversed percentage variance error for S horizon overburden

In order to ensure the data within the whole strip horizon followed similar trends to individual strips, an analysis was completed for strips six and seven. Due to the variability of the cycle time created from varying bench depth, only the increased horizontal block length was considered for this analysis. As can be seen in Figure 8 and Figure 9, the standard deviation graphs for strips six and seven followed a similar trend to that of the whole strip horizon with all block sizes up to half the size of the total strip producing relatively small differences within the standard deviation. The total strip standard deviation curve had a significant spike compared to that of the 700 m block size or less. This was due to the relatively small number of calculated cycle times and the effect of the variation of cycle time generated from the relative distance from the endwall. Therefore if only one data point is used for an individual bench height in the total strip, the variance of the combined benches is zero.

The inverse error percentage of strips six and seven followed a similar relationship to that of the total strips as seen in Figure 10, thus indicating that this trend was not an anomaly.

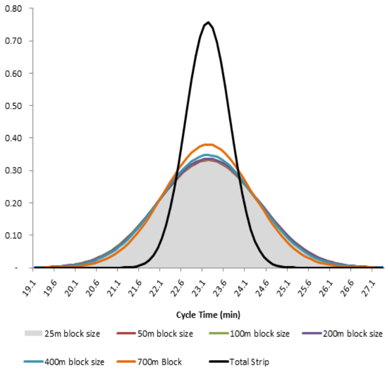

Figure 8 - Strip six horizon standard deviation for horizontal block size increase

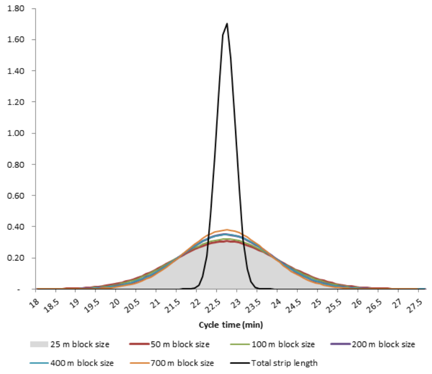

Figure 9 - Strip seven horizon standard deviation for horizontal block size increase

The variance error from the base case was then assessed for both the horizontal and vertical block sizes and plotted on a graph as seen in Figure 11. There was a noticeable difference between the varying horizontal block sizes and the varying vertical block sizes for the seam horizon. From the results presented in Figure 11, it can be assumed that it is more important to at least halve the horizontal block size for a given haulage calculation than to modify the vertical block size. This statement however is dependent upon the total seam horizon depth and the ramping structure. If there was a great variation in the cycle time between the vertical benches, it could be more beneficial to halve the block vertically than horizontally.

To present the effect block size used in a number of haulage scenarios, the variance error was calculated for the haulage cycle time. This analysis, depicted in Figure 12, showed a similar relationship to that presented by varying block sizes. In both instances, the total strip cycle time inverse error differences were significantly higher and did not appear to conform to the other detailed haulage analysis for block sizes 700 m and less. It was determined that this significant change was due to the fact that only one cycle time value was being generated for the total strip.

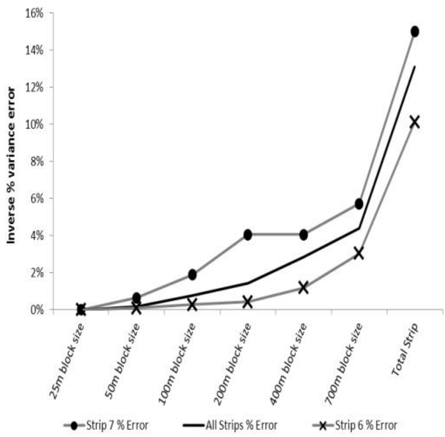

Figure 10 - Percentage variance difference for 5m bench height

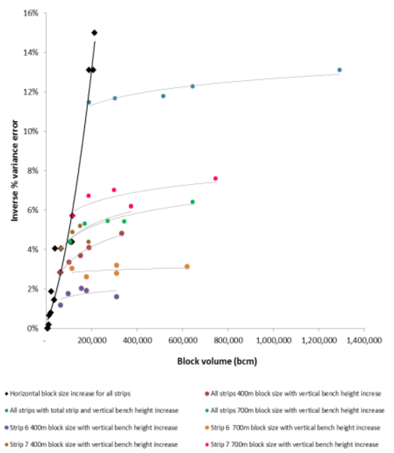

Figure 11 - Horizontal and vertical error comparison

A final analysis was completed to determine the influence of haul road calculation frequency on the mining schedule and mining rate. For this purpose an example case was generated, where the mean cycle time was utilised to calculate the overall truck requirements. In this case, six trucks were scheduled. As shown in Table 1, the average excavation rate, which is limited by the excavator mining rate, is 1270 BCM/hr. In mining the first half of the strip, where the cycle time is below the mean cycle time of 22.68 min per cycle, the excavator can maintain its maximum rate at 1270 BCM/hr. As the cycle time increases above the mean, the mining rate will be limited. This limitation is due to the fact that the excavator will no longer be loading at its maximum rate as it will be awaiting trucks for loading. Figure 13, illustrates the varying excavator rates within one strip depending on the increased cycle time throughout the strip. This example shows that due to incorrect truck allocation the mining rate can be reduced by 12%.

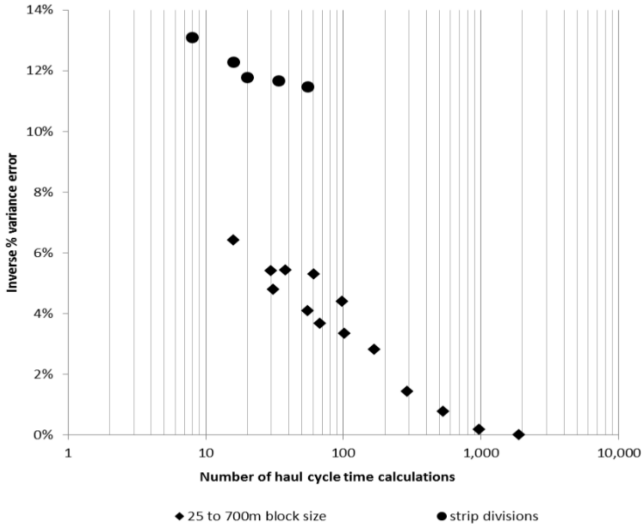

Figure 12 - Cycle time calculation frequency

Table 1 - Excavation rate variations

Parameter	Average	Min	Max
Cycle Time	22.68	25.90	19.92
bcm/load	80	80	80
Cycles/hr	2.6	2.3	2.8
bcm/hr	212	185	241
No Trucks	6	6	6
Mining Rate	1270	1112	1445
Difference		12%	14%

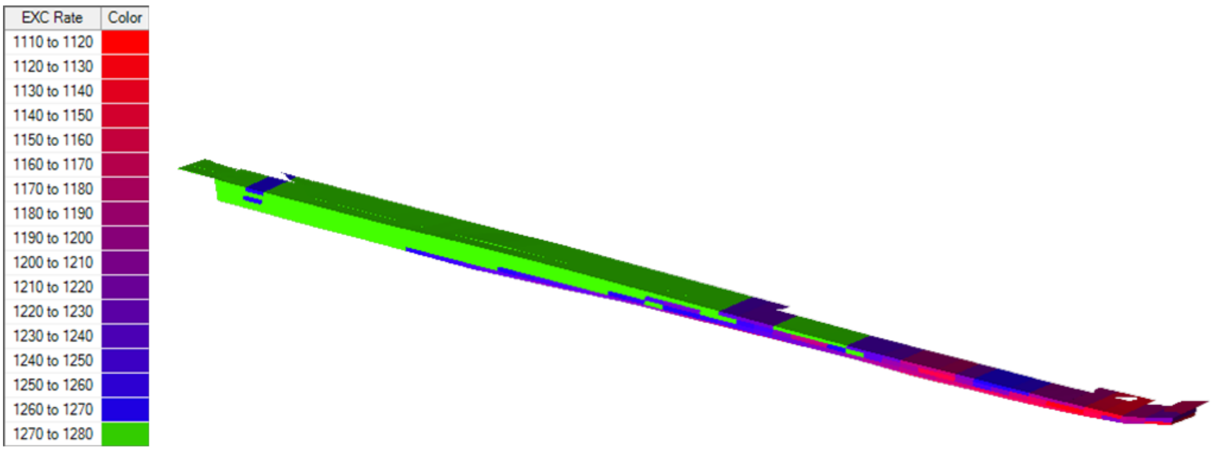

Figure 13 - Excavation rate variations

CONCLUSIONS AND RECOMMENDATIONS

Haulage is a major component of the overall cost of a surface mining operation. It is therefore imperative that adequate time, effort and resources be committed to ensuring accuracy in haulage analysis. Increasing the frequency of haul roads routes has become possible with advanced software such as Deswik.CAD®. The actual benefits of this additional detail in haulage analysis have not been quantified in the past. Many mining companies however have experienced the benefits of this software allowing them to account for required trucks, identify areas of limited dump room, minimise cycle times and create strategic dumping strategies.

The aim of this study was to identify the value drivers for haulage analysis. The primary objective was to quantify the value added through additional detail in haulage analysis. From the haulage analysis conducted throughout this project, a correlation was identified between the size of a mining block within a strip and the change in variance error from the mean cycle time for a particular strip. The study found that the estimated cycle time can be significantly affected by the sampling frequency by up to 14%. The study also demonstrated the potential use of this result to calculate the percentage reduction of the mining rate due to under allocation of trucks. Therefore, the most significant result of the study was that sampling frequency can change truck requirement estimates. In practical terms, insufficient number of haul road routes can potentially push out mining schedules, reduce mining rates, decrease utilisation of truck fleets and decrease mine productivity and profits.

The data analysis conducted was limited to one seam horizon within a mining operation utilising one ramping and dumping system. It is therefore recommended that additional analysis be conducted to determine the effects on other ramping and dumping methods. Additionally, an analysis should be completed to determine the influence that fluctuation of truck cycle times has upon the performance and schedule of the entire site. Finally, a financial analysis should be conducted to determine the monetary value added through this additional detail.

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