A Simulation Approach to Constraints Management of an Underground Conveyor System

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Executive Summary

This final report researches and discusses the importance of managing and alleviating constraints placed upon underground mine conveyor haulage systems. As conveyor systems are paramount in the haulage of coal from underground operations to various designated areas within a mine, it is of utmost importance that these system are fully functional with minimal failure. It assumes a constraints analysis technique, in an effort to expose bottleneck areas and suggest best practice methods of managing these constraints to improve on production figures and by extension, profit margins.

This study is completed under the mentorship of Mr Stephen Gerard Ross, Asset Optimisation Officer of the Anglo American Inyosi Coal Group. This paper is specifically referenced to one of the group's underground operations, Zibulo Colliery. It not only seeks to research different tools, techniques and methods developed for constraints management, but it also employs a trade-off study to select the best method or tool to duplicate the existing mine process and make further recommendations with the aid of a proposed solution model.

It incorporates buffer technology, optimal location strategies and cost-volume relationship studies on buffering for protection of production and to meet operational and strategic objectives in the long run.

This paper points out that although simulation modelling doesn't generate optimal solutions, it remains to be the best approach for modelling storage capacity, as empirical and analytical approaches oversimplify haulage system networks and are derived from unrealistic assumptions. The accuracy of results in simulation modelling is entirely dependent on the data file constructed by the user and the level of detail that the model considers.

Furthermore, this paper highlights the importance of continued process improvement in the mining industry, irrespective of its traditional structures with a specific focus on conveyor systems reliability improvement, taking into account those constraints which remain unchanged.

Contents

List of Figures 5
List of Tables 6
List of Equations 7
1. Introduction and Background 8
2. Problem Statement 10
3. Data analysis – problem statement 13
3.1. Part I: On surface 13
3.2. Part II: Underground 18
4. Project Aim 19
5. Project Approach and Scope 19
6. Deliverables 20
7. Literature Review 21
7.1 Bunker sizing 23
7.1.1 Empirical Modelling 23
Sevim Model 23
Flaws of the Sevim model 30
7.1.2 Stochastic Process Modelling 30
The Thompson-Carnahan Model 30
Flaws of the Thompson and Carnahan model 33
The Baral-Daganzo Model 34
Flaws of the Baral – Daganzo Model 37
7.1.3 Discrete event simulation 37
7.2 Bunker Location 38
7.2.1 Theory of Constraints 39
Flaws of the TOC model 40
7.2.2 FMEA using VIKOR and fuzzy numbers 40
Stochastic Process Model – Location 48
Baral Location Model 48
8. Literature Review Conclusion 51
9. Simulation Model Requirements 52
10. Surface As-Is Model 53
10.1. As-Is Surface Model - Silo operating levels and logic 53
10.2. Modelling belt system failure 55

10.3 Surface model – solution requirements 57
10.4 Simulation results – As – Is Surface Model 59
10.5 Proposed Surface Solution Model 60
11. The Underground As-Is Model 62
11.1 Strategic location of underground storage bunkers 62
11.1.1 Alternative 1 - FMEA with VIKOR under a fuzzy environment 62
11.1.2 Alternative 2 – Decision Trees 66
11.2 Underground model – solution requirements 68
11.3 Simulation Model for Bunker sizing 69
11.3.1 Alternative Solution to Bunker 1 Arrangement 72
12. Production Gain ROM (Run-Off Mine tonnes) 74
12.1 Production gain to problem 1 – surface throw-out silo 74
12.2 Production gain to problem 2 – underground bunkers 74
13. Cost analysis 75
14. Profit Analysis 75
15. Conclusion 77
16. References 78
Appendix A: Industry form 79
Appendix B: Total Availability Model 80
Appendix C: Expense Revenue figures for 2012 81
Appendix D: Belt Information 82
Appendix E: Inter-arrival time distributions 83
.... 83
Appendix F: Section belt downtime distributions 95
Appendix G: Trunk belts (Gathering belts) downtime distributions 101
Appendix H: Belt Mean Time Before Failure 108
.... 108
Appendix I: Simulation Results Opportunity Lost (Run Off Mine tonnes) 109
.... 110
Appendix J: Cost of a 2000T bunker 112
.... 112
Appendix K: Cost of a 6000T bunker 113

List of Figures

Figure 1. Schematic layout of the mine 10
Figure 2. Exploded view of mechanical systems between the two silos (6000T and the 2000T) 12
Figure 3. Exploded view of mechanical systems between the 2000T Silo and the Plant 12
Figure 4. Pareto analysis on Plant, overland conveyor and crusher and chute system downtimes 15
Figure 5. Average production rate loss in the event of system start-up 17
Figure 6. Pareto analysis on underground belts 18
Figure 7. Bunker used for surge 22
Figure 8. Bunker for buffering 22
Figure 9. Bunkering for segregating flow 22
Figure 10. Transition between various states 24
Figure 11. Probability profiles a room and pillar face (Sevim, 1987) 26
Figure 12. Probability profiles of the four flows merging onto one belt (Sevim, 1987) 27
Figure 13. Probability profiles from 9 flows which merged on a single belt (Sevim, 1987) 29
Figure 14. Cumulative probability distributions for bunker contents at different time intervals (Sevim, 1987) 29
Figure 15. Operational policy of bunker according to Sevim 30
Figure 16. (a) Load inter-arrival time constructed from time studies; (b) load discharge time constructed form time studies (Carnahan et al, 1991) 31
Figure 17. System with n conveyors 34
Figure 18. Simplified system with two conveyors 35
Figure 19. System states (Baral et al, 1987) 35
Figure 20. The different possible transition states that the model can assume (Baral et al, 1987) 36
Figure 21. Relationship chart between A and V (Baral et al, 1987) 37
Figure 22. System failure consequential 39
Figure 23. Conveyor system divided into 3 portions for critical path analysis 40
Figure 24. Trapezoidal fuzzy number A (Lui et al, 2012) 43
Figure 25. Membership functions for rating weights (Lui et al, 2012) 44
Figure 26. Membership functions for rating failure modes (Lui et al, 2012) 45
Figure 27. Flowchart of FMEA with fuzzy numbers and VIKOR (Lui et al, 2012) 48
Figure 28. Simulation model requirements (inputs), constraints, and projected outputs 52
Figure 29. Depicting the 6000 ton silo, conveyor scv 002 and the crusher and chute system 53
Figure 30. Depicting the 2000 ton silo, the Plant and dispatch train 53
Figure 31. 6000T silo logic 54
Figure 32. Modelling belt failure and repair logic of the crushers and chute system 55
Figure 33. Underground model depicting the 8 sections along with equipment 62
Figure 34. Silo 6000T full at upper warning limit of 5700 tons 57
Figure 35. Graph showing the time taken for the 6000T silo to reach full capacity 58
Figure 36. Box and whiskers diagram for surface systems (downtime durations) 58
Figure 37. 6000T silo with additional 3000T capacity 61
Figure 38. 6000T silo weight contents against time 61
Figure 39. Second silo weight contents .... **Error! Bookmark not defined.**
Figure 40. Decision tree analysis for risk mitigation 67
Figure 41. Box and whiskers of the top ranked belts which require risk mitigation strategies 69
Figure 42. Bunker 1 in operation whilst the East Main belt has failed 70

Figure 43. Graph of the optimal size capacity required for Bunker 1 after 10 replications 70
Figure 44. Bunker 2 in operation whilst Trunk 4 has failed 71
Figure 45. Graph of the optimal size capacity required for Bunker 2 after 10 replications 71
Figure 46. Alternative arrangement of bunker 1 72
Figure 47. Storage bunker 1 set at a maximum capacity of 1000 tons 73
Figure 48. Auxiliary storage bunker set at a maximum capacity of 1000 tons during a simulation run 73

List of Tables

Table 1. Downtimes with specific reference to the Plant, overland conveyor belt, and crusher and chute system 15
Table 2. Profit Loss for underground production halt for the year 2012 16
Table 3. Downtimes on all underground belts 18
Table 4. Conventional FMEA scale for occurrence 41
Table 5. Conventional FMEA scale for severity 41
Table 6. Conventional FMEA scale for detection 42
Table 7. Fuzzy numbers for rating the weights of O, S, and D (Lui et al, 2012) 44
Table 8. Fuzzy numbers of risk factors (failure modes) (Lui et al, 2012) 45
Table 9. Summary statistics table of downtime duration surface systems 59
Table 10 Silo weight contents 60
Table 11. Silo 6000T weight flow out 60
Table 12. Average duration of failure of surface systems 60
Table 13. The identified failure modes from the system 63
Table 14. The different departments within the mining industry 63
Table 15. Weighting of risk factors O, S,and D 63
Table 16. Fuzzy numbers for weighting risk factors 63
Table 17. Risk judgement of failure modes 64
Table 18. fuzzy numbers of failure modes 64
Table 19. Transforming fuzzy numbers into crisp values for the different risk factors 64
Table 20. Transforming fuzzy numbers into crisp values for the different failure modes 65
Table 21. Value of criterion for alternatives 65
Table 22. Ranking failure modes 65
Table 23. Comparison between FMEA with VIKOR and Decision Tree analysis 68
Table 24. Optimal bunker capacities (in tons) required underground for the East Main belt and Trunk 4 72
Table 25. Capacity of bunker 1 (Silo6000T) and the auxiliary bunker (Tank2) 73
Table 26. Percentage Failed time for East Main belt and Trunk 3 75
Table 27. Projected annual profit 76

List of Equations

Equation 1. Fundamental rule of flow rates 21
Equation 2. Exponential density function (Sevim, 1987) 24
Equation 3. Weibull density function (Sevim, 1987) 24
Equation 4. New probability profile of gathering belt (Sevim, 1987) 27
Equation 5. Calculating the average outflow from a surge bin (Sevim, 1987) 28
Equation 6. Cycle time computation (Carnahan et al, 1991) 32
Equation 7. Convoluted pdf of X_i (Carnahan et al, 1991) 32
Equation 8. Cycle time constraints (Carnahan et al, 1991) 32
Equation 9. Pdf for the discharge of the i-th load (Carnahan et al, 1991) 32
Equation 10. Pdf for loads *i* expected within the time interval range of (0, t) from a single loading source (Carnahan et al, 1991) 32
Equation 11. Simplified pdf of the number of loads *i* expected within the time interval range of (0, t) (Carnahan et al, 1991) 33
Equation 12. Availability of a belt system (Baral et al, 1987) 34
Equation 13. System availability expression for a two system conveyor (Baral et al, 1987) 35
Equation 14. Derived states probability function (Baral et al, 1987) 36
Equation 15. (Baral et al, 1987) 36
Equation 16. System availability as a function of bunker capacity (Baral et al, 1987) 36
Equation 17. Dimensionless value (Baral et al, 1987) 37
Equation 18. Grade of membership piecewise function (Lui et al, 2012) 43
Equation 19. Transformation formula (Lui et al, 2012) 45
Equation 20. Grade of membership for rating failure modes (Lui et al, 2012) 46
Equation 21. Lower limit, mode intervals, and upper limit of the universe *X* for rating failure modes 46
Equation 22. Grade of membership for rating weight criteria (Lui et al, 2012) 46
Equation 23. Lower limit, mode intervals, and upper limit of the universe X for rating failure modes 46
Equation 24. Criteria rating for *j* criteria allocated to alternative *Ai* based on equation 21 46
Equation 25.Ri and Si computations for ranking alternatives (Lui et al, 2012) 47
Equation 26. Computing Qi, the final unit for ranking the alternatives (Lui et al, 2012) 47
Equation 27. System availability for a system with n conveyors (Baral, 1987) 49
Equation 28. Availability of the system at the asymptotic point (Baral et al, 1987) 49
Equation 29. determination of the optimal location j of the bunker (Baral et al, 1987) 49

1. Introduction and Background

Believed to be one of the most fundamental processes ever to be developed, the coal mining industry is almost set in stone with institutional structures and legacy processes. Since the discovery of this non-renewable resource (dating back to the 1880s), the mining industry has played a pivotal part in the growth, sustainability and development of numerous countries. According to Amadi-Echendu, Lephauphau and Maswanganyi (2011), the industry continues to face technological environments which are continuously changing, indicating that the necessary changes need to be made to bridge the gap.

One of two methods of mining underground coal is employed by collieries around the world, namely; the Long wall mining method and the Continuous mining method. In Longwall mining, a shearer is used to cut coal and load onto a chain conveyor, whereas in Continuous mining a continuous miner cuts the face and loads onto a shuttle car or battery hauler, which then offloads onto a conveyor belt.

Wang (1998) in his dissertation on Mine Belt Systems highlights that underground coal mining haulage systems require intensive capital investment as they are representative of complex systems in their application in this regard. Wang (1998) further states that these systems are critical in the transportation of coal from the mining sections to the surface because ultimately, the production of a mine is almost entirely dependent on the reliability, economics and efficiency of conveyor systems.

Zibulo Colliery, situated in the Witbank area, is one of Anglo American's South African coal mines. This underground operation adopts the Continuous mining method and is comprised of 8 mining sections. It is set to deliver 8 Mtpa (million tonnes per annum) for export and domestic purposes, over a life of mine of 20 years. Like most coal mines, Zibulo's coal is transported via a series of linked conveyor belts from the mining sections to a 6000 ton silo on surface (used for buffering purposes) and from there on dispatched, on a continuous basis, to a Plant where the coal is further washed, processed and carried via rail to meet various customer demands. The Plant is a 50:50 joint venture between BHP Billiton and Anglo American Inyosi Coal and is managed and maintained by a third party company, Minoplex.

Golratt's Theory of Constraints, reiterated by (Smith and Pretorius, 2002), is based on the premise that every company or operation must have a constraint; else that company would bear similarities to a perpetual system producing infinite profits, (an impossible concept). This principle, according to Womack and Flower (1999), is based on the following: 1.

Systems are a series of events which are dependent on one another. 2. All systems are subject to a constraint which constitutes the bottleneck of the process. 3. Improved constraint performance results in an overall system performance improvement. 4. Constraints are a resultant of organisational rules, training, or measures referred to as policy constraints. 5. Improvement in a non-constraint is a mirage.

(Baral, Daganzo and Hood, 1987), stresses that in any serially dependent conveyor system, the failure of one component consequentially leads to the entire preceding system to be put on stop, the ramifications of these events, stretching far and wide in the said organisation. Baral et.al (1987) further puts emphasis on the provision of additional storage capacity at strategically located areas within a conveyor system for the preservation of production in the event of system failure. Wang (1998) also notes that the three models used in designing belt capacity, namely the "peak loading rate model, the average loading rate model, and the random loading rate model" have very little, if no impact, on designs for bunkering capacity. Therefore; as a producer of a valuable commodity, it is paramount for as asset-intensive organisation to have structured approaches and processes within its systems aligned with its strategic objectives for improved performance on both long and short-term goals.

2. Problem Statement

Wang (1998) notes that a mine conveying system may lack sufficient capacity to convey coal on one of the following reasons (1) the belt capacity may have been poorly designed from the onset, (2) a change in the mine plan which involved shifting to long-wall mining after the conveyor system was placed, (3) failing to anticipate the level of productivity or system capability that would eventually be reached through continuous improvement measures.

Wang also emphasises that these bottlenecks can be resolved through increasing belt capacities or installing bunkers, the former being significantly more expensive as it may require complete system re-configuration or replacement of the conveyor system completely.

The following figure represents a schematic of the current layout of the mine in question. As stated above, the underground operation consists of 8 sections with the arrows indicating the flow of coal from the mining sections to the Plant.

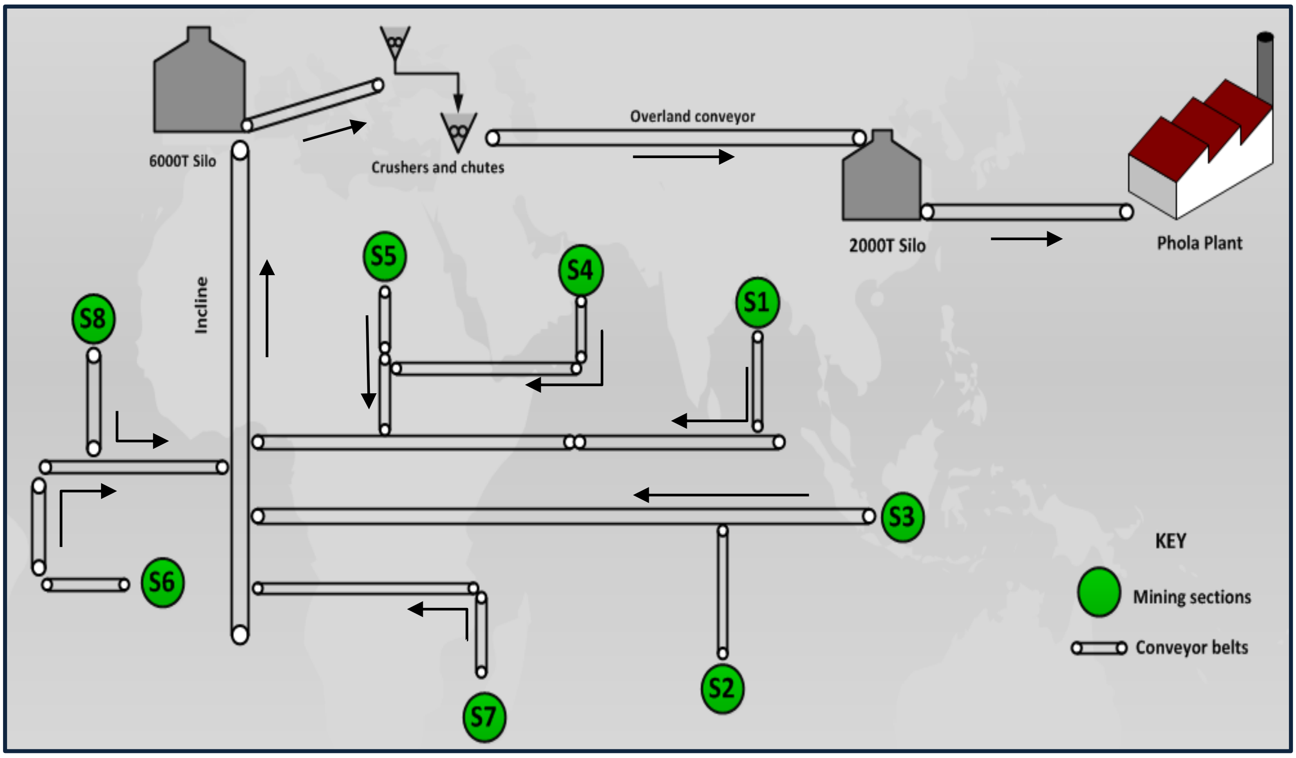


Figure 1. Schematic layout of the mine

The mine faces two major problems:

- A number of mechanical systems and processes exist between the two storage facilities (figure 2) and between the 2000T storage facility and the Plant (figure 3), namely a chain feeders, belts, crushers, magnets, metal detectors, screens and chutes and a stacker and reclaimer for transferring coal from the stockpile to the Plant were it is washed and dispatched via railway to Eskom and East London (for shipment) in differing variations. Should any of these serially configured mechanical systems fail at any given point in time, an associated risk exists in which the entire mine may have to stop production for the said duration depending on the occupied capacities of both silos at that point in time. Like the domino effect, which assumes “a chain reaction occurs when a change causes a similar change nearby which, in turn, causes another change, and so on, in a linear sequence.” (Stronge, 2004). Another factor that exacerbates the situation is that the mine’s designated area of responsibility ends just before the 2000 ton silo, that is, since the Plant is managed by a third party company, Minoplex, Zibulo management cannot seek to effect any changes in terms of maintenance schedules (preventative, predictive and proactive inclusive) and since the Plant is partly shared between Anglo Thermal Coal and BHP Billiton, it is therefore only made available to Zibulo Colliery three days of the week. During that three day period, the mine cannot afford to be experiencing any downtime measures that can be attributed to the Plant. Therefore, the onus rests solely on management to develop a credible process to mitigate the above-mentioned risk, which has had dire consequences on the mine’s financial standings in recent years. It should also be noted that this problem is somewhat twofold: in situations where the Plant is operating efficiently i.e. no breakdowns for a specified period, the mine itself also experiences problems stemming from those systems or pieces of equipment which exists between the two storage buffer facilities (as shown in figure 2). Breakdowns experienced on these systems as well as planned maintenance procedures carried out also pose a threat to the underground operation. Hypothetically, this situation may indicate that the 6000 ton storage facility may be insufficient for resolving systems failure since when such events occur, the storage facility is most likely to reach full capacity before the breakdown/problem is resolved either at the Plant (the systems after the 2000 ton storage facility), overland conveyor or at the systems existing between the two buffer storage facilities, hence production of all eight sections comes to a complete standstill.

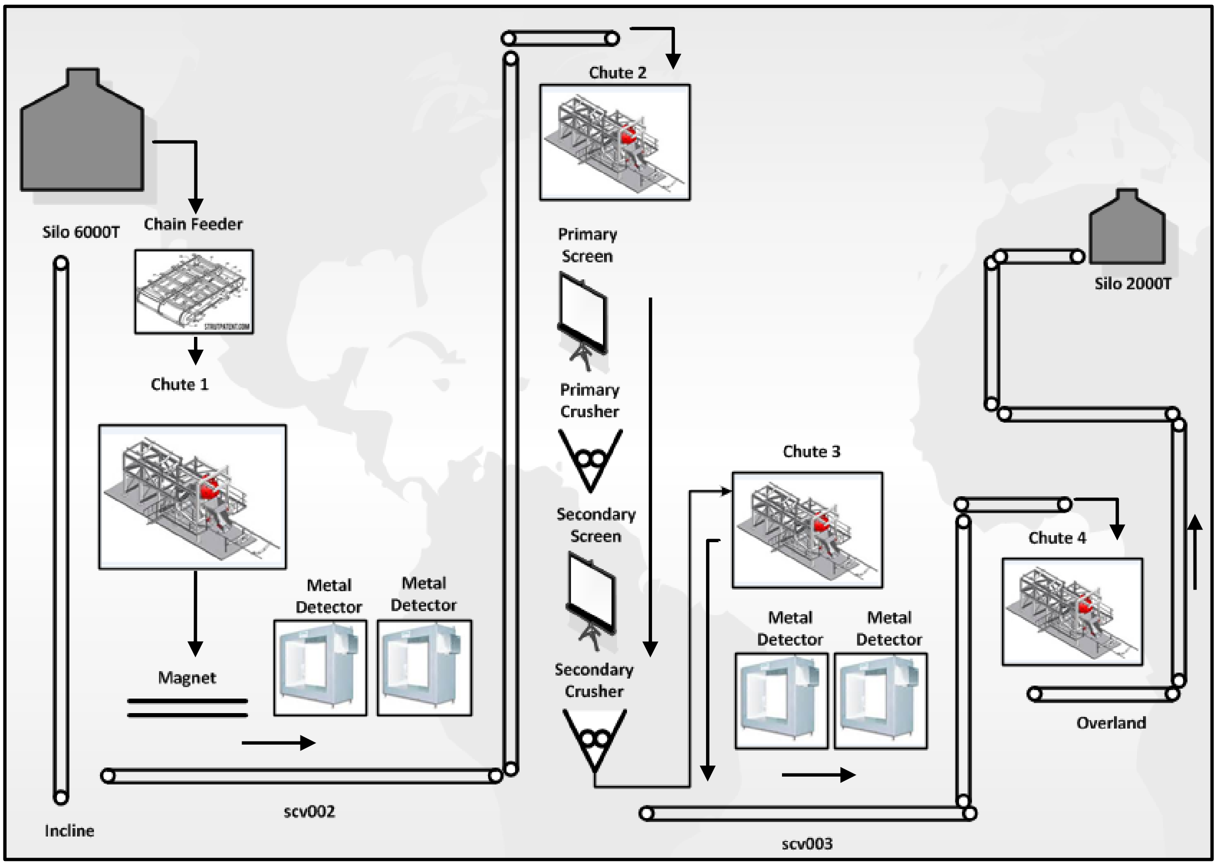


Figure 2. Exploded view of mechanical systems between the two silos (6000T and the 2000T)

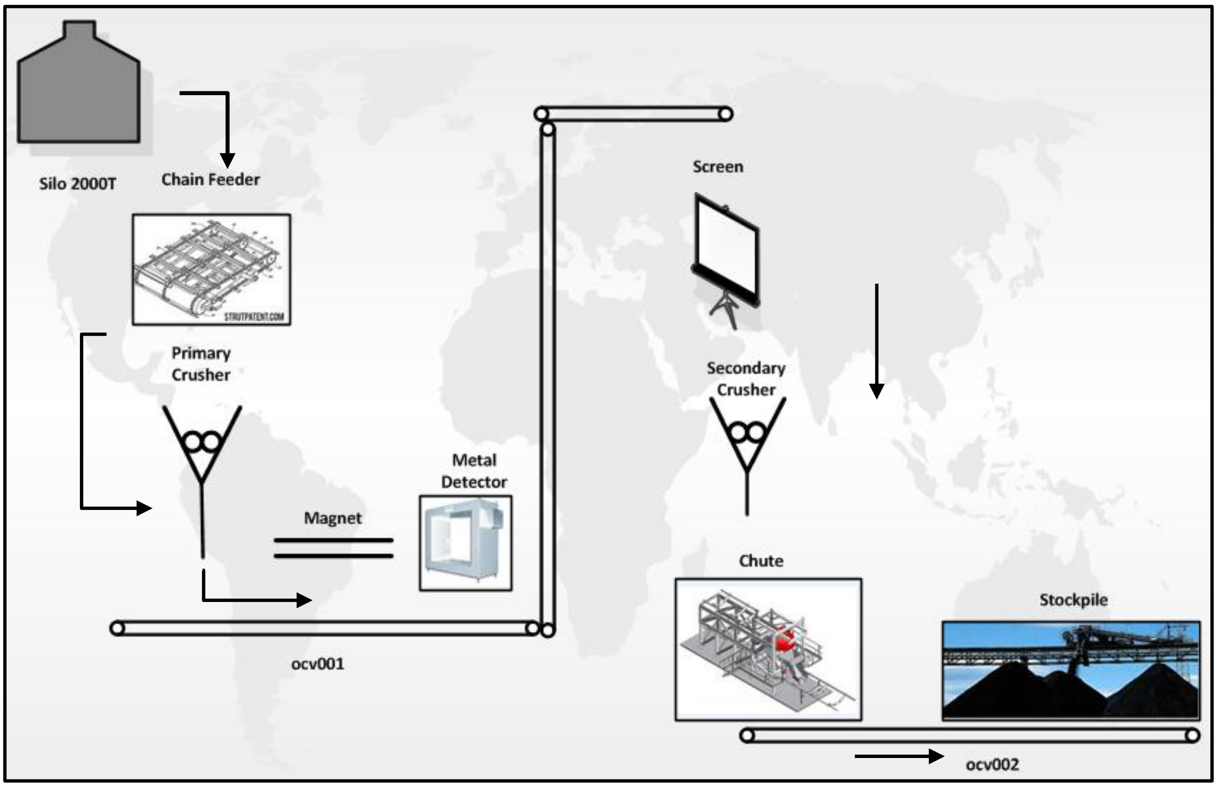


Figure 3. Exploded view of mechanical systems between the 2000T Silo and the Plant

- As was mentioned earlier, the failure of one component/conveyor in the underground operation leads to the failure of all systems preceding that component and ultimately a section/s will need to be halted on production (depending on where the breakdown occurred) until that component can be attended to. It has occurred to management that preventative maintenance procedures aren't sufficient to carry all breakdowns; therefore system improvement measures are sought to resolve this problem.

The two problems highlighted above show that continuous improvement measures are required in order to improve the productivity and ultimately profitability of the company with the said constraints in mind. It should also be noted that the problems posed, moreover the first problem, is not a "lack of capacity" problem on the Plant side, but rather, a problem of excessive discontinuity in the flow of commodity. It is evident that an ad hoc solution will need to be developed as it is a unique problem in the mining environment. Also, for brevity and clarity purposes, the contents of this report, with referral to the data analysis section, will be divided in two sections, the first section addressing the surface problems and the second section addressing the underground problems noted above.

Note: the two words 'breakdown' and 'downtime' will be used interchangeably from here forth as essentially 'downtime' is the duration of a breakdown.

3. Data analysis – problem statement

3.1. Part I: On surface

In order to be able to analyse the effects of breakdowns that the systems on surface have on the mine's production, it is important to first analyse the method in which the mine classifies certain breakdowns and other time consuming activities which affect production. The mine employs a Total Availability Model (Appendix B) in an effort to determine; on a typical day or month, how much time was available for production, and of the total production time, how much was lost due to uncontrollable circumstances. Breakdowns (downtimes) that occurred over a specific period are categorised as either engineering or production related downtimes for tracking, control and resolutory purposes. The model also takes into account consequential lost time due to the serial dependence of the system as a whole and production delay times. Its use effectively draws attention to 'controllable time' segments that occupy the largest percentage of total time. Thus, it serves as a tool or input for effecting change or designing improvement measures that aim at continuously reducing controllable time by simultaneously increasing production time. To further simplify the process, a systems approach is considered by dividing the surface into three significant areas of concern, that is,

1. The crushers and chutes system. This system comprises of all mechanical equipment that exists between the 6000T storage facility and the overland (figure 2). A holistic approach will be considered for data analysis purposes.
2. The 16.4km overland conveyor connecting the crusher and chute system to the 2000T storage facility.
3. The Plant's mechanical systems, existing between the 2000T storage facility and the stockpile facility (figure 3). A holistic approach will be considered here to for data analysis purposes.

A Pareto analysis, also known as the 80-20 rule will be used in the analysis of data. According to Cervone (2009:77), this method not only applies to the economic distribution of a country but can also be used to determine which problems in a system are in need of being resolved. It assumes that 20 per cent of the causes in a system carry an 80 per cent overall effect on the system. Therefore in this respect, this analysis technique seeks to find and emphasize that 20 per cent which causes 80 per cent of the problems. This tool allows for management to devise a strategy of placing the necessary measures and controls in place to continuously reduce or resolve that 20 per cent thus alleviating the 80 per cent effect. This technique is represented through a chart with the problems ranked in descending order

In the data analysis below, downstream downtime factors were considered i.e. Plant, overland conveyor, and crusher and chute downtimes, as a collective over a period of 1 year (2012)

Table 1. Downtimes with specific reference to the Plant, overland conveyor belt, and crusher and chute system

Downtime code	Downtime (hrs)	Cumulative Frequency
LEPL	909.79	48%
LOPL	536.43	77%
INST	103.76	82%
OBLO	78.79	86%
OREP	73.91	90%
DONO	65.47	94%
ELEC	43.93	96%
MECH	37.01	98%
OFRN	21.18	99%
BMAK	5.46	99%
HYDR	4.75	100%
DOAC	3.25	100%
DOWS	2.78	100%
OROC	1.47	100%
DODP	0.93	100%
DOME	0	
Total	**1888.91**	

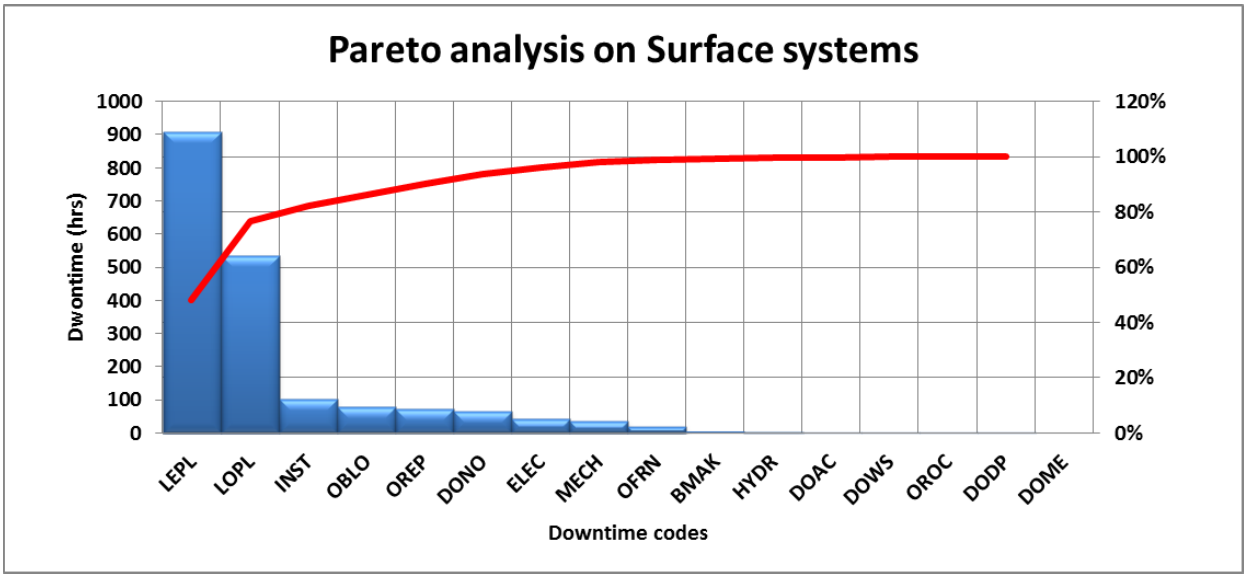


Figure 4. Pareto analysis on Plant, overland conveyor and crusher and chute system downtimes

The following should be noted from the above Pareto analysis:

- The following codes LEPL, LOPL and INST (as per appended Total Availability Model) are consequential engineering, production and instrumentation downtimes respectively as a result of the Plant. 'Consequential' in this context is sequence related as per serial dependence of the conveyor belt system

- All three downtime codes mark the top 20 per cent of total downtime causes that carry an 80 per cent impact on production.
- The 20 per cent 'causes' mark the vital few for which improvement and control measures need to be initiated to reduce their effects on production.

In order to expose the total losses that the mine experienced from the previous year (2012) also to highlight exactly how much of an effect this problem has had on the mine's financial standings, additional data was gathered reflecting the number of times the mine had to go on a complete stop i.e. all 8 sections had to stop production as a result of breakdowns experienced by the Plant. In the computations of lost profit (see table below), considerations were made for:

- Export and domestic quality coal (Middlings) according to their apportioned ratios (63.95% of the total Run Off Mine tons produced are allocated to export quality coal, 16.06% to Eskom and 19.99% is discard (poor quality coal))
- Direct Operating Hours (DOH), these are the allocated hours for which production was expected to be made.
- The average production rates of each section during the months in which the instances of complete underground production stoppages occurred.
- Sale per ton figures for the relative months with considerations made for the exchange rate for export quality coal.(see Appendix C)
- Expenses incurred for rail, warfage, FOB (Free On Board) selling expenses as well as cash costs (Appendix C)

Table 2. Profit Loss for underground production halt for the year 2012

Lost ROM	11877.529
Lost ROM :Export	7595.6801
:Middlings	1907.5312
Revenue:	**R 5 635 826**
Export	5319862.4
Middlings	315963.47
Expenses:	**R 3 726 340**
Export	3273206.4
Middlings	453134.04
Total Profit Loss	**R 1 909 485**

In the event that production is started up again after complete system shut down, the graph below, based on data analysis, typically represents the average production rate of the mine as monitored from the incline. Production is considered to be stable at a rate ranging between 2357 t/hr. and 2800 t/hr. in any given 9 hour shift, in light of system production recovery, it takes an average of 2.3 hours to reach system stability. The graph shows that the loss curve shown in the red in the event of system start-up, follows a parabolic function, thus the opportunity tons lost can be computed from this curve as follows:

Opportunity Tons Lost = system stability production rate*time taken to reach stability- $\int \frac{1}{2} \int_0^{2.3} ax^2\, dx$ where $a = 874.24$

Considerations similar to those stated above were taken into account in the computation of lost profit as well as the average DOH expressed as a percentage and the total number of shutdown occurrences (17 instances). From this a profit loss of R3 925 372.40 was experienced.

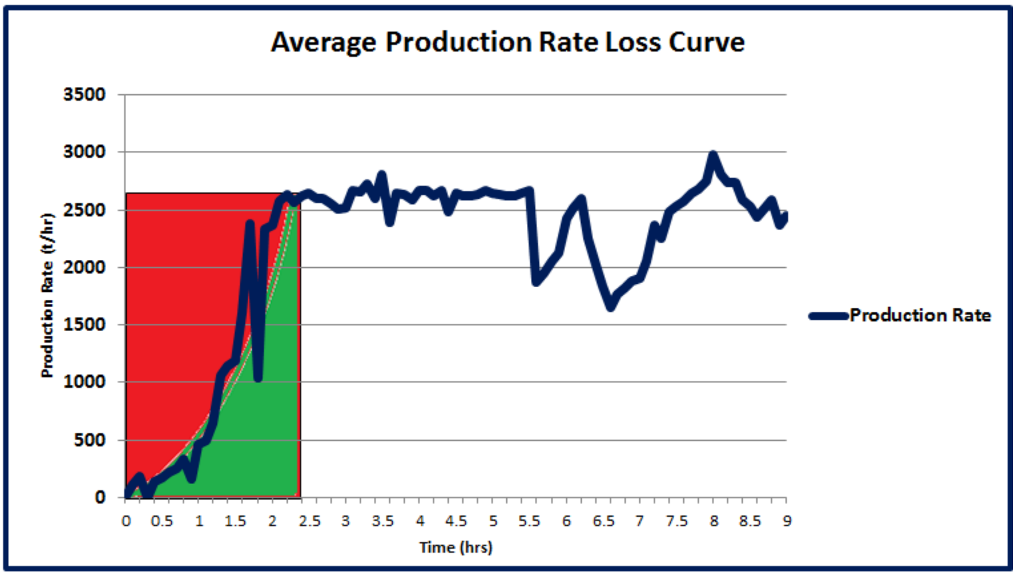


Figure 5. Average production rate loss in the event of system start-up

The following is an anecdote expressing just how far and wide this problem stretches and the consequences accompanying the risk of complete system shut down:

> In 2011, the stacker and reclaimer at the Plant collapsed. This event resulted in 8 days of production lost equivalent to 24 shifts. From this unfortunate turn of events, the mine lost a staggering R11 146 200 in profits. In addition to this, there were rollover effects. For a period of 5 months thereafter the mine continued to lose an average of 3 production days per month, reflecting an additional R18 150 000 profit loss. In its entirety, just from one instance of a breakdown at the Plant, the mine lost R29 296 200 in profits.

It should once again be reiterated that in the case of these surface system failures; with specific reference to Plant breakdowns, management cannot seek to effect any changes at the Plant as their managing power is restricted to the mine processes and not those of the Plant and that the Plant is shared between Zibulo Colliery and BHP Billiton thus the time that is made available to the colliery should be utilised effectively with minimal setbacks.

3.2. Part II: Underground

Once again the Pareto analysis tool was employed to determine which causal factors are considered 'the vital few' and which are considered 'the trivial many.' With the use of the Total Availability Model as appended, the following were the results:

Table 3. Downtimes on all underground belts

Underground conveyor downtimes for 2012		
Downtime Code	Downtime (hrs)	
OREP	618.22	29%
ELEC	438.12	21%
INST	342.88	16%
OBLO	184.45	9%
MECH	180.4	8%
DOWS	139.65	7%
XDEL	121.66	6%
DOAC	47.34	2%
DODP	25.87	1%
XDOL	12.86	1%
BMAK	12.05	1%
AART	2.25	0%
Total	2125.75	

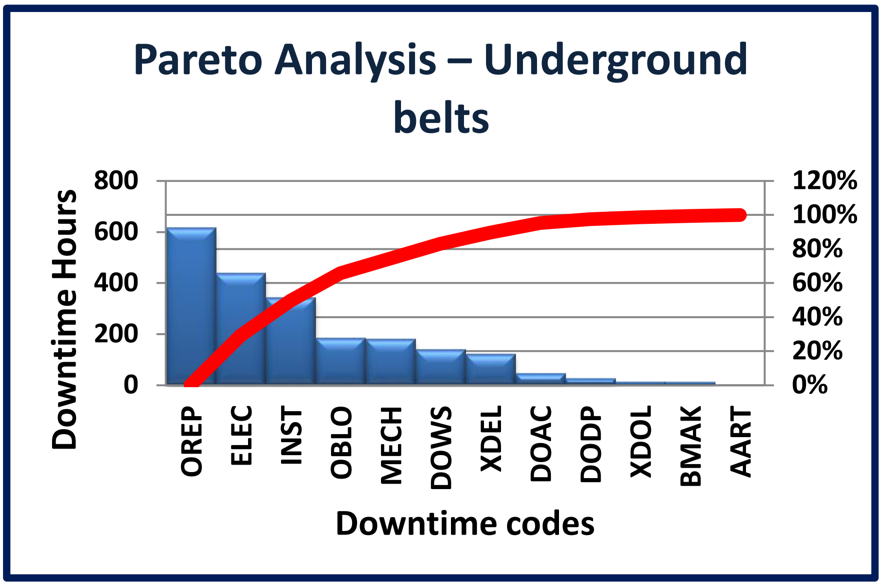


Figure 6. Pareto analysis on underground belts

The following conclusion can be drawn from the above figure:

Unplanned production stoppages, engineering downtime related to electrical, instrumentation, and mechanical issues and blockage require a further in-depth analysis into their prevalence. It is difficult to quantify these downtimes in terms of production as the quantification may not necessarily be a true reflection of the exposed losses.

From the data analysis in its entirety, it is quite evident as to why an efficient and reliable system needs to be developed. Intensive research and analysis tools and techniques are required to establish a credible solution to resolve these issues.

4. Project Aim

The aim of this project is to mitigate the risk of complete operation shut down as a result of Psurface activities as well as individual section shutdowns as a result of serially dependant conveyor systems.

- To address the first problem, the project will seek to determine the size of additional buffer capacity required to ensure that mining continues in the event of a breakdown occurring on surface. This additional capacity will be reflective of additional tonnages that the mine stands to produce and sell on such events occurring. Data analysis will be done using IE tools to determine the storage capacity required. A simulation model will be used to determine the validity of the assumptions made in this phase.
- The second phase of the project will seek to research alternative ways in which capacity can be built in the underground operation in the form of storage bunkers (required capacity) as well as optimal strategic location points of these bunkers in an effort to improve system availability, reliability and productivity. This second phase will consider a number of models and make a trade-off analysis on each to determine the best model suited for the projection and development of a realistic solution.

Therefore, on a strategic level, objectives include: to develop a “To-Be” model from the “As-Is” model and to determine and project additional tonnages that can be produced on a yearly basis considering the life-of-mine as well as a feasibility study considering project spend. These two objective will be traded off against one another in selection of the one with the highest return on investment

5. Project Approach and Scope

The Project scope, with reference to the DMAIC Model will include:

1. Gather an understanding of the mine’s business model and the importance of this model.
2. An in-depth study into the mine’s current state of operation along with the constraints that it’s exposed to with specific reference to utilisation of the conveyor system, current performance and capabilities.
3. Exploring downtimes, their classifications and distributions, and what effect they have on production coupled with probabilistic models of occurrence and consequence.

4. Determining system behaviour through the introduction of additional capacity to account for downstream breakdowns as a result of Plant operations.
5. A study into risk mitigation and analysis on underground storage bunkers sizes, and optimal strategic location points as well as an in depth study and trade-off analysis into the models that have been developed to address problems of this nature.
6. An understanding of system sensitivity and correlation of production times and production rates upon introduction of buffer capacity.
7. Recommendations, as to size additional capacity required to account for downstream breakdowns as well as additional capacity required and strategic positioning of storage bunkers for the underground conveyor system which will result in a global optimum model.

6. Deliverables

1. A trade-off analysis on different models from literature and selection of a model which will best represent the current and proposed situation of the mine.
2. An "As-Is" model of the mine along with a detailed analysis of results
3. A risk analysis of the current state of the mine.
4. A proposed/ "To-Be" model of the mine with detailed results.
5. Projections of Return on Investment on project spend should the project be implemented considering life-of-mine. (project payback period)

7. Literature Review

In an underground coal mine, an outbye haulage system, which consists of conveyor belts that are serially configured, exists which have the capability of increasing the availability of the system if carefully planned, designed and maintained. Baral et al. (1987) believe that a technique exists which is set to reduce the serial dependence of one conveyor upon another. This technique involves the installation and strategic placement of bunkers along the haulage route, enabling the production process to carry on even in the event of a belt failing.

Baral, et al (1987) also highlights the fact that although the implementation of storage facilities such as bunkers underground can improve system availability, production capability may not be reached if the capacities of the bunkers are not correct or if they are located inappropriately. It is thus on this basis that this literature review seeks to discuss and compare different modelling techniques for bunker location and sizing within the mining environment.

Bunkering activity in coal mines may be used for a number of purposes. Not only is the bunker/silo capacity dependant on the flow rate of coal into the bunker or silo but also the discharge rate. Wang (1998) elaborates that the objective of introducing bunkering activity into a system is to keep spillage to a minimal, improve system availability, and maintain an economically feasible system. Wang (1998) also further iterates that regardless of the application of the bunker, that is, be it for protection, segregation or surge purposes, there exists a fundamental rule of flow rates that must be adhered to:

Equation 1. Fundamental rule of flow rates

$$Qin - flow\ < Qout - flow$$

Generally, as stated above, bunkers have three types of employment:

- They reduce surge by decreasing the belt capacity requirements thus making flow smooth. This type of application mostly addresses the situation of spillage or the likelihood of overloading the belts beyond their specified design limits. (See figure 7 below)
- The second application is used to enhance availability of the belt system; this is where the bunkering activity is employed for buffering purposes. This application is common when belts downstream from the bunker have failed to operate and one wishes to keep the in-bye belts feeding the bunker in operation. (Figure 8). Should belt 4 fail, belts 1 ,2 and 3 can continue operating for a time dependant on the capacity of the bunker

- Bunkers are also employed for segregating loads that come from two or more loading points. (Figure 9). Alternating flow between bunker A and B to keep coal from belt 1 and 2 separate from that of belt 3 and 4. This method could be evident in the production of different qualities of coal.

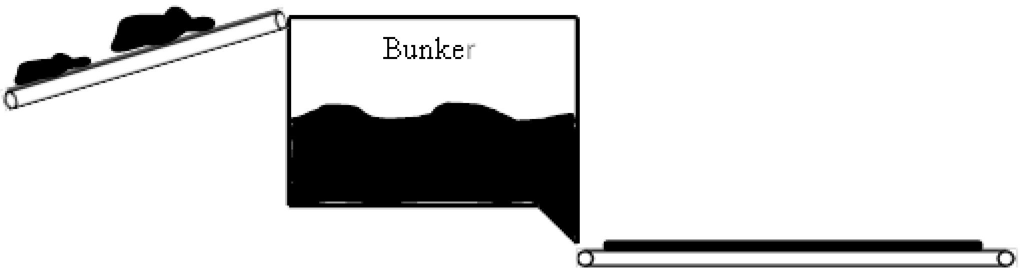


Figure 7. Bunker used for surge

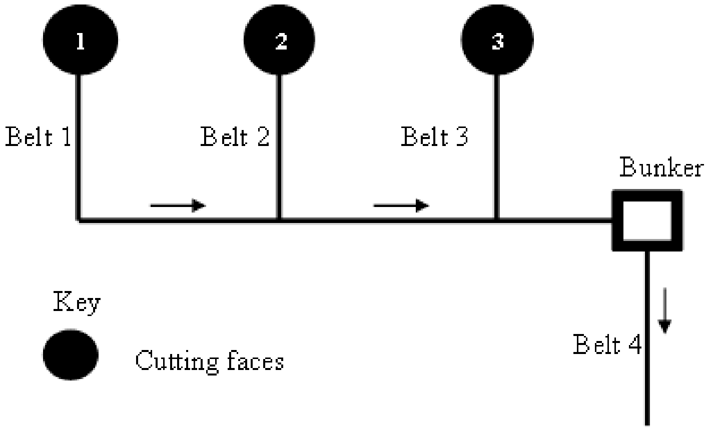


Figure 8. Bunker for buffering

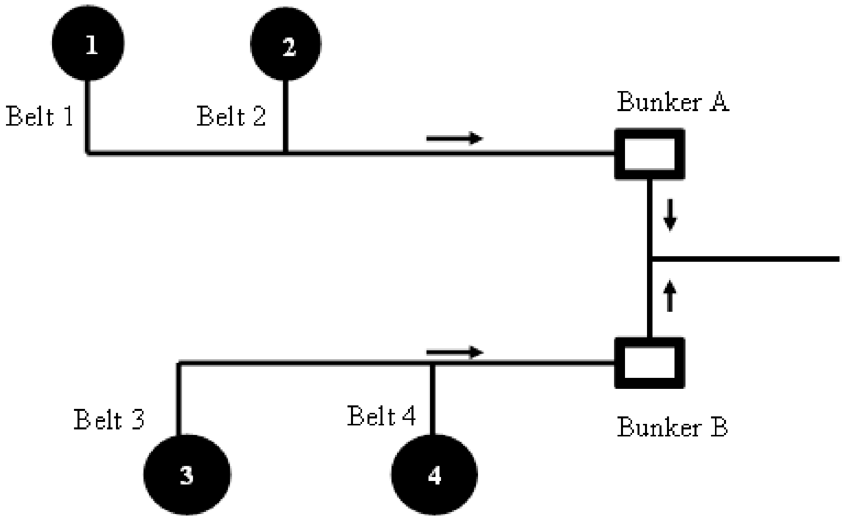


Figure 9. Bunkering for segregating flow

Wang (1998), further highlights that whichever method is used, the objective should be to keep spillage and costs to a minimum, improve on production rates as well as system availability.

For the purpose of this report, specific focus will be placed on bunkering used for buffering purposes to address the issues at hand. A number of models will be discussed and evaluated; these will include analytical models, empirical models, and simulation modelling for determining the capacity of bunkers.

7.1 Bunker sizing

Over the years empirical, stochastic and simulation models have been developed for sizing bunkers. Each of these will be discussed thoroughly in the following section and a suitable method of modelling will be selected based on model requirements, constraints and capabilities.

7.1.1 Empirical Modelling

Numerous of empirical models have been created over the years for sizing bunkers. These models are based on experimental data and some tend to have very little theoretical inference.

Sevim Model

Sevim (1987) derived a method assuming a heuristic approach to bunkering activity. He explained the model in three fundamental steps for bunker sizing. In the first step he proposed that the coal face operation can be explained in four fundamental stages, namely:

1. The commencement of a 9 hour shift
2. The actual mining activity of coal
3. Downtimes due to machine breakdowns
4. Tramming (the act of finishing cutting a face and moving onto the next face)

In this method, he devised that coal flow follows a semi-Markovian process and that one could generate a transition probability matrix for which the process could move through the various states mentioned above based on the transition matrix. There is also a time dependant variable associated with each probability which allocates the time in which the system will spend in one state before it can move to the next state.

The figure below represents the four stages. Once the system leaves state 1 to state 2 it is highly unlikely that it will return to state 1 until the commencement of the next shift. From state two the system may either move to state 3 or 4. This move is dependent on whether or not a breakdown has occurred or if a new coal face has to be cut and equipment repositioning needs to be made. Regardless of whether the system is in state 2 or state 3 it will return to state 2, which is the mining state.

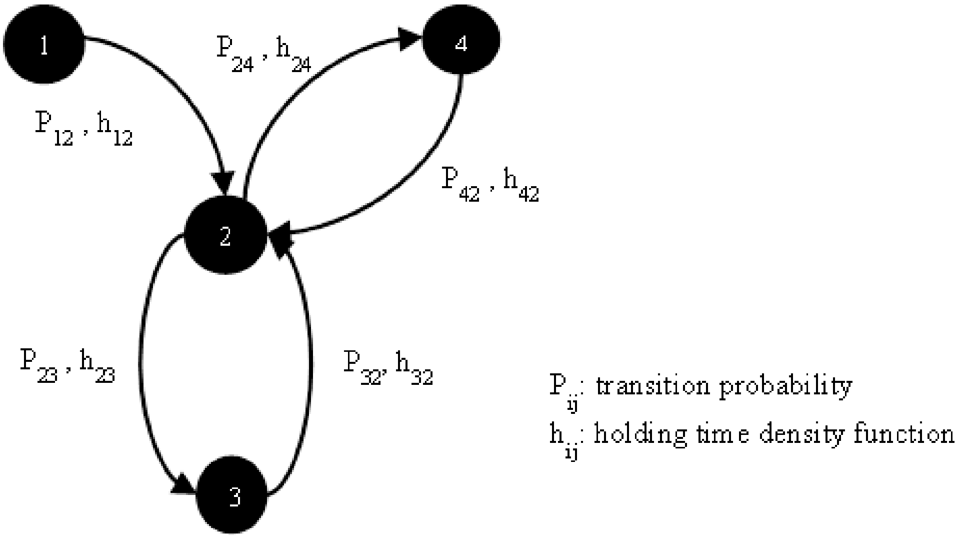


Figure 10. Transition between various states

The values for the transition probabilities and holding time density functions are gained from experimental published data on machine availability and breakdowns. In the application of room and pillar mining (continuous mining), the transition matrix is given the following manner:

h_{ij}: time density function of staying in state *i* before moving to state *j*.

h_{12} = exponential density function where α = 15 and β = 30

The exponential density function has the following expression:

Equation 2. Exponential density function (Sevim, 1987)

$$f_x(t) = (1/\beta)e^{-(t-\alpha)/\beta},$$

h_{23} = Weibull distribution with α = 0.0, β = 3.76 and γ = 2.0

The Weibull density function is expressed as follows:

Equation 3. Weibull density function (Sevim, 1987)

$$f_x(t) = \frac{\gamma}{\beta}\left(\frac{t-\alpha}{\beta}\right)^{\gamma-1} e^{-\left(\frac{t-\alpha}{\beta}\right)^{\gamma}},$$

h_{42} = Weibull distribution where α = 25.0, β = 60.0 and γ = 3.0

P_{ij} : Probability of transitioning from *i* to *j*

$P_{12} = 1.0$, $P_{23} = 0.83$, $P_{24} = 0.17$, $P_{32} = 1.0$, $P_{42} = 1.0$ and $P_{ij} = 0$ for all other transitions

From here forth, one needs to determine the liklihood that the process will be in state *j* at time t on condition that it entered state *i* at time 0. This variable is given the symbol $\varphi_{ij}(t)$. This transition probability determines when there is coal loaded on a belt and when there isn't in the following fashion: states 1, 3 and 4 are the states which represent no coal mining, therefore, in those states it is expected that there won't be coal flow on the section belts.

$\varphi_{no\text{-}coal}(t) = \varphi_{11}(t) + \varphi_{13}(t) + \varphi_{14}(t)$ where t ranges from 0 to 240 minutes for a typical half shift. Sevim considered a half shift of 240 minutes for the following reasons:

- To represent a continuous flow pattern with minimal discontinuities for analytical purposes.
- Generally, after 240 minutes employees would take a 30 minute tea break, splitting the shift into two. Sevim (1987) believes that the second half shift would bear similarities to the first with regards to moving through the various states, except with differing variables.

Sevim further analysed the process in the following technique:

The probability of there being coal on a belt from a single loading source can be reduced to the following expression:

$\varphi_{coal}(t) = \varphi_{12}(t)$

The results of the above expressions can be shown in the following figure:

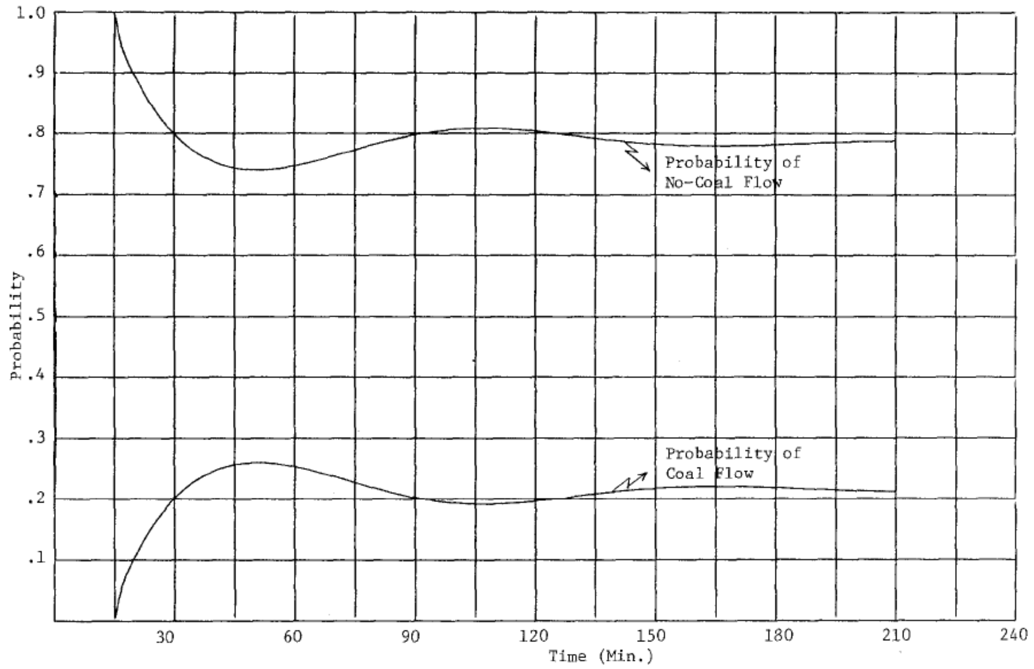


Figure 11. Probability profiles of coal, no-coal flows from a room and pillar face (Sevim, 1987)

The following is deduced from the above graph:

- The first 15 minutes of a shift are allocated to shift change; 30 minutes are reserved for a tea break after the 210th minute.
- Production begins to stabilise after 150 minutes of a half shift and the probability of there being no coal on a belt (at certain points in time) as the system reaches stability is 78% and that of there being coal on the belt is 22%. This is due to the temporality of coal flow.

In step two Sevim defines the relationship when coal merges from different faces. The belts from these different faces merge onto one gathering belt and so do their respective probabilities. For instance, taking into consideration two different cutting faces, room and pillar face and long-wall face, each following different cutting cycle distributions, four flows can be seen, with each flow having its own pdf (figure 12). Namely, these flows are:

- No flow
- Coal flow from the long-wall operation
- Coal flow from room and pillar
- Coal flow from both long-wall and room and pillar

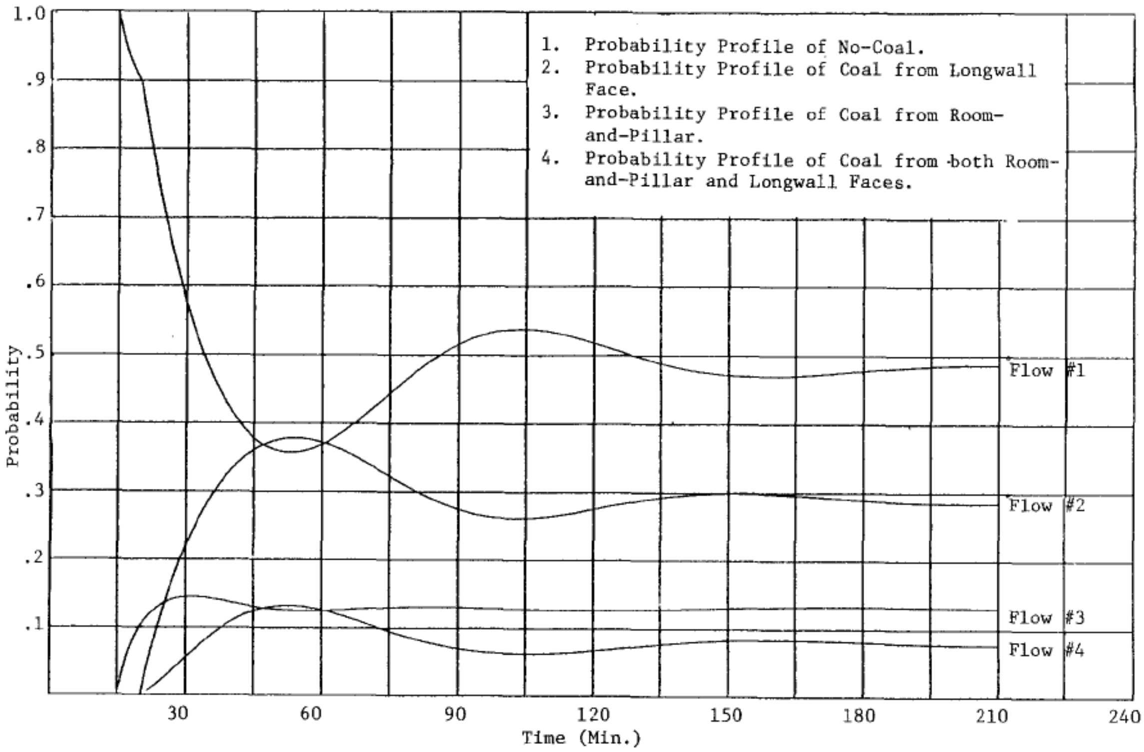


Figure 12. Probability profiles of the four different flows merging onto one belt (Sevim, 1987)

The above figure considers probability profiles of coal flow on a single gathering belt. It should be noted that the probability of there being no coal flow on a gathering belt (approximately 49% at system stability) is significantly less than the probability of there being no coal flow on a section belt (approximately 78% at system stability) due to the increase in the expected number of coal flows as a result of the increasing number in loading sources. The new probability of coal flow on the gathering belt can be expressed as a product in the following manner:

Equation 4. New probability profile of gathering belt (Sevim, 1987)

$$P^*(n, t) = \prod_{i=k}^{m} P(i, t)$$

Where *P*(n, t)*: the probability of the new flow *n* on the gathering belt at time *t*

P(i, t): the probability of section belt *i* at time *t*

k: first section belt

m: last section belt

To determine bunker capacity, he explains the need to firstly compute the required outflow rate of the bunker as follows:

Equation 5. Calculating the average outflow from a surge bin (Sevim, 1987)

$$\text{AVG} = \sum_{K=1}^{N} Q(k) \sum_{t=1}^{n} P(k, t)/T$$

Where

Q(k): the rate of the k-th flow in m^3

P (k, t): the probability of the k-th flow at time t (φ_{12}(t)), note: *P(k, t) = P*(n, t)*

N: the number of flows.

n: duration of active mining during a half shift (state 2).

AVG: average outflow rate of coal from the bunker (m^3/min)

T: the sum of active mining and inactive mining

Equation 5 is then used to determine the capacity of the bunker and its operating policy by determining the inflow rate and outflow rate through minute by minute examination. For instance, Sevim considered a small scale theoretical mine consisting of two long-wall and two room-and-pillar cutting faces. It would therefore be assumed that the gathering belt would carry 16 different coal flows (2^4) which would have to tip into the bunker (bin), but however, if it is assumed that the two room and pillar faces are similar in cycle distribution and that the two long-wall cutting faces are also similar (follow identical distributions), the loads could then be reduced to 9 different coal flows with different probability profiles, as can be seen in figure 13 below. From this, there will be 9 flows being discharged into the bunker, each with probability *P(k, t)*.Therefore, the first sample space will consist of 9 values. During the second minute another 9 flows will discharge into the bunker each with their own probability profiles, making the sample space increase to 81 (9*9), during the third minute the sample space would increase to 729. As can be seen, minute by minute examination would create an unimaginably large sample space, increasing exponentially with an increase in the number of cutting faces. Sevim (1987) then proposed that the sample space be redistributed over various cells. Each cell would carry a specified weight interval. The value of each cell would represent the weighted average of the sample values falling into that cell interval; however, after a number of iterations Sevim believes that the integrity of the data would be subject to increasing variations due to the growing sample space, thus precision most likely becomes a trade-off for manageable data. He further explains that the bunker capacity can be determined by examining the cumulative minute-by-minute probability distributions (see figure 14 below). For example, the contents of the bunker were assessed at different time intervals of 20, 50, 100 and 70 minutes for attaining a 90% probability of coal flow. At 90% probability, the bin content is highest at Q_{70} therefore Q_{70} would be the desired capacity of the bunker.

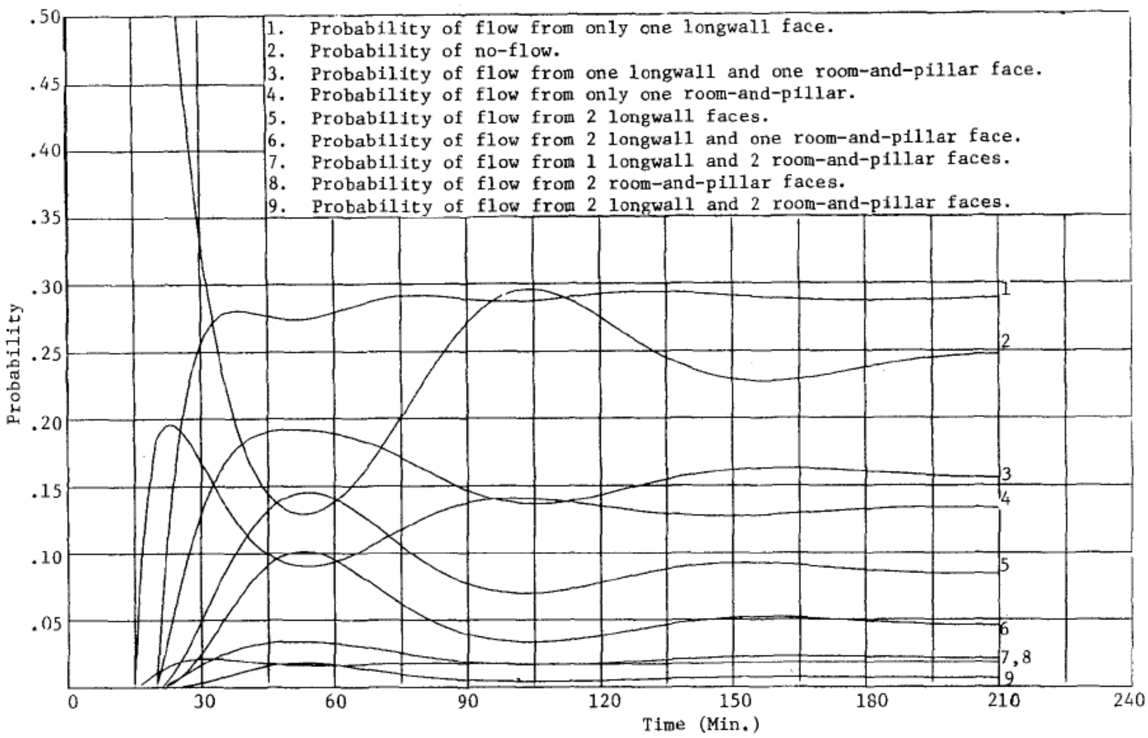


Figure 13. Probability profiles from 9 flows which merged on a single belt (Sevim, 1987)

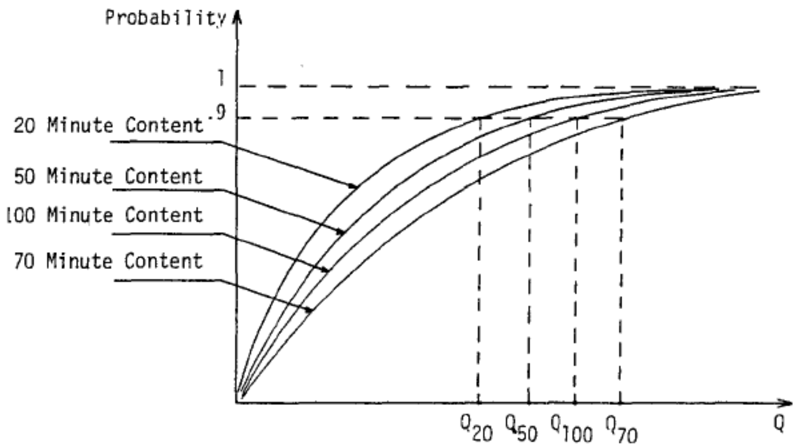


Figure 14. Cumulative probability distributions of the bunker at different times intervals (Sevim, 1987)

As per figure below, Sevim defines three operating policies of the bunker for surge purposes. When β is reached, it is proposed that the discharge rate be increased to (1+α)*Average outflow rate. When the contents of the bunker decreases to a value below the warning limit, the outflow will cease until the contents of the bunker increases to above the lower warning limit. β = 75% and α = 20%

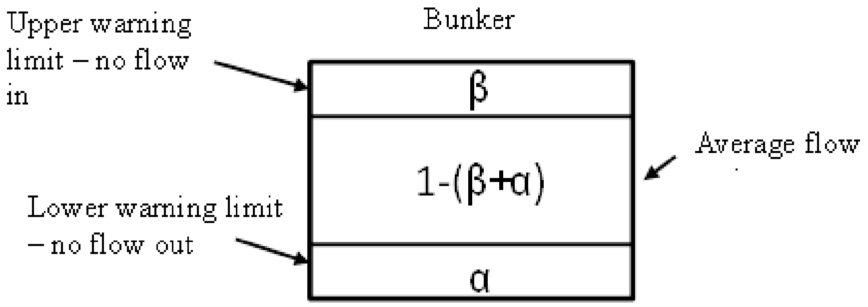


Figure 15. Operational policy of bunker according to Sevim

Flaws of the Sevim model

This model assumes that the time between transitions from one state to the next follows an exponential distribution, as a semi-markovian chain is used. This may not necessarily hold in reality. An average production rate is used for a shift; Wang (1998) believes that this measurement fails to address the temporality of coal flow nor does it address the possibility of belt failure. It should also be noted that examining the bunker contents on a minute to minute basis would create a very large sample space which may prove difficult to handle, though this can be addressed by condensing the sample space, a trade-off would have to be made against result accuracy. Furthermore, increasing the outflow rate when the bunker contents reaches β typically violates the equation on flow rates above (equation 1), also there is no sound reasoning for setting $\beta = 75\%$ and $\alpha = 20\%$

7.1.2 Stochastic Process Modelling

The Thompson-Carnahan Model

Thompson and Carnahan (1991) proposed a model for discrete loading of coal onto belts that are fed by more than one loading source (gathering belts). They based this on Khintchine's theory which suggests that a Poisson process exists for approximating superimposed loads that arrive at a bunker. From this, a probability density function (pdf) can be derived for these loads after a series of convolution sums. Carnahan, et al. (1991) further states that these pdfs can be expressed in terms of the number of cutting faces, collection time period in which the bin receives coal but doesn't discharge, and the individual pdfs from the said loading points or cutting faces. The model is designed in the following manner:

K independent loading sources are considered, with each loading source fashioned in a discrete manner. For analytical purposes, a single loading source is considered over a time interval (0, t) with a number of discharges onto a belt from that loading source is determined through time and motion studies. Probability density functions are then developed for the time that is needed for a certain number of coal loads that need to be discharged onto the belt during that time interval. Carnahan (1991) then suggests that superimposing the individual pdfs of the loading sources leads to the convoluted pdf of loads accumulating in the bunker. Carnahan (1991) denotes the following variables:

A_i: the time interval between the offloading of the (i-1) load and arrival of the i-th load

W_i: time taken to offload onto a conveyor system

A_i and W_i are considered to be independent of one another and identically distributed following a pdf of f_A and f_w respectively. They are also defined over an interval of [A_{min}, A_{max}] and [W_{min}, W_{max}] respectively and can be approximated via a histogram. (figure16).

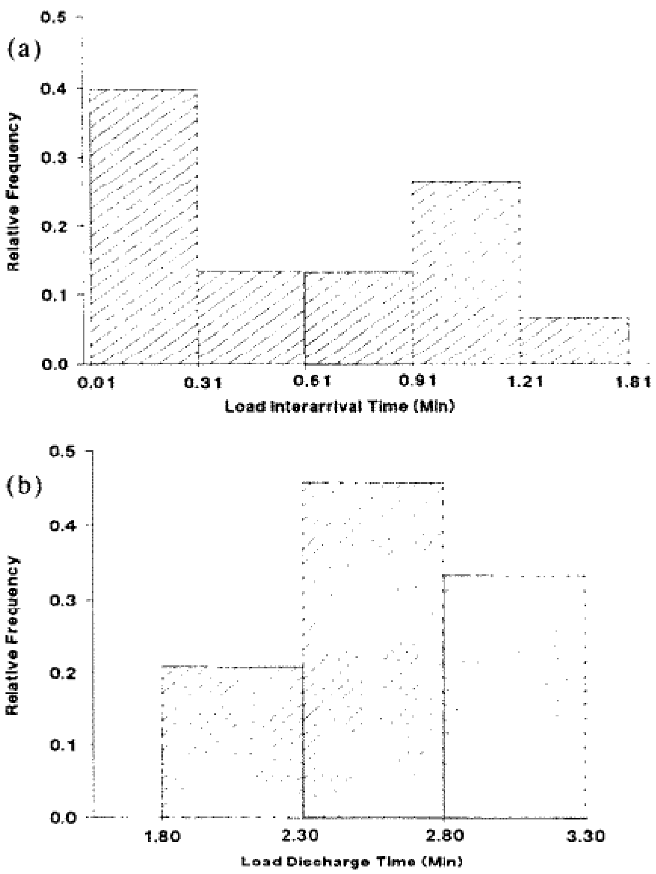


Figure 16. (a) Load inter-arrival time taken from time studies; (b) time to discharge constructed form time studies (Carnahan et al, 1991)

Carnahan (1991) further deduced that total cycle time comprised of the sum of these two random variables as:

Equation 6. Cycle time computation (Carnahan et al, 1991)

$$Xi = Ai + Wi$$

Due to the independence of the two random variables stated above, the probability density function of X_i can be deduced through the convolution of the pdfs of the two random variables, thus the pdf of X_i stands as:

Equation 7. Convoluted pdf of X_i (Carnahan et al, 1991)

$$f_X(x) = \mathrm{P}[X = x] = \sum_{r=0}^{x} f_A(x-r) f_W(r)$$

Equation 8. Cycle time constraints (Carnahan et al, 1991)

Where $X_{min} \leq x \leq X_{max}$ and $X_{min} = A_{min} + W_{min}$ and $X_{max} = A_{max} + A_{min}$

The time when offloading of the *i*-th component is completed is given by the symbol S_i . It is related to the previous cycle time as: $S_i = S_{i+1} + X_i$. Therefore, the pdf of S_i can be computed as a series of convolutions in the following manner:

Equation 9. "Pdf of the time when the discharge of the i-th load is completed" (Carnahan et al, 1991)

$$f_{S_i}(s) = \mathrm{P}[S_i = s] = \sum_{u=0}^{s} f_{S_{i-1}}(s-u) f_X(u)$$

The maximum number of arrivals from a single loading point can be given as t/X_{min} between time interval (0, *t*). To further clarify the model, Carnahan (1991) states that the density function for the total number of loads within the specified time interval from a single loading source, denoted by M_t, is given as:

Equation 10. Pdf for the number of loads *i* expected within the time interval range of (0, t) from a single loading source (Carnahan et al, 1991)

$$\begin{aligned} \mathrm{P}[M_t < i] &= \mathrm{P}[S_i > t] \\ &= 1 - F_{S_i}(t) \\ &= 1 - \sum_{s=0}^{t} f_{S_i}(s) \end{aligned}$$

$F_{si}(t)$ is the cumulative distribution function for S_i with its pdf given above (equation 7). Through substitution and power of elimination the pdf given in equation 8 simply reduces to the following notation:

Equation 11. Simplified pdf of the number of loads *i* expected within the time interval range of (0, t) (Carnahan et al, 1991)

$$
\begin{aligned}
f_{M_i} &= \mathrm{P}[M_t = i] \\
&= \mathrm{P}[i \leqslant M_t < i+1] \\
&= F_{S_i}(t) - F_{S_{i+1}}(t).
\end{aligned}
$$

Since this expression forms the pdf for the number of loads that can be discharged form a single loading source, Carnahan (1991) believes that an expression can be deduced to determine the number of loads expected within a said time interval *t* from *K* loading sources. In a more simplistic format, a normal approximation can be made which can aid in the determination of the bunker capacity in the following manner:

Let *Ci* = the size of the loads which are independent and identically distributed random variables

Mkt = Number of loads which can accumulate in a time interval t from k loading sources

Q = size of the load that the bunker is expected to accommodate in tons

$$Q = \sum_{i=1}^{Mkt} Ci$$

Flaws of the Thompson and Carnahan model

Though this model is fairly simple and quick to use, it is not entirely representative of a real life mining situation as it does not differentiate between different types of loading source distributions. Assuming that all loading sources follow a similar distribution may lead one to incorrect results. On a more pressing note, Wang (1998) believes that this model doesn't account for delays or interruptions in loading and although coal flow is discrete, belt failures are not accounted for in Carnahan's model. Wang further highlights that the time it takes to determine the accurate size of the bunker would depend entirely on experience.

The Baral-Daganzo Model

Baral, et al (1987) also developed a method for determining bunker sizes in an underground coal operation. In this approach, his main objective was optimising the availability of the haulage system through the assumption that only one production section was active with a number of serially cascaded belts which assumed an exponential distribution for the time between failures and time to repair of the belts. Also, Baral assumed that the belts operated independent of one another. In his approach, he modelled the availability of the system through the following formula which he would later modify by introducing a bunker:

Equation 12. Availability of a belt system (Baral et al, 1987)

$$A = \frac{1}{1 + \sum_{i=1}^{i=n} \rho_i}$$

A = System availability

Where,

$\rho = \frac{\lambda i}{\mu i}$

$\mu_i = \frac{1}{MTTRi}$ repair rate of conveyor *i*

$\lambda_i = \frac{1}{MTBFi}$ failure rate of conveyor *i*

$MTTR_i =$ mean time to repair conveyor *i*

$MTTF_i =$ mean time to failure of conveyor *i*

In modelling the system, Baral (1987) believes that a system such as the one in figure 17 can be simplified to just a system with a conveyor on either side of the bunker (shown in figure 18) on condition that the repair rates of all the conveyors on either side are similar.

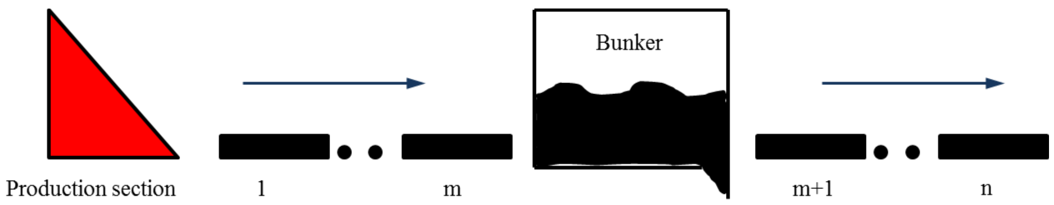


Figure 17. System with n conveyors

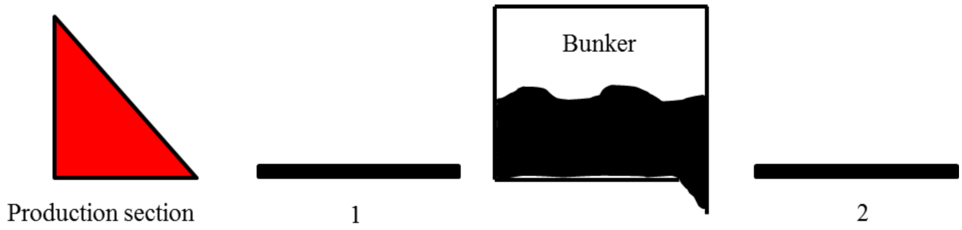


Figure 18. Simplified system with two conveyors

In the analysis, two conveyors are considered first with a system availability expression of:

Equation 13. System availability expression for a two system conveyor (Baral et al, 1987)

$$A_{1,2} = \frac{1}{1+\rho 1+\rho 2}$$

The introduction of a bunker with capacity V between the conveyors would increase the availability of the system. The new availability expression would be expressed as a function of the bunker capacity. Baral (1987) denotes that the volume of the bunker at any time *t* is given the random variable X_t . He also defines two states of any given conveyor. The first state termed the 'up' state; this state is when the conveyor is in operation or is capable of being in operation. The 'down' state is the state when the conveyor has failed and thus cannot operate. The states are given the random variable $C_{i,\,t}$ which is allocated the values of either one or zero. This depends on whether the conveyor is up or down during a certain time *t*. C_i would be the notation given for conveyor *i*. Baral et al (1987) further assigns a state variable for the system, given by S_t, which would be dependent on the values for $C_{1,\,t}$, $C_{2,\,t}$ and X_t. All conveyors have capacities of Q tons per minute.

Following the above assignments, the system is said to comprise of eight states noted as:

$S_t = 1 = C_1$ up, C_2 up, $0 < X_t < 1$; $(0, 0, x)$

$S_t = 2 = C_1$ up, C_2 down, $0 < X_t < 1$; $(0, 1, x)$

$S_t = 3 = C_1$ down, C_2 *up*, $0 < X_t < 1$; $(1, 0, x)$

$S_t = 4 = C_1$ down, C_2 down, $0 < X_t < 1$: $(1, 1, x)$

$S_t = 5 = C_1$ up, C_2 up, $X_t = 1$; $(0, 0, 1)$

$S_t = 6 = C_1$ up, C_2 down, $X_t = 1$; $(0, 1, 1)$ (input subsystem shut down)

$S_t = 7 = C_1$ up, C_2 up, $X_t = 0$; $(0, 0, 0)$

$S_t = 8 = C_1$ down, C_2 up, $X_t = 0$; $(1, 0, 0)$ (discharge subsystem shut down)

Figure 19. System states (Baral et al, 1987)

States (1, 0, 1) and (1, 1, 1) have been left out since they are highly improbable states, i.e. if conveyor 1 has failed, and conveyor 2 is operational, the bunker cannot be assigned a value of 1 as it cannot be operational in this state, the like for (1, 1, 1). A diagram showing all possible transitions between the states is shown below.

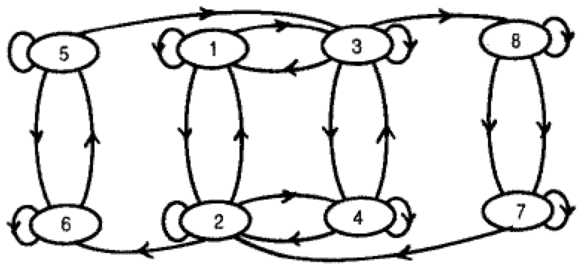


Figure 20. The different possible transition states that the model can assume (Baral et al, 1987)

From this transition diagram, probability function can be derived as follows:

Equation 14. Derived states probability function (Baral et al, 1987)

$$
\begin{aligned}
P_i(t,x) &= \lim_{\delta x \to 0} \frac{1}{\delta x} Pr\{S_t = i, x \le X_t \le x + \delta x\} \\
&= \lim_{\delta x \to 0} \frac{Pr(S_t = i,\ X_t \le x + \delta x) - Pr(S_t = i, X_t \le x)}{\delta x} \\
&= \frac{\mathrm{d}}{\mathrm{d}x} [Pr(S_t = i, X_t \le x)]
\end{aligned}
$$

$P_i\ (t, x)\ dx$ probability of the system being in state i given that $x \le x \le\ x+dx$. (Baral et al, 1987).

Pr, probability that $0 \le x \le 1$; $t \ge 0$; and $i = 1, 2, 3, 4$. (Baral et al, 1987).

The above function reduces to

Equation 15. (Baral et al, 1987)

$$\int_0^1 P_i(t,x)\mathrm{d}x = P_i(t)$$

As the system reaches steady state, the probability function will become independent of time and the availability will become a function of bunker capacity (Baral et al, 1987).

Equation 16. System availability as a function of bunker capacity (Baral et al, 1987)

$$A(V) = \frac{[\rho_1 - \rho_2 \exp(-\gamma V)]}{\rho_1 A_1^{-1} - \rho_2 A_2^{-1} \exp(-\gamma V)}$$

Equation 17. Dimensionless value (Baral et al, 1987)

$$\gamma = \frac{(\mu_1 + \mu_2 + \lambda_1 + \lambda_2)(\lambda_1 \mu_2 - \lambda_2 \mu_1)}{(\mu_1 + \mu_2)(\lambda_1 + \lambda_2)Q}$$

$A_i = \frac{1}{1+\rho i}$ (Availability of conveyor i)

A(V) is asymptotic in nature, therefore it can be seen from the figure below (figure 21), that the more one increases the volume of the bunker, the less benefit there is to gain from it, thus the optimum capacity of the bunker can be seen to be the asymptotic point where the system availability starts to stabilise.

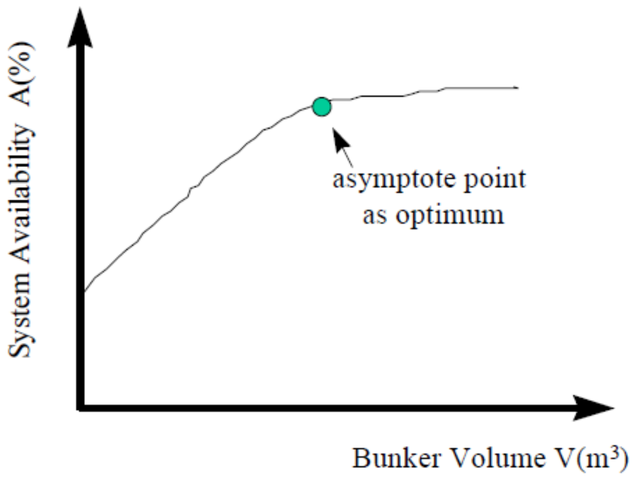


Figure 21. Relationship chart between A and V (Baral et al, 1987)

Flaws of the Baral – Daganzo Model

A number of assumptions have been made in the generation of this model. Evidently, assuming that the time between repairs and the average time to failure of a component follows an exponential distribution may be flawed. This may not necessarily hold as it will depend on the analysis of data. Additionally, this model only assumes one loading source. Though alterations may be made in this regard, the model is still based on the inclination that all belts have equal carrying capacity, thus neglecting the concept of "gathering belts" which require larger capacities than normal belts. The model also forfeits the issue of loading variations and their distributions.

7.1.3 Discrete event simulation

Discrete event simulation is a technique that is employed for modelling conveyor and bunker capacities. This approach is used for representing an actual real problem on a computer and performing different experiments to arrive at a feasible solution. Simulation approaches are considered to be flexible and versatile. They are highly effective and well advocated when it comes to modelling complex conveyor systems. Baral, et al (1987) believes that simulation models do not give optimal solutions towards solving a problem since not all external influences and environmental impacts can be modelled. Baral further enunciates that simulation techniques should only be used where the problem is so complex that it cannot be solved by analytical techniques.

Some of the earliest simulation programs developed are BELTSIM and SIMBELTM. BELTSIM was originally designed for simulating belts and not bunkers. It was later modified to include bunker size simulation, however; these computations were based on the probability of spillage at conveyor intersections. Baral (1987) argues that although bunkers can be used to prevent spillage at belt intersections, this method is not conventional or universally practiced. Bunkers are generally used for the protection of production systems.

SIMBELTM on the other hand, generates negative exponential variables to simulate breakdowns and uptime for production. SIMBELTM only requires that one inputs the average percentage of time a day that conveyors are idle or failed. This downtime per day has different mean values and follows an exponential distribution. Sizing the capacity of the bunker is dependent on the distribution of operational time and downtime, thus a lot of deviation can be noted from this model. The results of this model therefore need to be assessed vigorously.

Simio simulation modelling allows one to model the system based on a data file. The time between failures and the time to repair are not assumed to follow exponential distributions but rather are dependent on the data analysis. It tends to give more of a visual experience which is able to mimic reality. It also allows for experimental procedures to be conducted and bunker sizing availability.

Baral,et al (1987) emphasizes that simulation modelling comes with one major disadvantage: it is a time consuming approach. It is also important to note that simulation modelling doesn’t provide one with the optimal solution, however; one can arrive at a workable solution and it is the best that one can hope for with these approaches.

7.2 Bunker Location

A number of tools and techniques will be evaluated for locating bunkers within the system. The strategic positioning of these bunkers will be so as to improve system availability, reliability and economics. Strategic bunker locationing is integral in the determination of bunker sizing as incorrect placement of bunkers can lead to exorbitant investments with no real benefit. The following approaches will be discussed: Theory of constraints, Decision tree analysis, Failure Mode and Effects Analysis (FMEA) using VIKOR and fuzzy numbers, and the Baral Technique.

7.2.1 Theory of Constraints

In order to determine where bunkers must be placed using the theory of constraints, one needs to consider critical paths from an activity network diagram. Chen (1995) elaborates the importance of finding the longest path in an activity network diagram because the arcs present in the said path denote the most critical activities. Chen (1995) further states that if the activities on the critical path are delayed then the entire project will be delayed. In the below figure, a reproduction of the conveyor belt system layout of Zibulo is shown, with the arrows indicating the consequential effects of a breakdown on surface (starting from the Incline belt) or even on any of the trunk belts. Based on the production rate of each cutting section as well as the downtime duration of belts, one can determine a critical path which would indicate where a buffer should be placed. The conveyor belt system should thus be divided into 3 portions in the determination of where to buffer or not and the activity network diagrams of each section draw.

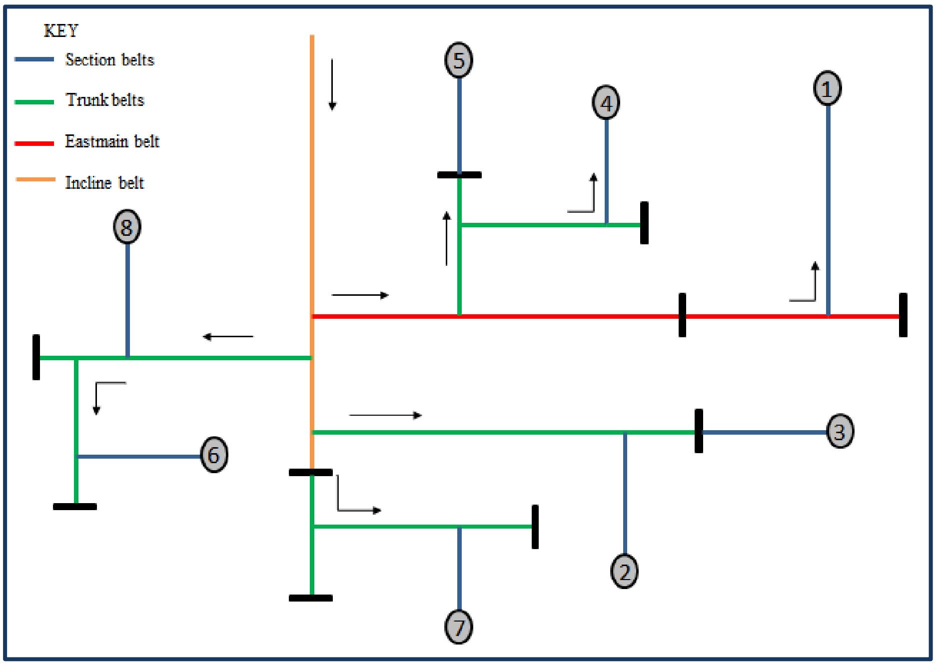


Figure 22. System failure consequential

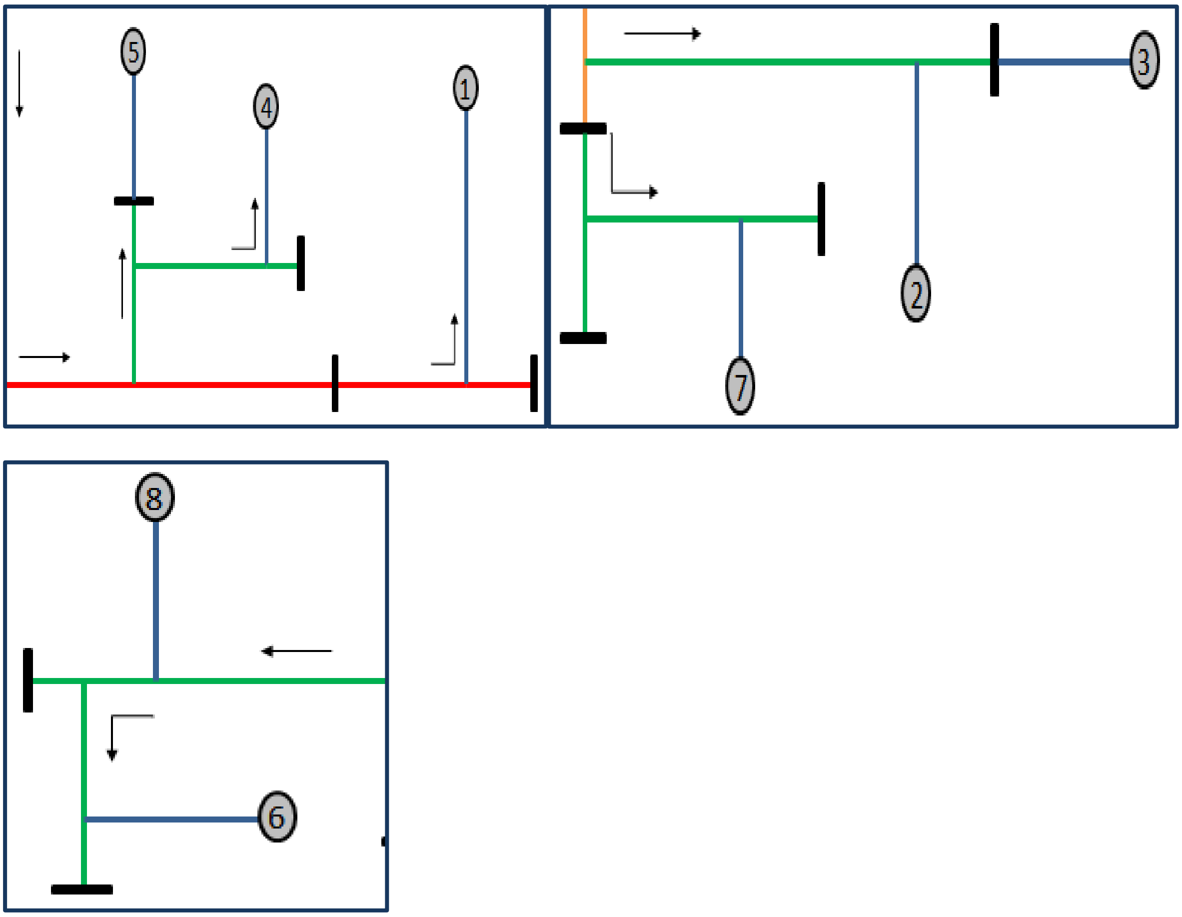


Figure 23. Conveyor system divided into 3 portions for critical path analysis

Flaws of the TOC model

Although this method may suggest where a bunker should be placed by determination of the critical path, it does not give the exact location of the bunker, but rather along which path it should be placed. This method may also suggest that, based on the 3 portions, since a critical path may exist for each one, a bunker should definitely be made present in all three portions. It fails to take into account the feasibility of the system. Also, Chen (1995) in his paper on *Critical path in an activity network with time constraints,* highlights that an activity network has a unique source as well as a unique destination, the latter not holding in this regard.

7.2.2 FMEA using VIKOR and fuzzy numbers

Failure mode and effect analysis (FMEA) using VIKOR and fuzzy numbers is a tool used for identifying and eliminating potential failures in a system. This risk assessment tool uses Risk Priority Numbers (RPNs) to assess system failure modes and can be said to be the product of probability of occurrence (O), consequence or severity (S) and likelihood of failure detection (D). According to Liu and Mao (2012), FMEA using VIKOR and fuzzy numbers was developed after much criticism was placed on the conventional method of assessing risk as the latter illustrated some important weaknesses when it came to real world applications. In the following section, the conventional method of assessing risk will be discussed, also highlighting the shortcomings of this approach. This will be followed by an introduction into how the conventional method was modified to include VIKOR and fuzzy numbers.

The conventional method

FMEA is a highly effective tool employed by management for providing them with critical information for making decisions. This tool is specifically designed to improve system reliability and enhance the safety of complex systems. In the conventional method of assessing risk, all potential failure modes of a system need to be identified and assigned risk priority numbers. To calculate the RPN of a potential failure, the three risk factors O, S and D need to be assessed and ranked using a 10 point scale (see tables 4,5 and 6), the one with the highest product ranking requiring immediate action (RPN = O*S*D).

Table 4. Conventional FMEA scale for ranking occurrence (Ford Motor Company, 1988)

Probability of failure	Possible failure rates	Rank
Extremely high: Failure almost inevitable	$\geqslant$ in 2	10
Very high	1 in 3	9
Repeated failures	1 in 8	8
High	1 in 20	7
Moderately high	1 in 80	6
Moderate	1 in 400	5
Relatively low	1 in 2000	4
Low	1 in 15,000	3
Remote	1 in 150,000	2
Nearly impossible	$\leqslant$1 in 1,500,000	1

Table 5. Conventional FMEA scale for ranking severity (Ford Motor Company, 1988)

Effect	Criteria: severity of effect	Rank
Hazardous	Failure is hazardous, and occurs without warning. It suspends operation of the system and/or involves noncompliance with government regulations	10
Serious	Failure involves hazardous outcomes and/or noncompliance with government regulations or standards	9
Extreme	Product is inoperable with loss of primary function. The system is inoperable	8
Major	Product performance is severely affected but functions. The system may not operate	7
Significant	Product performance is degraded. Comfort or convince functions may not operate	6
Moderate	Moderate effect on product performance. The product requires repair	5
Low	Small effect on product performance. The product does not require repair	4
Minor	Minor effect on product or system performance	3
Very minor	Very minor effect on product or system performance	2
None	No effect	1

Table 6. Conventional FMEA scale for ranking detection (Ford Motor Company, 1988)

Detection	Criteria: likelihood of detection by design control	Rank
Absolute uncertainty	Design control does not detect a potential cause of failure or subsequent failure mode; or there is no design control	10
Very remote	Very remote chance the design control will detect a potential cause of failure or subsequent failure mode	9
Remote	Remote chance the design control will detect a potential cause of failure or subsequent failure mode	8
Very low	Very low chance the design control will detect a potential cause of failure or subsequent failure mode	7
Low	Low chance the design control will detect a potential cause of failure or subsequent failure mode	6
Moderate	Moderate chance the design control will detect a potential cause of failure or subsequent failure mode	5
Moderately high	Moderately high chance the design control will detect a potential cause of failure or subsequent failure mode	4
High	High chance the design control will detect a potential cause of failure or subsequent failure mode	3
Very high	Very high chance the design control will detect a potential cause of failure or subsequent failure mode	2
Almost certain	Design control will almost certainty detect a potential cause of failure or subsequent failure mode	1

Based on literature by Lui, et al (2012), the conventional FMEA method has a number of shortcomings, including:

- O, S and D are assumed to carry equal weights
- Different sets of O, S and D can produce the same RPN but with totally different risk implications. This could result in high risk failure modes going unnoticed as well as investment in resolving failure modes that don’t have a significant impact or carry a large weighting factor.
- The three parameters are difficult to evaluate as they are usually vague and linguistically expressed (such as serious, likely, medium) with no mathematical background inference.
- It may be misleading to represent the results as a multiplicative operation as they are evaluated according to discrete ordinal scales of measure

In the next section, the fuzzy set theory will be introduced along with the VIKOR method to compliment the conventional method.

The fuzzy set theory

This traditional method of FMEA was later improved by Zadeh (1965) to include a fuzzy set theory. In this theory, Zadeh developed fuzzy numbers which would be representative of uncertain, imprecise, fuzzy situations. (Lui et al, 2012). It was developed to clear ambiguity in concepts, since a human being's rationale tends to be subjective when having to measure the effects of a failure. These fuzzy numbers could be likened to what a human being would rank as being "very low, medium, high, very high etc." In his literature, Zadeh describes a universe X comprising of variables $\{x_1, x_2, x_3, \ldots x_n\}$ and fuzzy set Ã. He highlights that the fuzzy set Ã is a member of universe X and is therefore given the function value symbol $\mu_{\tilde{A}}(x)$. This function value is called the grade of membership and the larger the value the stronger the relationship between the universe and the fuzzy number. This grade of membership lies between 0 and 1, represented in the following piecewise function:

Equation 18. Grade of membership piecewise function (Lui et al, 2012)

$$\mu_{\tilde{A}}(x) = \begin{cases} 0, & x < a_1, \\ \frac{x-a_1}{a_2-a_1}, & a_1 \leqslant x \leqslant a_2, \\ 1, & a_2 \leqslant x \leqslant a_3, \\ \frac{x-a_4}{a_3-a_4}, & a_3 \leqslant x \leqslant a_4, \\ 0, & x > a_4. \end{cases}$$

Lui et al (2012) also states that the most commonly used types of fuzzy numbers in both theory and practice are trapezoidal in nature as they are a more accurate form of measurement. Fuzzy number Ã contains numbers (a_1, a_2, a_3, a_4) as can be seen from figure 24.

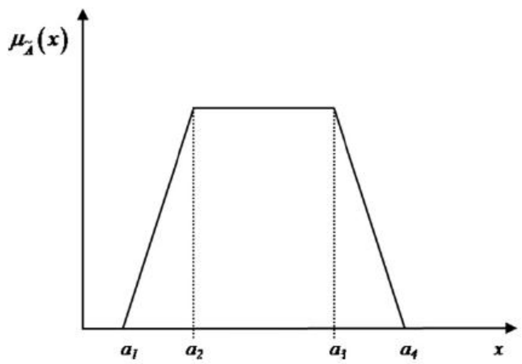


Figure 24. Trapezoidal fuzzy number A (Lui et al, 2012)

In the figure, a_1 and a_4 are called the upper and lower limits of Ã, whilst a_2 and a_3 are referred to as the mode interval. According to Lui et al (2012), “Linguistic variables are variables whose values may be expressed in linguistic terms.” These variables are extremely vague and difficult to express in a quantitative manner therefore they are represented as fuzzy numbers using the trapezoidal function. These variables are set for two purposes, that is, to weight risk factors as well as rating the failure modes of a system. Linguistic variables for weighing risks can be deduced from figure 25 according to their membership functions and converted to fuzzy numbers shown in table 7 below. The same methodology is followed in rating failure modes of a system, shown in figure 26 below, and transforming them into fuzzy numbers. (Table 8)

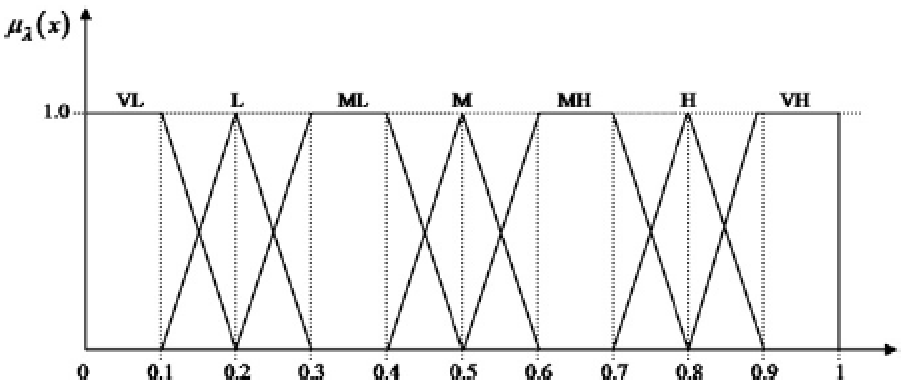


Figure 25. Membership functions for rating weights (Lui et al, 2012)

Table 7. Fuzzy numbers for rating the weights of O, S, and D (Lui et al, 2012)

Linguistic variables for rating the weights of risk factors.

Linguistic variables	Fuzzy numbers
Very low (VL)	(0,0,0.1,0.2)
Low (L)	(0.1,0.2,0.2,0.3)
Medium low (ML)	(0.2,0.3,0.4,0.5)
Medium (M)	(0.4,0.5,0.5,0.6)
Medium high (MH)	(0.5,0.6,0.7,0.8)
High (H)	(0.7,0.8,0.8,0.9)
Very high (VH)	(0.8,0.9,1,1)

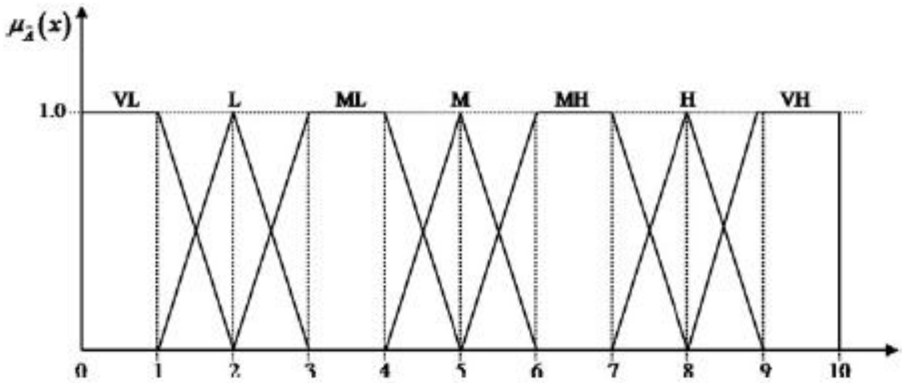


Figure 26. Membership functions for rating failure modes (Lui et al, 2012)

Table 8. Fuzzy numbers for rating the weights of risk factors (failure modes) (Lui et al, 2012)

Linguistic variables for rating the failure modes.

Linguistic variables	Fuzzy numbers
Very low (VL)	(0,0,1,2)
Low (L)	(1,2,2,3)
Medium low (ML)	(2,3,4,5)
Medium (M)	(4,5,5,6)
Medium high (MH)	(5,6,7,8)
High (H)	(7,8,8,9)
Very high (VH)	(8,9,10,10)

These fuzzy numbers are then transformed into crisp numbers using a process called 'defuzzification.' This defuzzification step utilises the centroid formula (equation 19) for trapezoidal functions for transforming a fuzzy number into a crisp value for later mathematical analysis.

Equation 19. Transformation formula (Lui et al, 2012)

$$\bar{x}_0(\tilde{A}) = \frac{1}{3}\left[a_1 + a_2 + a_3 + a_4 - \frac{a_4 a_3 - a_1 a_2}{(a_4 + a_3) - (a_1 + a_2)}\right].$$

The VIKOR method

The VIKOR method, developed by Opricovic (1998), is used for ranking and selecting a solution from a set of alternatives. (Lui, 2012). Here a multi criterion ranking index is created which measures how close one is to the ideal solution. It also suggests compromise solutions for a problem with conflicting criteria. This model requires the input of decision makers who rate all the alternatives (A_i) with respect to the criteria at hand. These decision makers can be seen as different departments in an organisation. It seeks to balance departmental weights before arriving at a matrix with a number of proposed hierarchical solutions. The VIKOR method works in the following step manner:

- **Step 1:** determine the departmental linguistic weight ratings of O, S, and D as well as the linguistic ratings of the failure modes and transform these into fuzzy numbers using the trapezoidal form of measure. Assuming that there's *K* departments/decision makers in an organisation, Lui proposes that the grade of membership or aggregated fuzzy rating for universe X be expressed as:

Equation 20. Grade of membership for rating failure modes (Lui et al, 2012)

$$\tilde{x}_{ij} = (x_{ij1}, x_{ij2}, x_{ij3}, x_{ij4})$$

Where

Equation 21. Lower limit, mode intervals, and upper limit of the universe *X* for rating failure modes

$$x_{ij1} = \min_k x_{ij1}^k, \quad x_{ij2} = \frac{1}{K}\sum_{k=1}^{K} x_{ij2}^k, \quad x_{ij3} = \frac{1}{K}\sum_{k=1}^{K} x_{ij3}^k, \quad x_{ij4} = \max_k x_{ij4}^k.$$

Here x_{ij} is the rating of alternative A_i given analysis criteria C_i. Similarly, the fuzzy weights of the criteria can be expressed in the following way:

Equation 22. Grade of membership for rating weight criteria (Lui et al, 2012)

$$\tilde{w}_j = (w_{j1}, w_{j2}, w_{j3}, w_{j4})$$

Where

Equation 23. Lower limit, mode intervals, and upper limit of the universe X for rating failure modes

$$w_{j1} = \min_k w_{j1}^k, \quad w_{j2} = \frac{1}{K}\sum_{k=1}^{K} w_{j2}^k, \quad w_{j3} = \frac{1}{K}\sum_{k=1}^{K} w_{j3}^k, \quad w_{j4} = \max_k w_{j4}^k.$$

- **Step 2:** apply the defuzzification method for transforming the fuzzy numbers of the failure modes as well as the weight criteria into crisp values. This step is completed using the trapezoidal centroid formula (equation 19).
- **Step 3:** calculate f_j* and f_j^-. These are the values allocated to criteria *j* for alternative $A_{i.}$ and is given by:

Equation 24. Criteria rating for *j* criteria allocated to alternative *Ai* based on equation 21

$$f_j^* = \left\{ \begin{array}{ll} \max_i x_{ij}, & \text{for benefit criteria} \\ \min_i x_{ij}, & \text{for cost criteria} \end{array} \right\}, \quad i = 1, 2, \ldots, m,$$

$$f_j^- = \left\{ \begin{array}{ll} \min_i x_{ij}, & \text{for benefit criteria} \\ \max_i x_{ij}, & \text{for cost criteria} \end{array} \right\}, \quad i = 1, 2, \ldots, m.$$

- **Step 4:** calculate S_i and R_i. These two variables are regarded as ranking measurements. The minimum value obtained in the computation of Si represents the 'majority rule' function, whilst the minimum value obtained from the R_i computation represents the minimum regret of the opposing alternatives for the individual department/decision maker.(Lui et al, 2012).

Equation 25.Ri and Si computations for ranking alternatives (Lui et al, 2012)

$$S_i = \sum_{j=1}^{n} \frac{w_j(f_j^* - x_{ij})}{f_j^* - f_j^-},$$

$$R_i = \max_j \left(\frac{w_j(f_j^* - x_{ij})}{f_j^* - f_j^-} \right).$$

- **Step 5:** calculate Q_i.

Equation 26. Computing Qi, the final unit for ranking the alternatives (Lui et al, 2012)

$$Q_i = v\frac{S_i - S^*}{S^- - S^*} + (1 - v)\frac{R_i - R^*}{R^- - R^*},$$

$S* = min_iS_i$ (minimum of all S values for the different alternatives)

$S^- = max_iS_i$ (maximum of all S values for the different alternatives)

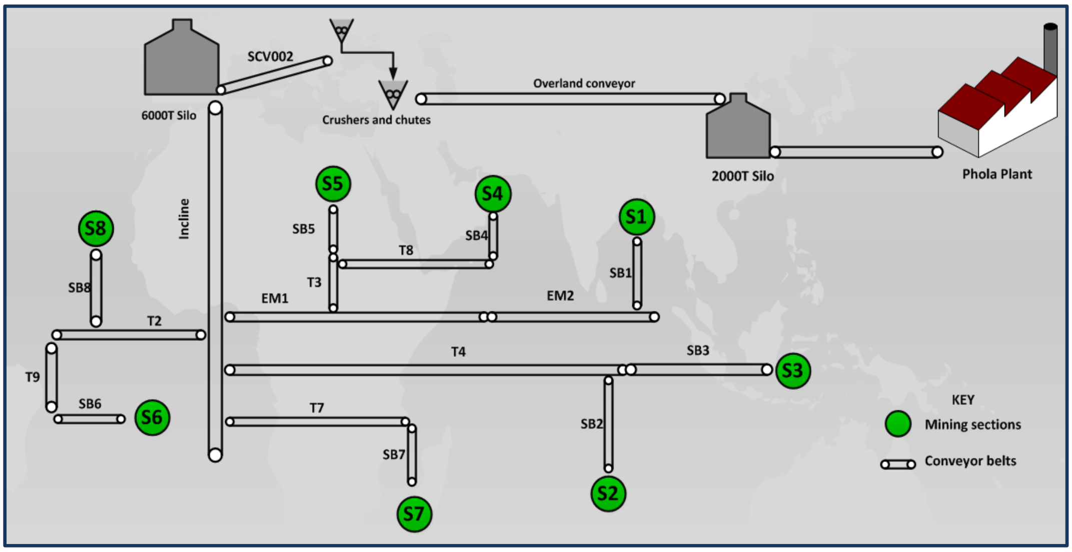
$R* = min_iR_i$ (minimum of all R values for the different alternatives)

$R^- = max_iR_i$ (maximum of all R values for the different alternatives)

v = utility constant set to 0.5

- **Step 6:** the alternatives need to be ranked in decreasing order; firstly by S, R and Q, tis step produces three ranking lists.

For simplicity, FMEA with VIKOR and fuzzy numbers can be presented as a flow chart (see figure 28)



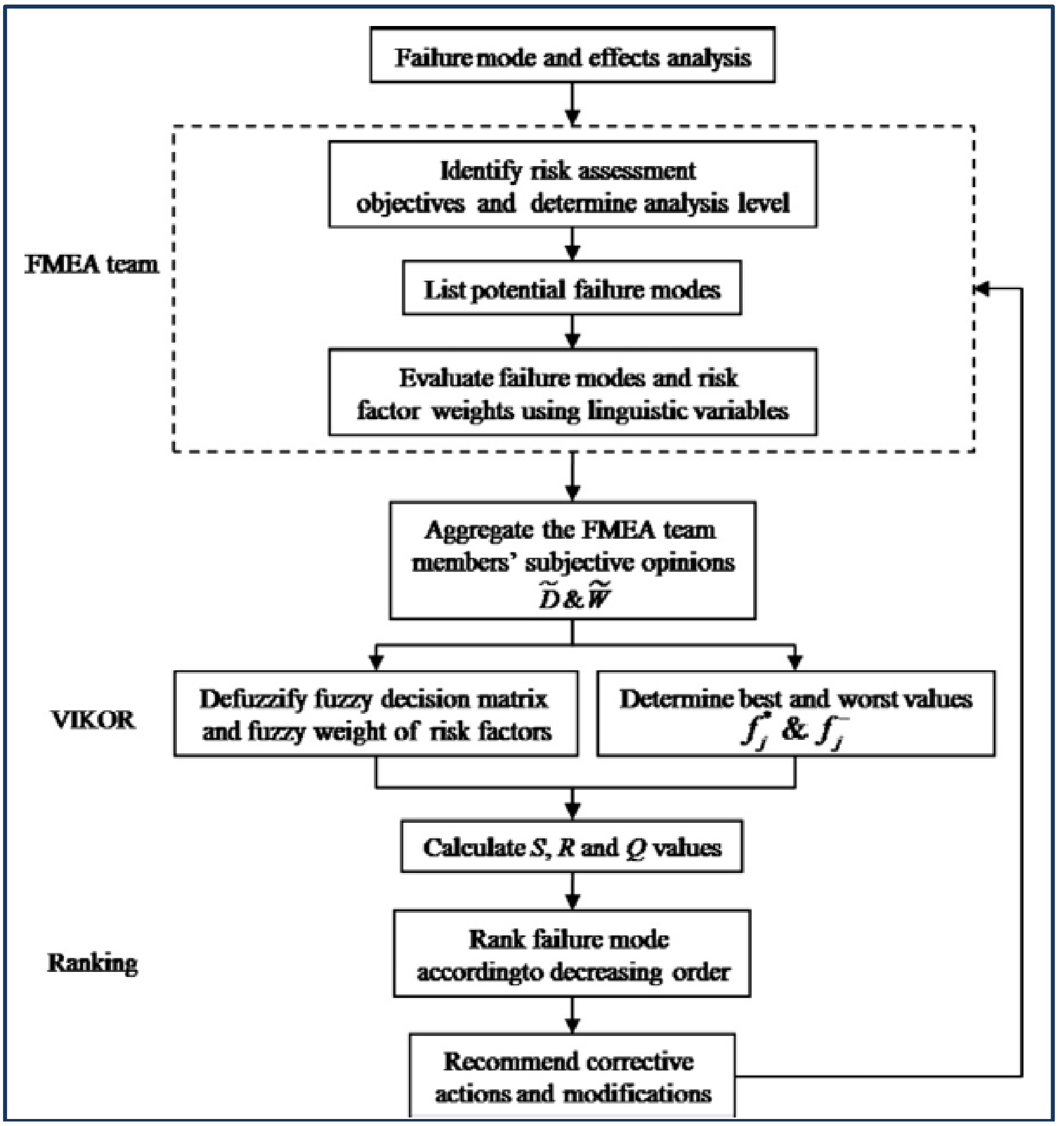


Figure 27. Flowchart of FMEA with fuzzy numbers and VIKOR (Lui et al, 2012)

This model has been extensively improved through a number of analysis techniques. It proves to be more detailed and accurate when it comes to ranking risks than the traditional risk analysis technique of simply multiplying probability of occurrence with consequence and detection. It successfully consolidates and considers the effects of failure modes on different departments from a multi-view perspective, resolving conflicts of interest between departments and the human intuition of ranking failure modes and weighting criteria for evaluation.

Stochastic Process Model – Location

Baral Location Model

Baral, et al (1987) proposes a method for bunker location within a system of conveyors. It is based on the expression for system availability discussed earlier. In this model, Baral et al (1987) states that in a system comprising of *n* conveyors, there are *n-1* possible location points for placing a bunker. One needs to find the location that would enable maximum system availability. Recalling that for a system of *n* conveyors with a bunker placed after the *m*-th conveyor, the system availability is calculated as follows:

Equation 27. System availability for a system with n conveyors (Baral, 1987)

$$A_{1,m,n} = \frac{\rho_{1,m} - \rho_{m+1,n}\exp^{(-\gamma V)}}{\rho_{1,m}A_{1,m}^{-1} - \rho_{m+1,n}A_{m+1,n}^{-1}\exp^{-\gamma V}}$$

The general rule is obeyed:

$A_{1,m,n}(0) \leq A_{1,m,n}(V) \leq A_{1.m.n}(\infty)$

Where $A_{1,m,n}(0)$ is the availability of the system when there is no bunker

$A_{1,m,n}(V)$ is the availability of the system when there is a bunker with capacity V

$A_{1,m,n}(\infty)$ is the availability of the system after the asymptotic point has been reached.

Equation 28. Availability of the system at the asymptotic point (Baral et al, 1987)

When $V \to \infty$:

$$\lim_{V \to \infty} A_{1,m,n}(V) = \mathrm{Min}\{A_{1,m}, A_{m+1,n}\}$$

The above equation is so because, according to Baral et al (1987), once the system reaches stability, should $A_{1,m}$ be greater than $A_{m+1,n}$, the bunker will be fed by the input belt for a time duration proportional to $A_{1,m}$ but however, since the fundamental rule of flow states that the input availability of the system should be greater than the output availability, the output of the bunker will be made available for a time duration proportional to $A_{m+1,n}$. therefore, for $(A_{1,m} - A_{m+1,n})$ of the time, the contents of the bunker will be increasing at $A_{1,m}$ fraction of the time and for $(A_{m+1,n} - A_{1,m})$ the bunker will discharge at a fraction of $A_{m+1,n}$ of the time. According to equation 28 above, a pair of subsystems exists for each location if we partition the conveyor system into n-1 pairs (Baral, et al, 1987). The optimal location (*j*) can therefore be said to be the maximum of all the minimum availabilities of the subsystems. (Equation 29)

Equation 29. determination of the optimal location j of the bunker (Baral et al, 1987)

$$\mathrm{Min}\{A_{1,j}, A_{j+1,n}\} = \mathrm{Max}[\mathrm{Min}\{A_1, A_{2,n}\}; \mathrm{Min}\{A_{1,2}, A_{3,n}\}; \mathrm{Min}\{A_{1,3}, A_{4,n}\}; \ldots, \mathrm{Min}\{A_{1,n-1}, A_n\}]$$

Thus it is the location where the subsystems are partitioned into two subsystems where the availabilities of these two subsystems are relatively close to one another (Baral, et al, 1987).

This method, though efficient and relatively easy to use, is still based on the fact that in the calculation of availabilities provision is only made for exponential distributions in the time between failures and time to repair. This may not be a true reflection of reality (refer to Appendix H for time between failures and time to repair distributions of the mine in question)

8. Literature Review Conclusion

In conclusion, it is highlighted that although the mining environment is plagued with traditional inertia to change in process mining, new technologies/ methods should be implemented to support these traditional structures and/or processes. It is believed that continuous improvement initiatives are a key driver in ensuring that companies reach both their operational and strategic objectives.

In terms of buffering for protection of production purposes, literature reveals that bunkering activity may be the industry standard.

The problems discussed required in-depth research on best practices for bunker sizing and strategic locationing, with the solution re-modelled to best fit the mining environment of the company. It was found that simulation modelling would be the best tool to employ to model the current and proposed solution coupled with FMEA with extended VIKOR method under a fuzzy environment (for the purpose of determining optimal location points). However, for alternative solution build, risk analysis using decision trees will be used to determine optimal location points for placing buffer storage capacity. This is due to the fact that all the other models are subject to a number of assumptions that may not necessarily hold true. A comparison will be made between FMEA with extended VIKOR under a fuzzy environment and decision tree analysis to determine if the two solution methods are in agreement with each other or not

Highlight should once again be made that although simulation modelling does not give one the optimal solution, it does enable one to reach a workable solution.

Also there is the suggestion that, regardless of new implementations carried out in any environment, it is to the detriment of the company if monitoring, measuring and risk analysis techniques are not in place for control purposes.

9. Simulation Model Requirements

The simulation model for the mine will require a number of inputs for each mining section (see figure 28 below). For these input distributions, Arena Input Analyser will be used to build a data file on inter-arrival time, belt downtime, belt mean time before failure etc. (See appended documentation). This model will also be subject to constraints as those noted in the below figure. The model is expected to generate optimal buffer storage sizes thus reflecting additional tonnage figures that the mine may expect to obtain from the implementation of this project. A cost volume analysis will then need to be conducted to determine the feasibility of the number of storage bunkers as well as their storage capacities in relation to costs. This will be followed by a calculation of the project payback period taking into consideration a life-of-mine of 20 years.

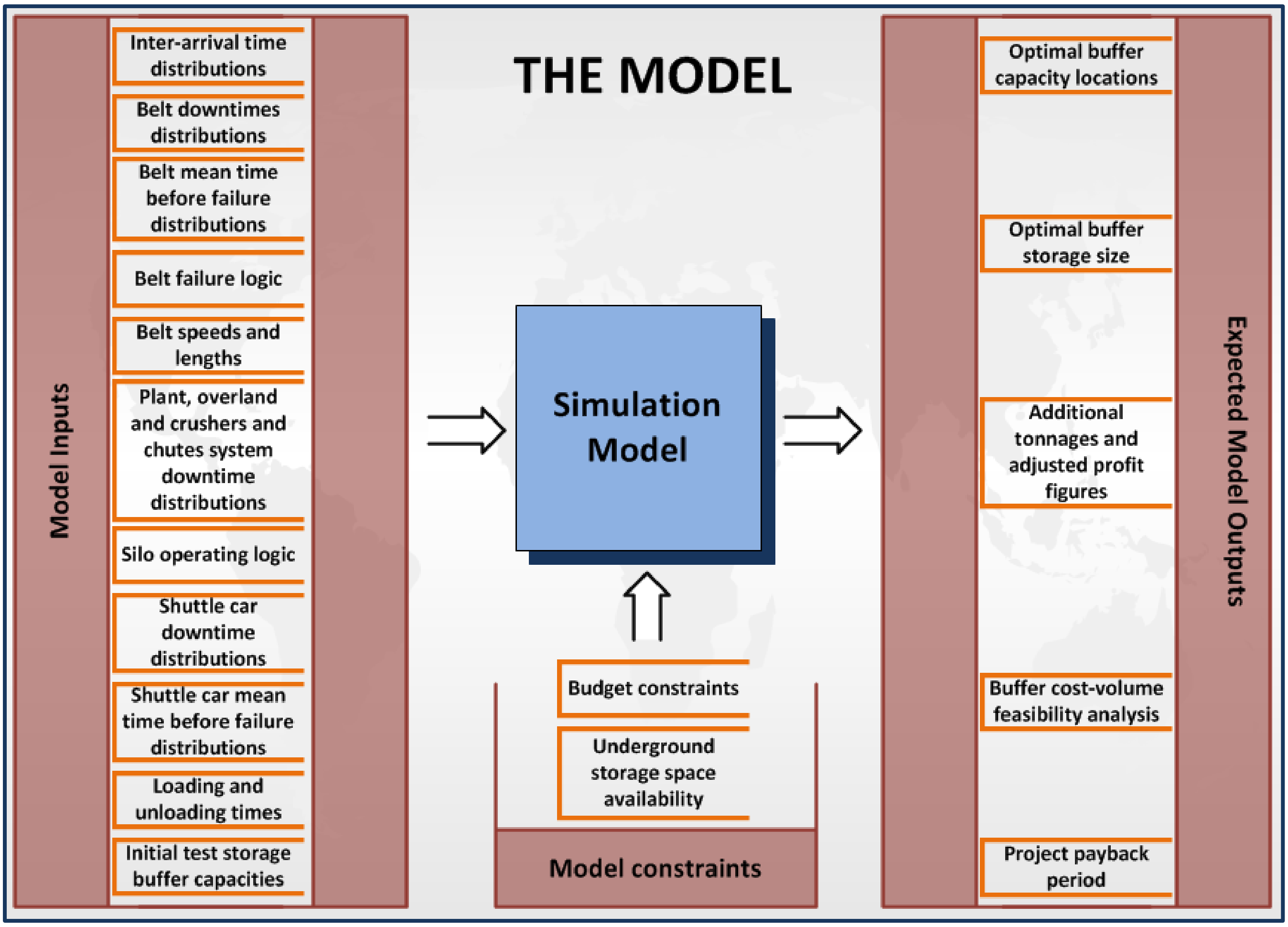


Figure 28. Simulation model requirements (inputs), constraints, and projected outputs

10. Surface As-Is Model

The model is expected to generate coal at all eight sections of the mine from 10 replications to ensure results integrity. Coal generation follows cyclic distributions (see appendix E) specific to each section. Each section (except section 8) has three allocated shuttle cars/battery haulers each with a carrying capacity of 18 tons, with section 8 having 2 shuttle cars. These shuttle cars/battery haulers carry the coal from the continuous miner, which cuts the coal, to the feeder breaker which is responsible for cutting the coal to a smaller size and loading it onto the conveyor system. The distances that the shuttle cars travel to the feeder breaker have not been considered in the simulation because all cutting faces are continuously moving and after a lengthy period of time it is assumed that these distances will have no effect on the results of the mining operation. The coal travels via a series of conveyors each having different carrying capacities, lengths and distances (see appendix D) to a silo of capacity equivalent to 6000 tons. The simulation also considers the distributions for mean time before failure and time to repair, per conveyor belt as well as the time taken to move between coal faces.

10.1. As-Is Surface Model - Silo operating levels and logic

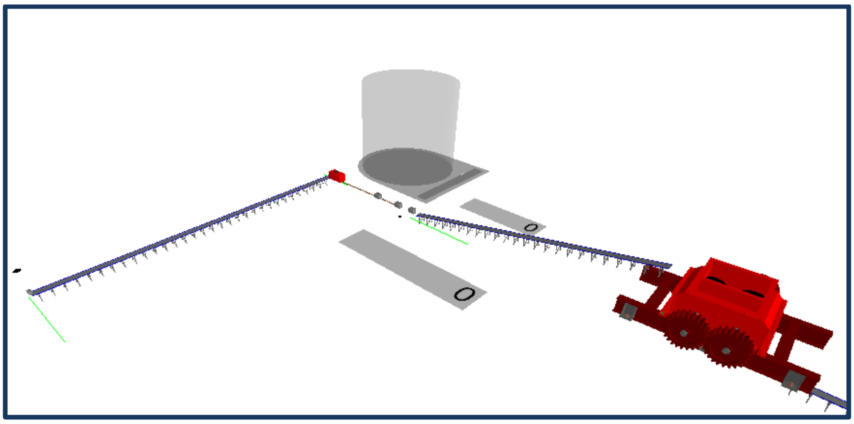


Figure 29. Depicting the 6000 ton silo, conveyor scv 002 and the crusher and chute system

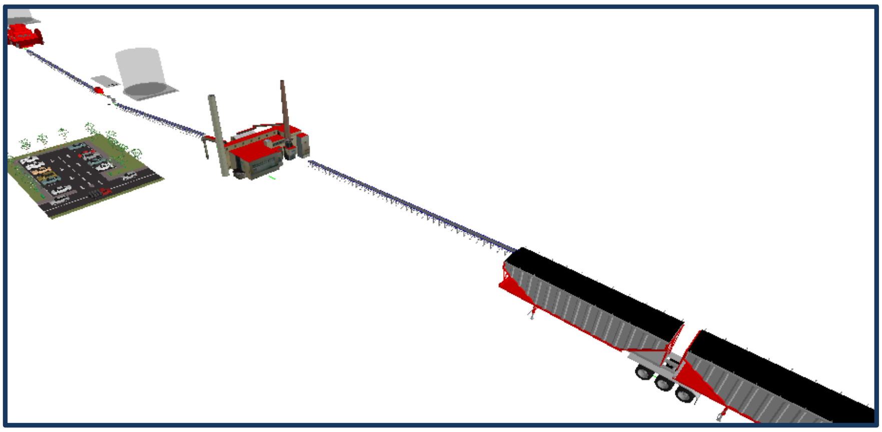


Figure 30. Depicting the 2000 ton silo, the Plant and dispatch train

Three operating levels are defined for the silos, namely:

- Lower warning limit of 14% of the total capacity (low mark)
- Mid-mark of 50% of the total capacity
- Upper warning limit of 95% of the total capacity (high mark)

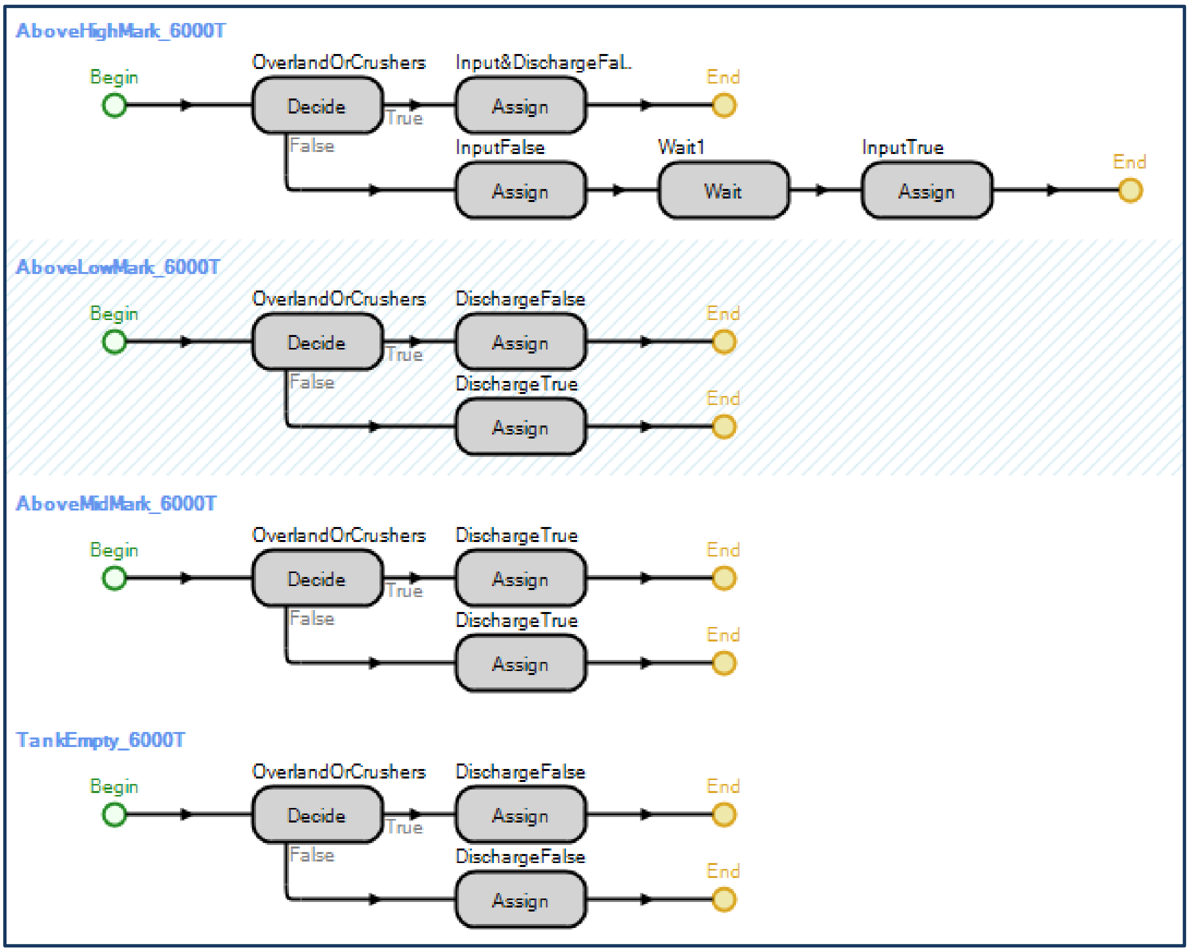


Figure 31. 6000T silo logic

In the above figure, the 6000T silo logic is modelled in the following way:

When the silo content is rising (the silo is filling up), the model monitors whether or not a failure, either at the crushers and chute system, or the overland conveyor has occurred by using decision steps in the following way:

- ‘Above high mark’ process, which is an add-on process for determining whether the silo contents has reached a high mark of 5700 tons (95% upper warning limit), the model evaluates if any of the above-mentioned failures are active. If this step returns a true value, then the model will automatically assign both the input and output flow regulator of the silo to false. These flow regulators determine whether or not flow must be allowed into or out of the silo. In the case that any of the failures are active, the silo may not discharge flow, but rather the silo current holding capacity should increase. Should the decide step monitor that no failures are active from either the crushers and chutes system or the overland then the model assigns the output of the silo regulator to true, then a wait step to wait until a certain event condition is met (in this case wait until the silo contents decreases to below the upper warning limit) then assign the input of the flow regulator to true stating that coal can flow into the silo.

- ‘Above low mark’ process. This process determines the logic that needs to be modelled should the silo contents increase to above the lower warning limit. If a failure occurs (either the crushers and chute or the overland conveyor) the silo should once again not discharge flow, but if a failure is not active then the outflow of the silo should be toggled to true.
- ‘Above mid-mark’ process. Once again this process determines whether or not a failure is active and if the flow regulators of the silo need to be adjusted similarly to the ‘Above low mark’ process.
- When the silo is empty, whether or not the failures are currently active, the model will toggle the discharge rate of the silo to false.

This same logic is applied to the 2000 ton silo, but instead of monitoring the crushers and chutes and the overland failures the only failure that would need to be monitored is the failure relative to the Plant. This same logic is also applied to when either of the silo’s weights contents is decreasing i.e. when the silo level is falling.

10.2. Modelling belt system failure

Due to the serial dependence of the conveyor system, modelling failure and repair steps also needs to be serially configured.

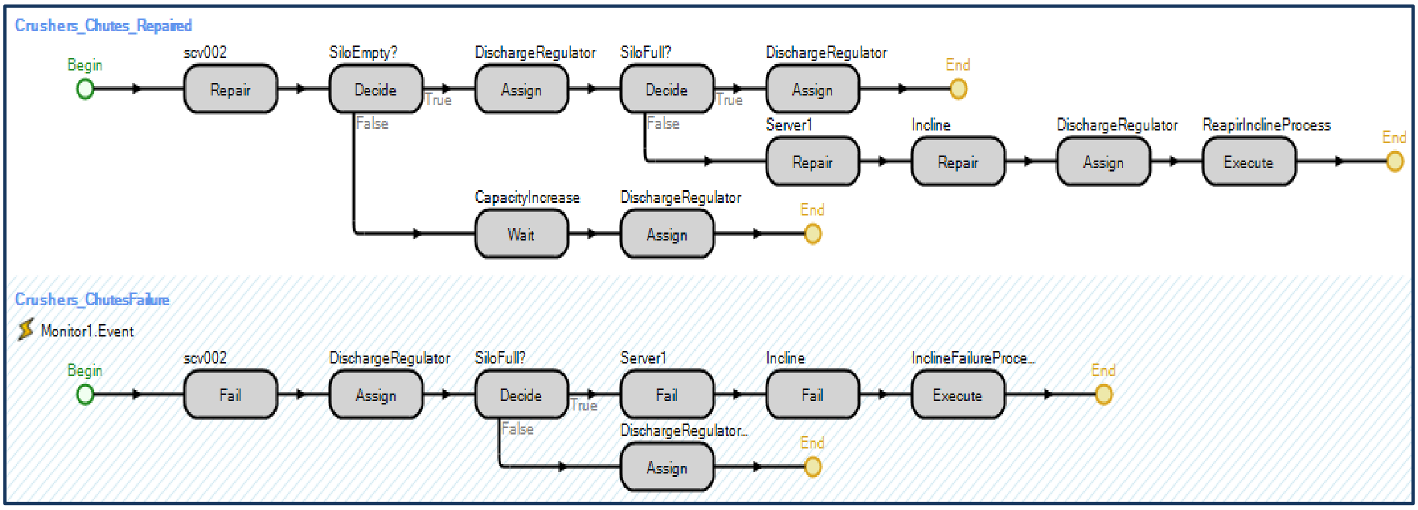


Figure 32. Modelling belt failure and repair logic of the crushers and chute system

From the above 'Crushers and Chutes Failure' logic process steps, it can be seen that there is a specific event which fires this process. This event is a monitor which monitors whenever failure of the crushers and chutes takes place. Upon crusher and chute failure, the following steps take place:

- Conveyor scv 002 (which links the 6000 ton silo to the crushers and chutes) needs to be put on stop (failed) and the output flow regulator of the silo must be set to false to ensure that the silo doesn't discharge flow while the failure of the system is still active
- A decision needs to be made to determine whether the silo has reached its full capacity. If it has reached its full capacity (true branch), then all preceding systems need to be failed or put on stop for the duration of the said failure. This implies that the entire mining operation needs to be halted until the breakdown can be attended to.
- If the system moves to the false branch, then the output flow regulator of the silo will be assigned to false, since the failure is still active.

The repair step is activated when the failure downtime has lapsed. This process is carried out in the following way:

- A repair will need to be made to conveyor scv 002. Furthermore, the model needs to decide if the silo weight content is above the lower warning limit before it can discharge flow onto conveyor scv002. If the silo's current capacity is above the lower warning limit, then an assignment must be made to change the output of the flow regulator of the silo to true. A further decision must be made to assess, at a specific point in the simulation run, if the silo was full (reached the upper warning limit). If true, the output of the flow regulator must be assigned to false, if not true then repairs must be made to all preceding systems, therefore the mining operation can commence again and the output flow regulator will be assigned to allow silo discharge. However, in the first decision step, if the silo's current capacity was below the lower warning limit, then a waiting condition must be effected to tell the model to wait for the silo capacity to increase to above the lower warning limit before it can discharge onto conveyor scv 002.

The same principles are followed for failure modes with respect to the overland conveyor as well as the Plant failure, with the Plant failure requiring significantly longer and more detailed failure and repair logic.

10.3 Surface model – solution requirements

As discussed in the data analysis of the problem statement, significant losses are experienced by the mine due to the lack of sufficient buffer capacity to carry or to account for surface downtimes. It is therefore necessary to increase the capacity of the buffer station in an effort to prevent total system failure. The following provides validation for increasing buffer capacity:

- From figure 35 it can be seen that it takes approximately 3.29 hours to fill up the 6000T silo when one of the surface system have broken down (full at 5700 tons, slope of graph=0), it is thus necessary to determine the duration of downtimes for the surface systems, if they are lengthier than the 3.29 hours, that would be an indication that additional buffer storage is required, if the duration is shorter than the said 3.29 hours then additional buffer is not required. To be able to make this comparison, box plots will be used to measure the length of surface system downtimes.
- Additionally, as indicated by figure 35, the 2000T silo only reaches a maximum capacity of 774 tons before the 6000T silo fills up, indicating that the 2000T silo capacity is sufficient after all.

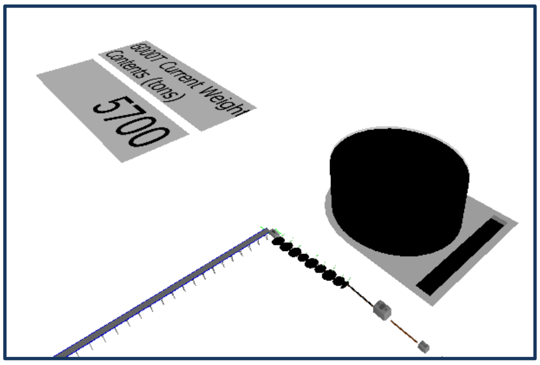


Figure 33. Silo 6000T full at upper warning limit of 5700 tons

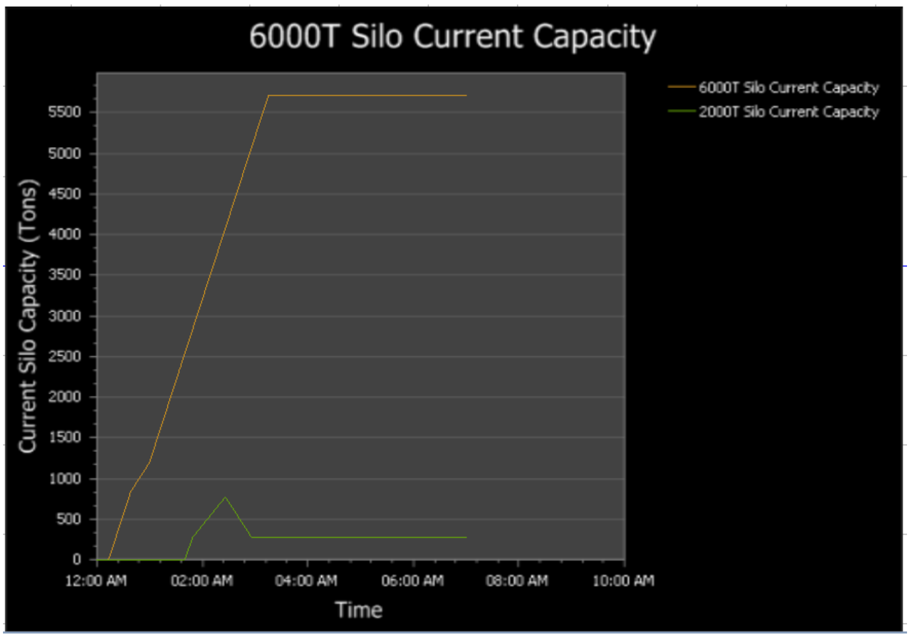


Figure 34. Graph showing the time taken for the 6000T silo to reach full capacity

A box and whiskers diagram (see figure 36 below) approach is used to determine the duration of downtimes caused by the surface systems in support of the hypothesis that additional storage capacity on surface is required to prevent system shutdown.

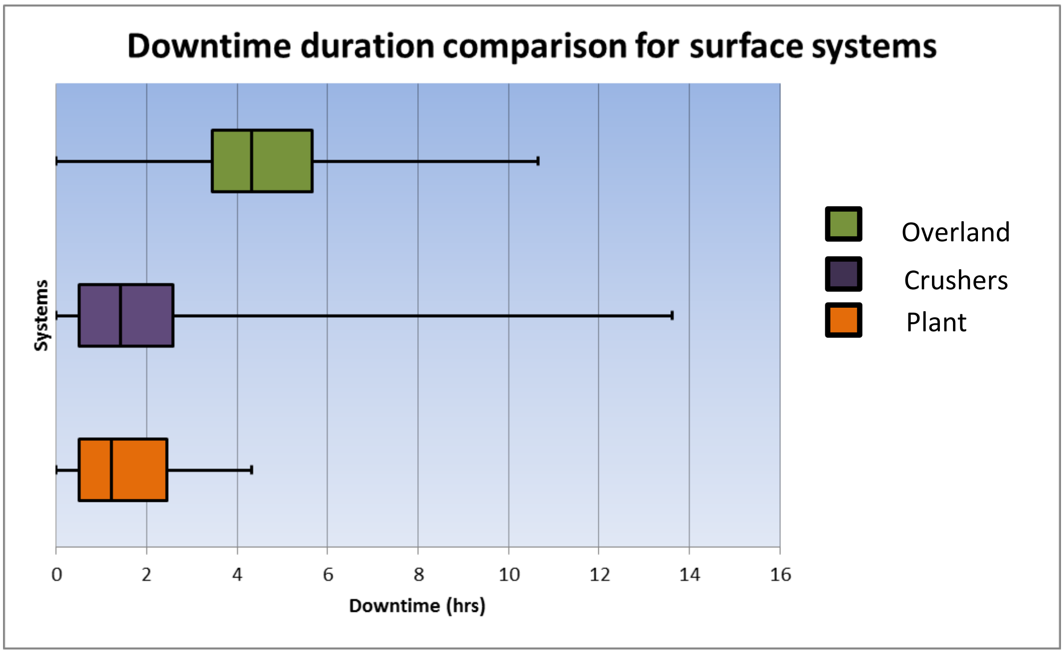


Figure 35. Box and whiskers diagram for surface systems (downtime durations)

Table 9. Summary statistics table of downtime duration surface systems

Summary statistics					
Plant		Crushers and chutes		Overland conveyor	
Min	0	Min	0	Min	0
Q1	0.5	Q1	0.5	Q1	3.45
Med	1.22	Med	1.42	Med	4.33
Q3	2.45	Q3	2.58	Q3	5.67
Max	4.33	Max	13.62	Max	10.66

From the above figure and table, the following deductions should be made:

- At any given point in time, the size additional capacity that this project wishes to propose should, at the least be, able to cover the downtime durations of one of the systems mentioned above. Based on the box and whiskers plot shown, it is evident that typically the longest downtime durations arise from the overland conveyor. Hence the capacity build that this project wishes to address should be able to increase system availability largely through the coverage of the most extensive downtime durations. Since it wouldn't be financially feasible to build enough storage capacity to cover maximum value downtimes, it is proposed that the storage capacity built cover 75% of all downtime durations. Thus it is selected, through analysis, that overland conveyor downtime coverage be addressed, spanning a duration of 5.67 hours (Q_3), (as per table above). As was indicated that 'time to fill' for the 6000T silo is 3.29 hours, significantly shorter than Q_3 of the overland conveyor, it can now be deduced that this 'time to fill' wouldn't be able to cover even 25% of all downtimes for the overland conveyor, hence, through data analysis it is evident that additional storage capacity is required to keep the system in operation in the event of surface system failure.

10.4 Simulation results – As – Is Surface Model

Table 10 shows, over the simulation run, the number of tons that were contained in the silo. It is evident that the 6000 ton silo reached its full capacity of 5700 tons, which was set to be the 95% upper limit. Table 11 indicates that only 1134 tons were dispatched by the 6000 ton silo. This highlights that a failure occurred from the crushers and chutes or the overland conveyor or the Plant of proportions relative to those in table 12 that rendered the silo unable to deliver more throughput. It also shows that since the final value of the silo reached full capacity the underground operation came to a complete halt as this current buffer capacity is insufficient to cover the surface system downtimes.

Table 10 Silo weight contents

WeightLevel - FinalValue			
Object Name	Data Source	Category	Value
Silo2000T	FlowContainer	Content	22.27858
Silo6000T	FlowContainer	Content	5700

Table 11. Silo 6000T weight flow out

WeightFlowOut - Total			
Object Name	Data Source	Category	Value
FlowSink2	FlowContainer	Throughput	0
Path1	[Flow]	Throughput	6834.89866
Path2	[Flow]	Throughput	22.27858
Silo2000T	FlowContainer	Throughput	0
Silo6000T	FlowContainer	Throughput	1134.89866

Table 12. Average duration of failure of surface systems

FailedTime - Average			
Object Name	Data Source	Category	Value
CrushersAndChutes	[Resource]	ResourceState	3.21238
Plant	[Resource]	ResourceState	4.30988
Server2	[Resource]	ResourceState	3.21238

10.5 Proposed Surface Solution Model

It is therefore proposed that an additional 9200T silo be built next to the 6000T silo (figure below and table 13) as figure 38 indicates a continuous flat line showing that the silo was full and couldn't discharge during that time due to the prolonged failure of the overland. The process, augmented with a 9200T silo (figure 37) shows that the mine may stand to benefit an additional 9200 tons for a failure that can be attributed to either of the surface systems causing the 6000 ton silo to fill up. The additional 9200 ton silo would effectively prevent the mine from having to go on stop in the event that the main silo fills up due to consequential breakdowns.

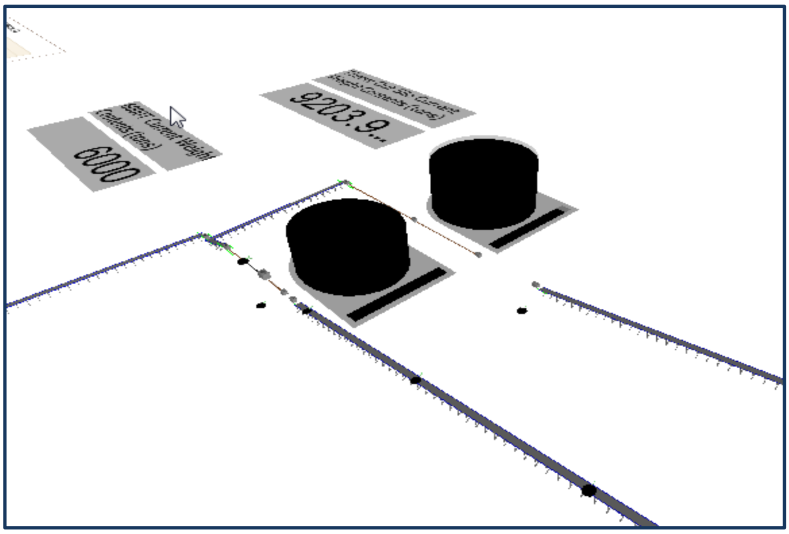


Figure 36. 6000T silo with additional 3000T capacity

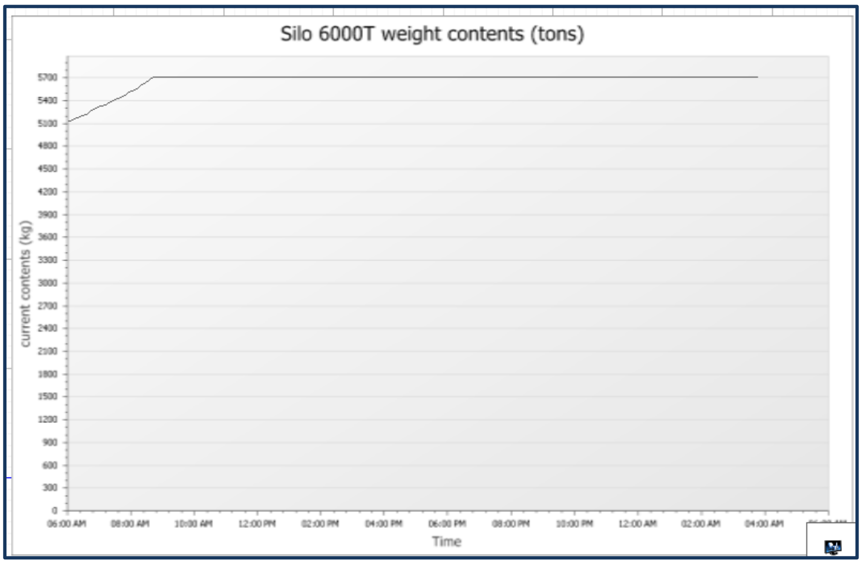


Figure 37. 6000T silo weight contents against time

Table 13. Throw-Out Silo weight contents

WeightLevel - Maximum

Object Name	Data Source	Category	Value
Silo2000T	FlowContainer	Content	772.3936
Silo6000T	FlowContainer	Content	6000
ThrowOutSilo	FlowContainer	Content	9203.95158

11. The Underground As-Is Model

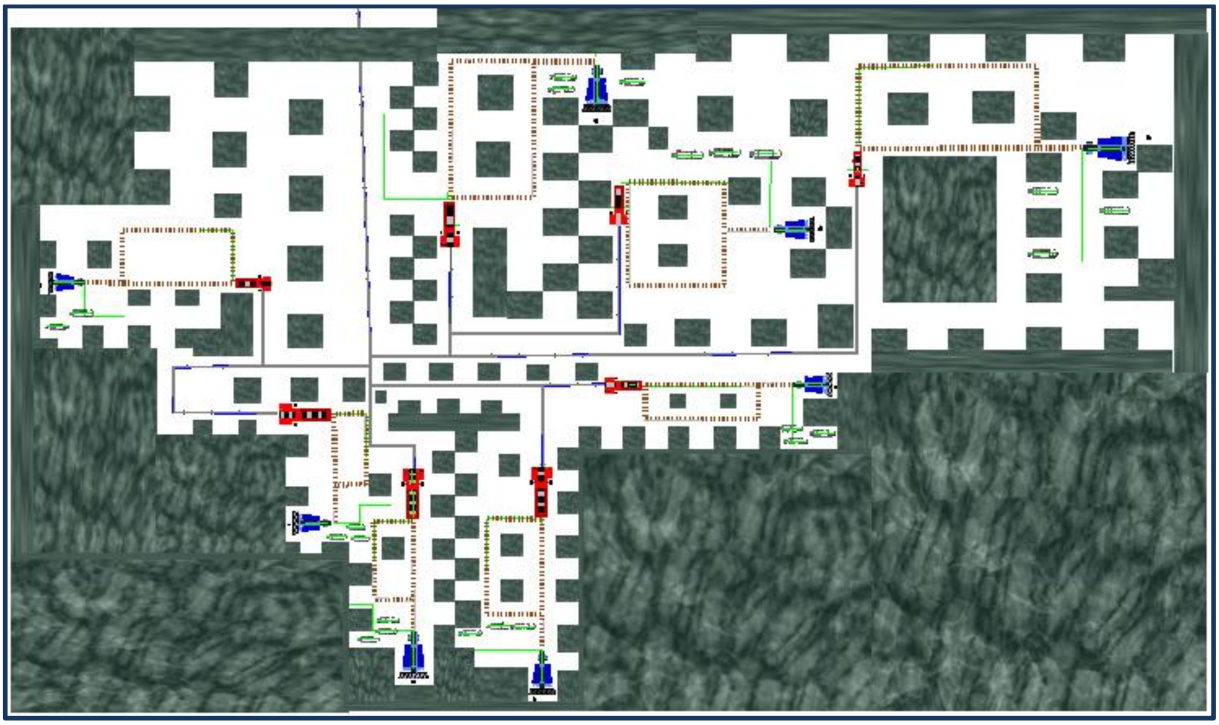


Figure 38. Underground model depicting the 8 sections along with equipment

The underground As-Is model is modelled with time to repair and mean time before failure of all belts. The process modelling that exists here is of a sequential nature, meaning the failure of one belt is modelled to fail all sequential belts preceding the said belt, the repairing sequence following the same logic.

11.1 Strategic location of underground storage bunkers

In order to determine optimal size capacity for bunkers to resolve the second problem underground, it is first important to analyse which points within the mine model would prove to be optimal for placement of bunkers. Two alternative methods will be used to deduce if one validates the other, namely 'FMEA with VIKOR under a fuzzy environment' and decision tree analysis.

11.1.1 Alternative 1 - FMEA with VIKOR under a fuzzy environment

The FMEA technique is used to analyse and rate failure modes and weighting failure criteria. Based on the failure of the incline belt (which connects the underground operation to the 6000T silo), all preceding belts leading to the sections will fail. Hence the different failure routes were identified as failure modes (Table 13).

Table 14. The identified failure modes from the system

Failure Modes	
FM1	East Main 1
FM2	Trunk 2
FM3	Trunk 4
FM4	Trunk 5
FM5	Trunk 3

Additionally, the FMEA team was constructed of the different departments within the mine in question, as can be seen in table 15.

Table 15. The different departments within the mining industry

Decision Makers (DM)	
DM1	Asset optimisation dept.
DM2	Mining dept.
DM3	Engineering dept.
DM4	Finance dept.
DM5	HR dept.

The weight of risk factors and judgement of failure modes are completed using table 5, 6 and 7. Each decision maker/department decides which risk factor (either O, S, or D) is most or least important to measure for that department (done in table16), then transformation is done on changing the risk factors and failure modes into fuzzy numbers using the trapezoidal method (see table 18)

Table 16. Weighting of risk factors O, S,and D

Importance weight of risk factors from FMEA team members					
			Team Members		
Risk Factors	**DM1**	**DM2**	**DM3**	**DM4**	**DM5**
Occurrence	VH	VH	VH	VH	ML
Severity	VH	VH	MH	VH	M
Detection	H	H	VH	VL	VL

Table 17. Fuzzy numbers for weighting risk factors

Liguistic variables for rating the weights of risk factors					
Importance weight of risk factors from FMEA team members					
			Team Members		
Risk Factors	**DM1**	**DM2**	**DM3**	**DM4**	**DM5**
Occurrence	0.8, 0.9, 1, 1	0.8, 0.9, 1, 1	0.8, 0.9, 1, 1	0.8, 0.9, 1, 1	0.2, 0.3, 0.4, 0.5
Severity	0.8, 0.9, 1, 1	0.8, 0.9, 1, 1	0.5, 0.6, 0.7, 0.8	0.8, 0.9, 1, 1	0.4, 0.5, 0.5, 0.6
Detection	0.7, 0.8, 0.8, 0.9	0.7, 0.8, 0.8, 0.9	0.8, 0.9, 1, 1	0, 0, 0.1, 0.2	0, 0, 0.1, 0.2

The same process followed above for ranking the risk factors per department is also followed for ranking the different failure modes identified in table 14. These linguistic variables are once more transformed into fuzzy numbers using the trapezoidal method (refer to table 9), these are completed in table 18 and 19 below.

Table 18. Risk judgement of failure modes

Judgement on six failure modes by FMEA team members under risk factors															
Risk Factors	O					S					D				
Failure modes	DM1	DM2	DM3	DM4	DM5	DM1	DM2	DM3	DM4	DM5	DM1	DM2	DM3	DM4	DM5
FM1	L	H	M	L	VL	VH	VH	VH	VH	M	ML	MH	H	VL	VL
FM2	VH	VH	VH	M	M	L	MH	M	M	ML	ML	M	MH	L	L
FM3	VH	VH	VH	M	ML	VH	VH	H	VH	M	ML	M	H	L	L
FM4	VH	VH	VH	ML	M	M	M	L	ML	L	L	M	M	L	L
FM5	VH	VH	VH	H	M	H	H	H	VH	ML	ML	M	H	L	L

Table 19. fuzzy numbers of failure modes

Judgement on six failure modes by FMEA team members under risk factors															
Risk Factors	O					S					D				
Failure modes	DM1	DM2	DM3	DM4	DM5	DM1	DM2	DM3	DM4	DM5	DM1	DM2	DM3	DM4	DM5
FM1	1,2,2,3	7,8,8,9	4,5,5,6	1,2,2,3	0,0,1,2	8,9,10,10	8,9,10,10	8,9,10,10	8,9,10,10	4,5,5,6	2,3,4,5	5,6,7,8	7,8,8,9	0,0,1,2	0,0,1,2
FM2	8,9,10,10	8,9,10,10	8,9,10,10	4,5,5,6	4,5,5,6	1,2,2,3	5,6,7,8	4,5,5,6	4,5,5,6	2,3,4,5	2,3,4,5	4,5,5,6	5,6,7,8	1,2,2,3	1,2,2,3
FM3	8,9,10,10	8,9,10,10	8,9,10,10	4,5,5,6	2,3,4,5	8,9,10,10	8,9,10,10	7,8,8,9	8,9,10,10	4,5,5,6	2,3,4,5	4,5,5,6	7,8,8,9	1,2,2,3	1,2,2,3
FM4	8,9,10,10	8,9,10,10	8,9,10,10	2,3,4,5	4,5,5,6	4,5,5,6	4,5,5,6	1,2,2,3	2,3,4,5	1,2,2,3	1,2,2,3	4,5,5,6	4,5,5,6	1,2,2,3	1,2,2,3
FM5	8,9,10,10	8,9,10,10	8,9,10,10	7,8,8,9	4,5,5,6	7,8,8,9	7,8,8,9	7,8,8,9	8,9,10,10	2,3,4,5	2,3,4,5	4,5,5,6	7,8,8,9	1,2,2,3	1,2,2,3

The fuzzy numbers are transformed into crisp numbers for weighing risk factors using equation 19; the results can be seen in table 19. The same equation is used for weighing the different failure modes (see table 20).

Table 20. Transforming fuzzy numbers into crisp values for the different risk factors

Weight of each risk factor		
Risk Factors		
Occurrence	Severity	Detection
0.2, 0.78, 0.88, 1	0.4, 0.76, 0.84, 1	0, 0.5, 0.56, 1
Crisp numbers		
0.685	0.737	0.511

Table 21. Transforming fuzzy numbers into crisp values for the different failure modes

Aggregated fuzzy rating of six failure modes and aggregated fuzzy weight of risk factors.

Failure modes	O	S	D
FM1	0, 3.4 ,3.6 ,9	4, 8.2, 9, 10	0, 3.4, 4.2, 9
FM2	4, 7.4, 8, 10	1, 4.2, 4.6, 8	1, 3.6, 4, 8
FM3	2, 7, 7.8, 10	4, 8, 8.6, 10	1, 4, 4.2, 9
FM4	2, 7, 7.8, 10	1, 3.4, 3.6, 6	1, 3.2, 3.2, 6
FM5	4, 8, 8.6, 10	2, 7.2, 7.6, 10	1, 4, 4.2, 9

Crisp values for decision matrix and weight of each risk factor.

Failure modes	O	S	D
FM1	4.16	7.6	4.25
FM2	7.25	4.46	4.25
FM3	6.51	7.47	4.69
FM4	6.51	3.5	3.4
FM5	7.47	6.49	4.69
Weight	0.685	0.737	0.511

Table 22. Value of criterion for alternatives

S*	0.486
S^-	1.733
R*	0.486
R^-	0.737

Best f*	
f_O*	4.16
f_S*	3.5
f_D*	3.4

Worst f^-	
f_O^-	7.47
f_S^-	7.6
f_D^-	4.69

Values for S and R are calculated for each failure mode. These are the ranking measurements that are used to calculate Q, which ranks the failure modes.

Table 23. Ranking failure modes

	Failure modes				
	FM1	FM2	FM3	FM4	FM5
S	1.074	1.149	1.711	0.486	1.733
R	0.737	0.639	0.714	0.486	0.685
Q	0.734	0.571	0.945	0	0.896

	FM1	FM2	FM3	FM4	FM5
by S	4	3	2	5	1
by R	1	4	2	5	3
by Q	3	4	1	5	2

From this approach, the top three rankings are selected which indicate where a bunker is required in order of FM3, FM5 and FM1, which are Trunk belt 4, Trunk belt 3 and East Main 1 respectively. However capacity is constrained to two bunkers and based also on simulation bunkers should be provided for the East Main belt and the Trunk 4 belt. The bunker should be placed before these belts in order to increase system availability and reliability and protect the production of sections 1, 4, 5, and 2, and 3.

11.1.2 Alternative 2 – Decision Trees

Decision tree analysis for risk mitigation is used in this section to determine the optimum location of bunkers for the underground operation. This technique is employed to determine if different results will be rendered to the first approach. The figure below depicts a decision tree that is used to represent the serial dependence of one conveyor upon another. From the incline failure, the possible modes of failure are represented which lead to a complete system halt, along with section production rates, section belt downtime averages and standard deviations. This is computed to determine which sections/belt configurations incur the highest losses. This method uses the risk assessment tool of Probability of failure*consequence to rank which failures appear to be the highest so that risk mitigation strategies may be employed by locating bunkers at those points. As can be seen from the figure below, the highest losses can be ranked as follows:

- Losses due to the failure of Trunk belt 4
- Losses due to Trunk belt 2
- East Main 1 belt
- Trunk belt 5

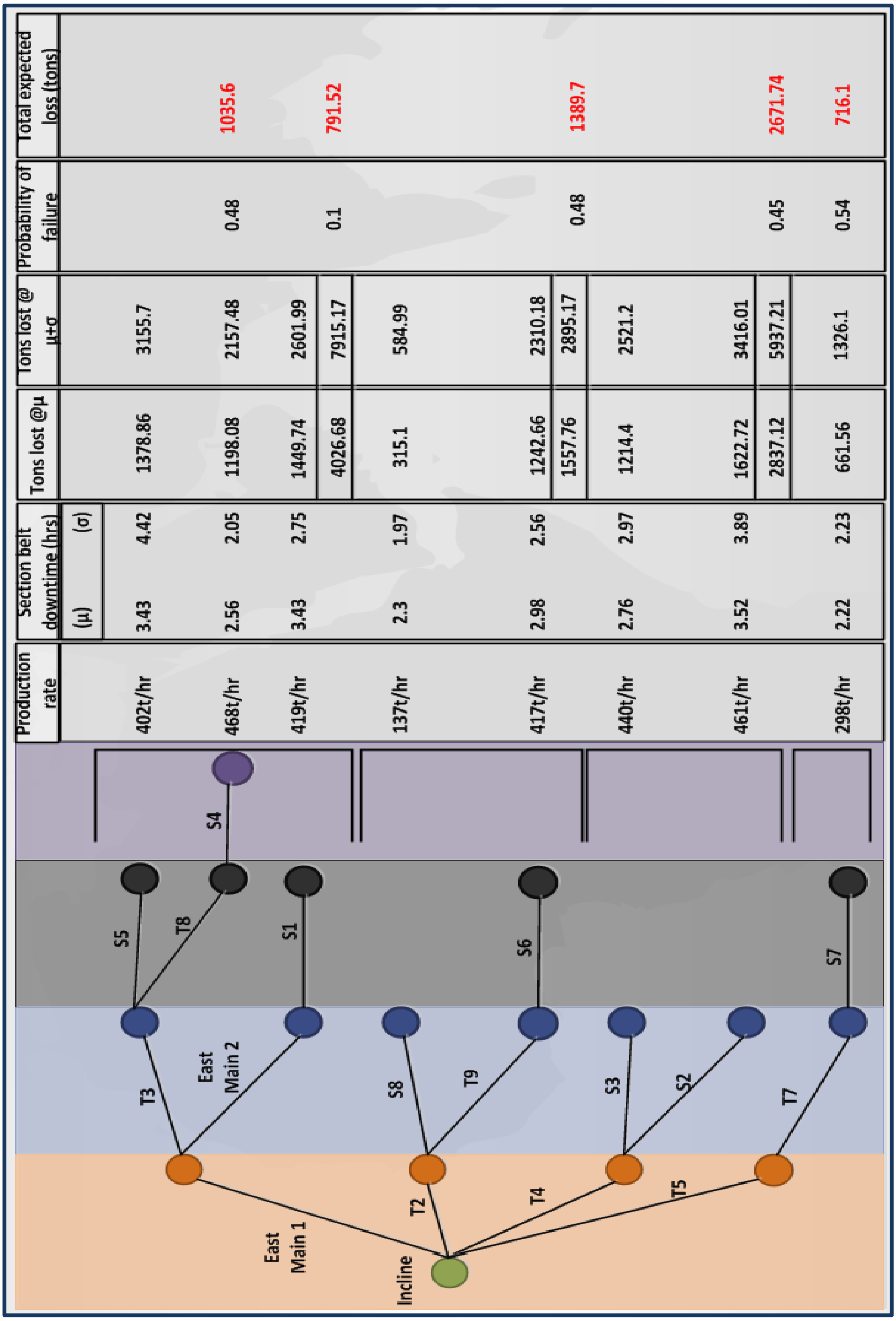


Figure 39. Decision tree analysis for risk mitigation

Comparing the two methods bears the following results:

Table 24. Comparison between FMEA with VIKOR and Decision Tree analysis

	Comparison	
Ranking	**FMEA with VIKOR**	**Decision Trees**
1	Trunk 4	Trunk 4
2	Trunk 3	Trunk 2
3	East Main belt	East Main belt

The following notes can be made:

- The two methods analysed bear similarities with two failure modes, namely, Trunk 4 and the East Main belt.
- FMEA using VIKOR is more extensive in analysis as it recognises Trunk 3 belt as an important failure mode that needs to be mitigated separately to the East Main belt.
- Trunk 2 ranking second in the decision tree analysis may seem to be questionable as section 8 has not yet reached full production ramp up and is therefore a poor performing section as it is mainly plagued by dykes which makes mining difficult. Its ranking may be indicative of the short-comings that were highlighted in earlier pages of the traditional FMEA method of ranking, as it came into use in this analysis. This highlights the importance of having a cross-functional team spanning across all departments for ranking failure modes. Thus FMEA with VIKOR is seemingly more accurate as the analysis involved is in depth and may bear little, if no shortcomings.
- The number of bunkers required need to be kept to a maximum of two bunkers. Simulation shows that bunkers should be provided for the East Main belt as well as the Trunk 4 belt since the mean time before failure of the trunk 3 belt is rather extensive.

11.2 Underground model – solution requirements

Three optimal location points for placing storage buffers to protect production have been identified; these are the points just before trunk 4, trunk, and the east main belt. The second phase involves determining optimal buffer capacities for these locations. To do this it is important to know how much downtime coverage will need to be accounted for, for each of these belts in order to be able to validate results later. Once again box and whisker diagrams will be used for this. (See figure 41 and table 24).

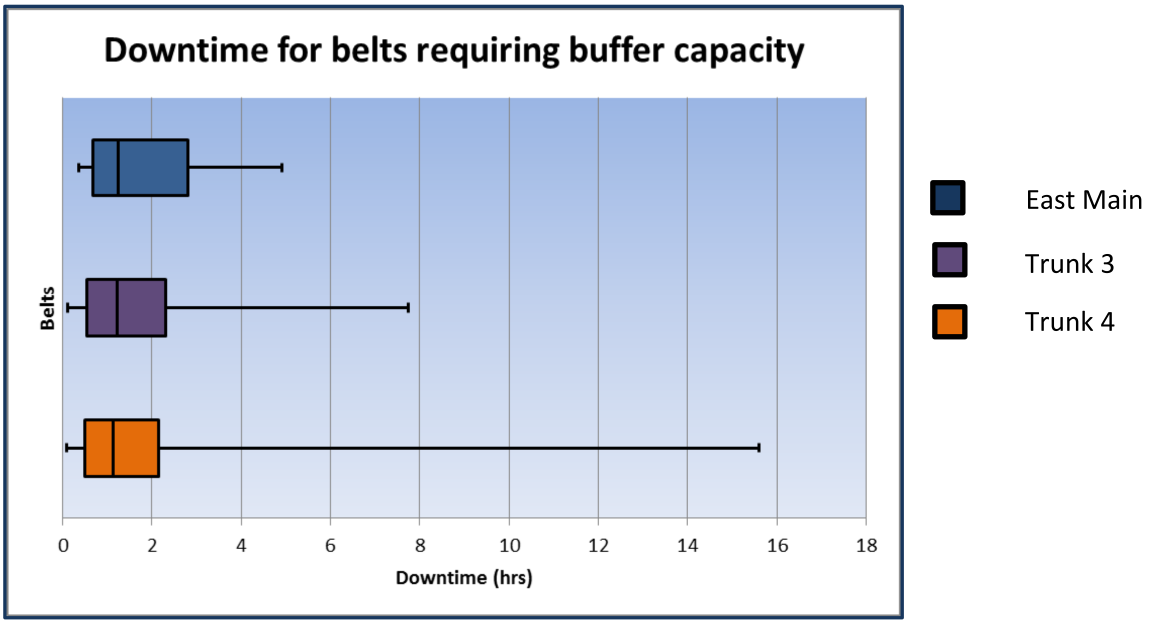


Figure 40. Box and whiskers of the top ranked belts which require risk mitigation strategies

Table 24. Summary statistics for the three belts

Summary statistics					
Trunk4		Trunk3		East Main	
Min	0.08	Min	0.1	Min	0.35
Q1	0.5	Q1	0.54	Q1	0.67
Med	1.125	Med	1.22	Med	1.25
Q3	2.155	Q3	2.31	Q3	2.8
Max	15.6	Max	7.75	Max	4.92

It is highly improbable that maximum downtime values can be accounted for in building storage bunkers as very large bunkers would be required, space underground is highly limited and large bunkers may pose a safety threat as pillars holding the roof of the mine may need to be excavated, however 75% of all downtimes can be accounted for, for each belt and these considerations are taken into account in determining bunker sizes.

11.3 Simulation Model for Bunker sizing

In determining the optimal bunker size for protection of production in the simulation, the bunkers are placed at the identified strategic locations. E.g. figure 42. The red object placed in the simulation is a monitor placed on the East Main belt which monitors the failure of the belt. This monitor changes to the colour red as an indication that the belt has failed and thus, the simulation redirects the coal flow onto the belt leading to the bunker.

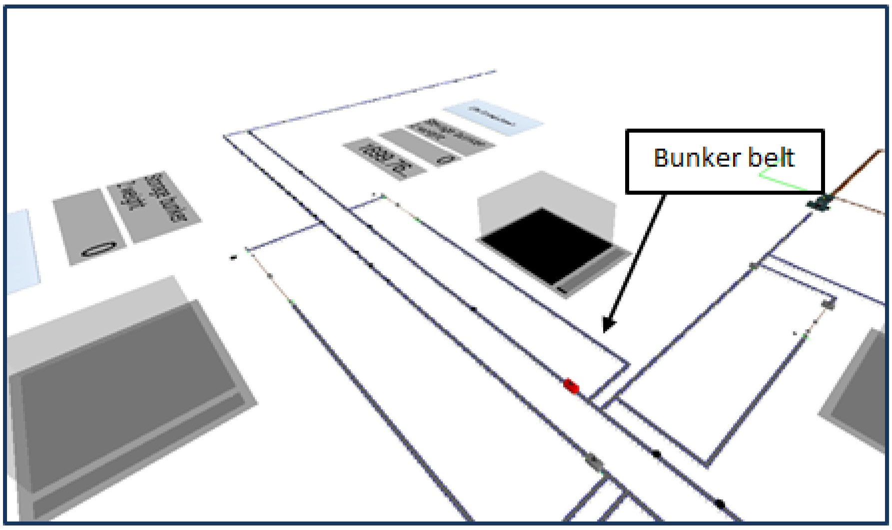


Figure 41. Bunker 1 in operation whilst the East Main belt has failed

The bunkers weights are made infinitely large and have monitors which monitor the level of coal that is in the bunker. In the graph below, the highest points of the graph indicates the size capacity that the bunker needs to be in order to cover the East Main belt down times after 10 replications.

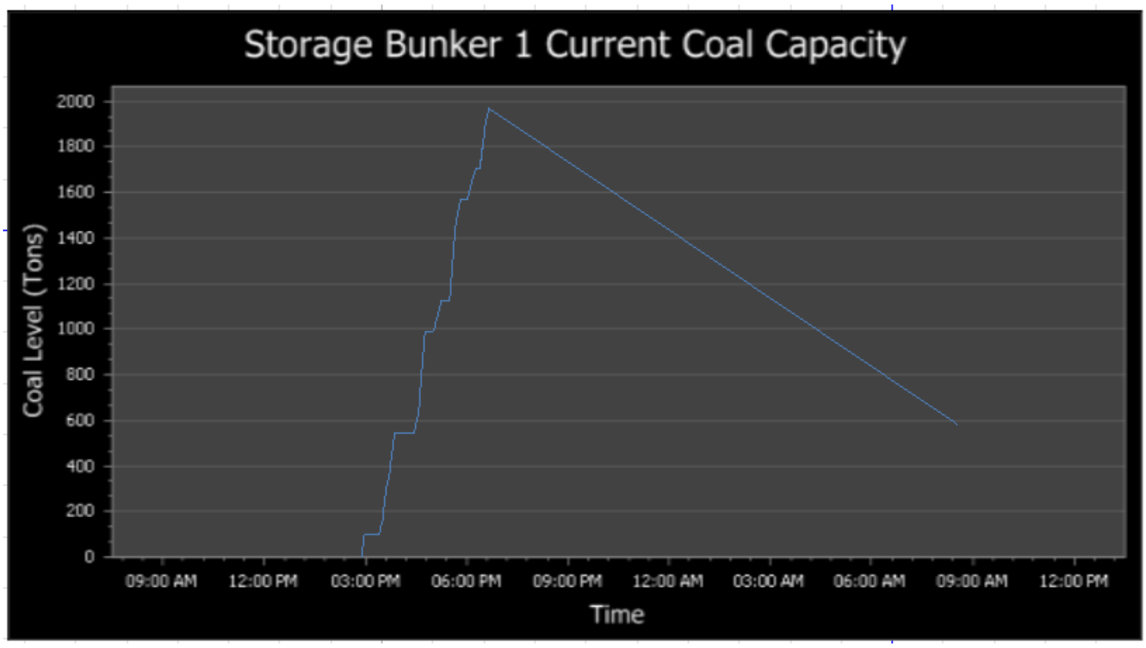


Figure 42. Graph of the optimal size capacity required for Bunker 1 after 10 replications

Storage bunker 2 is placed before trunk 4 and monitors the failure of the belt. Figure 44 shows the optimal size of bunker 2 required in order to protect the production of sections 3 and 2.

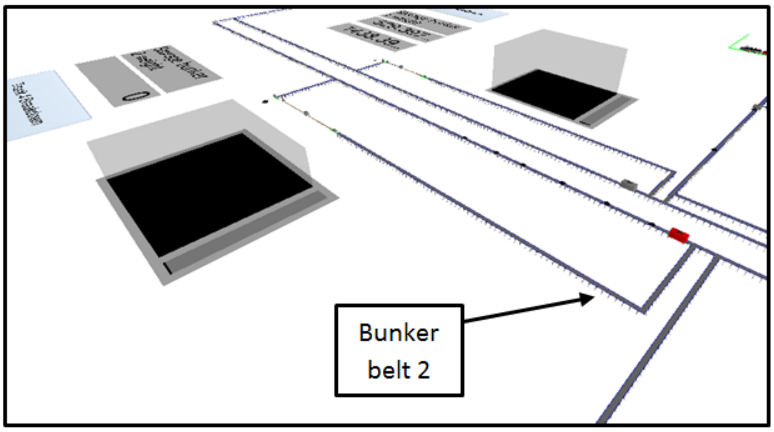


Figure 43. Bunker 2 in operation whilst Trunk 4 has failed

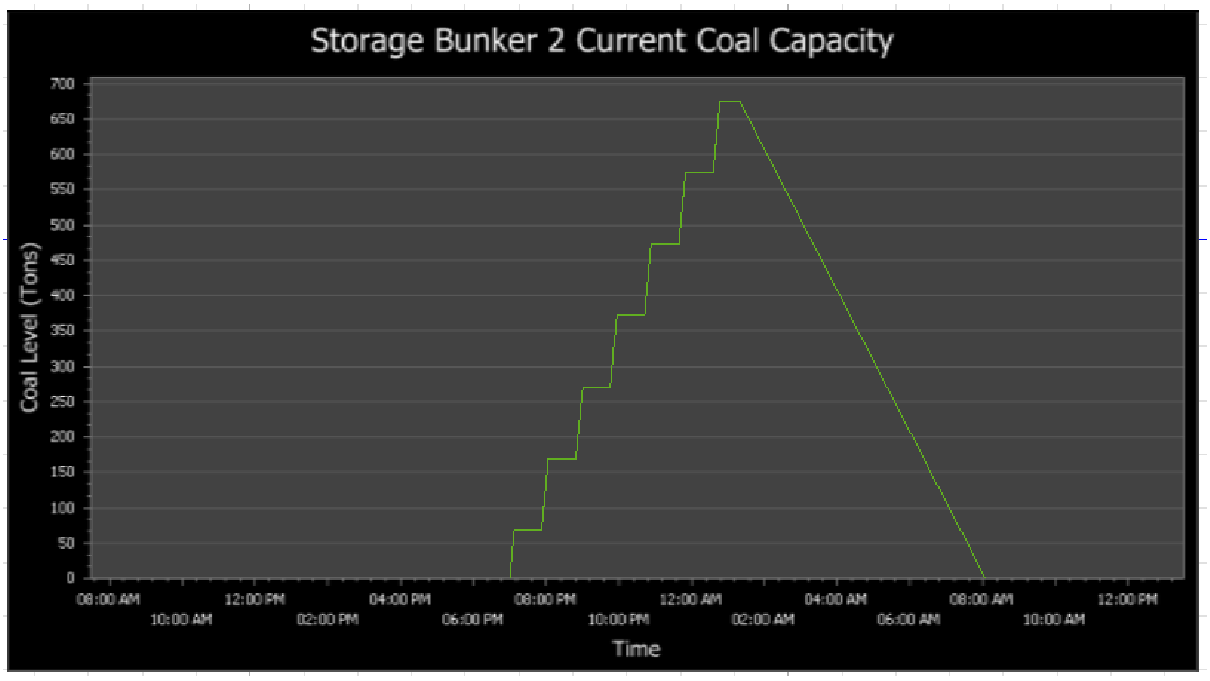


Figure 44. Graph of the optimal size capacity required for Bunker 2 after 10 replications

The following table is an excerpt from the simulation report which reflects the optimal capacities required for the above-mentioned bunkers. ‘Silo2000T’ refers to bunker 2 whilst ‘Silo6000T’ refers to bunker 1 in this instance. It is evident that a storage capacity equivalent to 2000 tons is required for bunker 1 and that of approximately 700 tons required for bunker 2. Although the FMEA with VIKOR method proposes that a bunker be placed before trunk 3, the simulation results reveal that this bunker may be unnecessary as it never filled up after 10 replications.

Table 25. Optimal bunker capacities (in tons) required underground for the East Main belt and Trunk 4

WeightLevel - Maximum			
Object Name	**Data Source**	**Category**	**Value**
Silo2000T	FlowContainer	Content	675.027
Silo6000T	FlowContainer	Content	1966.42199
Tank1	FlowContainer	Content	0

11.3.1 Alternative Solution to Bunker 1 Arrangement

According to Wang (1998), the capacity of bunkers in underground coal mines range between 200 tons and 1000 tons, however much larger bunkers have been employed by mine, up to capacities of 2000 tons. Since bunker 1 requires a capacity of approximately 2000 tons, an alternative solution could be to instead have two bunkers protecting the production of sections 1, 4, and 5, as can be seen in the figure below. Bunker 1 and the auxiliary bunker will both have a maximum capacity of 1000 tons. The auxiliary bunker comes into employment when bunker 1 has reached full capacity and the East Main belt is still in failure mode

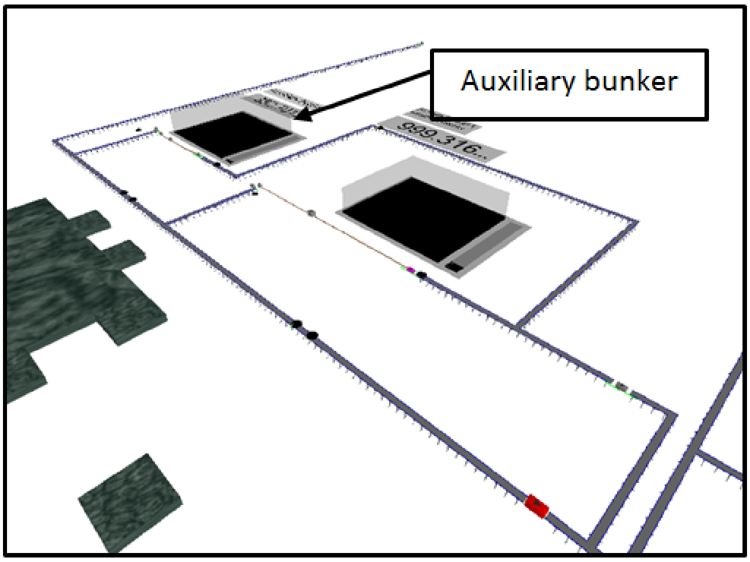


Figure 45. Alternative arrangement of bunker 1

From the graphs below it is seen that the East main belt broke down twice within a very short space of time. This is due to the fact that breakdowns are random and can never truly be anticipated. Through the introduction of an auxiliary bunker both breakdowns were able to be covered, the first breakdown causing the auxiliary bunker to have to come into operation and the second breakdown purely carried by the auxiliary bunker

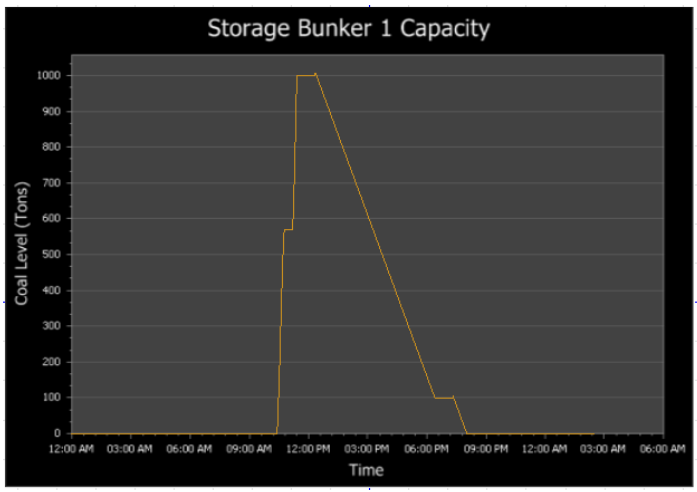


Figure 46. Storage bunker 1 set at a maximum capacity of 1000 tons

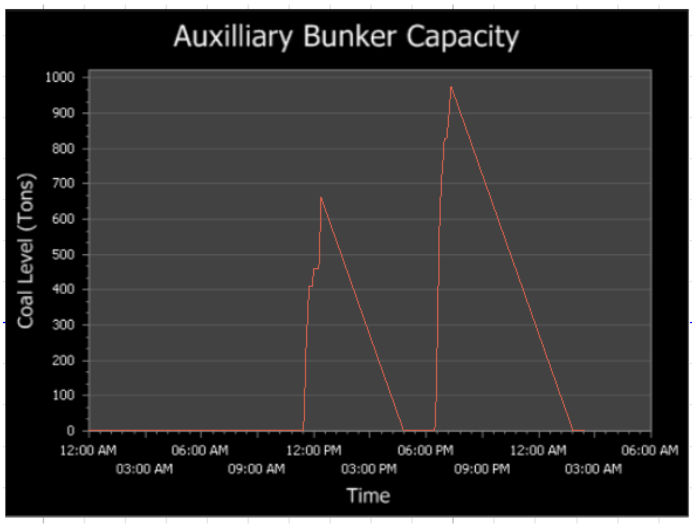


Figure 47. Auxiliary storage bunker set at a maximum capacity of 1000 tons during a simulation run

Table 26. Capacity of bunker 1 (Silo6000T) and the auxiliary bunker (Tank2)

WeightLevel - Maximum			
Object Name	**Data Source**	**Category**	**Value**
Silo6000T	FlowContainer	Content	1007.96374
Tank1	FlowContainer	Content	0
Tank2	FlowContainer	Content	975.06945

The graphs may however be deceiving. When there are two bunkers (bunker 1 and an auxiliary bunker), the total throughput between the two bunkers totals to 1982 tons during the breakdown of the East Main belt (see table above). The total throughput along the East Main belt for the entire simulation run accumulates to 3718 tons, resulting in an additional production rate of 182.70 t/hr along the East Main belt. This is no different to the 3783 tons that were throughput for bunker 1 when it had a capacity of 1966 tons in alternative 1.

Conclusively, the best solution is to keep bunker 1 at 1966 tons and not have bunker 1 at 1000 tons and an auxiliary bunker. The latter would result in increased operational costs.

12. Production Gain ROM (Run Of Mine tonnes)

12.1 Production gain to problem 1 – surface throw-out silo

The installation of a throw out silo of capacity 9200 tons would increase the overall production rate of the mine by 196.69 t/hr. As per simulation results, a breakdown on the surface system has an expected probability of 0.48. With the direct operating hour allocated at 39%, an additional 149 437 tons can be expected from implementation with a payback period of 2.88 years and a return on investment of 65.26%.

12.2 Production gain to problem 2 – underground bunkers

According to the table below (an excerpt from the simulation results), the East Main belt (monitored by Server 1) is in the 'failed state' 11.6% of the time. Under the assumption that the mine is in operation for 344 days of the year (public holidays not accounted for), for 24 hours a day, with Direct Operating Hours (DOH) allocated at 39% (the time that production is expected to be made for the underground operation), and an additional production rate of 131.19 t/hr. due to the installation of bunker 1 at the said location with the said capacity, the mine stands to reflect approximately 48 999.65 additional tonnes. (Computed below)

$$\frac{11.60}{100} \times 344\ days \times 24\ hrs \times 0.39\ DOH \times additional\ prod\ rate\ of\ 131.19 t/hr$$

Similarly, the placement of bunker 2, monitored by server 3 with a failed time percentage of 19.96% at the said location would increase production figures by an additional 39 203 tonnes annually via the equation:

$$\frac{19.96}{100} \times 344\ days \times 24\ hrs \times 0.39\ DOH \times prod\ rate\ of\ 61\ t/hr$$

Accumulating to 88 203 ROM tonnes a year additionally.

Table 27. Percentage Failed time for East Main belt and Trunk 3

FailedTime - Percent			
Object Name	Data Source	Category	Value
FB2	[Resource]	ResourceState	70.38501
FB4	[Resource]	ResourceState	51.17666
Server1	[Resource]	ResourceState	11.59814
Server3	[Resource]	ResourceState	19.95989
sc1_1[1]	[Resource]	ResourceState	5.72622
sc1_2[1]	[Resource]	ResourceState	39.62341
sc1_3[1]	[Resource]	ResourceState	19.83732
sc1_5[1]	[Resource]	ResourceState	21.86216
sc2_1[1]	[Resource]	ResourceState	96.78314
sc2_2[1]	[Resource]	ResourceState	18.7946

13. Cost analysis

Project cost plus facility modifications for the underground storage bunkers amounts to R47 845 867.81 whilst that of the throw-out silo on surface costs R134 508 684 (See appendix J and K, 4000 ton storage facility excluded and provisions made for 700 and 9200 ton storage facility in calculations as well as an escalation cost allowance of 28.5% from 2008 to 2013). A total project cost of R182 354 552.50 is expected for the project in its entirety.

14. Profit Analysis

In computing profit, considerations need to be made for the apportioned ratios between Export and domestic quality coal, as well as exchange rates and the costs associated with mining a ton of coal

Table 28. Projected annual profit from underground bunkers and surface throw-out silo

		Bunker 1	Bunker 2	Throw-out Silo
Export (tons)	63.95%	29 186	19 159.69	194 400.94
Eskom (tons)	16.06%	7 329.49	4 811.65	48 821
Revenue				
Export	700.38	20 441 003.52	13 419 063.68	136 154 530.36
Eskom	165.64	1 214 056.72	797 001.71	8086649.153
Total revenue		21 655 060.25	14 216 065.39	144 241 179.51
Expenses				
Export	193.38	5 643 909.39	3 705 100.85	37593253.78
Eskom	237.55	1 741 120.35	1 143 007.46	11597340.66
Total expenses		7 385 029.74	4 848 108.31	49190594.43
Profit		**14 270 030.50**	**9 367 957.08**	**95 050 585.08**
Total Expected Profit				**118 688 572.66**

An additional annual profit of R23 637 987 can be seen from the implementation of the underground operation. The expected payback period of this project is 1.22 years with an overall increase in utilisation for sections 1, 4, and 5 of 9.83% from 46.83% and sections 3 and 2 experiencing an increase in utilisation of 7.46% from 60.84% (Appendix B).

That of the surface operation enjoys an annual profit of R95 050 585 with an expected payback period of 2.88 years, increasing system utilisation by 29.83%.

15. Conclusion

System requirements for downtime coverage of the surface and the underground model have been met in the following manner:

The surface model requires an additional silo of capacity 9200 tons in order to increase system availability and to negate the effects of complete system shut down. This capacity ties up with the tools and techniques employed at arriving at a workable solution.

The model shows that FMEA with VIKOR and fuzzy numbers for determining optimal strategic points tends to be much more vigorous and detailed as opposed to the conventional way of rating risks. The simulation model augments this method for validity of results by proposing two storage bunkers instead of three after carefully examining system behaviour.

Furthermore the 2 proposed bunkers require the following capacities: bunker 1 with a capacity of 2000 tons installed before the East Main belt to protect the production of section 4, 5, and 1, this will see an increase of almost 10% in system utilisation for those sections. Bunker 2 with a capacity of 675 tons installed before Trunk 4 to protect the production of section 3 and 2 will increase system utilisation by 7.5%. The implementation of the surface solution will see significant profit figures with a payback period of 2.88 years and a return on investment of almost 66%. Thus it can be seen that this project is most definitely viable with tremendous gains for Anglo Thermal Coal as a whole.

16. References

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Appendix A: Industry form

Department of Industrial & Systems Engineering
Final Year Projects
Identification and Responsibility of Project Sponsors

All Final Year Projects are published by the University of Pretoria on *UPSpace* and thus freely available on the Internet. These publications portray the quality of education at the University and have the potential of exposing sensitive company information. It is important that both students and company representatives or sponsors are aware of such implications.

Key responsibilities of Project Sponsors:

A project sponsor is the key contact person within the company. This person should thus be able to provide the best guidance to the student on the project. The sponsor is also very likely to gain from the success of the project. The project sponsor has the following important responsibilities:

1. Confirm his/her role as project sponsor, duly authorised by the company. Multiple sponsors can be appointed, but this is not advised. The duly completed form will considered as acceptance of sponsor role.
2. Review and approve the Project Proposal, ensuring that it clearly defines the problem to be investigated by the student and that the project aim, scope, deliverables and approach is acceptable from the company's perspective.
3. Review the Final Project Report (delivered during the second semester), ensuring that information is accurate and that the solution addresses the problems and/or design requirements of the defined project.
4. Acknowledges the intended publication of the Project Report on UP Space.
5. Ensures that any sensitive, confidential information or intellectual property of the company is not disclosed in the Final Project Report.

Project Sponsor Details:

Company:	ANGLO AMERICAN INYOSI COAL - ZIBULO COLLIERY
Project Description:	A SIMULATION APPROACH TO CONSTRAINTS MANAGEMENT OF AN UNDERGROUND COAL MINE CONVEYOR SYSTEM
Student Name:	KATLEGO MANKGE
Student number:	29646751
Student Signature:	K. Mankge.
Sponsor Name:	Stephen Ross
Designation:	Asset Optimisation.
E-mail:	stephen.ross@angloamerican.com.
Tel No:	(013) 643 4648
Cell No:	072 374 3341
Fax No:	
Sponsor Signature:	

Appendix B: Total Availability Model

DESCRIPTION	UCM6006	UCM6007	UCM6008	UCM6010	UCM6011	UCM6012	UCM6013	UCM6802
TOTAL TIME	**720**	**720**	**720**	**720**	**720**	**720**	**720**	**720**
NON PRODUCTION TIME	**136**	**128**	**136**	**136**	**128**	**136**	**136**	**128**
NON PRODUCTION SHIFT	136	128	136	136	128	126	136	128
SCHEDULED NONS MAINTENANCE						10		
SECTION MOVES								
SHUTDOWN & PROJECT								
PUBLIC HOLIDAY								
UNSCHEDULED NONS MAINTENANCE								
UNCONTROLABLE TIME	**1.33**	**1.33**	**1.33**	**1.33**	**1.33**	**1.33**	**1.33**	**1.33**
RAW MATERIALS								
U/C SCHEDULED MAINTENANCE								
U/C UNSCHEDULED MAINTENANCE								
MARKET CONDITIONS								
NON SCHEDULED HOLIDAY								
U/C LABOUR								
ENVIRONMENTAL U/C								
EXTERNAL UTILITIES	1.33	1.33	1.33	1.33	1.33	1.33	1.33	1.33
CONTROLLABLE TIME	**582.67**	**590.67**	**582.67**	**582.67**	**590.67**	**582.67**	**582.67**	**590.67**
ENGINEERING DOWNTIME	**34.82**	**35.74**	**70.75**	**72.76**	**55.07**	**79.66**	**39.09**	**29.51**
UNSCHEDULED ENG MAINTENANCE	**19.82**	**22.49**	**50.25**	**58.09**	**40.15**	**59.99**	**19.09**	**18.01**
ENGINEERING DELAYS	5.33		1.17	1	1.5	7.83	5.67	
BOILER MAKING		4.17	5.08					0.75
ELECTRICAL	1.83	3.17	15.92	27.5	8.99	24.41	0.75	3.43
HYDRAULIC	4.83	3.17	0.75	11.09	2.75	20.33	4.33	10.5
CONTROL & INSTRUMENTATION								
MECHANICAL	7.83	11.98	27.33	18.5	26.91	7.42	8.34	3.33
SCHEDULED ENG MAINTENANCE	**15**	**13.25**	**20.5**	**14.67**	**14.92**	**19.67**	**20**	**11.5**
FOLLOW-ON WORK								
DELAYS								
MODIFICATION								
INSPECTIONS								
SERVICES & SHUTDOWN	15	13.25	20.5	14.67	14.92	19.67	20	11.5
ENGINEERING AVAILABILITY%	94.02%	93.95%	87.86%	87.51%	90.68%	86.33%	93.29%	95.00%
PRODUCTION DOWNTIME	**40.28**	**18.67**	**20.83**	**48.5**	**42.92**	**38.25**	**59.56**	**22.55**
ACCIDENT / DAMAGE	14.67	1.42	3.58	17.73	4	4.92	22	8.34
DIESEL / ELECT / AIR / WATER SUPPLY	14.59	13.75	9.92	14.35	13.92	20.17	26.74	7.34
TYRES & ROPES								
GET	4.27	2.33	0.58	12.17	8.66	9.41	7.07	5.01
BLOCKAGE / STUCK			4.5		13.34	0.5		
UNALLOCATED								
OPS CONSUMABLE					1.83		2.58	
PROCESS INTERUPTION	6.75	1.17	2.25	4.25	1.17	3.25	1.17	1.86
PRODUCTION AVAILABILITY %	92.65%	96.64%	95.93%	90.49%	91.99%	92.40%	89.04%	95.98%
EQUIPMENT AVAILABILITY%	87.11%	90.79%	84.28%	79.19%	83.41%	79.76%	83.07%	91.19%
PRODUCTION DELAYS	**25.58**	**9.66**	**10.16**	**10.16**	**29.68**	**9.24**	**4.92**	**7.4**
ENVIRONMENTAL OPS								
NO OPERATOR / SUP		6.41						
OPERATIONAL DELAY	24.58	3.25	10.16	10.16	29.68	9.24	0.67	7.4
EQUIPMENT MOVES								
STAFF RELATED								
EXTERNAL DELAYS							4.25	
EXTERNAL OPS DELAY								
EXTERNAL ENG DELAY							4.25	
SERVICE DELAY								
SAFETY	1							
SUPPLY CHAIN OPS								
LOST TIME	**196.11**	**185.07**	**108.81**	**178.21**	**199.6**	**181.74**	**141.91**	**187.71**
CONSEQUENTIAL	**196.11**	**185.07**	**108.81**	**178.21**	**199.6**	**181.74**	**141.91**	**187.71**
CONSEQUENTIAL ENGINEERING	43.08	42.65	12.83	67.26	42.64	54.38	30.4	75.06
CONSEQUENTIAL PRODUCTION	153.03	142.42	95.98	110.95	156.96	127.36	111.51	112.65
STANDBY								
SERVICE METER HOURS	**285.85**	**341.53**	**372.1**	**273.01**	**263.35**	**273.74**	**337.17**	**343.47**
DIRECT OPERATING TIME	**285.85**	**341.53**	**372.1**	**273.01**	**263.35**	**273.74**	**337.17**	**343.47**
PRODUCTION	285.85	341.53	372.1	273.01	263.35	273.74	337.17	343.47
NON PRODUCTION								
ERROR TIME	**0.03**		**0.02**	**0.03**	**0.05**	**0.04**	**0.02**	**0.03**
SYSTEM EFFICIENCY %	56.32%	63.69%	75.77%	59.17%	53.45%	58.90%	69.66%	63.77%
USE OF ENGINEERING AVAILABILITY%	52.18%	61.54%	72.69%	53.54%	49.17%	54.42%	62.03%	61.21%
OVERALL UTILISATION%	49.06%	57.82%	63.86%	46.85%	44.58%	46.98%	57.87%	58.15%

Appendix C: Expense Revenue figures for 2012

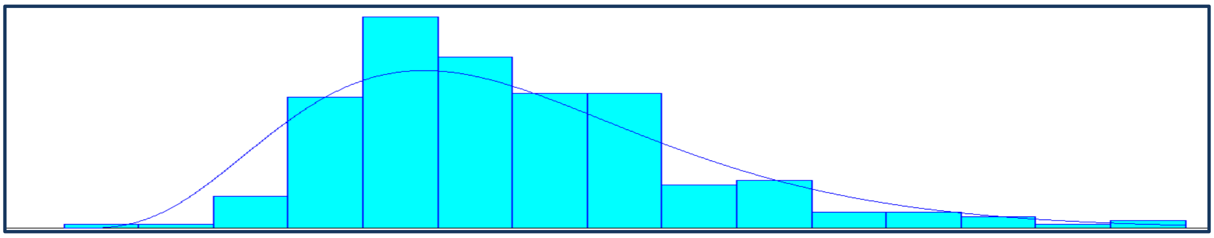
Actual cost figures for 2012													
Revenue		**Jan**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**
$ Price		99.08	95.82	100.36	92.67	86.44	75.51	80.42	80.97	85.39	73.16	75.4	81.99
Exchange Rate ®		7.97	7.68	7.58	7.83	8.19	8.42	8.25	8.27	8.26	8.9	8.7	8.58
R price per ton (export)		789.67	736.06	760.73	725.61	708.13	635.42	663.15	669.62	705.32	651.06	656.31	703.47
Eskom price per ton ®		208.93	182.29	183.99	163.98	159.74	177.09	152.13	171.24	181.83	109.98	88.67	207.84
Expenses													
Cash costs (R/ton):													
	Salable for PRT	267.45	256.52	223.76	237.48	213.97	204.66	236.99	243.42	235.04	201.1	216.28	313.89
	Salable Total Mine*	305.66	293.16	255.72	271.41	244.54	233.9	270.84	278.19	268.62	229.83	247.18	358.73
Total Export Selling Expenses (R/ton)		202.07	221.86	157.54	157.27	157.27	139.31	157.27	260.45	302.88	395.2	242.37	-72.97
	Rail	108.46	140.06	90.45	124.72	124.72	124.72	124.72	124.72	124.72	124.72	124.72	54.09
	Warfage	9.6	9.6	9.6	9.6	9.6	9.6	9.6	9.6	9.6	25.84	2.37	9.6
	FOB Selling Expenses	84.01	72.2	57.49	22.95	22.95	4.99	22.95	126.13	168.54	244.64	115.26	-136.66

Appendix D: Belt Information

Belt speed, capacity and lengths			
Section belts			
	Speed (m/s)	Capacity (t/hr)	Length (m)
SB1	3.5	feeder breaker set @ 830 t/hr	1252
SB2	3.1	feeder breaker set @ 830 t/hr	1013
SB3	3.5	feeder breaker set @ 830 t/hr	352
SB4	3.5	feeder breaker set @ 830 t/hr	550
SB5	3.1	feeder breaker set @ 830 t/hr	890
SB6	2.1	feeder breaker set @ 830 t/hr	200
SB7	3.5	feeder breaker set @ 830 t/hr	660
SB8	3.1	feeder breaker set @ 830 t/hr	320
Gathering belts			
	Speed (m/s)	Capacity (t/hr)	Length (m)
Incline	4.5	4200	980
EM1	4.1	3600	1713
EM2	3.1	1000	869
T2	3.5	1200	1269
T3	2.5	800	350
T4	3.1	1200	1818
T5	3.5	1200	280
T7	3.5	1200	150
T8	3.2	1200	140
T9	2.5	800	260
scv002	4.5	4200	120
Overland	4.5	4200	16400

Appendix E: Inter-arrival time distributions

Section 1			Date: 01 Jan 2012 -04 December 2012									
CM Code:	6802											
Inter-arrival time (seconds)												
142	123	138	157	144	172	169	118	179	177	189	185	187
149	129	127	136	101	157	160	150	180	179	172	180	215
138	152	120	183	125	154	175	144	224	223	195	132	185
169	133	140	180	159	158	133	117	146	198	217	167	182
164	143	120	160	120	173	168	159	180	148	173	174	
138	175	142	165	142	210	160	156	262	194	237	180	
162	140	124	160	146	200	130	204	178	132	216	178	
141	123	135	161	127	158	166	168	177	213	209	188	
139	117	143	147	153	168	135	155	210	159	211	188	
128	168	134	146	140	156	146	169	285	149	249	151	
148	138	133	153	142	154	137	158	185	195	229	210	
149	117	119	147	152	184	147	183	253	143	286	199	
81	133	145	163	131	160	177	149	232	142	156	169	
151	142	136	176	153	216	150	175	178	176	164	162	
135	130	145	144	151	192	130	152	147	159	225	165	
145	127	131	163	139	182	146	182	174	173	181	131	
145	133	156	153	155	186	156	235	202	240	159	137	
123	164	143	122	145	185	141	211	176	179	144	185	
136	182	132	149	133	183	171	192	250	159	155	127	
137	131	169	160	163	185	147	202	206	169	127	153	

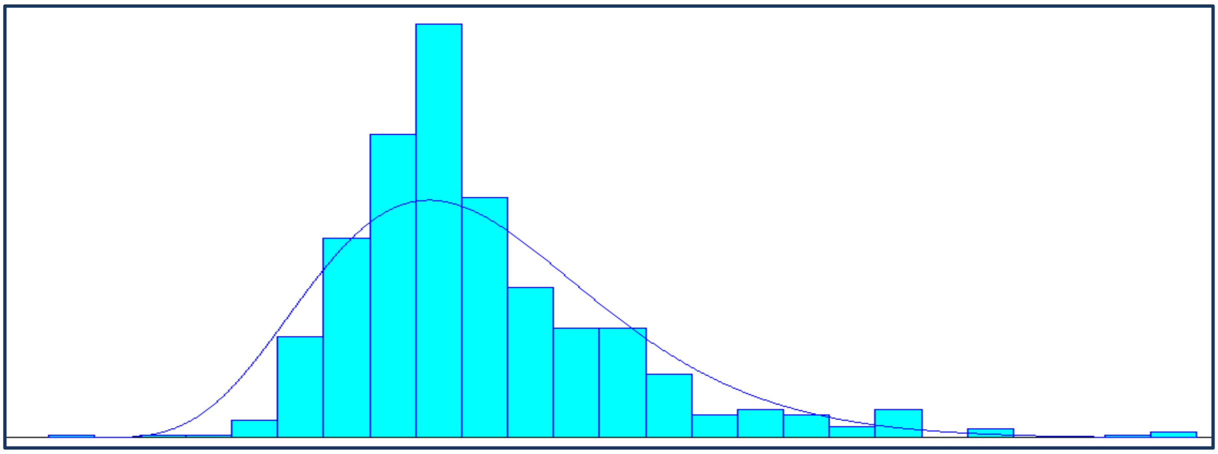


```
        Distribution Summary

Distribution:   Erlang
Expression:     81 + ERLA(16.4, 5)
Square Error:   0.009034

Chi Square Test
  Number of intervals    = 9
  Degrees of freedom     = 6
  Test Statistic         = 25.1
  Corresponding p-value  < 0.005

Kolmogorov-Smirnov Test
  Test Statistic         = 0.0875
  Corresponding p-value  = 0.0459

        Data Summary

Number of Data Points    = 244
Min Data Value           = 81
Max Data Value           = 286
Sample Mean              = 163
Sample Std Dev           = 31.7

        Histogram Summary

Histogram Range          = 81 to 286
Number of Intervals      = 15
```

Section 2 Date: 01 Jan 2012 -04 December 2012
CM Code: 6012
Inter-arrival time (seconds)

144	140	148	145	145	149	150	129	133	168	142	180	142	165	120	123
132	136	158	172	110	137	143	115	125	121	170	128	130	142	149	137
155	134	139	152	135	154	156	135	131	150	135	171	130	162	158	116
131	79	135	124	156	122	161	127	140	142	152	153	131	172	128	145
139	232	147	155	130	142	152	122	129	103	168	162	134	132	145	146
162	150	119	118	127	138	145	134	139	92	141	169	133	137	136	145
150	186	147	159	149	139	177	115	133	137	173	151	122	147	144	133
154	151	134	145	129	150	55	143	117	115	145	185	136	145	119	133
115	133	141	157	134	140	127	146	147	134	118	125	162	153	154	142
149	141	157	157	145	132	134	142	115	135	177	137	116	153	116	139
125	162	149	117	130	150	126	166	142	117	121	143	132	144	129	163
141	121	147	140	162	115	118	122	190	146	125	156	179	126	145	130
119	120	160	142	182	141	152	138	121	126	150	140	147	136	131	124
150	132	146	113	145	203	136	110	131	158	154	145	150	150	136	123
124	127	136	132	149	155	140	127	122	165	134	124	150	125	116	139
131	136	181	151	165	162	105	137	153	193	176	152	158	132	135	109
197	149	147	112	149	159	132	169	112	134	160	162	148	137	142	137
132	129	117	138	142	145	147	141	142	147	137	144	151	108	158	132
143	160	170	142	125	160	117	111	138	137	131	142	131	156	116	114
167	142	151	136	117	211	122	125	154	199	219	123	148	143	154	113

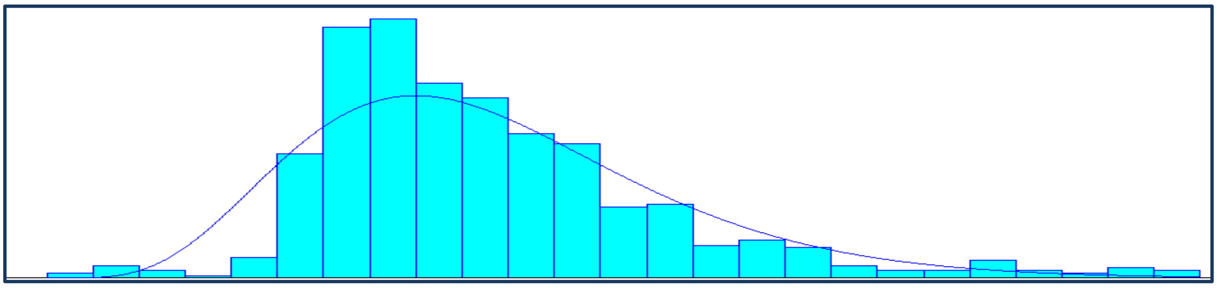
126	141	132	222	159	196	126	182	165	149	153	191	184	205	188	151
102	133	142	169	189	139	152	248	162	175	144	249	234	224	163	243
133	113	104	231	170	170	189	127	161	171	169	147	196	248	174	139
136	160	140	151	134	167	127	172	190	158	136	132	157	190	220	151
126	137	153	157	170	218	149	195	146	133	196	157	181	186	177	180
168	162	143	155	255	186	177	185	119	187	197	123	141	234	182	146
142	155	187	171	225	190	141	273	179	144	147	157	230	205	195	169
133	146	126	179	193	150	178	181	187	166	179	139	149	151	154	138
148	152	136	185	323	205	129	150	205	166	122	189	185	171	178	144
127	144	157	171	194	150	151	149	186	197	134	151	143	167	179	166
148	124	130	170	148	195	155	146	172	280	250	187	181	178	172	151
132	108	144	173	316	169	213	166	128	218	136	251	223	303	214	140
148	121	147	174	208	151	159	146	148	227	209	148	169	225	179	160
158	128	140	272	143	172	168	155	127	197	209	164	249	227	169	141
141	140	145	180	222	185	248	203	191	164	247	237	173	248	189	192
130	136	148	147	166	147	172	204	157	137	160	188	249	186	155	142
132	127	155	183	124	173	161	149	185	149	190	174	184	172	180	146
115	129	137	144	161	133	206	150	152	168	197	161	141	168	240	190
189	152	178	127	160	186	139	159	133	195	150	136	160	128	147	140
167	149	120	173	152	197	158	203	154	174	170	242	191	209	145	173



```
        Distribution Summary

Distribution:   Gamma
Expression:     55 + GAMM(12.1, 8.33)
Square Error:   0.014323

Chi Square Test
  Number of intervals   = 14
  Degrees of freedom    = 11
  Test Statistic        = 102
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0826
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 640
Min Data Value          = 55
Max Data Value          = 323
Sample Mean             = 156
Sample Std Dev          = 32.5

        Histogram Summary

Histogram Range         = 55 to 323
Number of Intervals     = 25
```

Section 3	Date: 01 Jan 2012 -04 December 2012														
CM Code:	6007														
Inter-arrival time (seconds)															
141	185	130	159	138	133	137	192	135	157	156	145	118	146	132	138
149	126	84	156	145	138	177	121	219	85	131	170	148	127	138	121
175	139	121	112	175	134	152	139	111	139	153	153	144	158	135	133
127	132	162	131	133	148	136	155	125	145	130	136	130	118	119	128
160	145	123	152	141	131	186	117	175	167	114	150	92	124	154	111
206	181	138	145	160	137	141	136	163	134	140	124	122	166	126	127
120	135	167	180	124	163	134	183	160	144	163	131	162	118	121	94
132	138	133	128	122	131	166	139	121	138	154	165	126	160	133	119
191	182	136	194	152	135	151	123	143	166	126	128	138	120	142	140
131	144	177	131	127	137	162	136	162	157	189	130	143	123	133	142
137	144	149	206	158	127	130	126	156	129	189	139	129	117	132	166
193	182	167	149	125	170	141	151	168	144	114	156	157	147	137	74
197	144	141	145	136	143	184	131	177	118	162	139	133	126	156	140
210	80	119	144	153	150	126	178	130	138	120	133	199	118	122	133
147	178	204	158	148	158	142	167	179	162	153	130	122	148	116	124
136	146	131	141	160	145	157	130	196	151	139	145	162	128	133	142
155	196	154	134	131	132	131	163	127	127	130	148	157	135	145	124
175	136	143	157	133	171	132	182	127	78	159	118	165	140	156	115
138	94	184	127	162	143	140	126	194	139	132	134	129	119	138	138
137	171	155	146	130	147	139	151	124	128	134	149	136	157	160	147

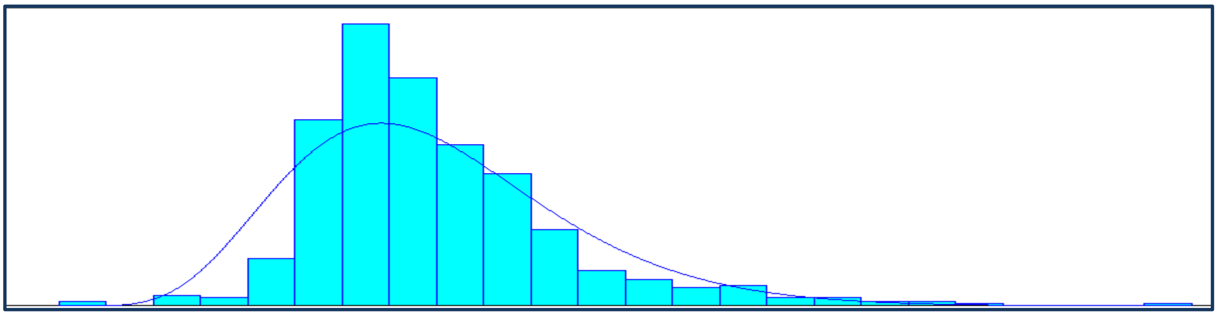
137	126	108	129	216	140	152	145	143	182	171	175	183	177	206	221	153
123	119	121	118	168	142	235	171	163	183	140	234	164	226	163	150	142
138	136	132	119	217	217	260	168	149	215	154	151	227	167	196	175	193
147	164	124	140	220	265	229	163	156	163	207	173	158	299	295	139	155
122	117	125	143	288	179	206	221	190	147	133	166	216	172	303	193	289
67	116	135	143	161	175	163	146	139	181	175	147	212	144	198	191	235
85	141	122	129	199	198	151	183	154	132	257	128	171	228	167	170	226
123	117	119	135	196	146	138	164	173	157	223	142	180	176	150	153	174
165	117	125	120	138	135	203	174	172	177	155	157	154	168	173	218	195
139	144	138	137	149	156	219	179	181	139	239	161	206	187	186	161	172
128	111	143	142	182	187	220	168	146	190	146	246	281	215	138	133	203
139	120	142	131	259	132	150	175	174	199	150	165	267	157	251	134	257
107	130	135	169	190	152	179	155	176	158	171	137	286	165	261	211	199
140	125	132	130	151	157	192	177	200	150	175	176	293	215	256	151	243
135	144	131	130	172	193	150	163	195	161	174	166	267	176	204	190	
137	121	102	132	201	156	209	140	160	155	183	157	153	198	217	170	
142	141	128	212	121	187	166	165	164	135	142	253	151	267	250	184	
140	136	144	115	159	138	164	156	175	151	133	185	175	183	196	162	
132	174	129	130	170	150	171	179	208	174	185	151	183	158	276	166	
120	131	119	161	157	180	148	162	220	143	140	179	159	226	210	169	



```
        Distribution Summary

Distribution:   Erlang
Expression:     67 + ERLA(15.1, 6)
Square Error:   0.008721

Chi Square Test
  Number of intervals   = 16
  Degrees of freedom    = 13
  Test Statistic        = 85.9
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0851
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 654
Min Data Value          = 67
Max Data Value          = 303
Sample Mean             = 157
Sample Std Dev          = 35.6

        Histogram Summary

Histogram Range         = 67 to 303
Number of Intervals     = 25
```

Section 4	Date: 01 Jan 2012 -04 December 2012																
CM Code:	6010																
Inter-arrival time (seconds)																	
142	176	129	147	143	140	131	126	144	124	144	135	144	145	123	120	130	143
146	162	126	148	167	139	138	60	132	122	122	123	130	138	147	156	124	127
126	135	151	115	111	128	182	132	129	150	112	125	133	136	125	136	214	145
152	156	134	156	173	127	149	136	157	132	133	135	129	144	146	143	140	149
155	165	160	148	141	130	164	87	127	124	139	109	143	124	191	137	145	127
112	160	134	124	84	155	165	129	145	144	135	130	139	128	128	127	205	168
154	158	158	150	152	115	123	138	150	137	122	142	136	148	140	139	135	148
147	92	169	147	149	156	152	142	150	125	125	127	131	164	145	129	141	182
121	140	137	170	125	144	199	129	171	129	125	128	129	161	138	137	144	143
149	158	137	149	142	127	234	134	129	136	131	113	148	147	129	135	139	170
139	142	179	145	140	99	218	145	171	127	124	128	138	141	159	138	132	138
129	145	162	128	128	144	223	159	136	142	132	115	165	145	71	123	157	191
148	158	162	152	167	118	176	136	139	149	119	128	140	144	143	132	136	172
119	148	142	162	145	154	169	131	142	130	140	122	126	84	133	119	144	154
142	144	178	144	140	134	173	153	113	139	139	122	184	137	147	126	171	150
161	152	159	133	127	132	152	137	100	137	122	117	139	111	137	124	211	191
155	153	139	178	123	170	114	96	128	128	134	121	171	118	138	163	138	238
156	158	133	111	111	119	110	177	127	150	136	111	137	136	161	85	204	240
156	127	152	162	121	137	145	139	135	143	124	141	134	145	169	156	145	259
141	148	121	97	136	142	184	137	132	121	129	141	131	130	162	144	133	238

176	138	141	167	158	180	168	176	180	191	165	160	134
263	245	157	180	199	138	158	260	148	165	163	148	157
247	151	141	163	200	148	153	274	348	154	159	148	136
236	146	137	176	181	133	194	171	189	221	147	144	120
209	131	148	157	152	124	138	216	183	177	125	139	165
190	167	154	139	150	175	220	175	183	156	155	162	142
169	227	179	152	145	154	205	191	144	167	180	181	184
175	159	223	124	144	147	249	183	189	276	144	160	143
153	207	158	175	151	203	146	229	187	190	138	232	150
223	176	183	137	145	176	173	160	234	189	165	167	165
142	140	170	173	143	164	255	189	179	205	198	142	184
169	149	130	151	133	177	158	187	189	174	162	181	153
166	175	157	143	157	201	119	176	216	233	198	132	
144	138	132	296	146	241	165	179	137	180	200	227	
169	155	142	125	185	285	219	165	168	184	195	190	
193	154	160	127	178	179	179	160	169	187	137	139	
144	133	192	196	164	153	169	139	153	177	177	121	
235	135	171	156	169	198	185	171	179	169	161	152	
144	143	166	198	168	213	202	164	169	173	128	133	
157	157	216	154	164	170	175	207	192	286	179	140	



```
        Distribution Summary

Distribution:   Gamma
Expression:     60 + GAMM(13, 7.31)
Square Error:   0.013892

Chi Square Test
  Number of intervals    = 13
  Degrees of freedom     = 10
  Test Statistic         = 91.3
  Corresponding p-value  < 0.005

Kolmogorov-Smirnov Test
  Test Statistic         = 0.105
  Corresponding p-value  < 0.01

        Data Summary

Number of Data Points    = 612
Min Data Value           = 60
Max Data Value           = 348
Sample Mean              = 155
Sample Std Dev           = 32.5

        Histogram Summary

Histogram Range          = 60 to 348
Number of Intervals      = 24
```

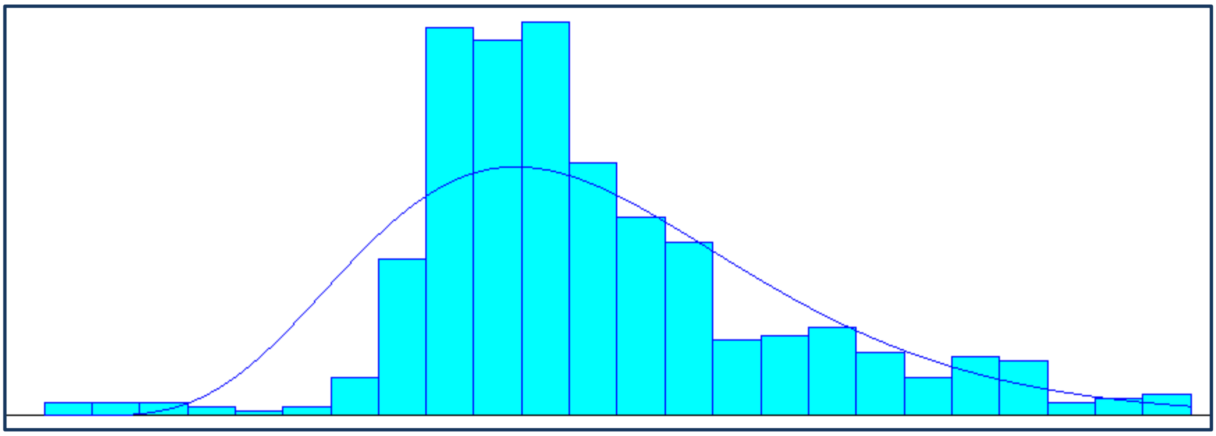
Section 5 Date: 01 Jan 2012 -04 December 2012

CM Code: 6008

Inter-arrival time (seconds)

144	140	186	181	138	151	138	151	78	161	133	67	147	144	152	162	148
170	159	91	160	206	70	148	153	133	171	198	105	151	143	147	170	131
179	162	171	183	146	151	150	155	148	131	207	167	158	149	164	150	152
190	172	143	154	144	145	156	113	143	135	209	134	153	147	153	175	156
136	162	175	123	154	137	155	135	119	116	85	149	143	155	163	161	172
160	152	174	184	126	140	160	132	139	127	149	138	172	164	161	137	156
179	143	145	191	145	140	148	95	145	145	160	162	146	175	137	172	159
161	185	171	168	225	134	144	144	139	145	140	131	163	141	149	158	149
145	183	148	167	134	156	161	154	146	163	140	140	149	162	160	142	134
172	148	146	151	152	148	163	162	125	151	143	160	175	166	147	177	150
146	144	142	170	165	140	147	74	141	68	169	159	145	136	185	135	135
166	164	162	172	146	126	134	136	152	185	147	158	134	158	159	141	148
168	161	171	64	161	140	140	172	134	246	131	152	155	163	161	172	147
165	271	152	165	171	138	154	147	160	152	161	157	151	158	160	164	135
160	159	167	133	136	150	156	134	159	149	147	134	151	160	139	159	147
156	155	167	171	147	146	156	142	136	139	165	162	147	145	159	171	142
137	147	169	207	140	138	147	153	119	146	151	151	156	159	152	146	153
162	192	201	167	126	150	153	148	145	160	155	240	160	191	179	156	157
122	152	166	160	148	151	143	162	156	143	141	160	58	158	180	146	196
146	153	158	141	159	82	136	152	139	144	140	138	147	137	174	154	167

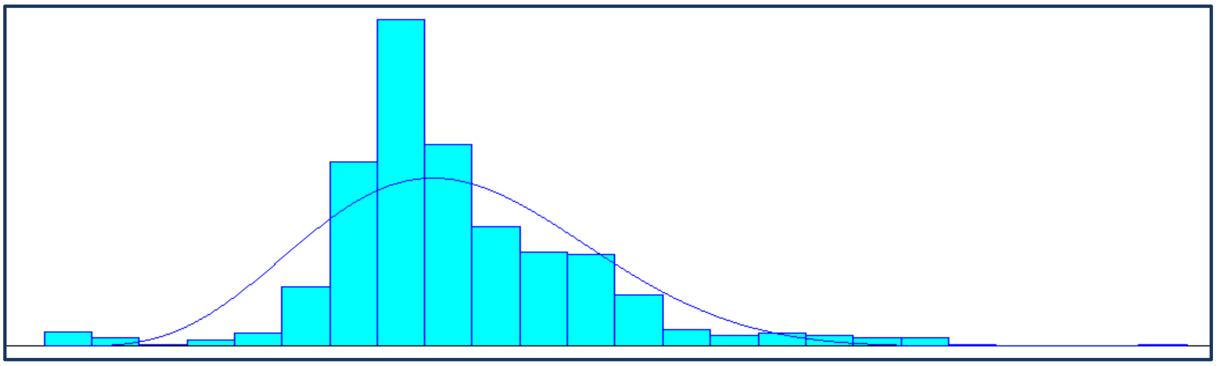
199	163	190	228	191	217	234	202	227	197	243	155	236	145
192	196	177	178	151	166	255	244	202	260	178	151	145	158
186	250	255	162	180	214	197	237	266	219	218	136	250	259
192	165	151	185	192	144	226	174	265	240	289	151	174	
177	211	179	193	234	234	201	186	239	244	222	139	188	
236	290	220	197	177	209	203	262	238	269	166	171	182	
198	197	195	154	206	182	185	289	195	270	175	230	188	
194	298	257	173	197	171	218	197	210	181	174	164	171	
218	195	248	184	180	182	210	222	229	209	195	162	195	
251	160	186	188	137	264	186	183	253	220	183	182	187	
183	179	221	221	196	229	251	201	162	174	185	143	150	
221	221	199	220	143	201	260	160	169	267	190	259	165	
187	222	159	185	175	225	172	199	279	180	226	288	183	
232	176	191	181	214	172	193	217	298	134	141	233	174	
226	192	165	177	157	169	173	167	257	146	149	257	182	
252	194	280	153	199	266	255	236	251	171	195	209	216	
262	192	156	178	160	191	201	170	200	212	142	167	224	
256	221	260	190	152	236	175	193	194	192	151	180	247	
217	162	212	210	152	282	232	177	162	171	157	165	173	
208	169	193	223	158	172	191	265	175	168	135	183	144	



```
        Distribution Summary

Distribution:   Erlang
Expression:     58 + ERLA(16.4, 7)
Square Error:   0.015645

Chi Square Test
  Number of intervals   = 17
  Degrees of freedom    = 14
  Test Statistic        = 130
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.134
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 603
Min Data Value          = 58
Max Data Value          = 298
Sample Mean             = 173
Sample Std Dev          = 38.5

        Histogram Summary

Histogram Range         = 58 to 298
Number of Intervals     = 24
```

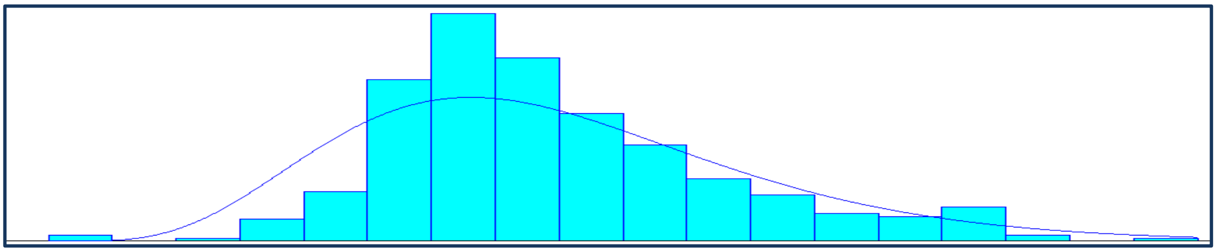
Section 6	Date: 01 Jan 2012 -04 December 2012															
CM Code:	6006															
Inter-arrival time (seconds)																
132	128	145	175	145	144	146	130	155	161	202	151	148	157	127	194	178
156	136	128	184	146	139	160	157	155	113	128	143	157	147	128	160	265
138	139	132	125	150	163	146	152	193	138	126	136	143	147	160	132	172
128	110	133	131	129	149	142	146	119	135	129	75	164	136	124	56	186
111	125	112	130	136	161	116	172	157	205	142	80	184	151	123	148	297
131	138	129	143	165	139	156	165	98	152	166	63	170	147	135	49	197
126	150	153	134	165	144	154	138	153	119	131	188	157	149	147	159	180
139	148	157	152	154	145	149	188	165	154	142	151	132	142	62	129	250
146	127	122	172	184	170	136	117	151	147	135	147	157	128	109	141	205
140	62	147	140	187	142	136	129	143	135	137	138	150	167	146	141	181
154	144	146	148	161	153	129	130	182	147	141	144	163	162	133	143	279
158	119	131	129	166	157	145	62	150	141	158	152	115	149	148	149	148
137	125	151	151	153	180	138	159	128	142	143	139	134	147	66	178	183
153	131	147	159	175	153	145	146	170	138	155	141	162	139	121	254	202
122	140	146	148	112	61	131	153	159	124	137	131	133	150	139	149	197
141	151	145	159	149	147	147	152	154	126	134	134	159	126	163	186	182
151	116	201	150	143	136	146	152	159	139	137	139	143	143	201	143	188
145	145	168	153	142	158	142	161	160	145	75	145	137	136	137	162	178
156	161	165	134	168	144	135	137	136	94	149	152	149	140	197	153	201
115	140	162	149	148	152	50	193	148	121	153	156	148	142	176	148	186

160	181	126	196	144	153	199	139	127	168	196	196	139
280	180	166	166	145	163	232	176	149	183	171	165	177
217	153	162	184	119	210	250	148	137	172	216	198	
190	170	146	242	167	175	203	151	167	199	155	158	
151	153	198	149	155	217	196	137	288	181	162	192	
198	169	175	180	126	274	173	163	138	203	196	164	
139	194	154	164	183	153	218	199	201	178	150	227	
156	187	363	148	245	182	186	167	221	194	152	193	
163	170	168	142	164	157	200	208	189	202	147	151	
166	221	145	155	181	158	184	167	195	259	168	167	
152	184	147	131	212	149	168	160	217	264	146	149	
177	226	157	193	138	173	213	176	210	259	174	146	
186	248	194	243	166	158	217	215	151	175	144	207	
205	211	176	171	180	184	209	229	302	175	156	174	
203	207	175	253	214	153	177	220	159	174	161	141	
204	205	184	157	194	151	190	261	156	144	166	159	
154	168	207	141	242	181	201	168	183	186	174	162	
205	296	213	162	224	189	176	248	182	171	168	163	
280	243	213	161	212	213	177	183	216	168	148	141	
165	293	101	150	212	162	162	149	201	175	145	191	



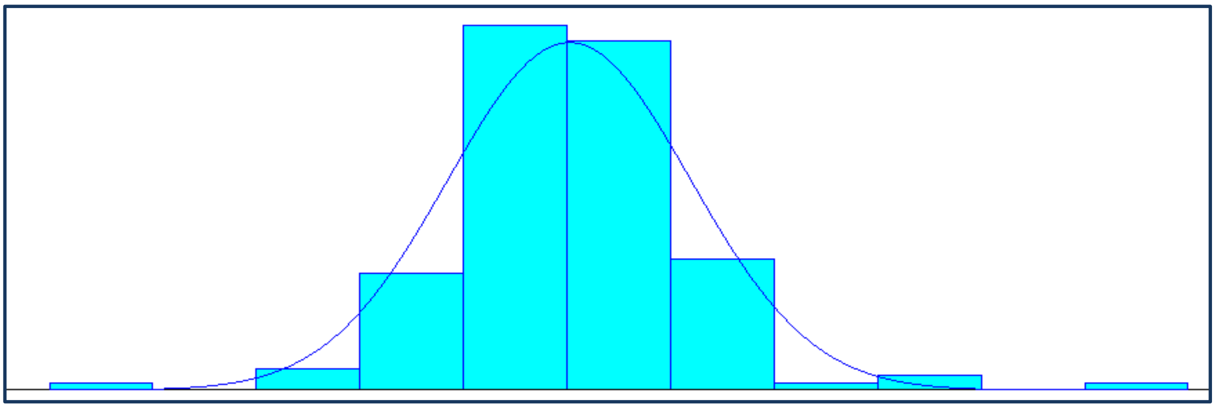
```
        Distribution Summary

Distribution:   Beta
Expression:     49 + 314 * BETA(5.68, 10.1)
Square Error:   0.025298

Chi Square Test
  Number of intervals   = 12
  Degrees of freedom    = 9
  Test Statistic        = 139
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.103
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 582
Min Data Value          = 49
Max Data Value          = 363
Sample Mean             = 162
Sample Std Dev          = 36.8

        Histogram Summary

Histogram Range         = 49 to 363
Number of Intervals     = 24
```

Section 7	Date: 01 Jan 2012 -04 December 2012																
CM Code:	6011																
Inter-arrival time (seconds)																	
115	202	271	179	171	199	148	140	156	149	123	230	153	169	157	214	245	182
131	145	179	160	158	139	125	152	142	162	142	254	156	162	228	219	248	173
144	267	208	177	143	174	131	169	142	162	165	238	149	163	185	205	200	189
110	73	234	186	165	183	143	158	150	197	177	159	258	158	191	218	192	253
148	246	209	170	205	162	160	156	156	157	175	227	231	151	179	160	168	185
132	173	218	150	196	267	168	178	158	159	176	171	162	121	148	193	228	167
123	268	158	160	199	203	153	163	172	167	165	179	186	203	178	211	191	204
188	205	226	147	273	146	157	204	160	144	170	200	201	164	184	176	192	221
157	178	230	152	167	158	162	176	160	209	178	198	157	130	187	181	168	
142	250	145	163	194	168	223	183	164	267	214	217	216	177	205	179	138	
139	115	148	163	196	154	315	163	187	140	168	165	189	207	160	173	237	
163	249	179	155	182	155	161	151	188	126	170	165	150	237	176	171	149	
149	281	152	164	208	166	159	171	264	146	215	149	200	157	175	170	142	
165	264	174	158	188	135	163	145	225	162	186	159	132	142	138	177	232	
133	182	167	72	165	148	161	151	227	143	213	178	177	197	229	190	140	
161	237	182	154	183	146	153	176	173	201	146	188	181	208	144	157	200	
129	153	172	180	202	141	130	236	191	185	193	158	159	273	138	204	176	
179	210	176	183	209	217	154	222	178	185	158	187	177	198	152	178	200	
154	238	177	147	180	151	149	162	205	122	285	208	178	266	209	193	201	
162	193	161	146	185	138	172	241	148	147	150	185	151	250	264	196	249	



```
        Distribution Summary

Distribution:   Erlang
Expression:     72 + ERLA(17.8, 6)
Square Error:   0.013741

Chi Square Test
  Number of intervals   = 11
  Degrees of freedom    = 8
  Test Statistic        = 49.4
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.117
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 348
Min Data Value          = 72
Max Data Value          = 315
Sample Mean             = 179
Sample Std Dev          = 35.9

        Histogram Summary

Histogram Range         = 72 to 315
Number of Intervals     = 18
```

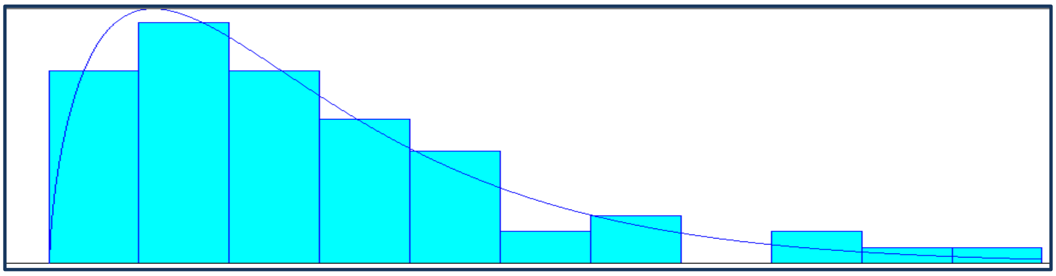
Section 8	Date: 01 Jan 2012 -04 December 2012					
CM Code:	6013					
Inter-arrival time (seconds)						
148	148	165	143	157	152	148
141	135	151	145	136	154	150
165	161	122	173	196	106	146
130	147	159	131	138	117	153
157	126	135	144	120	120	164
115	137	127	146	144	129	61
96	159	122	157	133	125	130
143	147	131	175	198	132	144
144	149	156	139	153	131	166
143	164	159	177	139	134	163
154	130	126	164	148	138	162
133	158	161	144	127	139	138
148	133	144	145	123	142	142
147	155	155	143	147	150	119
243	149	173	141	130	141	168
138	149	122	142	120	153	133
149	131	141	141	142	161	137
137	141	105	132	133	159	169
167	137	119	146	143	161	132
129	157	146	149	142	154	136



```
Distribution:    Normal
Expression:      NORM(144, 19.4)
Square Error:    0.007969

Chi Square Test
  Number of intervals   = 4
  Degrees of freedom    = 1
  Test Statistic        = 5.8
  Corresponding p-value = 0.0178

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0771
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 140
Min Data Value          = 61
Max Data Value          = 243
Sample Mean             = 144
Sample Std Dev          = 19.4

        Histogram Summary

Histogram Range         = 61 to 243
Number of Intervals     = 11
```

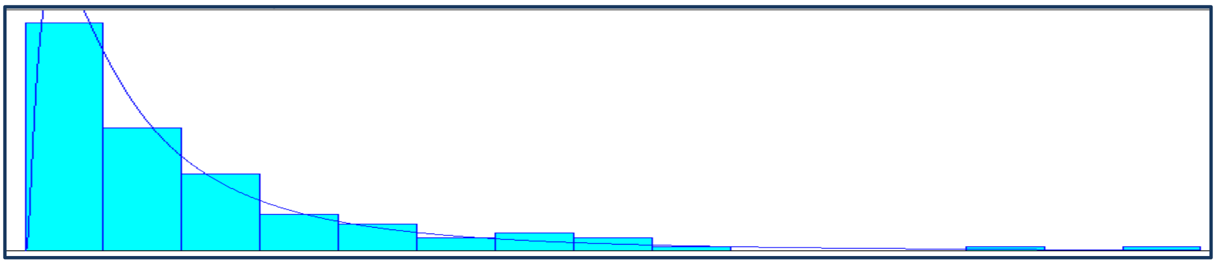
Appendix F: Section belt downtime distributions

Section belt 1



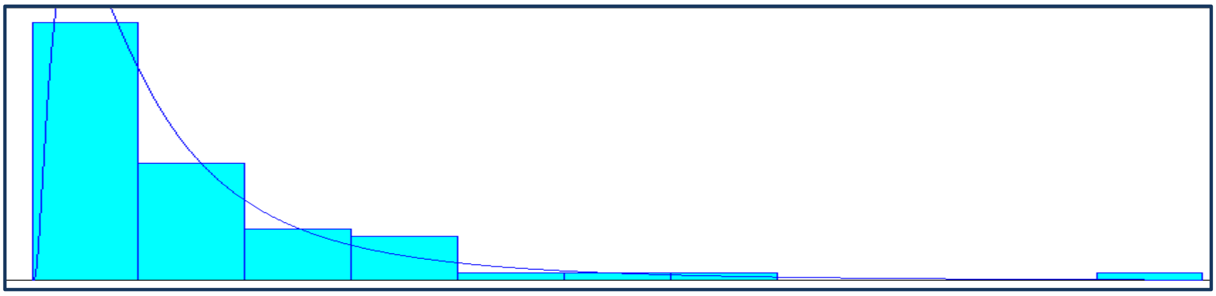
```
        Distribution Summary

Distribution:   Gamma
Expression:     GAMM(2.1, 1.65)
Square Error:   0.002241

Chi Square Test
  Number of intervals   = 6
  Degrees of freedom    = 3
  Test Statistic        = 1.04
  Corresponding p-value > 0.75

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0485
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 128
Min Data Value          = 0.32
Max Data Value          = 13
Sample Mean             = 3.46
Sample Std Dev          = 2.75

        Histogram Summary

Histogram Range         = 0 to 13
Number of Intervals     = 11
```

Section belt 2



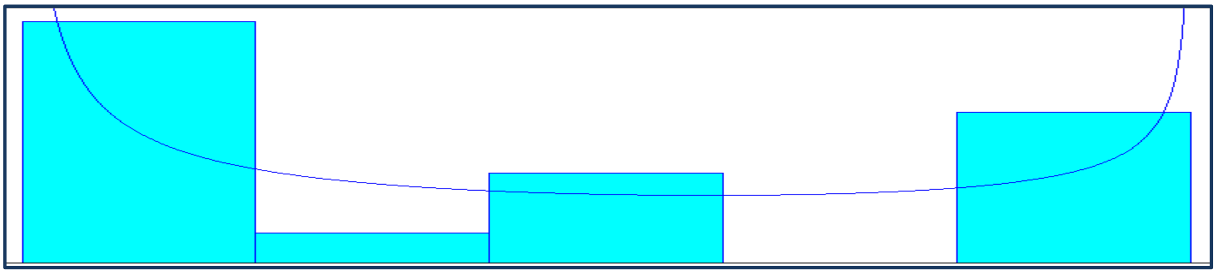
```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(3.7, 5.45)
Square Error:   0.001702

Chi Square Test
  Number of intervals   = 7
  Degrees of freedom    = 4
  Test Statistic        = 2.34
  Corresponding p-value = 0.678

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0569
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 242
Min Data Value          = 0.17
Max Data Value          = 24
Sample Mean             = 3.52
Sample Std Dev          = 3.89

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 15
```

Section belt 3



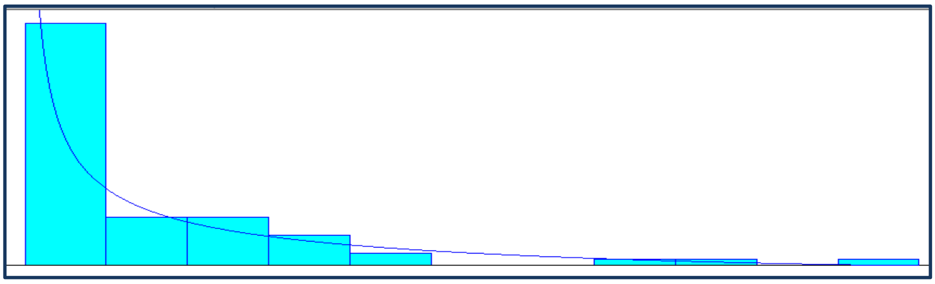
```
        Distribution Summary

Distribution:    Lognormal
Expression:      LOGN(2.81, 3.4)
Square Error:    0.002890

Chi Square Test
  Number of intervals   = 4
  Degrees of freedom    = 1
  Test Statistic        = 2.94
  Corresponding p-value = 0.0897

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0817
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 136
Min Data Value          = 0.17
Max Data Value          = 19.1
Sample Mean             = 2.76
Sample Std Dev          = 2.97

        Histogram Summary

Histogram Range         = 0 to 20
Number of Intervals     = 11
```

Section belt 4



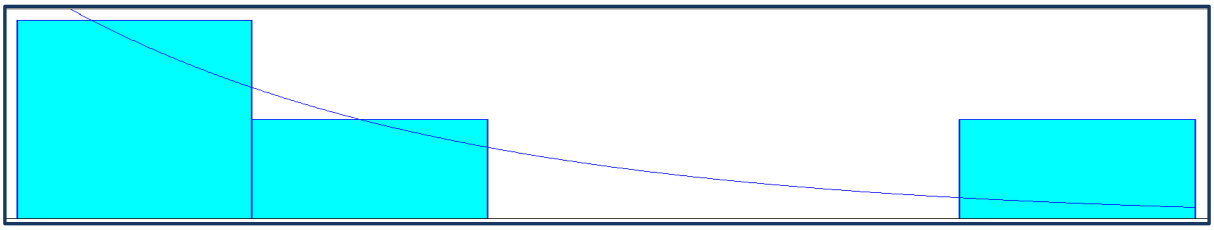
```
        Distribution Summary

Distribution:   Beta
Expression:     6 * BETA(0.469, 0.629)
Square Error:   0.033617

Chi Square Test
  Number of intervals   = 4
  Degrees of freedom    = 1
  Test Statistic        = 3.73
  Corresponding p-value = 0.0547

Kolmogorov-Smirnov Test
  Test Statistic        = 0.177
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 34
Min Data Value          = 0.17
Max Data Value          = 5.58
Sample Mean             = 2.56
Sample Std Dev          = 2.05

        Histogram Summary

Histogram Range         = 0 to 6
Number of Intervals     = 5
```

Section belt 5



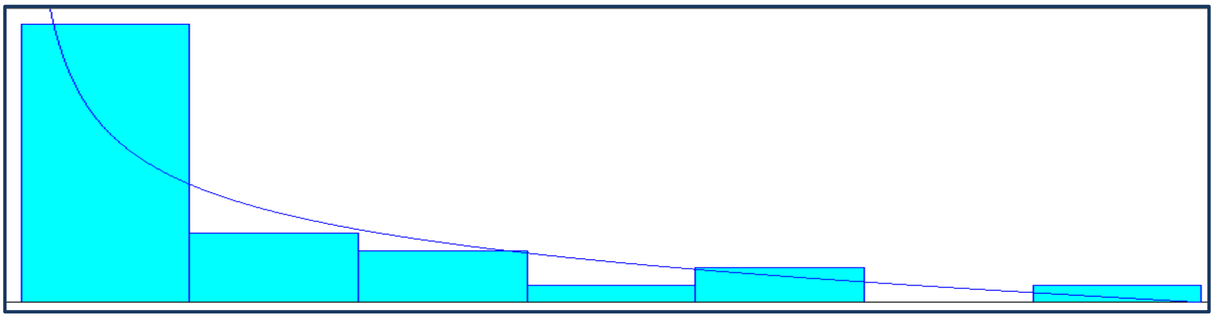
```
        Distribution Summary

Distribution:   Beta
Expression:     24 * BETA(0.372, 2.23)
Square Error:   0.010158

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 9.35
  Corresponding p-value = 0.00951

Kolmogorov-Smirnov Test
  Test Statistic        = 0.2
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 132
Min Data Value          = 0.17
Max Data Value          = 24
Sample Mean             = 3.43
Sample Std Dev          = 4.42

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 11
```

Section belt 6



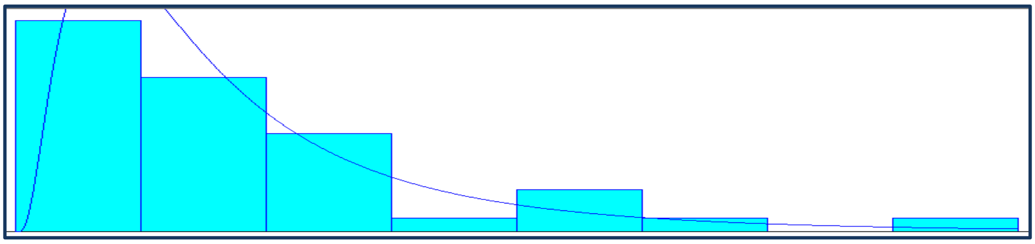
```
        Distribution Summary

Distribution:   Exponential
Expression:     0.999 + EXPO(1.98)
Square Error:   0.069772


Kolmogorov-Smirnov Test
  Test Statistic          = 0.221
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 8
Min Data Value          = 1
Max Data Value          = 7
Sample Mean             = 2.98
Sample Std Dev          = 2.56

        Histogram Summary

Histogram Range         = 0.999 to 7
Number of Intervals     = 5
```

Section belt 7



```
        Distribution Summary

Distribution:   Beta
Expression:     10 * BETA(0.553, 1.94)
Square Error:   0.007969

Chi Square Test
  Number of intervals    = 4
  Degrees of freedom     = 1
  Test Statistic         = 1.69
  Corresponding p-value  = 0.21

Kolmogorov-Smirnov Test
  Test Statistic         = 0.292
  Corresponding p-value  < 0.01

        Data Summary

Number of Data Points    = 54
Min Data Value           = 0.42
Max Data Value           = 10
Sample Mean              = 2.22
Sample Std Dev           = 2.23

        Histogram Summary

Histogram Range          = 0 to 10
Number of Intervals      = 7
```

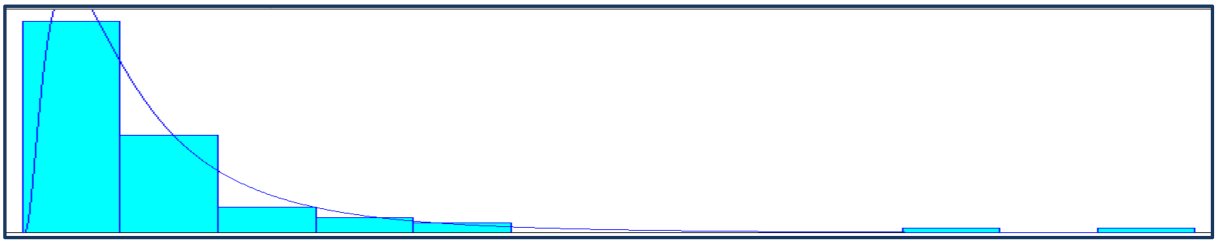
Section belt 8



```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(2.33, 2.25)
Square Error:   0.008034

Chi Square Test
  Number of intervals    = 4
  Degrees of freedom     = 1
  Test Statistic         = 1.17
  Corresponding p-value  = 0.294

Kolmogorov-Smirnov Test
  Test Statistic         = 0.0738
  Corresponding p-value  > 0.15

        Data Summary

Number of Data Points    = 78
Min Data Value           = 0.33
Max Data Value           = 9.75
Sample Mean              = 2.3
Sample Std Dev           = 1.97

        Histogram Summary

Histogram Range          = 0 to 10
Number of Intervals      = 8
```

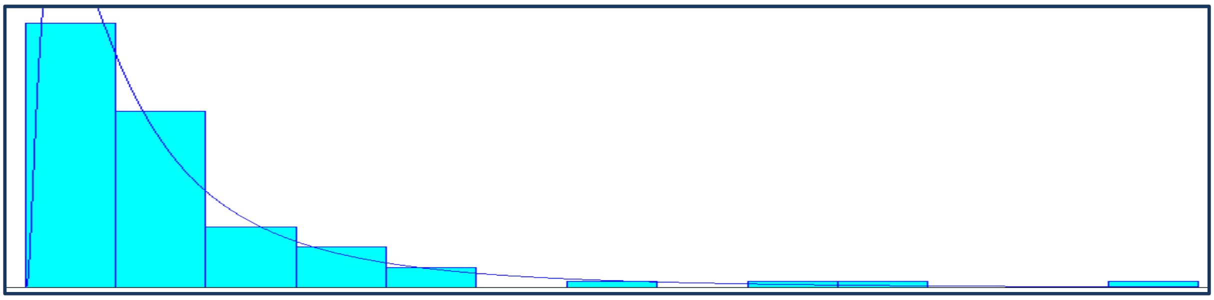
Appendix G: Trunk belts (Gathering belts) downtime distributions

Trunk belt 8



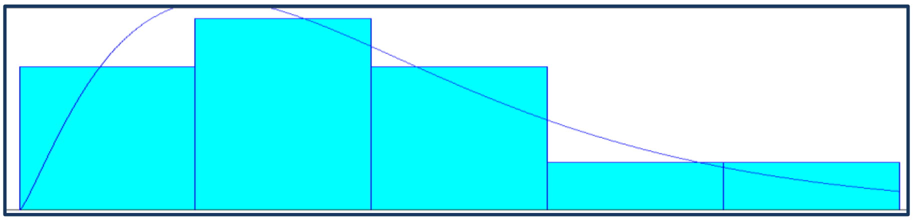
```
        Distribution Summary

Distribution:    Lognormal
Expression:      LOGN(2.91, 3.07)
Square Error:    0.007692

Chi Square Test
  Number of intervals   = 4
  Degrees of freedom    = 1
  Test Statistic        = 3.84
  Corresponding p-value = 0.0502

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0547
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 144
Min Data Value          = 0.33
Max Data Value          = 24
Sample Mean             = 3.02
Sample Std Dev          = 3.68

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 12
```

Trunk belt 3



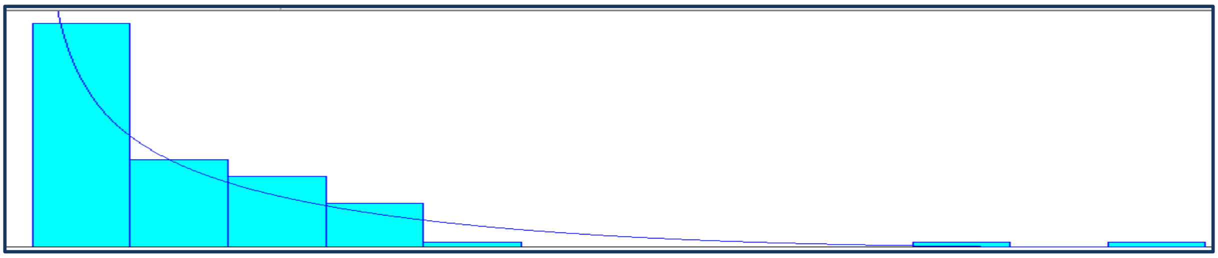
```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(3.29, 4.35)
Square Error:   0.002966

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 2.82
  Corresponding p-value = 0.246

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0564
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 174
Min Data Value          = 0.17
Max Data Value          = 24
Sample Mean             = 3.23
Sample Std Dev          = 3.85

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 13
```

East Main 1



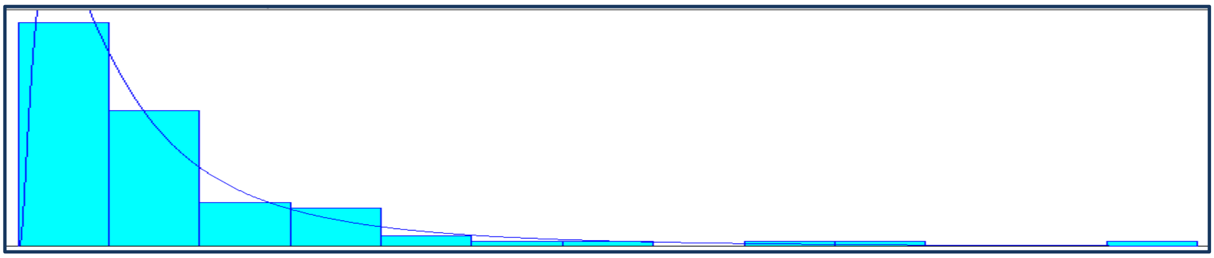
```
        Distribution Summary

Distribution:   Gamma
Expression:     GAMM(1.28, 2.24)
Square Error:   0.003089

Chi Square Test
  Number of intervals   = 3
  Degrees of freedom    = 0
  Test Statistic        = 0.212
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.176
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 24
Min Data Value          = 0.25
Max Data Value          = 6.5
Sample Mean             = 2.85
Sample Std Dev          = 1.69

        Histogram Summary

Histogram Range         = 0 to 7
Number of Intervals     = 5
```

Trunk belt 4



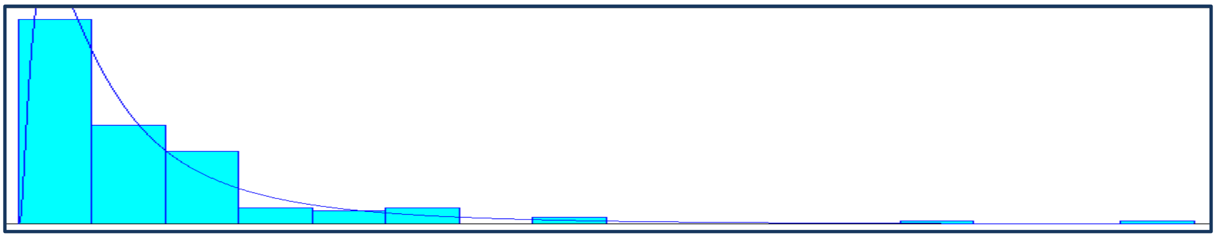
```
        Distribution Summary

Distribution:   Beta
Expression:     24 * BETA(0.569, 3.54)
Square Error:   0.007227

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 23
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.153
  Corresponding p-value < 0.01

        Data Summary

Number of Data Points   = 162
Min Data Value          = 0.17
Max Data Value          = 24
Sample Mean             = 3.32
Sample Std Dev          = 3.67

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 12
```

Trunk belt 2



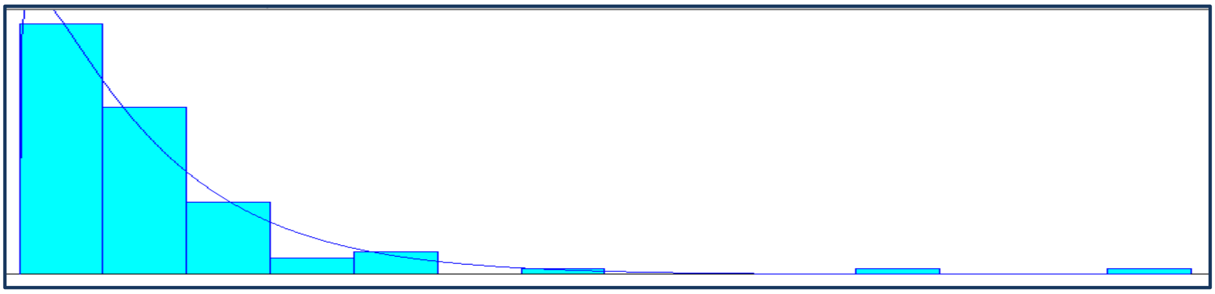
```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(3.2, 4.11)
Square Error:   0.001969

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 3.54
  Corresponding p-value = 0.187

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0656
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 176
Min Data Value          = 0.17
Max Data Value          = 24
Sample Mean             = 3.19
Sample Std Dev          = 3.8

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 13
```

Trunk belt 9



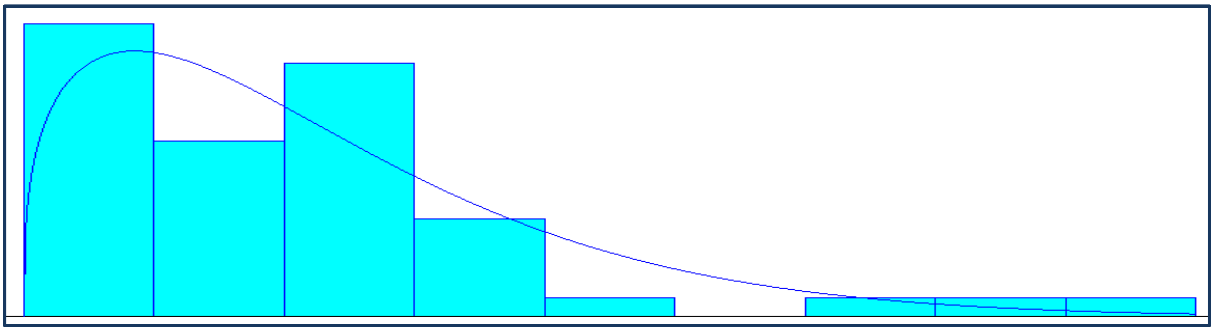
```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(2.76, 3.57)
Square Error:   0.005199

Chi Square Test
  Number of intervals   = 6
  Degrees of freedom    = 3
  Test Statistic        = 12.4
  Corresponding p-value = 0.00649

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0436
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 264
Min Data Value          = 0.12
Max Data Value          = 24
Sample Mean             = 2.71
Sample Std Dev          = 3.22

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 16
```

Trunk belt 5



```
        Distribution Summary

Distribution:   Gamma
Expression:     GAMM(2.35, 1.13)
Square Error:   0.004043

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 6.11
  Corresponding p-value = 0.0479

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0736
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 196
Min Data Value          = 0.25
Max Data Value          = 24
Sample Mean             = 2.66
Sample Std Dev          = 3.36

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 14
```

Trunk belt 7

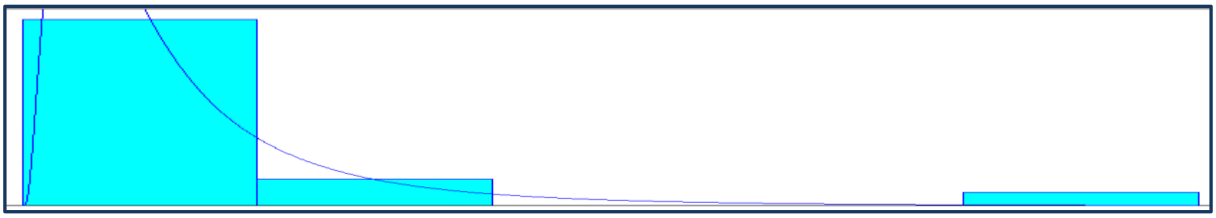


```
        Distribution Summary

Distribution:   Weibull
Expression:     WEIB(2.48, 1.32)
Square Error:   0.022208

Chi Square Test
  Number of intervals   = 5
  Degrees of freedom    = 2
  Test Statistic        = 16.2
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0883
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 92
Min Data Value          = 0.33
Max Data Value          = 9
Sample Mean             = 2.27
Sample Std Dev          = 1.87

        Histogram Summary

Histogram Range         = 0 to 9
Number of Intervals     = 9
```

Incline

```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(1.94, 2.84)
Square Error:   0.001601

Chi Square Test
  Number of intervals   = 4
  Degrees of freedom    = 1
  Test Statistic        = 4.64
  Corresponding p-value = 0.033

Kolmogorov-Smirnov Test
  Test Statistic        = 0.0524
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 208
Min Data Value          = 0.08
Max Data Value          = 24
Sample Mean             = 1.99
Sample Std Dev          = 3.14

        Histogram Summary

Histogram Range         = 0 to 24
Number of Intervals     = 14
```

Mug Incline



```
        Distribution Summary

Distribution:   Lognormal
Expression:     LOGN(2.48, 2.63)
Square Error:   0.004256

Chi Square Test
  Number of intervals   = 2
  Degrees of freedom    = -1
  Test Statistic        = 0.427
  Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
  Test Statistic        = 0.122
  Corresponding p-value > 0.15

        Data Summary

Number of Data Points   = 34
Min Data Value          = 0.5
Max Data Value          = 18.3
Sample Mean             = 2.79
Sample Std Dev          = 4.12

        Histogram Summary

Histogram Range         = 0 to 19
Number of Intervals     = 5
```

Appendix H: Belt Mean Time Before Failure

Belt code	Mean Time Before Failure distribution (hrs)
SB1	5+exponential(23.4)
SB2	Normal(4.54, 4.1)
SB3	-7+lognormal(28.6, 28.5)
SB4	-6+exponential(15.6)
SB5	-4+lognormal(25.7, 28.4)
SB6	Triangular(-6, 3.82, 27)
SB7	-8+gamma(1.96, 10.4)
SB8	-7+lognormal(19, 15.5)
T8	-6+lognormal(24.6, 26.3)
T3	-7+lognormal(25.1, 24)
EM1	-3+lognormal(8.85, 9.83)
T2	-8+weibull(1.28, 32.7)
T9	-8+lognormal(28.4, 29.1)
T5	-6+lognormal(26.5, 28.1)
T7	-9+lognormal(25.9, 23.5)
Incline	Expo(13)
MUGICV1	-11+lognormal(24.4, 19.1)

Appendix I: Simulation Results Opportunity Lost (Run Off Mine tonnes)

Date	East Main	Production Rate (t/hr)
	Downtime (hrs)	234.93
2012/10/09	4.5	1057.185
2012/10/10	5	1174.65
2012/10/13	1.08	253.7244
2012/10/17	2.8	657.804
2012/10/18	4.75	1115.9175
2012/10/22	0.47	110.4171
2012/10/25	2.5	587.325
2012/10/26	4.5	1057.185
2012/10/30	0.58	136.2594
2012/10/31	2.23	523.8939
2012/11/01	4	939.72
2012/11/02	3	704.79
2012/11/06	0.92	216.1356
2012/11/07	0.75	176.1975
2012/11/08	8.92	2095.5756
2012/11/10	0.35	82.2255
2012/11/11	12	2819.16
2012/11/13	3	704.79
2012/11/15	1.33	312.4569
Total	**62.68**	**14725.4124**

Date	Trunk 3	Production Rate (t/hr)	Date	Trunk 3	Production Rate (t/hr)
	Downtime (hrs)	121.88		Downtime (hrs)	121.88
2012/02/21	1.47	179.1636	2012/04/02	3.67	447.2996
2012/02/22	8.42	1026.2296	2012/04/03	6.57	800.7516
2012/02/23	6.89	839.7532	2012/04/04	1.66	202.3208
2012/02/24	0.92	112.1296	2012/04/05	0.2	24.376
2012/02/25	5.48	667.9024	2012/04/06	0.5	60.94
2012/02/27	6.83	832.4404	2012/04/07	2.25	274.23
2012/02/28	3.08	375.3904	2012/04/09	3.93	478.9884
2012/02/29	15.9	1937.892	2012/04/10	1	121.88
2012/03/01	0.23	28.0324	2012/04/11	1.07	130.4116
2012/03/02	7.15	871.442	2012/04/12	1.98	241.3224
2012/03/03	0.77	93.8476	2012/04/13	5.92	721.5296
2012/03/06	0.5	60.94	2012/04/14	2.67	325.4196
2012/03/08	1.17	142.5996	2012/04/16	4.25	517.99
2012/03/09	0.87	106.0356	2012/04/17	6.67	812.9396
2012/03/10	0.83	101.1604	2012/04/18	2.29	279.1052
2012/03/12	1.55	188.914	2012/04/19	1.52	185.2576
2012/03/13	7.17	873.8796	2012/04/20	2.87	349.7956
2012/03/14	3.28	399.7664	2012/04/21	0.75	91.41
2012/03/15	0.25	30.47	2012/04/25	3.08	375.3904
2012/03/17	0.33	40.2204	2012/04/26	1.25	152.35
2012/03/20	0.5	60.94	2012/05/04	2.58	314.4504
2012/03/22	11.75	1432.09	2012/05/08	1.92	234.0096
2012/03/23	1.68	204.7584	2012/05/09	16.42	2001.2696
2012/03/24	0.5	60.94	2012/05/10	1.08	131.6304
2012/03/26	4.72	575.2736	2012/05/11	2.25	274.23
2012/03/28	4.92	599.6496	2012/05/14	0.2	24.376
2012/03/29	7.34	894.5992	2012/05/15	0.25	30.47
2012/03/30	4.42	538.7096	2012/05/16	14.02	1708.7576
2012/03/31	1.5	182.82	2012/05/17	0.67	81.6596
			2012/05/19	7.5	914.1
			2012/05/21	3.84	468.0192
			2012/05/22	0.5	60.94
			2012/05/23	3.5	426.58
			2012/05/24	0.25	30.47

Date	Trunk 3	Production Rate (t/hr)	Date	Trunk 3	Production Rate (t/hr)
	Downtime (hrs)	121.88		Downtime (hrs)	121.88
2012/05/28	5.73	698.3724	2012/08/01	0.67	81.6596
2012/05/29	0.25	30.47	2012/08/02	3.42	416.8296
2012/05/30	1	121.88	2012/08/08	0.75	91.41
2012/05/31	14.25	1736.79	2012/08/11	0.67	81.6596
2012/06/01	1.9	231.572	2012/08/13	3.42	416.8296
2012/06/04	0.7	85.316	2012/08/15	1.08	131.6304
2012/06/05	1.67	203.5396	2012/08/16	2	243.76
2012/06/06	0.35	42.658	2012/08/17	2.58	314.4504
2012/06/07	1.57	191.3516	2012/08/18	3.33	405.8604
2012/06/08	4.17	508.2396	2012/08/20	1	121.88
2012/06/09	0.5	60.94	2012/08/21	0.75	91.41
2012/06/11	2.75	335.17	2012/08/23	1.58	192.5704
2012/06/13	2.42	294.9496	2012/08/24	0.17	20.7196
2012/06/19	0.5	60.94	2012/08/25	2	243.76
2012/06/26	0.83	101.1604	2012/08/27	1.75	213.29
2012/07/03	1.72	209.6336	2012/08/28	2.25	274.23
2012/07/04	0.33	40.2204	2012/08/29	3.5	426.58
2012/07/05	3.83	466.8004	2012/08/30	8.12	989.6656
2012/07/06	0.75	91.41	2012/08/31	3.89	474.1132
2012/07/07	2.37	288.8556	2012/09/01	0.67	81.6596
2012/07/09	1.3	158.444	2012/09/03	4.58	558.2104
2012/07/10	0.67	81.6596	2012/09/04	3.07	374.1716
2012/07/16	3.58	436.3304	2012/09/05	5.92	721.5296
2012/07/17	2.5	304.7	2012/09/06	1.83	223.0404
2012/07/21	2	243.76	2012/09/07	4.7	572.836
2012/07/24	0.33	40.2204	2012/09/08	3.92	477.7696
2012/07/25	1.75	213.29	2012/09/11	0.08	9.7504
2012/07/26	1.58	192.5704	2012/09/13	1.59	193.7892
2012/07/27	5.25	639.87	2012/09/14	0.5	60.94
2012/07/28	1.95	237.666	2012/09/18	4.41	537.4908
			2012/09/19	1.92	234.0096
			2012/09/20	4	487.52
			2012/09/22	0.1	12.188
			2012/09/25	5.09	620.3692
			2012/09/26	0.92	112.1296
			2012/09/27	2	243.76
			2012/09/28	0.83	101.1604
			2012/10/01	0.25	30.47
			2012/10/02	3.83	466.8004
			2012/10/04	8.58	1045.7304
			2012/10/05	5.91	720.3108
			2012/10/08	3.5	426.58
			2012/10/09	6.82	831.2216
			2012/10/10	5	609.4
			2012/10/13	1.08	131.6304
			2012/10/17	2.8	341.264
			2012/10/18	5.75	700.81
			2012/10/22	0.47	57.2836
			2012/10/24	0.83	101.1604
			2012/10/25	2.5	304.7
			2012/10/26	1.83	223.0404
			2012/10/30	0.58	70.6904
			2012/10/31	2.23	271.7924
			2012/11/02	3	365.64
			2012/11/06	0.92	112.1296
			2012/11/07	0.75	91.41
			2012/11/08	8.92	1087.1696
			2012/11/10	0.35	42.658
			2012/11/13	5.5	670.34
			Total	**448.46**	**54 658.30**

Appendix J: Cost of a 2000T bunker

F.A. EZOU 3253 Base date 2008

Contract No.: ZON 000 07

Item No.	Short Description	Unit	Quantity	Rate	Amount
	FINAL ACCOUNT				
	FINAL SUMMARY				
SCHEDULE NO.		PAGE			
1.	PRELIMINARY AND GENERAL				
	Fixed-Charge Items	1			1 995 691.66
	Value-Related Items	1			0.00
	Time-Related Items	1			9 932 777.44
2.	2000 TONNE SURGE SILO	6			5 034 824.20
3.	~~6000~~ 4000 TONNE SURGE SILO	12			10 571 514.61
4.	DAYWORKS AND SITE MODIFICATIONS	13			47 060.83
5	PILING	14			4 123 700.90
6	ESCALATION	15			1 855 917.92
	SUB-TOTAL :			R	33 561 487.56
	VALUE ADDED TAX @ 14%			R	4 698 608.26
	TOTAL CARRIED TO CERTIFICATE OF AGREEMENT :			R	38 260 095.82

- Escalation from 2008 - 2013 Allow 28,5%

Ⓐ R.

- Study Cost Allow 7% of Ⓐ
- Design and Implementation 12% of Ⓐ

R.

Please note this does not include any Struct/Mechanicals.

Appendix K: Cost of a 6000T bunker

6000t Silo storage facility - Provisional Budget Estimate 2013/01/07 11:26 AM

ANGLO AMERICAN INYOSI COAL

ZIBULO COLLIERY

EXISTING 6000T SILO STORAGE FACILITY - NEW SILO , OVERSPILL ARRANGEMENT ON STOCKPILE AND FEL LOADING ARRANGEMENT

PROVISIONAL BUDGET ESTIMATE

JULY 2012 MONEY VALUES

ESTIMATE NUMBER	DESCRIPTION	TOTAL COST
0110	Site Establishment	236 250
0120	Shaft Sinking - Incline Shaft	-
0121	Shaft Sinking - Upcast / Downcast Shaft	-
0140	Mining Access - Opencast	-
0141	Main Development - Pre-Production	-
0210	Mining Equipment - Overburden Stripping	-
0211	Mining Equipment - Coal Winning and Loading	3 946 985
0212	Mining Equipment - Coal Hauling and Road Construction	-
0220	Arterial Transport - Overland Conveyor	-
0290	Bulk Storage and Allied Transport	31 723 644
0291	Rail Load-Out	-
0320	Crushing, Screening and Sampling - Tip	-
0321	Process Plant	-
0560	Fuel Handling and Storage	530 000
0570	Pollution Control	2 711 445
0620	Permanent Roads and Terraces	695 800
0621	In Pit Roads	-
0640	Stream Diversions and Stormwater Drainage	144 250
0650	Water Supply	980 437
0680	Sewage	-
0720	Low Density Housing	-
0740	Stores, Offices and Amenity Buildings	-
0770	Security	240 000
0780	Medical Facilities and Protective Equipment	-
0840	Workshops	-
0850	Electrical Reticulation	-
0851	Electrical Reticulation - Pit	-
0860	Instrumentation and Control	2 500 000
0880	Pumping and Dewatering In Pit	-
0890	Surface Transport	-
	Sub-Total : R	43 708 810
0910	Outside Consultants	7 053 568
0952	Pre-Production Technical Investigations	-
0954	Lease Area and Property Rights	87 500
0956	Current Assets	2 500 000
0960	Company Operating Costs	50 000
9952	Reimbursables	3 120 000
	Sub-Total : R	56 519 878
9990	Contingencies	11 414 443
	TOTAL : R	67 934 321
0953	Foreign Exchange Variations	-
9980	Escalation	9 868 150
	TOTAL : R	77 802 471