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Analysis of new tools for equipment scheduling in underground mining and ore safety stock management

Master's thesis, which has been submitted for Master of Science degree.

Espoo, 09.11.2015
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Tiivistelmä

Tämän diplomityön tarkoitus on tutkia vuonna 2014 kehitetyn tietokoneavusteisen tuotannonohjausohjelmiston vaikutusta maanalaisen kaivoksen laitteiston aikataulutukseen. Tämän työn toisena keskeisenä tutkimuskohteena on kaivoksien varastonhallinta ja tarkemmin malmin varmuusvarastotason optimointi uuden "real options" – menetelmän avulla. Molemmat edellä mainitut menetelmät on koestettu Agnico Eagle Kittilän kaivoksella, käyttäen todellisia muuttujia.

Tässä työssä esitellään kirjallisuuskatsaus kehitetystä tietokoneavusteisesta tuotannonohjausohjelmistosta sekä varastonhallinta menetelmistä. Tietokoneavusteisen tuotannonohjausohjelmiston kirjallisuuskatsaus esittelee ohjelmiston taustat sekä käytetyt algoritmit. Varastonhallintaa koskeva kirjallisuuskatsaus esittelee uuden "real options" – menetelmän sekä perinteisen menetelmän. Tämän lisäksi työssä esitellään katsaukset varastoidun malmin sekoitusmenetelmiin sekä muiden alojen soveltamiin varastonhallinta menetelmiin.

Tietokoneavusteista tuotannonohjausohjelmistoa koestettiin Kittilän kaivoksella ja saatuja tuloksia on arvioitu vertailemalla niitä todellisiin käytännön tuloksiin. Tässä työssä esitellään kaksi merkittävintä ratkaisua, jotka on saatu käyttämällä tuotannonohjausohjelmistoa. Saadut tulokset antavat ymmärtää, että tuotannonohjausohjelmistoa käyttämällä, laitteiston käyttämä kokonaismatka ja kokonaisaika voi laskea jopa 21% vuoron aikana. Saadut tulokset osoittavat myös, että peränajon tuotantotehokkuus voi kasvaa jopa 5% vuoron aikana.

Tässä työssä on kehitetty uusi menetelmä optimoimaan malmin varmuusvarastotaso. Tämä menetelmä keskittyy tarjonnan ja kysynnän eroavaisuuksiin. Tämä menetelmä on sovitettu yhteen "real options" – menetelmän kanssa, jotta on pystytty määrittämään tarvittava malmin varmuusvarastotaso entistä optimaalisemmin. Kyseinen menetelmä on koestettu yhtenevillä muuttujilla Kittilän kaivoksen kanssa. Saadut tulokset luovat perustan vuosittain tarvittavalle malmin varmuusvarastotasolle. Tämän lisäksi, tuloksissa esitellään kehitetty päätöksenteko työkalu avustamaan käytännön varastonhallintaa.

Tässä työssä esitellyt menetelmät ja mallit edustavat ratkaisuja entistäkin optimoidumpaan kaivostoimintaan. On kuitenkin huomioitavaa, että kyseiset menetelmät ja mallit on koestettu vain tietyissä olosuhteissa ja siksi niitä ei pidä yleistää.

Avainsanat maanalainen kaivos, laitteiston aikataulutus, tuotannonohjaus, varmuusvarasto

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Abstract of master's thesis

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Abstract

This Master's thesis was carried out to test the efficiency of an equipment scheduling optimization software, developed by Song et al., in 2014 for underground mining. Another focus of this thesis is in mine inventory management. More specifically in the optimization of safety stock levels of ore supply, using a real options model, developed by Song in 2015. Both the equipment scheduling software and the safety stock optimization model were tested at Agnico Eagle Kittilä mine with real mine parameters.

A literature study for both topics was conducted. The study on equipment scheduling optimization software presents the background of the software and the used algorithms. The study on safety stock optimization presents the new real options model as well as the conventional model. Other focuses of the study include the possible blending strategies used in stockpiling and inventory systems that model safety stock in other industries.

The equipment scheduling optimization software was tested at Kittilä mine and the discrepancies between actual and planned (software) operations were evaluated. This thesis presents the two most significant scenarios generated by the equipment scheduling optimization software. The results obtained from the tests suggest that the equipment scheduling software can decrease the travel distance and travel time of mining machines by as much as 21% on a daily basis. Additionally the results suggest that the operating efficiency at workfaces can be increased by 5% on a daily basis.

A new method was created to optimize safety stock. This method focuses on the discrepancies in ore supply and mill demand. The new method was analyzed and integrated with the real options model to optimize safety stock levels. This method was tested with parameters consistent to the parameters of Kittilä mine. The result of the test provides a foundation for the needed safety stock levels on a yearly basis. Additionally, a decision model for mine inventory management in practice has been created and is presented in the results.

The methods and models presented in this thesis represent solutions towards more optimized operations. However, these methods and models have been only tested for specific circumstances and therefore should not be generalized.

Keywords underground mine, optimization, equipment scheduling, safety stock

Foreword

This Master's thesis was produced as a part of the I^2Mine project at Aalto University. The funding of this project was provided by the European Commission 7^{th} Framework Programme. The main focus of this thesis was in the testing of an equipment scheduling software for underground mines, developed by Zhen Song D.Sc. This thesis focuses also in the optimization of safety stocks in mining operations, using new methods.

I would like to express my gratitude to Professor Mikael Rinne from Aalto University for giving me the opportunity to work as a part of this project. I would also like to express my gratitude to my advisor, Zhen Song D.Sc. from Aalto University. The significant guidance and feedback from Zhen Song made this thesis possible. I would also like to thank Agnico Eagle Kittilä mine for making it possible to carry out the testing of the equipment scheduling software at the mine. Special thanks go to André van Wageningen, Elen Toodu, Tuula Koivuniemi and Kimmo Koppström from Agnico Eagle Kittilä mine for providing the needed material.

I would additionally like to express my gratitude to the following people from Aalto University for all these years: Jari Airaksinen, Heli Hakamäki, Frans Ritala, Mateusz Janiszewski, Lauri Uotinen, Joni Sirkiä, Mikko Honkonen, and Otto Hedström.

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Espoo 09.11.2015

Juha Karstila

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List of Abbreviations

DU	Demand Uncertainty
EPS	Enhanced Production Scheduler
ESOS	Equipment Scheduling Optimization Software
FOQ	Fixed-Order Quantity
FTP	Fixed-Time Period
ICT	Information and Communications Technology
I^2Mine	Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future
LHD	Load Haul Dump
LTU	Lead Time Uncertainty
LOM	Life of Mine
MAPE	Mean Absolute Percentage Error
NPV	Net Present Value
SU	Supply Uncertainty
SS	Safety stock
VT	Inclined drift (Vino tunneli)

1 Introduction

1.1 Background

There is a continuous need to develop and improve the efficiency of mine-wide applications. The rapid development of Information and Communications Technology (ICT) during the 21st century has opened up numerous possibilities to ensure the efficiency of mining operations and thus bring economical value and stability to mining projects. However, the emphasis of developing and improving ICT has been implemented mostly towards open-pit mining operations. This has been due to the fact that e.g. communications technology and automation are easier to implement on the surface than underground, where data systems are more complex and require more infrastructure.

In recent years, however, there has been an increasing desire to develop and improve real-time monitoring applications for underground mining operations. An excellent example of this is the I^{2}Mine project (Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future), which secured funding from the European Commission 7th Framework Programme in November 2011 (I^{2}Mine, 2013). In the beginning of 2014, Aalto University had developed an equipment scheduling optimization software (ESOS) for underground hard rock mines, as part of the I^{2}Mine project (Song et al., 2014). The developed equipment scheduling optimization software is consistent with the goal to improve the efficiency of underground mines in Europe, set for the I^{2}Mine project (I^{2}Mine, 2013, Song et al., 2013).

The equipment scheduling optimization software has not yet been implemented into commercial use for mining operations. However, there have been preliminary software tests conducted with real historic mine parameters, gathered from the Agnico Eagle Kittilä mine in 2014. (Uripto, 2014) The results presented in the study by Uripto (2014), indicate that the software still needs to be improved, modified and tested before it can be presented for commercial use. The second test phase of the equipment scheduling optimization software should be an iterative process where the feedback and improvement suggestions are applied into the software during the test. This phase will be implemented physically at the Agnico Eagle Kittilä mine, in order to achieve results that are as realistic as possible.

In mining operations, as described by Song (2015a), there is a tendency to keep safety stock (ore stockpiles) of a specific quantity and quality of ore, to prevent stockout in the downstream processes of the supply chain. The main reason for mining operations to have safety stocks is the high uncertainty in mining operations. The cause of operational uncertainty is usually a result of equipment failure and safety issues. In addition to the uncertainty related with operational aspects, there are also high uncertainties in the geological conditions of mine operations. These uncertainties, related with geological conditions, are described in terms of ore grade, ore quantity and ore quality. Although these critical uncertainties can be mitigated with thorough investigation and mine planning, it will not guarantee a constant ore supply in the long run or life of mine (LOM). Since stockouts are not tolerated in the mining industry, due to their negative financial impact, ore safety stocks are considered as a necessary evil.

In the mining industry, mineral processing (mill) and metallurgical processing (refinery) are usually considered of having a static or constant demand, whereas the mining operation (ore supply) is considered to produce ore in a more stochastic nature. There have been very few studies carried out and published regarding the supply chain uncertainty in inventory and supply-chain management. One of the main reasons for this is the fact that most industries encounter demand uncertainty (DU) and lead time uncertainty (LTU) rather than supply uncertainty (SU) (Snyder & Shen, 2006).

Due to these characteristics in the supply chain of a mining operation, inventory management has to be reconstructed, in comparison to the widely studied inventory management of the retail industry. The supply chain in the mining industry faces an uncertainty in the supply side, whereas the retail industry faces uncertainty in the demand side. It could be considered that the supply chain management for the mining industry could just be implemented as a mirror image from the supply chain management of the retail industry. However, this might not always be the case, and therefore a further study of the inventory and supply chain management in mining operations will be implemented.

1.2 Motivation

Preliminary tests have been carried out by Uripto (2014) for the equipment scheduling optimization software with historical data from the Kittilä mine. Since then, modifications and improvements to the first phase of the equipment scheduling optimization software have been carried out by Song et al. (2014). These improvements are based on the suggestions presented by Uripto (2014). The results of the study indicate that further tests of the equipment scheduling optimization software should be implemented and tested on mine-site, with real time mine data. This will hopefully provide a comprehensive and reliable view of the advantages of the equipment scheduling optimization software for underground hard rock mines.

In mining, inventory management and supply chain management are topics that haven't received the attention they might need, in order to maximize the financial profit and at the same time secure continuous mine-wide operation. Therefore, there is a significant need to study the supply uncertainties encountered in mining operations and evaluate the safety stocks needed for optimal operation.

1.3 Objectives

There are three main objectives for this thesis.

1. Equipment scheduling optimization software (*ESOS*)
    - Implement the ESOS created by Song et al. (2014) at Kittilä mine with real time mine operating data and analyze the discrepancies of actual and planned equipment scheduling.
2. Inventory management in the mining industry
    - Study and analyze the new real options model created by Song (2015a) to determine safety stock. Create a practical decision model for evaluating safety stock, and create a real options blending strategy for safety stock.
3. Equipment scheduling optimization software and inventory management
    - Linking ESOS with inventory management to reconcile safety stock with mining operations, so that there is no need to constantly prepare large safety stocks.

1.4 Scope

The equipment scheduling optimization software, which will go through second phase testing, is by no means the only underground mine equipment scheduling software. Similar software available on the market are e.g. Geovia's MineSched, Deswik's Deswik.Sched and MineRP's EPS (Enhanced Production Scheduler). There are however differences between these software. The focus in this Master's thesis is only on the equipment scheduling optimization software developed by Song et al. (2014). This software is strictly aimed for hard rock underground mining. It is not meant for underground continuous mining e.g. coal mining or for surface mining operations.

Inventory management is focused on supply uncertainty, which is compared with the literature review, mainly of retail industries, where uncertainty occurs in lead time (the time between making an order and delivering the order) and demand. In this study we make the assumption that lead time and demand are constant in the mining industry and that supply is the main cause of uncertainty.

Mining operations might hold safety stocks in multiple sections of the supply chain to ensure a continuous operation without delays. However, in this thesis the focus of optimizing safety stocks is only considered for the part between actual mine production and mill feed.

1.5 Research problem

The following research problems were formed for this thesis.

1. (i) Are there discrepancies in the production efficiency with actual and planned equipment scheduling?

   (ii) Will the equipment scheduling optimization software minimize production duration and maximize both production and equipment efficiency?

   (iii) Does the software; lack critical features, encounter problems or bugs, provide a user-friendly interface and provide significant benefit in view of management?

2. (i) How to evaluate safety stocks that apply to real life mining operations with supply chain dependencies, and high service levels?

   (ii) Are there operational and financial benefits of the new real options model of ore supply in comparison with the conventional model?

   (iii) How to evaluate a real options model for blending strategies?

3. (i) How to create a link between the ESOS and safety stock management for a more optimized operation?

   (ii) How can the real options model for safety stock be optimized to meet the demand in a cumulative and continuous mining operation?

1.6 Structure

This thesis consists of six chapters. The first chapter is the introduction to this Master's thesis. It presents the background, motivation, objectives, scope and research problems related with this thesis. The second chapter is literature study on both topics, the equipment scheduling optimization software and safety stock management. The literature study gives a clear view of what the current situation is, regarding both topics, and suggests new opportunities within the topics.

Chapter three focuses on the methods that were executed for the actual testing of the equipment scheduling optimization software at Kittilä mine. In this chapter, two scenarios that represent crucial operations are presented with detail. Additionally, the user experience and functionality concerning the software are discussed.

Chapter four focusses on the methods created and used for the optimization of safety stock. In this chapter the supply and demand uncertainties related with safety stock are analyzed. A new method for evaluating safety stock is presented and further applied together with the real options method.

Chapter five presents the results and discussion for the efficiency of the equipment scheduling optimization software as well as the results and discussion for determining safety stock using the new method together with the real options method. The results section also includes a decision model for inventory management, and furthermore safety stock management. The final section in the results chapter presents an approach of how the linking on the equipment scheduling optimization software and safety stock optimization increases the efficiency of an underground mining project.

The final chapter of this thesis is the conclusion. This chapter sums up what has been achieved with this study and what could be done in the future, regarding the topics dealt with.

2 Literature study

2.1 Equipment scheduling optimization software

At the start of 2014, Song et al. had developed an equipment scheduling optimization software for underground hard rock mines, as part of the I^{2}Mine project. The main objective for the development of the equipment scheduling optimization software is to improve the efficiency and productivity of underground mining machinery and production scheduling. The decision support instrument should be able to initiate schedules of mobile mining equipment for mine managers on a shift-to-shift basis, as well as propose a solution for a new schedule when encountering unexpected events that can disturb underground mining (Song et al., 2015).

2.1.1 About software

The equipment scheduling optimization software is going through second phase testing, in order to determine the actual benefits of the software before possibly launching it for commercial use. The equipment scheduling optimization software uses Java programming language as the underlying platform and is run through a web browser.

Several changes have been implemented into the software after the first phase tests were carried out by Uripto (2014). The main changes that have been implemented into the software, since phase one, include initial input parameters, bug fixes, and interface appearance. The changes in initial input parameters include the additional option for adding break times and the additional option for determining the curing time for shotcreting.

2.1.2 Algorithms

The used algorithms are the core foundation of the equipment scheduling optimization software, while the input data actually works as constraints for the software. The algorithms emphasize minimizing the timespan of the entire mining process, for a specific workload within a time period, under the given constraints. It assists mine managers to optimally schedule underground mobile mining equipment in order to improve the performance of the entire mining process. (Song, 2015b)

There are several relationships between the algorithms used in the equipment scheduling optimization software. These relationships are shown in *Figure 1*. After all initial input parameters are given for the software, the software first processes a grouping algorithm and then a sequencing algorithm. Once both algorithms have been processed, in the described order, the software produces an optimized schedule for the underground mining equipment.

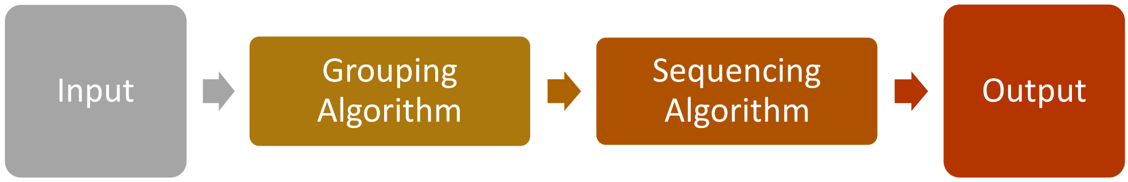

Figure 1. *Processing sequence of the equipment scheduling optimization software.*

The operating time of the underground mobile mining equipment is determined by the workload and the operating rate. Therefore, by minimizing the entire working time, it will not only reduce the waiting time of underground mobile mining equipment, but it will also reduce the moving distance between workfaces. (Song, 2015b)

A brief overview of the used algorithms are presented in the following sub-chapters. For a more detailed coverage, please refer to the reference Song et al., 2015.

Grouping algorithm

The grouping algorithm is an essential part of the software, since it will significantly improve the efficiency of the software itself. The requirement of developing numerous workfaces within a short period of time, without the grouping algorithm, would create a huge amount of permutations for the software to process. This would result in slow processing of the software or even incapacity to work.

The main objective of the grouping algorithm is to reduce the computing time and workload of the software. By organizing all the workfaces into smaller sets of unique groups, the number of permutations needed for the software to process, will be minimized significantly. For example, if there are 20 workfaces then there are 20! permutations, which results in a total of 2 432 * 10^{15}. Instead of processing all those permutations, which would be hard for even a super computer, the grouping algorithm will find a way to divide the workfaces into e.g. 5 groups, as shown in *Figure 2*. This would reduce the number of permutations to 3!+4!+5!+2!+6! = 872 or even 3!+4!+5!+2!+2!+4! = 178. The grouping algorithm will thus reduce the computing workload of the sequencing algorithm and the computing time (Song et al., 2015).

Figure 2 illustrates a grouping method, of the grouping algorithm, for a theoretical cross-section of an underground mine. The figure shows how a number of 20 workfaces have been grouped into 5 groups and additionally the group on the left has been grouped further into two subgroups. The maximum number of working faces for one subgroup has been empirically set to five working faces. The same rule applies for the maximum number of subgroups in a group. However, this setting can be modified according to the specific properties of the underground mine.

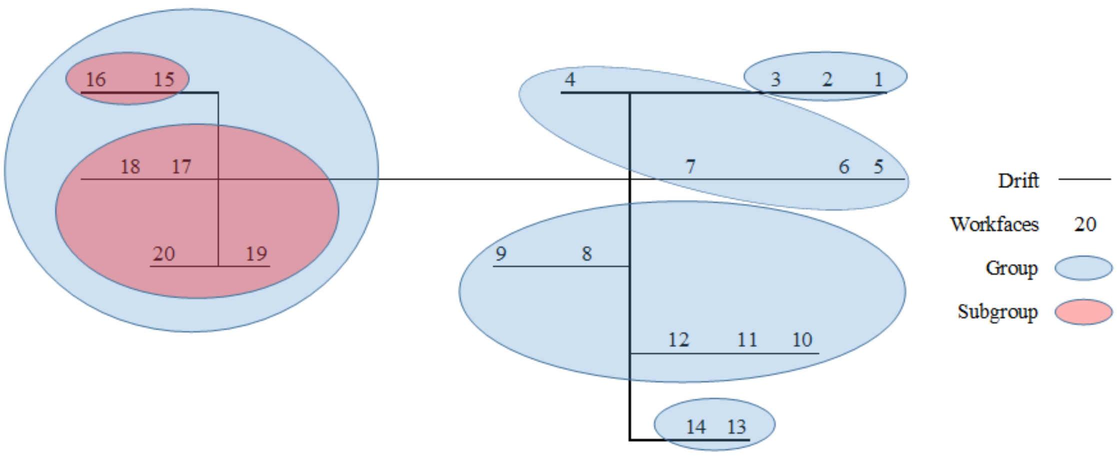

Figure 2*. Grouping method for a theoretical cross-section of an underground mine (modified: Song et al., 2015).*

Each workface, subgroup and group is organized according to the distances from one to another. The grouping algorithm is combined to work with the sequencing algorithm in order to reduce the computing workload. After the grouping algorithm has produced the groups, the sequencing algorithm will find the optimized sequence of workfaces in each individual group and subgroup. After this, the sequencing algorithm produces the optimal sequence for subgroups within that group. Now the sequencing algorithm will be able to find the optimized sequence of groups in order to create a sequence with the minimum timespan. (Song et al., 2015)

Sequencing algorithm

As Song et al. described in 2015, the sequencing algorithm is used to find the minimum timespan for the longest serially sequenced operation. The minimum timespan is calculated and optimized from the critical path of the operation. The timespan covers the entire time from the start of the first machine, at the first workface, to the end of the last machine, at the last workface.

For example, there are n number of workfaces (f_1-f_n) and m number of machine types (m_1-m_k), which are assumed to process each workface in the specific sequence of m_1-m_k. This results in a number of n! permutations of the sequences for the workfaces. In other words, this results in a number of n! timespans (T). After this the minimum timespan is further obtained by choosing the minimum ($T_{min.1}$, $T_{min.2}$,..$T_{min.n!}$) timespans from the n! timespans. This determines the optimal sequence for workfaces and thus gives the minimum timespan for the operation. (Song et al., 2015).

The sequencing algorithm is constructed from three matrices. These matrices consist of an end time matrix, an operating time matrix and a moving time matrix, as shown in *Equation 1* (Song et al., 2015).

$$\begin{array}{ccc} \text{End time} & \text{Operating time} & \text{Moving time} \\ \begin{bmatrix} \mathbf{et_{11}} & \mathbf{et_{12}} & \circ\circ\circ \\ \mathbf{et_{21}} & \mathbf{et_{22}} & \circ\circ\circ \\ \circ\circ\circ & \circ\circ\circ & \mathbf{et_{n,k}} \end{bmatrix} & \begin{bmatrix} \mathbf{ot_{11}} & \mathbf{ot_{12}} & \circ\circ\circ \\ \mathbf{ot_{21}} & \mathbf{ot_{22}} & \circ\circ\circ \\ \circ\circ\circ & \circ\circ\circ & \mathbf{ot_{n,k}} \end{bmatrix} & \begin{bmatrix} \mathbf{mt_{11}} & \mathbf{mt_{12}} & \circ\circ\circ \\ \mathbf{mt_{21}} & \mathbf{mt_{22}} & \circ\circ\circ \\ \circ\circ\circ & \circ\circ\circ & \mathbf{mt_{n,k}} \end{bmatrix} \end{array} \tag{1}$$

The elements in the end time matrix represent a time stamp, while the elements in the operating time matrix and the moving time matrix represent a time interval. The time stamp for end time et_{ij} is the time when that specific machine j stops working at that specific workface i. The time intervals of the elements ot_{ij} in the operating time matrix represent how much time it takes for a specific machine j to operate in that specific workface i. The same rule applies for the moving time matrix (mt_{ij}) as the operating matrix. It represents the time it takes for a specific machine j to move from one workface to another workface i. (Song et al., 2015).

The equation for the timespan T was presented by Song et al. in 2015 and is shown in *Equation 2*. The minimum timespan is obtained as a sum of the minimums $\Sigma(T_1, T_2, T_{n!})$.

$$\boldsymbol{T = et_{n,k} - st = \max(et_{n,k-1}, et_{n-1,k} + mt_{n,k}) + ot_{n,k} - st} \tag{2}$$

$et_{n,k-1}$ - End time of machine k-1 at workface n

$et_{n-1,k}$ - End time of machine k at workface n-1

$mt_{n,k}$ - Moving time of machine k from workface n-1 to workface n

$ot_{n,k}$ - Operating time of machine k at workface n

st - Start timestamp of the entire operations

The start timestamp of the entire operation can be set to 0, which will simplify the equation as shown in *Equation 3*.

$$T = et_{n,k} = \max(et_{n,k-1}, et_{n-1,k} + mt_{n,k}) + ot_{n,k} \tag{3}$$

2.1.3 Interface

The equipment scheduling optimization software is run through a web browser at a local host address http://188.166.91.42:3000/. As seen in *Figure 3*, the software interface consists of six main input buttons and two main execute buttons that are used to sort the produced Gantt chart either by individual machines or by workfaces.

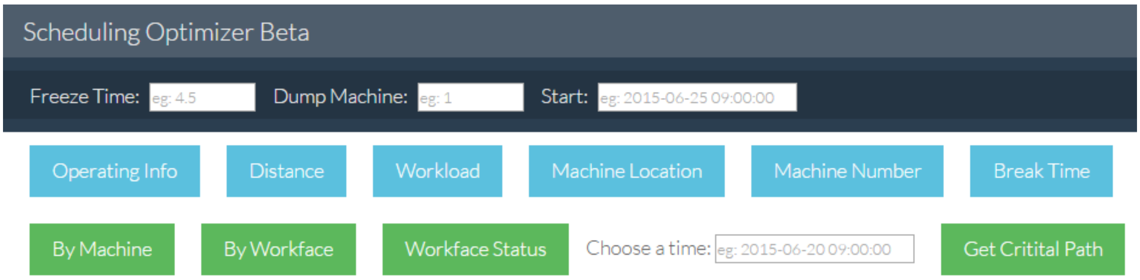

Figure 3*. Interface of the Equipment Scheduling Optimization Software.*

The six main input files are uploaded into the software in text file format *(Appendix 11)*, in order for the software to perform the algorithm calculations. These input files include:

- Operating Info (*Operating_Info.txt*)
- Distance (*Distance.txt*)
- Workload (*Workload.txt*)
- Machine Location (*Machine_Location.txt*)
- Machine Number (*Machine_Number.txt*)
- Break Time (*Break_Time.txt*).

The two main execute functions include:

- By Machine
- By Workface.

In addition to the six main input files, the software interface includes a "Freeze Time" option, a "Dump Machine" option, a "Start" option a "Workface Status" option and a

"Get Critical Path" option. The freeze time option is used as a constraint to adjust the curing time for shotcreting. The input for freeze time is given in hours. The software then optimizes the schedule in a way that no production can be carried out in a workface where the shotcrete is curing for the given time. The dump machine option is used to specify the machine type number of the shotcreters, in order for the software to apply the curing time for the right process. The start option is used to specify the initial start time for the scheduling optimizer e.g. the start of a shift. After the software has produced an optimized Gantt chart, the progress of each workface at a specific time, can be visualized by applying the desired time into the "Choose a time" and then clicking the "Workface Status" button. The software is also able to visualize the critical path of the operation. Mine managers can use this as a good indicator of which operations need the most attention.

Operating info

The first of the six main input files is "Operating info". This file consists of a row, of each machine type, in the order of which a development workface is operated in. In this example, shown in *Table 1*, the operating sequence of a development drift begins with scaling, then shotcreting, bolting, drilling and ends with charging. The chosen sequence should be used because the software does not account for mucking. If a mucking operation would be included into the software, the most practical operating sequence would begin with drilling, then blasting, mucking, scaling, shotcreting and bolting.

Table 1. *General view of "Operating Info" input file.*

			Operating rate historical [m/h]			Moving speed historical [m/h]		
Operating Info	Average operating rate [m/h]	Average moving rate [m/h]	Min.	Max.	Mode	Min. (Incline)	Max. (Decline)	Mode (Horizontal)
machine0(scaler)	3.87	12 400	3.29	4.32	4.00	7 000	17 000	13 200
machine1(shotcreter)	2.19	11 400	1.88	2.47	2.21	5 000	17 000	12 100
machine2(bolter)	1.78	11 400	1.50	1.97	1.85	5 000	17 000	12 100
machine3(driller)	1.66	10 400	1.41	1.85	1.72	5 000	15 000	11 000
machine4(charger)	2.20	10 400	1.88	2.47	2.26	5 000	15 000	11 000

Due to the fact that the software doesn't consider mucking, a continuous scheduling process can be obtained with the sequence shown in *Table 1*. The reason why the software doesn't take mucking into account is the unique cycle property of mucking operations. The mucking operation can be rather easily optimized and furthermore there are already a large number of mucking cycle optimization software available on the market. When running the software with the recommended operating sequence, it is crucial to re-run the software with the new updated data after a workface has been mucked. This will allow the operation to continue for a new round of operation in that specific workface.

The operating info file also consists of an operating rate and a moving speed, in meters per hour, for each machine type. The operating rate is a performance value, set from recorded historical operating data, for a specific drift cross section. For example, the operating rate for machine0(scaler) is 3.87 meters per hour, as shown in *Table 1*. This means that the scaler can operate 3.87 meters forward, in an hour, on a tunnel with a cross section of 25m^2 (5m * 5m). If the cross section is larger or smaller, the operating rate has to be adjusted accordingly. The moving speed is also a performance value, set from recorded historical moving data. It is adjusted as the average of the moving speed on an incline, decline and a horizontal drift. In addition to these parameters, the minimums, the maximums and the modes of both operating rates and moving speeds are added into the operating info input file. The sequence for the columns should be as shown in *Table 1*, in order for the software algorithms to process them correctly.

Distance

The second main input file needed for the software is "Distance". This file consists of distances between two workfaces. As shown in *Table 2*, the distance input file has a diagonal zero section going from the left top corner to the bottom right corner. This is because the workfaces are the same along the diagonal.

Table 2*. General view of "Distance" input file.*

Workface distances [km]		Workfaces									
		0	1	2	3	4	5	6	7	8	9
Workfaces	0	0.00	0.60	0.80	1.34	1.59	1.94	2.01	1.81	2.05	2.28
	1	0.60	0.00	0.65	1.18	1.44	1.79	1.86	1.65	1.90	2.12
	2	0.80	0.65	0.00	1.04	1.29	1.64	1.71	1.51	1.75	1.98
	3	1.34	1.18	1.04	0.00	1.48	1.83	1.90	1.69	1.94	2.16
	4	1.59	1.44	1.29	1.48	0.00	1.73	1.80	1.60	1.84	2.07
	5	1.94	1.79	1.64	1.83	1.73	0.00	1.80	1.60	1.84	2.07
	6	2.01	1.86	1.71	1.90	1.80	1.80	0.00	1.32	1.56	1.79
	7	1.81	1.65	1.51	1.69	1.60	1.60	1.32	0.00	1.01	1.23
	8	2.05	1.90	1.75	1.94	1.84	1.84	1.56	1.01	0.00	1.13
	9	2.28	2.12	1.98	2.16	2.07	2.07	1.79	1.23	1.13	0.00

The distance input file is therefore a mirrored image in respect to the zero diagonal. The distance between workfaces should be given in kilometers, in order for the algorithm to calculate them correctly. The sufficient accuracy for the parameters of the distance input file is two decimal places, which accounts for 10 meters. When running the software, it should be acknowledged that the first workface number is 0 and not 1.

Workload

The third main input file needed for the software is "Workload". The workload input file, shown in *Table 3*, consists of as many rows as there are different machine types. The sequence of the machine types should be the same as the sequence in the operating info input file. There is also a column for each workface and thus there can be a specific amount of workload at each workface for a specific machine type. The workload for each machine type and for each workface is given in meters. In order for the software to perform in the same way as real mining operations, the workload for one workface cannot exceed the maximum length of one blasted development round. This is because the software executes the whole workload, for that specific workface and machine type, in a row, at once. The maximum length of one blasted development round is a site specific parameter with no further restrictions in the size.

Table 3. *General view of "Workload" input file.*

	Workfaces									
Workload [m]	0	1	2	3	4	5	6	7	8	9
machine0(scaler)	0.0	**4.7**	0.0	0.0	**4.7**	0.0	**4.7**	**4.7**	0.0	0.0
machine1(shotcreter)	0.0	**4.7**	0.0	0.0	**4.7**	0.0	**4.7**	**4.7**	0.0	0.0
machine2(bolter)	**4.7**	0.0	**3.5**	0.0	**4.7**	0.0	**4.7**	**4.7**	0.0	0.0
machine3(driller)	**4.7**	0.0	0.0	0.0	0.0	0.0	**4.7**	0.0	**1.0**	**4.7**
machine4(charger)	**4.7**	0.0	0.0	0.0	0.0	0.0	**4.7**	0.0	**1.0**	**4.7**

For example, if the desired workload for one development drift would be a total of 47 meters for each machine type, the software would recommend an operating sequence where it would first scale 47 meters, then shotcrete 47 meters, etc. However, this is not the case in real mining operations, where the 47 meters of workload would be divided into ten executable rounds of 4.7 meters. The workload of each workface can be adjusted to site specific requirements. Usually the length of one development round is the same every time, in this example 4.7 meters. However, operations may need for example correction blasting which is indicated in *Table 3*, workface 8 with a 1.0 meter workload for drilling and blasting. The software should be executed on a shift basis to avoid situations where the same workface has more than one round of workload planned. When running the software on a shift basis, the time within the shift (10 hours) neglects the possibility of operating at a specific workface for more than 1 round (eg. scaling 2h, shotcreting 2h, curing 5h, bolting 2h, drilling 2h, charging 2h, total time 15 hours). The total time for all operations at one workface exceeds the time available during a shift.

In general, it is beneficial to remove all the workfaces from the input text file that don't have any workload planned for them. This will increase the performance of the software, and it will also make the produced Gantt chart visually more readable. However, if the machines are initially located at a workface which doesn't have any planned workload, that workface should be included with a zero workload in order for the software to know where to start the travel for that machine (*see. Machine location*).

Machine location

The fourth main input file needed for the software is "Machine Location", shown in *Table 4*. The machine location input file consists of three columns. The first column represents the machine type for each row. In order for the algorithms to work properly, the first machine type number should be set to 0. In this case machine type 0 refers to a scaler etc.

Table 4. *General view of "Machine Location" input file.*

Machine Location		
Machine type	Initial workface location (Start of travel)	Machine ID
0	1	1001
0	1	1002
1	4	2001
1	4	2002
2	2	3001
2	1	3002
3	4	4001
3	2	4002
4	0	5001
4	7	5002

The second column represents the initial workface location of that specific machine at the start of the operation. The initial workface location is not the same thing as the first operating workface for that machine. The initial workface location is the location where the machine was left after the previous shift. The third column represents the individual name or machine ID of that specific machine. The third column helps users detect which individual machine is which, within the same machine type.

Machine number

The fifth main input file needed for the software is "Machine Number", as can be seen in *Table 5*. The machine number input file consists of two columns, machine type and machine quantity. The machine type column represents each machine type in the same sequence as in operating info, workload and machine location input files. The machine quantity column specifies the amount of machines available for each machine type.

Table 5*. General view of "Machine Number" input file.*

Machine Number	
Machine type	Machine quantity
0	2
1	2
2	2
3	2
4	2

Break time

The sixth main input file needed for the software is "Break Time", shown in *Table 6*. The break time input file consists of only one column. Every odd number row represents the start time of a break, and every even number row represents the end time of that break. The break times are given for one day and should be given in chronological order starting from 00:00:00.

For the six main input files (*Table 1* - *Table 6*), the grey area info sections are neglected in the text file format. Only the colored and white input parameters are accounted for, in the text file format. For the input text files, each column is separated with a tab and each row is separated with enter. An example of the executable text files can be found in *Appendix 11*.

Table 6. *General view of "Break Time" input file.*

Break Time		
Break 1 (Shift change / Blast)	Break Start	04:45:00
	Break End	07:15:00
Break 2 (Lunch)	Break Start	10:30:00
	Break End	11:30:00
Break 3 (Shift change / Blast)	Break Start	16:45:00
	Break End	19:15:00
Break 4 (Lunch)	Break Start	22:30:00
	Break End	23:30:00

Gantt chart sorting

The equipment scheduling optimization software will produce a Gantt chart, sorted according to either machine ID or workface number. These two options are illustrated in *Appendix 1*, *Figure 34* and *Figure 35* respectively. The Gantt chart will provide a visually easy to read sequence of the optimized equipment scheduling. The software also has a possibility to move operations manually on the interface. For example, if there is a need for a specific machine to work at a specific workface. However, when playing around with the results manually, the modified sequence is no longer the most efficient sequence theoretically.

In addition, it should be acknowledged that some operations may be visually blocked underneath e.g. the waiting time of another machine. These operations can be visualized by moving the “operations” on top of each other, to the side momentarily.

2.2 Safety stocks

Safety stock, in its simplest form, is the inventory that is carried along the operation to prevent stockouts. As King described in 2011, determining appropriate inventory levels is one of the most important and most challenging tasks faced by operations managers. The challenge in the task is finding the optimal inventory- and safety stock level for a unique supply chain. For example, from a financial point of view, if an excessive amount of inventory is carried along the operation, a large amount of money is tied up in working capital. On the other hand, if there's not enough inventory carried along the operation, stockouts will occur, and thus lead to financial losses.

The use of safety stock has been widely observed in industrial supply chains to counter variability that may be present. However, little research has been carried out for the optimization of safety stocks in the mining industry. In industrial inventory management, stockouts are thought to occur due to factors such as fluctuating customer demand, forecast inaccuracy, and variability in lead times of raw material or manufacturing. (King, 2011) In other words, stockouts are a result of uncertainties in the supply chain, that haven't been considered correctly or accurately enough. The uncertainties can stem from random yields, processing times or fluctuating customer demands. The buffering mechanisms used to protect against these uncertainties are overestimation of demand (hedging), safety time or planned safety stock norms, or even a combination of these. (Wijngaard & Wortmann, 1985)

In the mining industry, the uncertainties don't necessarily follow the same trend as mentioned in the last paragraph. The uncertainties occurring in a supply chain of a mining operation stem from uncertainties in the supply, not demand. The supply of raw materials i.e. mined ore from an underground mine, tends to fluctuate quantitatively within a specific period of time. Another factor which influences the uncertainty of supply is the quality of ore. The ore grade in an ore body typically fluctuates and even though this is considered in the life of mine- (LOM) and other mine plans, this will have a qualitative uncertainty in the supply. Safety stock will therefore help the mines to meet the variations in the supply of raw materials (ore).

2.2.1 Conventional method for determining safety stock

A classic model for determining safety stock, illustrated in *Equation 4*, was first given by Silver and Peterson in 1985. Safety stock buffers the inventory in time during the lead time of supply. The classic method makes two major assumptions in the determination of safety stock. It makes the assumption that the lead time and demand follow a normal distribution and additionally that the lead time and demand are independent of one another. (Song, 2015a)

$$\boldsymbol{Safety\ stock} = \boldsymbol{Z}\sqrt{(\boldsymbol{\mu_L \sigma_D^2}) + (\boldsymbol{\mu_D \sigma_L})^2} \qquad \textbf{(4)}$$

Z - safety factor (i.e. standard score from inverse standard normal probability of service level 0~100%)

μ_D - mean of demand

μ_L - mean of lead time (the time between mining the ore and feeding the mill with the mined ore)

σ_D - standard deviation of demand

σ_L - standard deviation of lead time

The safety factor Z (Z-score) is determined from the inverse standard normal probability of a service level of 0-100%. Service levels are often used to describe system performance because of the difficulties in quantifying stockout costs. Service levels can serve as an indicator for the number of stockout occurrences or the size or even the size and duration of stockouts. (Schneider, 1981) Mining operations tend to have a zero tolerance for stockouts, which corresponds to a 100% service level. However, the correlation between service level and Z-score grows exponentially when closing in on a 100% service level. Therefore a Z-score value of 3.29 will be sufficient enough. As shown in *Figure 4*, a Z-score value of 3.29 corresponds to a 99.95% service level. Furthermore, this corresponds to one stockout in less than every 5 years, which can be financially tolerated.

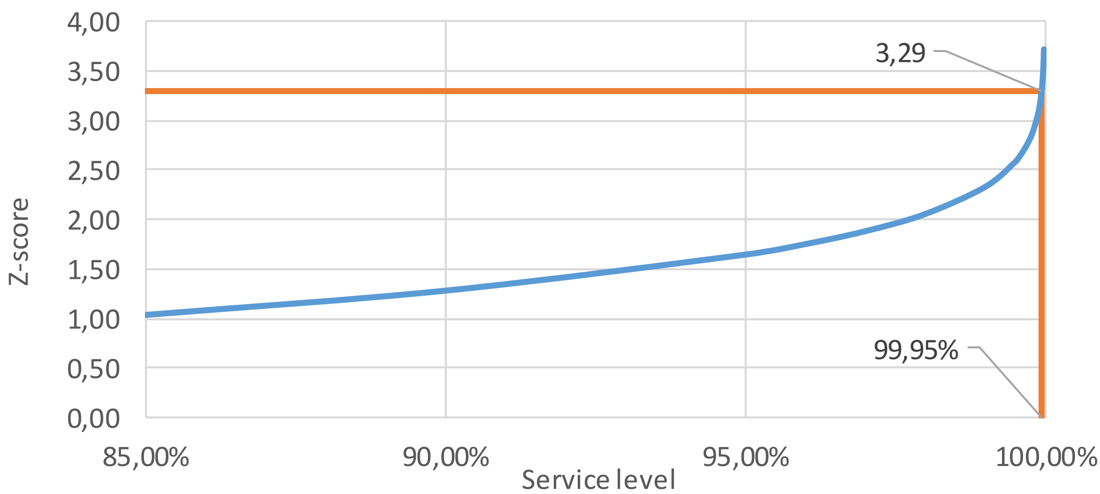

Figure 4*. Determination of Z - Safety factor with corresponding service levels.*

The classic model, however, requires further modifications in order to accurately represent a real mining operation (*Equation 5*).

$$\boldsymbol{Safety\ stock} = (\mu_D - \mu_S)L + 3,29\sqrt{(\sigma_D^2 + \sigma_S^2 + 2\rho\sigma_D\sigma_S)L} \qquad \textbf{(5)}$$

μ_D - mean of demand

μ_S - mean of supply

σ_D - standard deviation of demand

σ_S - standard deviation of supply

ρ - correlation between demand and supply

L - lead time

If the demand of the mill is assumed to be constant, *Equation 5* is simplified to *Equation 6*.

$$\boldsymbol{Safety\ stock} = (\mu_D - \mu_S)L + 3,29\sigma_S\sqrt{L} \qquad \textbf{(6)}$$

The conventional method is somewhat of a good indicator for the calculation safety stock levels. However, to get an even more accurate interpretation of the needed safety stock levels for a mining operation, an out of the box solution should be made. Therefore, a new real options approach has been created by Song (2015a).

2.2.2 Real options method for determining safety stock

Option is the right to execute an underlying asset by the due date of the asset. By the due date, an option holder can claim the right from a counter party, if the status of the underlying asset is in favor of the holder's own interest. The most well-known pricing formula (*Equation 6*) of the option was first developed and published by Black and Scholes in 1973.

A real option is the right to handle (e.g. contract, abandon, expand, switch) an underlying real asset. Real option valuation was first defined and developed by Stewart Myers in 1977. It was further used as a prevailing method to evaluate opportunities associated with the possibility of changing decisions, in order to solve uncertainty during utilizing real assets (Gertner et al., 1999). It can assist the decision process and improve management flexibility for many industries. It has been widely used to assess the value of a mining project using the Black-Scholes model. Mayer et al. (2007) have reviewed the applications of real options valuation used for mining production, and described several real options developed for solving the uncertainties commonly seen in mining: e.g. capacity option, shutdown option, and sequence decision option.

A new real options approach for determining the optimized safety stock levels was developed by Song (2015a). This new real options approach is shown in *Equation 7*. To quote Song (2015a), "The essence of this method is to evaluate the probability and impact of a future event and giving a solution associated with that event under the current circumstances".

$$SS_t = Fe^{-r(T-t)}\left(1-\Phi\left(\frac{ln\frac{Y_t}{F}+(r-\frac{\sigma^2}{2})(T-t)}{\sigma\sqrt{T-t}}\right)\right)-Y_t\left(1-\Phi\left(\frac{ln\frac{Y_t}{F}+(r+\frac{\sigma^2}{2})(T-t)}{\sigma\sqrt{T-t}}\right)\right) \quad \textbf{(7)}$$

$$= Fe^{-r(T-t)}(1-\Phi(d_2)) - Y_t(1-\Phi(d_1))$$

SS_t - ore stock at time t
F - mill's average feed
Y_t - mine's yield at time t
σ - standard deviation of yield
r - ore loss rate
T - due time
t - current time
Φ - cumulative standard normal distribution function

The total quantity of ore safety stock (SS_{period}) needed to cover a time period from t to T is shown in *Equation 8*.

$$SS_{period} = \sum_{t}^{T} SS_i \qquad where \quad i = t, \dots T \quad \textbf{(8)}$$

To further explain the new real options method, if the mill's average feed F is required at a due time T, the real options *Equation 7* returns how much ore safety stock SS_t there should be under the mine's yield at time t. The mine's yield at time t (Y_t) is assumed to obey a Wiener process, which assumes that the yield fluctuates around the previous yield in a Brownian motion. Song (2015a)

A study performed by Song (2015a), uses real ore supply and mill demand parameters from the Kittilä gold mine. The study shows the MAPE (mean absolute percentage error) of both the conventional method (*Equation 6*) and the real options method (*Equation 7*) for determining the needed safety stock levels. The results of the study, presented by Song (2015a) are shown in *Table 7* and *Table 8*.

Table 7*. MAPE of Conventional Method for evaluating safety stock levels (Song, 2015a).*

μ_s, σ_s	Conventional Method *Mean Absolute Percentage Error*		
	01.01~11.05	28.05~31.08	19.09~31.12
Up-to-date	8.09	5.05	6.38
Last 21-day	9.63	11.8	7.62
Last 7-day	6.63	10.13	5.87

Table 8*. MAPE of Real Options Method for evaluating safety stock levels (Song, 2015a).*

σ	Real Options Method *Mean Absolute Percentage Error*		
	01.01~11.05	28.05~31.08	19.09~31.12
Up-to-date	1.4	1.67	1.5
Last 21-day	1.53	1.72	1.39
Last 7-day	1.16	1.47	1.1

The results clearly indicate that the real-options method outperforms the conventional method. As can be seen from comparing the results of the conventional method and the new real options method in *Table 7* and *Table 8*, the mean absolute percentage error of each time period in the real options method is significantly lower than that of the conventional method.

Another significant notation that can be seen from both results in *Figure 5*, is that the most accurate estimation is achieved by using the last 7-day data, rather than the last 21-day data or up-to-date data.

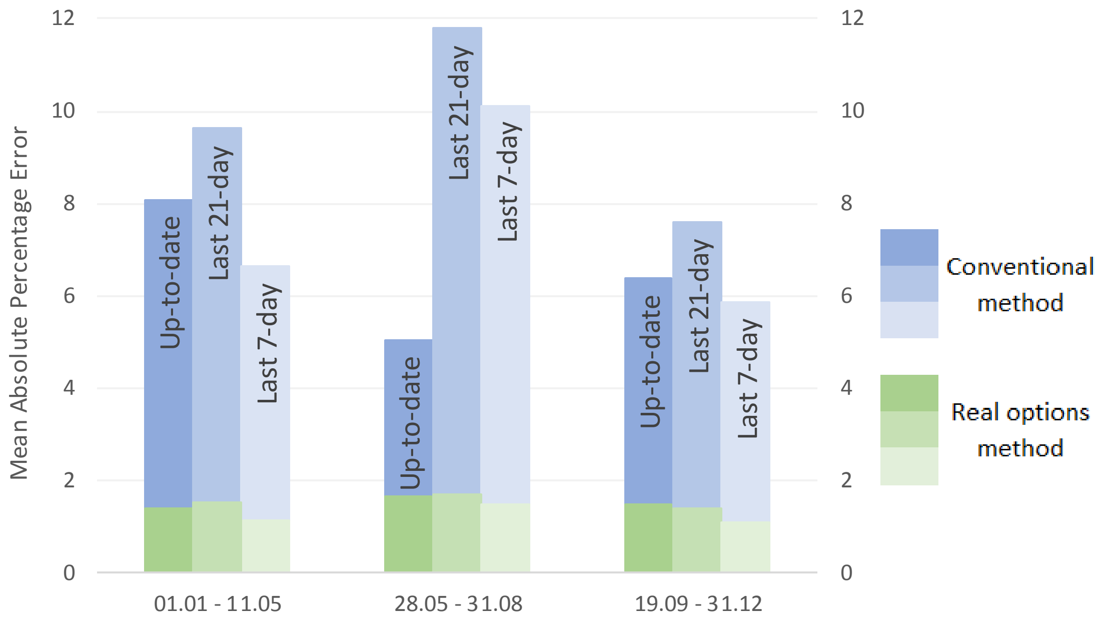

Figure 5*. Comparison of MAPE for conventional method and real options method.*

A conclusion from this can be that the most recent data provides also the most accurate estimation for both models. However, this should be further tested with different data sets to get a more reliable conclusion. A further analysis of the MAPE of the conventional method and the real options method will be presented further on in this thesis.

2.2.3 Ore blending strategies for mill feed

In mining operations, ore stockpiles often serve two purposes. The first reason for ore stockpiles is the fact that they act as buffers, so that processes before and after can operate without being constrained by each other. (Robinson, 2004) In other words, successive processes become independent of one another. This function often provides the primary economic justification for the expense of ore stockpiles. Thus blending options must be considered by looking at minor modifications to the stockpiles which have been planned in order to provide safety stock. The second reason for creating ore stockpiles is the fact that they can be used to reduce the amount of variation in ore grade within the supply. As mentioned by Robinson in 2004, a blending stockpile generally reduces the amount of grade variation between parcels of ore of a moderate size, because the parcels of ore on reclaiming are mixtures of the parcels of ore which were stacked. This means that the variations in the ore grade over time should be smaller in the flow of material which is reclaimed from the stockpile than that what it was in the flow of material which was stacked on the stockpile.

The significance of a constant ore grade feed to the mill should not be underestimated. For example, if the ore grade varies significantly over time, the demand of the mill will also vary largely. The reason for demand to vary largely is that the mill needs to produce a targeted amount of ore with a targeted ore grade. If the ore grade varies, or is lower than the target, the demand will then be higher to achieve the planned quantity of end product. However, the mill only has a limited amount of capacity it can handle, so if the ore grade drops too low and thus demand rises too high, the mill capacity will not suffice.

The blending of ore can be managed in two different ways. The ore can be either blended during the creation of ore stockpiles or it can be blended during the mill feed process. These two strategies are quite the opposite from each other, but eventually aim for the same result. Both strategies will be discussed and analyzed in the following sub-chapters.

Stockpile blending

The first strategy to blend the mined ore is to create a stacking strategy for the making of the stockpile. With this strategy, the mined ore is stacked onto the stockpile in a specific order and with specific quantities. For this strategy to work properly, ore grade distribution of the supplied ore needs to be monitored systematically. The monitoring of ore grade can be performed by analyzing and tagging truckloads of mined ore and then allocating the right place for the ore to be dumped on the stockpile (Koppström, 2015). The tags in the ore stockpile will then send information about their location. This will help the mine management to follow ore grade variation within the stockpile. It is worth mentioning that the ore grade of the mined ore from the underground mine is already known, with a significantly high level of confidence, before the actual mining takes place. Therefore the allocation of where the loaded trucks should be dumped can be planned prior to the operation. However, the high level of confidence in the planned ore grade still has room for error and therefore systematical monitoring of the mined ore should be performed to achieve the actual grade distribution.

A study performed by Robinson in 2004 demonstrates three unique stockpiling methods, two of which will reduce the variations in ore grade when reclaiming them in a certain pattern. The two good stockpiling methods are the chevron-stacked stockpile and the windrow-stacked stockpile. The third method is bed-blending. Bed-blending assumes that each reclaimed layer on a specific bench has the exact same amount of material. This might be a good method for some bed-blending systems, but it is not a good approximation for most mineral stockpiling systems. (Robinson, 2004)

The cross-section of a chevron-stacked stockpile is illustrated in *Figure 6*. The main idea in the chevron-stacked stockpile is that material, in this case ore, is stacked in layers according to the ore grade. The layers form a triangular shape stockpile with an approximate angle of 35.5°, as shown in *Figure 6*.

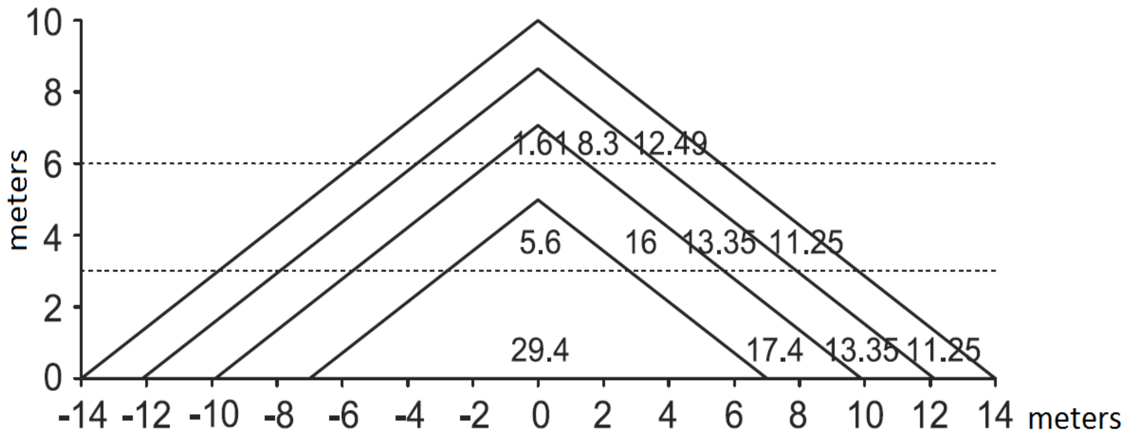

***Figure 6**. Cross-section of a chevron-stacked stockpile. The angle of the material is about 35.5 degrees. The dotted lines represent the reclaim benches and the numbers represent the cross-sectional area of that portion. Each of the four layers stacked has a total cross-sectional area of 35 m^2 (modified: Robinson, 2004).*

Each layer in the cross-section has the same total area. However, this cross-sectional area is no longer the same during reclamation, which is indicated with the dotted lines. In other words, when the stockpile was created, it consisted of four layers. At reclamation, it will consist of only three layers. The proportions of each layer for a specific reclaimed bench can be used to calculate the blended ore grade. A brief overview of the used equations is presented below. For a more detailed coverage, please refer to the reference (Robinson, 2004).

Suppose that the geometry of a stockpiling system is known and is specified by an $s * r$ matrix of masses which relate s blocks of stacked material to r blocks of reclaimed material. This will give you a matrix shown in *Equation 9*.

$$\begin{matrix} \boldsymbol{m_{s,1}} & \boldsymbol{m_{s,2}} & \vdots & \boldsymbol{m_{s,r}} \\ \vdots & \vdots & & \vdots \\ \boldsymbol{m_{2,1}} & \boldsymbol{m_{2,2}} & \cdots & \boldsymbol{m_{2,r}} \\ \boldsymbol{m_{1,1}} & \boldsymbol{m_{1,2}} & \cdots & \boldsymbol{m_{1,r}} \end{matrix} \tag{9}$$

The quantity $m_{i,j}$ specifies the mass of material from stacked block number i which is reclaimed in reclaim block number j. For each output block the proportion which comes from each input block can be calculated with *Equation 10*.

$$\boldsymbol{p_{i,j} = m_{i,j} / \sum_i m_{i,j}} \tag{10}$$

If we know the grades of stacked blocks as a vector $[g_1, g_2, \ldots g_s]$ and we are prepared to assume that the input blocks are fairly uniform, then the grades of output blocks can be found by calculating weighted averages. In general, output block number j has grade $\Sigma_i p_{i,j} g_i$. (Robinson, 2004) These calculations of weighted averages are illustrated in an example below.

As an example presented by Robinson (2004), if the grades of the four input layers are 13, 15, 14 and 17, the blended ore grades of the reclaimed output layers would get ore grade values of 15.26, 14.41, and 13.81. This method clearly indicates that it is possible to lower the variance in ore grade by a simple stockpiling technique.

The windrow-stacked stockpile has the same principal foundation as the chevron-stacked stockpile, and will therefore not be presented with such accuracy. *Figure 7* illustrates the cross-section of a windrow-stacked stockpile. There is a significant difference in the lay-out of the windrow-stacked stockpile to the chevron-stacked stockpile. The differences between benches are generally much larger for windrow-stacked stockpiles than for chev-ron-stacked stockpiles, because the benches are further from being an equal mixture of all of the input blocks.

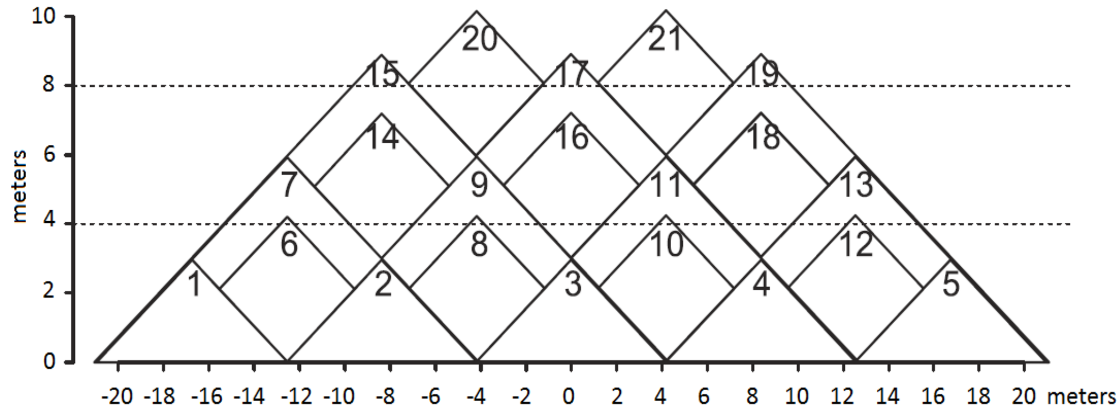

***Figure** 7. Cross-section of a windrow-stacked stockpile (modified: Robinson, 2004).*

To conclude, there are three steps in making useful predictions of the blending performance of stockpiles.

1. Prediction of the geometry of a stockpiling system, meaning the manner in which output portions of material are composed of input parcels of material.
2. Prediction of the variability of the grade of incoming material.
3. Combination of the predictions about geometry and grade in order to produce predictions about the variability in output grade. *(Robinson, 2004)*

Blending after stockpiling

The second strategy for blending the ore is to blend it previous to the mill feed process. An overview of this process will be presented below. The main idea in this method is to create multiple ore stockpiles, with each stockpile having different properties in ore grade and undesired mineral composition (*Figure 8*). It is crucial that all stockpiles have an ore grade which is at least the cut-off grade. This method is used at Agnico Eagle's Kittilä gold mine. The method is presented and analyzed with the help of an interview with Kittilä mine's mine geologist Koppström (2015).

Once multiple ore stockpiles have been generated, as shown in an example in *Figure 8*, the blending of ore feed can be managed by combining specific portions of some or all of the stockpiles. The aim is to produce a composition with the targeted quantity and ore grade quality. (Koppström, 2015)

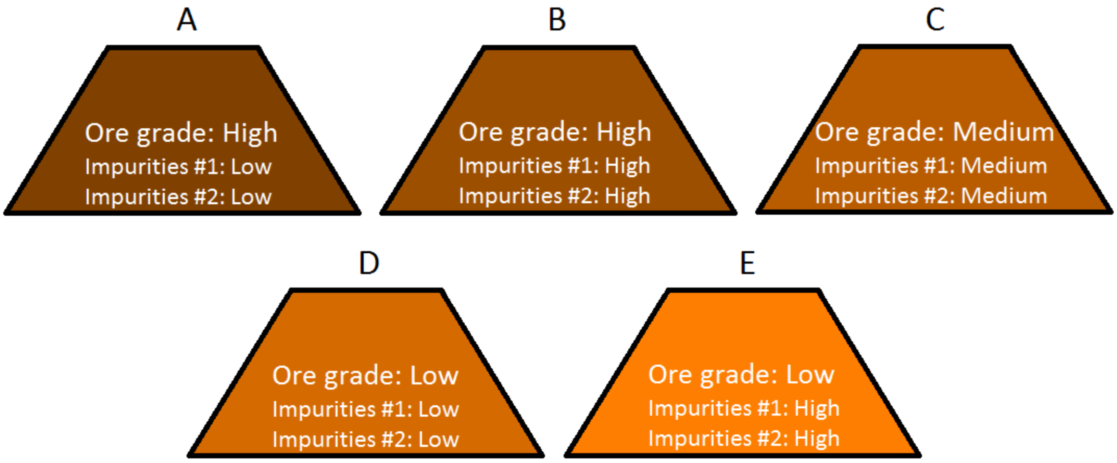

Figure 8*. Development of multiple ore stockpiles according to ore grade and mineral composition.*

An example of this blending strategy is presented in *Table 9*. The distribution of ore grades and impurities are presented in *Figure 9*. There is an indefinite amount of combinations that will result in the targeted ore grade and impurity properties. At this point, four scenarios are presented. To obtain the optimal blend of ore from multiple stockpiles, an iterative process should be implemented. This process should account for the minimum targeted ore grade and additionally the maximum allowed impurity contents.

Table 9*. A method to create the optimal blending strategy for multiple ore stockpiles.*

				Scenario 1	Scenario 2	Scenario 3	Scenario 4
Stockpile	Ore grade	Impurity #1	Impurity #2	Portion [%]	Portion [%]	Portion [%]	Portion [%]
X_A	10 %	1.0 %	1.5 %	**20 %**	**15 %**	**30 %**	**20 %**
X_B	10 %	2.7 %	3.0 %	**20 %**	**10 %**	**0 %**	**10 %**
X_C	7 %	1.9 %	2.1 %	**20 %**	**20 %**	**25 %**	**25 %**
X_D	5 %	1.1 %	1.0 %	**20 %**	**30 %**	**25 %**	**30 %**
X_E	5 %	3.0 %	2.9 %	**20 %**	**25 %**	**20 %**	**15 %**

Target		**Scenario 1**	**Scenario 2**	**Scenario 3**	**Scenario 4**
Quantity [tons]	4 000	4 000	4 000	4 000	4 000
Min. Ore grade [%]	7 %	7.40 %	6.65 %	7.00 %	7.00 %
Max. Impurities #1	2 %	1.94 %	1.88 %	1.65 %	1.73 %
Max. Impurities #2	2 %	2.10 %	1.97 %	1.81 %	1.86 %

In *Table 9*, there are four scenarios to blending of the ore. In scenario 1, an equal portion of 20% of the total targeted quantity is taken from each stockpile A-E. This results in a total ore grade of 7.4% which is within the limit of at least 7%. The impurity content of impurity #2 is however above the maximum allowed content. Therefore scenario 1 is not a tolerated mill feed composition. In scenario 2 the portions of ore from each stockpile have been modified in order to meet the restrictions in impurity content. In this scenario however, the minimum required ore grade is not accomplished. Therefore scenario 2 is also not a tolerated composition for the mill feed. Scenarios three and four represent compositions that can be used for an optimal blending of the ore. In both scenarios the minimum required ore grade of 7% is achieved and additionally both impurity content grades are within the restricted limits. Depending on the company strategy and possibly the amount of ore available on each stockpile, either scenario can be chosen. In scenario three, the blended ore is created from stockpiles A, C, D, and E, leaving stockpile B untouched, whereas scenario four establishes a similar ore blend using all the available stockpile A-E.

Figure 9*. Ore grade and impurity content distribution for ore stockpiles A-E. Bubble size indicates ore grade content.*

The process of finding the right scenario for a mining operation is an iterative process like mentioned before, and should be adjusted according to company strategy. The parameters presented in *Table 9* and *Figure 9* are imaginary and were chosen to show the principal logic behind this method.

2.2.4 Supply and demand uncertainty

In the mining industry, both ore supply and mill feed demand face uncertainties due to multiple reasons. There are however many strategies to cope with these uncertainties. In this chapter, an example from the Kittilä mine is presented, showing the variations within ore supply and mill demand.

Real parameters of Kittilä mine ore supply and mill demand for a time of one year were analyzed. By examining the ore supply and mill demand parameters, shown in *Figure 10*, it can be seen that both operations tend to follow an approximation of a normal distribution. In *Figure 10*, the daily mill feed has roughly 40 days of zero mill feed during the year. These 40 days are a result of scheduled and unscheduled maintenance to the mill, during which it cannot operate. During the one year period shown in *Figure 10*, the daily mined ore has varied from 100 tons to over 9 000 tons per day. However, these end values are extremely rare but do occur every now and then. The values for each day are not in chronological time order, they have been sorted from smallest to largest.

Figure 10 indicates that the total yearly ore supply is more or less consistent with the yearly mill feed demand. However, it also indicates that the mill feed demand is slightly higher than the ore supply if neglecting the time during which the mill is under maintenance. To conclude, during the time the mill is under maintenance, the ore supply will make up for the lower relation to the mill feed demand. Another factor is that the daily ore supply manages to get larger values in the high end of the curve. This will additionally balance the supply demand ratio during the time of one year.

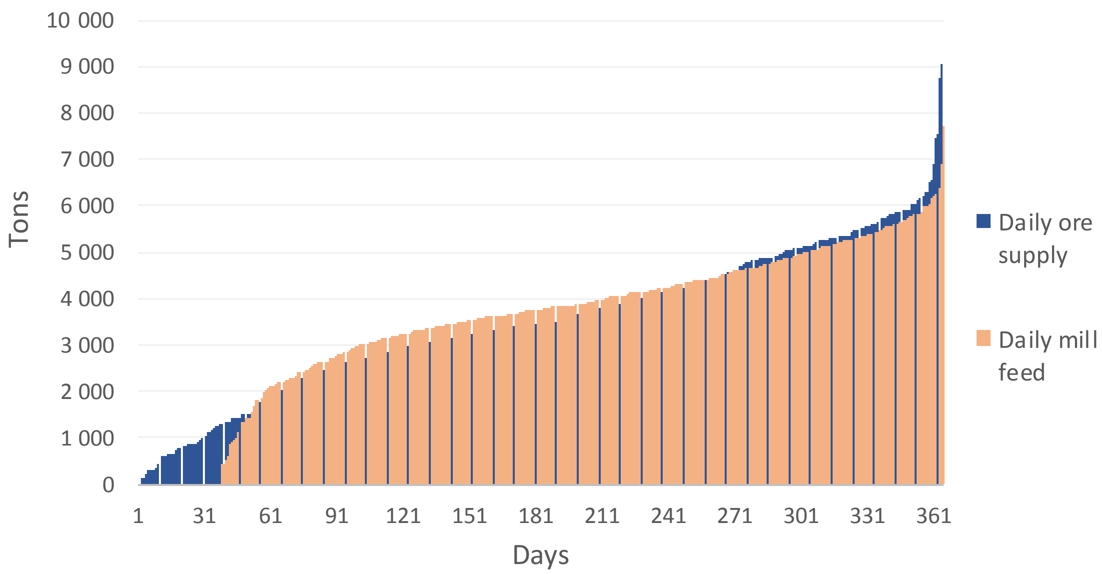

Figure 10*. Comparison of Kittilä mine ore supply and mill demand on a yearly basis.*

The parameters of ore supply and mill demand from Kittilä mine were modified in order to evaluate the statistical manner in which they behave. *Figure 11* shows that both the ore supply and mill demand approximately follow a normal distribution with different properties. In *Figure 11* the mill shutdown period of roughly 40 days has been neglected. *Figure 11* shows that mill demand follows a narrower and higher normal distribution than that of ore supplies normal distribution. The mill demand gets a mean value of 3 900 tons per day, indicated at the peak of the bell curve, whereas ore supply gets a mean value of 3 520 tons per day. This would result in running out of supply because the mean demand is larger than mean supply. However, if taking into account also the mill shutdown period, shown in *Figure 12*, the mean for the mill feed demand is 3 500 and the mean for the supply is 3 520. In this scenario the yearly ore supply would suffice for the mill feed demand.

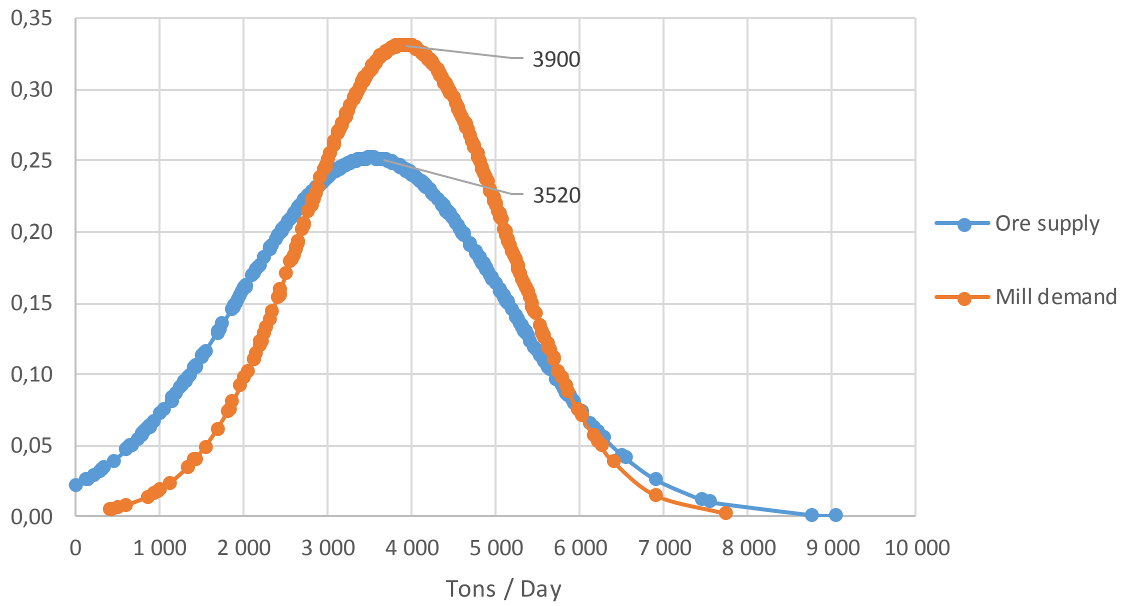

Figure 11*. Normal distribution of Kittilä mines ore supply and mill demand on a yearly basis (neglecting mill shutdown period).*

Figure 11 and *Figure 12* indicate that ore supply and mill demand follow approximately a normal distribution, and thus can be statistically estimated. The standard deviations of ore supply and mill demand in *Figure 11* are 1 590 tons and 1 200 tons respectively. The standard deviation of ore supply is significantly higher than that of mill demands standard deviation. This is also balanced in *Figure 12*, where the standard deviation of ore supply gets a value of 1 590 tons and the standard deviation of mill demand gets a value of 1 640 tons. The mean value and standard deviation of the mill demand is balanced to the near same values as the mean and standard deviation of ore supply. This is due to the fact that when taking into account the mill shutdown period (*Figure 12*) the mill demand is balanced to the near same values as for the ore supply.

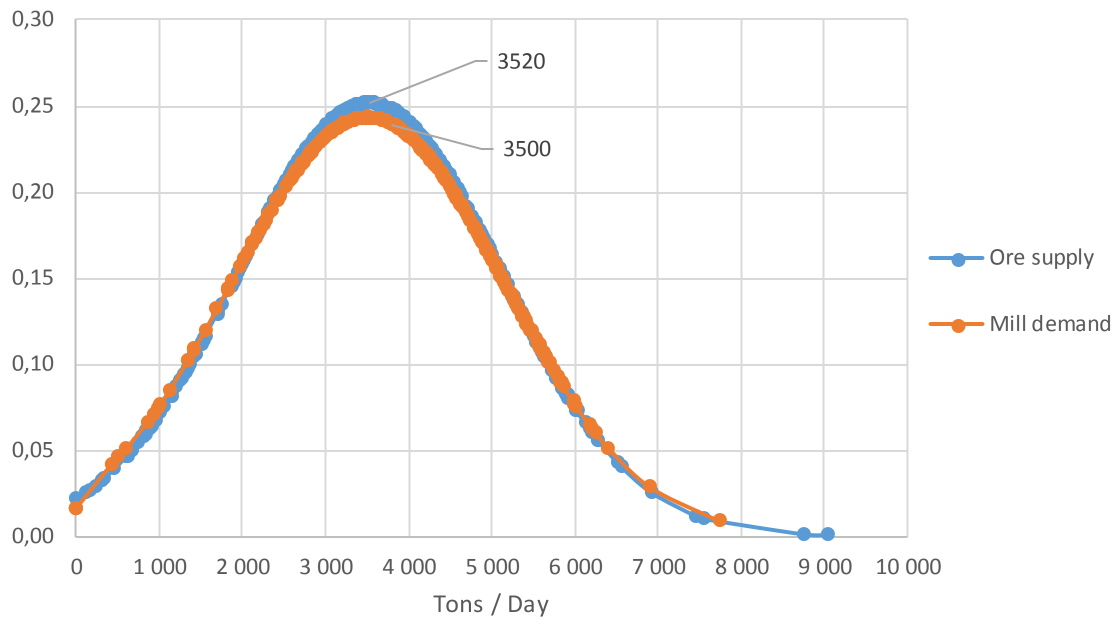

Figure 12*. Normal distribution of Kittilä mines ore supply and mill demand on a yearly basis (with mill shutdown period).*

The approximations of normal distributions indicate that both ore supply and mill demand receive values close to the mean values significantly more often than boundary values. However, the daily values of ore supply and mill demand can fluctuate dramatically within back to back days, shown in *Figure 13*. To conclude, the normal distributions represent long term scenarios. If observing short periods of time the values will represent of a stochastic variation rather than a normal distribution.

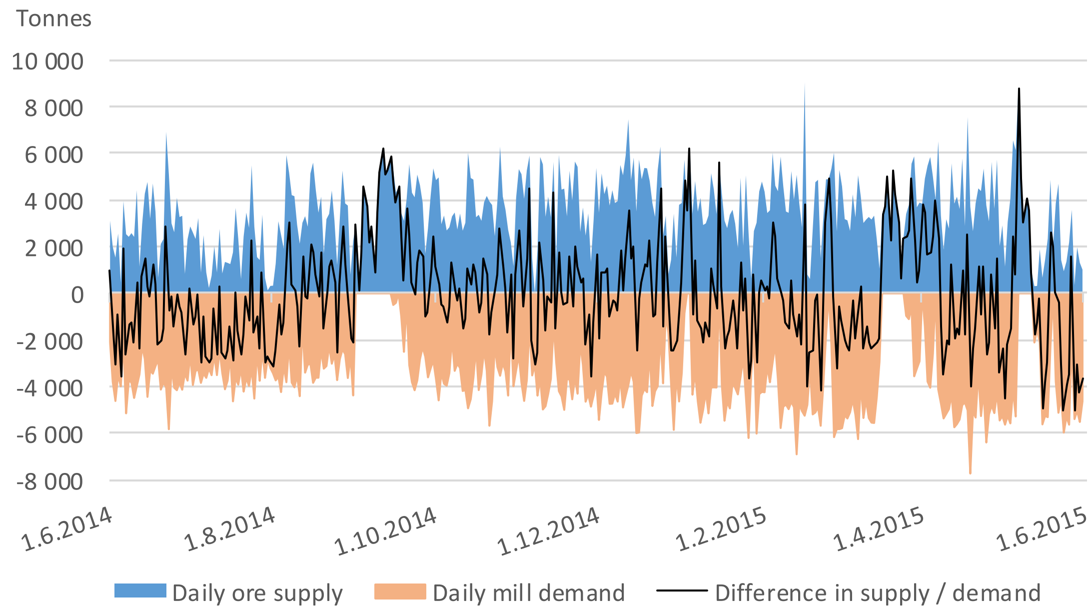

Figure 13*. Kittilä mine daily ore supply and mill demand from 1.6.2014 - 1.6.2015.*

2.2.5 Inventory systems that model safety stock

There are a few basic inventory systems that model safety stock for different retail and manufacturing industries. These models are however not meant for mining operations directly, but can be used as of assistance in creating a solution for mining operations. Since the principal logic behind modelling safety stock estimations through inventory systems is the same in both cases, these models for retail and manufacturing industries will be studied.

The most significant difference between the retail industry and the mining industry regarding safety stock is the fact that there is usually a zero tolerance for stockout in the mining industry. Whereas there might only be a 95% service level for a retail industry. This can however be dealt by increasing the z-score value to be over three, which will increase the service level to more than 99.9% as explained in chapter *2.2.1* on page *28*. Another significant difference between the different industries is that the main source of uncertainty in the retail industry occurs on customer demand, whereas the uncertainty in the mining industry occurs both on supply and demand. Demand uncertainty in the mining industry can however be dealt with blending strategies, as explained in chapter *2.2.3* from page *33* onwards. The main principle behind this is to maintain a steady mill demand with a targeted quantity and quality of ore.

For example an inventory model designed for retail industry with a service level of 95%, shown in *Figure 14*, there will be an expected 50% of the time that cycle stock will not be depleted and safety stock will not be needed. Additionally there will be an expected 45% of the time that cycle stock will be used but the amount of safety stock will suffice to keep the process running. That leaves 5% of the time in which both the cycle stock and safety stock will be depleted and result in stockout. (King, 2011) This model shows that when increasing the z-score (safety factor) it will result in fewer stockouts, but at the same time it will require a significantly larger safety stock.

It could be argued that safety stock levels should be increased to suffice a safety factor of 4 (service level 99.997%) and above in order to not run out of stock. However, the inventory costs and the storage capacities of such amounts of safety stock grow in the same respect. Therefore there must be a balance between inventory costs and the cost of stockout. The cost of stockouts are however extremely hard to evaluate, since there are so many factors involved. By using the equations in chapter *2.2.1 Conventional method for determining safety stock*, desired service- and safety stock levels can be calculated.

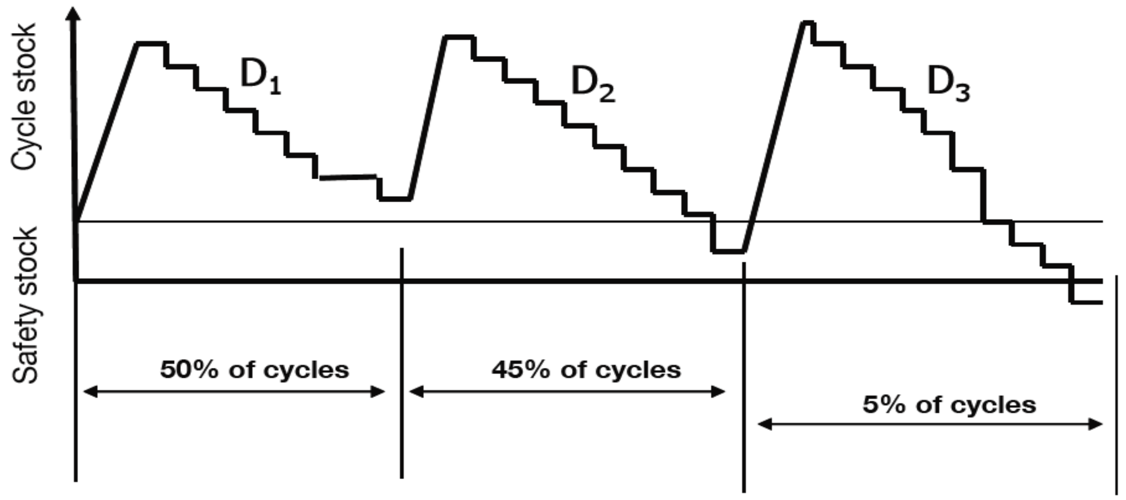

Figure 14*. Inventory model designed for a service level of 95% (King, 2011).*

The previously described scenario is an example of a Q-model inventory system. This Q-model inventory system, also known as fixed-order quantity model (FOQ) is further demonstrated in *Figure 15*. Additionally to the Q-model, another well-known inventory system model is the fixed-time period model (FTP). The FTP model is further demonstrated in *Figure 16*. Both of these models are mainly designed for retail and warehouse inventory systems, where materials or goods are ordered e.g. from a manufacturer. In the mining industry, there is naturally no such ordering of materials, because the ore is mined with own processes and not bought from outside. Therefore a reordering point, which is implemented in both models, does not apply for the mining industry.

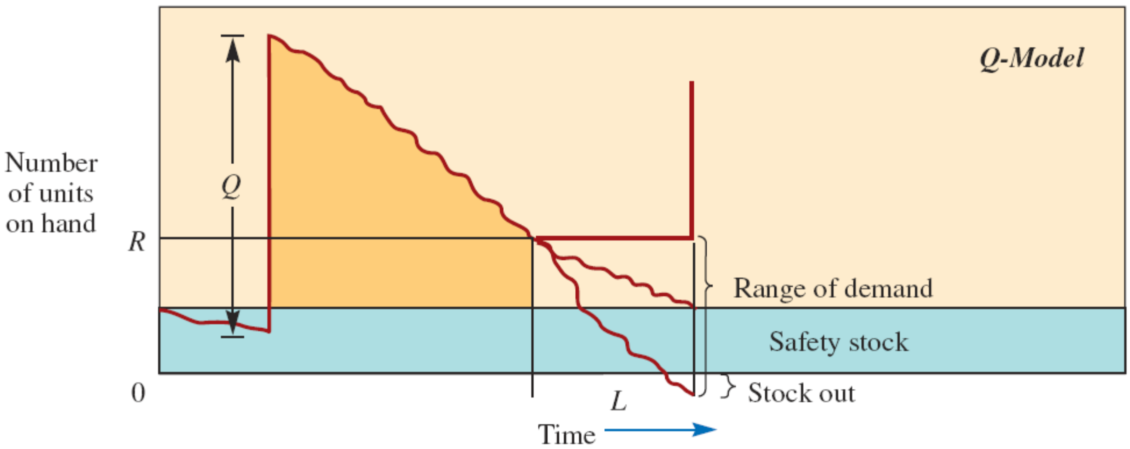

Figure 15*. Fixed-order quantity (FOQ) model for inventory management (Jacobs & Chase, 2013).*

The fixed-order quantity model (Q-model) is an inventory control model where the amount requisitioned is fixed and the actual ordering is triggered by inventory dropping to a specified level of inventory. Whereas the fixed-time period model (P-model) is an inventory control model that specifies that inventory is ordered at the end of a predetermined time period. In this model the interval of time between orders is fixed and the order quantity varies. (Jacobs & Chase, 2013)

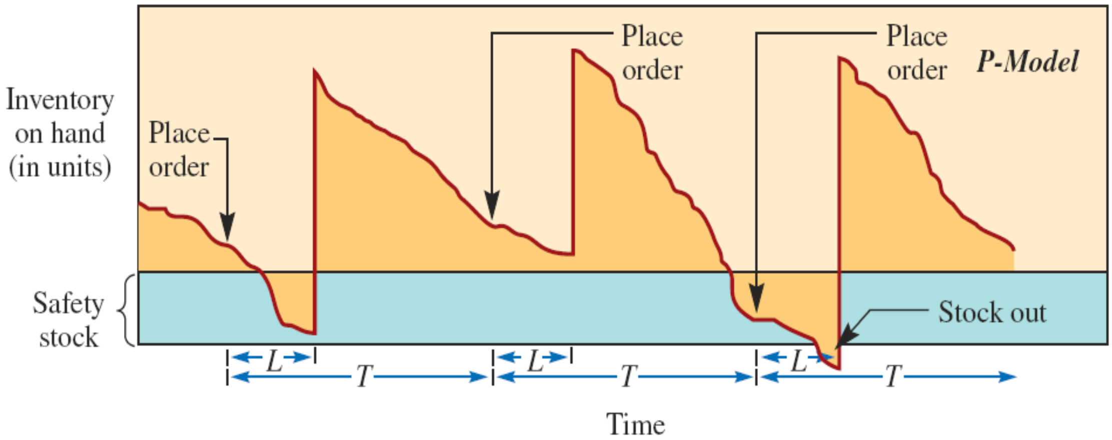

Figure 16*. A Fixed-time period (FTP) inventory model (Jacobs & Chase, 2013).*

The basic distinction of these models is that the FOQ models are event triggered and the FTP models are time triggered. For companies trying to establish optimum safety stock levels, it is common to simply state that a certain amount of supply e.g. 10% of the monthly demand need to be kept in safety stock. However, it is better to use a statistical approach that captures the variabilities in demand and supply. This can be managed by a method called the probability approach. In this probability approach, it is assumed that the demand over a period of time is normally distributed with a mean and a standard deviation. The probability of stocking out over a period of time can be solved by plotting the normal distribution of the expected demand and comparing it to what is on hand.

The equations presented by Jacobs & Chase in 2013 for evaluating safety stock for the fixed-order quantity model and the fixed-time period model are shown in *Appendix 3*. These equation are similar to equations presented in chapter *2.2.1 Conventional method for determining safety stock*. The principal idea behind these models will be implemented for creating a method to optimize safety stock for mining operations in chapter *4 Safety stock optimization*.

3 Implementation of ESOS at Kittilä mine

3.1 Kittilä mine

Kittilä mine is one of the largest underground gold mines in Europe to date. The mine is located in Lapland, northern part of Finland. The Kittilä mine is owned by a Canadian mining company Agnico Eagle Mines Limited. The mine started its production in the beginning of 2009 with two open-pit mines Suurikuusikko and Rouravaara. From 2010 onwards the development and production was gradually focused towards underground mining. By the end of 2012 both open-pits had been mined out and production continued only on underground mining. (Agnico Eagle, 2014)

The underground mine covers both Suurikuusikko and Rouravaara orebodies. The two orebodies, shown in *Figure 17*, are however so far apart that the mine layout consist of two main drifts, VT1 drift for Suurikuusikko and VT2 drift for Rouravaara. The basic layout of the Agnico Eagle Kittilä mine is presented in *Figure 18* and further with more detail and size in *Appendix 4*, *Figure 36*.

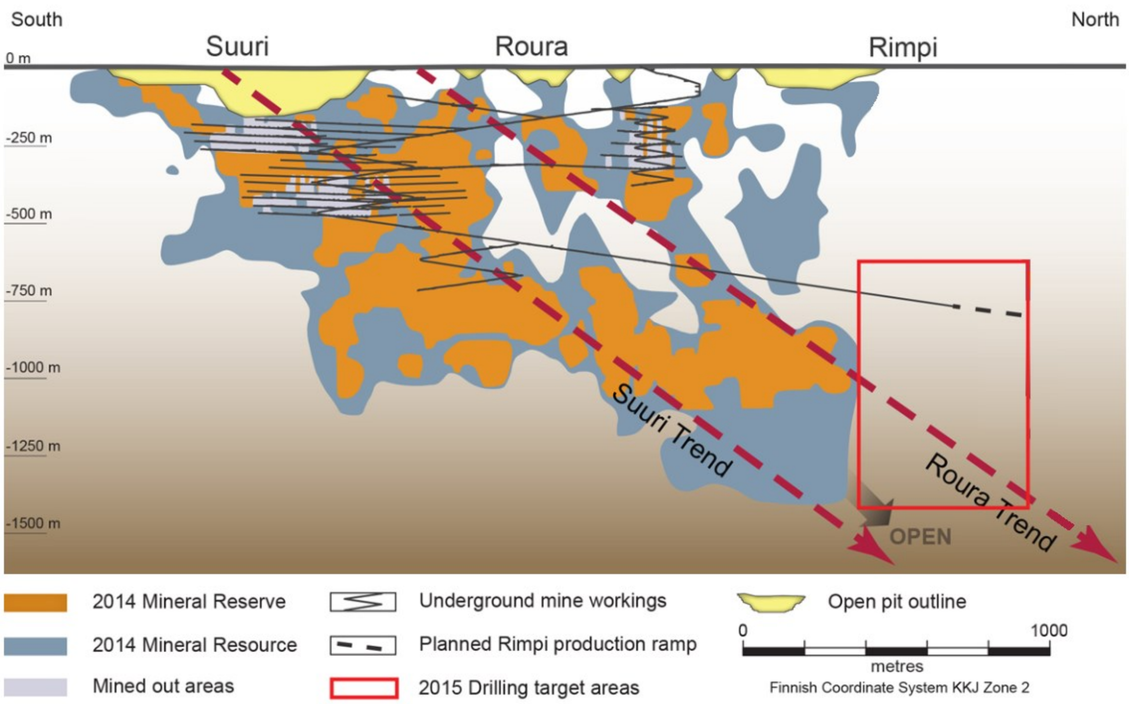

Figure 17*. Kittilä mine orebodies with underground workings (modified: Agnico Eagle, 2014).*

The Suurikuusikko orebody at the Kittilä mine has been developed with a 25 meter level difference. The first mining operations have been performed at a level of 125 meters from the ground surface. The development drifts reach up to 700 meters below the ground surface. The Rouravaara orebody at the Kittilä mine has been however developed with a 40 meter level difference. This is due to significantly different rock conditions within the two areas. The Rouravaara development drifts reach depths of 150 to 390 meters from the surface. Additionally to the two main drifts VT1 and VT2, there is a research drift (VT-TUT) which reaches more than 800 meters beneath the surface. VT-TUT is indicated in *Appendix 4*, *Figure 36*.

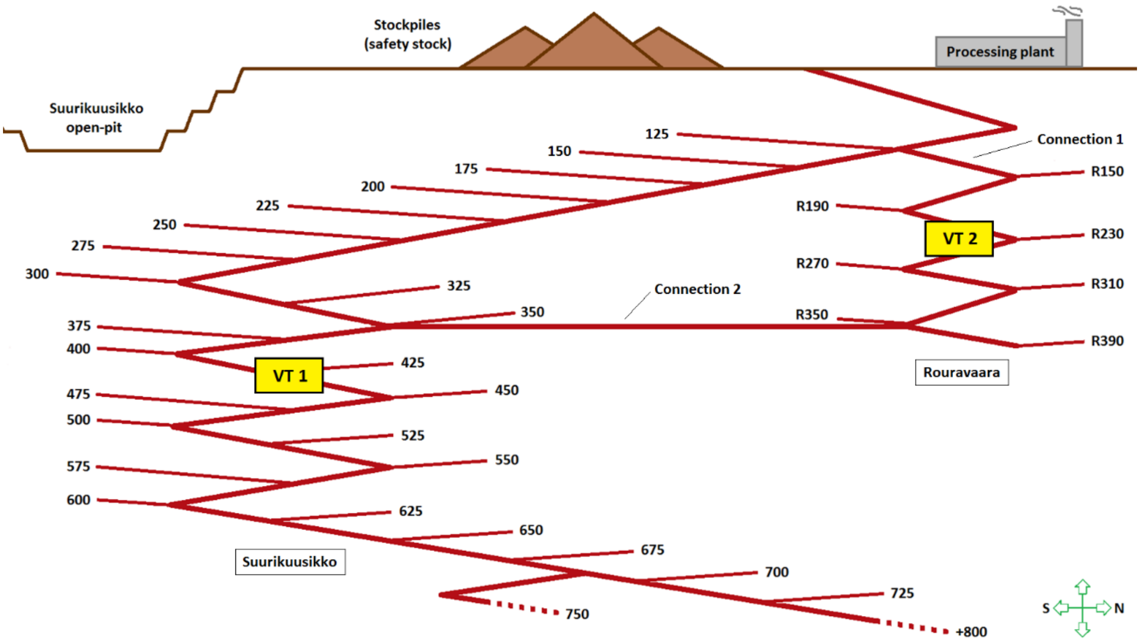

Figure 18*. Overview of Kittilä mine layout with main drifts and operating levels.*

The development drifts on each level have one or multiple workfaces, as indicated in *Appendix 4*, *Figure 36*. The equipment scheduling optimization software is tested only on optimizing the production of development drifts. It can however be further used to optimize the sequencing of stope production as well, by modifying the equipment operating rates and workface workloads.

3.2 Evaluating the needed parameters for ESOS

A significant workload was initially needed for the gathering of the input parameters for the equipment scheduling optimization software. The initial objective was to get an overview of the mine layout (*Figure 18*), in order to determine the best solution for calculating the distances between workfaces. The workface-workface distance input file was initially created in a way that the mine was divided into levels, according to the real mine layout (*Appendix 4*, *Figure 36*). In the building phase of the workface-workface distance input file, an assumption was made that the distances between workfaces located on the same level will be set to 0. This assumption was made not only to ease the already tedious workload but also due to the fact that the maximum distances between workfaces located on the same level are significantly smaller than the distances between workfaces located on different levels. Furthermore, the main drift connecting the different levels has an incline of 1:7 which means that the machines travelling on the inclined ramp will move significantly slower than on the horizontal drifts of each level. Moreover, the distance of the inclined ramp results in significantly greater distances than the distances between adjacent workfaces, which also supports the made assumption. As a result, each workface will have a distance of 0 meters to adjacent workfaces on the same level, as shown in *Table 28* and *Table 29* in *Appendix 6*. This might create a slight error of a few hundred meters in the travelled distances, depending on the mine layout. However, this additional distance and time which is not accounted for in the software can be saved in not having to change the water and electricity source when changing the workface at the same level.

As mentioned in the previous chapter, there are two main drifts at the Kittilä mine, the Suurikuusikko drift (VT1) and the Rouravaara drift (VT2). These two main drifts have two connections to one another at levels 150 meters and 350 meters, with horizontal distances of 400 meters and 700 meters respectively. The connections are indicated in *Figure 18* by connection 1 and connection 2. In order for the software to achieve the shortest time span for the equipment scheduling, a manual optimization had to be made for the workface distances between Suurikuusikko and Rouravaara. Because there are two connections that are of different lengths between Suurikuusikko and Rouravaara, it had to be manually calculated which route would give a shorter distance when travelling from e.g. Rouravaara level R390 to Suurikuusikko level 125. In this example, taking connection 1 at level 150 will result in a total distance of 2 255 meters whereas taking connection 2 at level 350 will result in a total distance of 2 555 meters. By performing this comparison with all the combinations of workface-workface distances between the two parts of the mine (Suurikuusikko and Rouravaara) the minimum distances where obtained for the distance input file. The whole distance input file is shown in *Table 28* and *Table 29* in *Appendix 6*. This distance input file is however later scaled down to consist of only the needed workfaces which are prioritized before running the software. In order for the software to process the distance input file (matrix) with ease, it is recommended that all workfaces, with no workload for any machine type, should be neglected from the final distance and workload input files. However, if the initial location of any machine is at a workface that has no operation planned for it, this workface must be kept in the distance and workload input file. In this case the workload for that workface is zero for all machines. This will ensure the software to be able to calculate the distance travelled from the initial location to the first workface. An example of this is shown in *Table 2*.

The operating rates for all machine types were gathered from historical data. The historical data, together with manufacturer's specifications were averaged in order to get as realistic rates as possible. These rates were then used for the actual testing. However, this creates a slight problem when comparing the results of actual operations and the software's planned operations. The problem occurs when real machines operate always in different rates even though they are theoretically set to work at a constant rate. The comparison between actual and plan has to be adjusted by taking the average time for one round for each machine type. For example, if a machine operates in a drift in one round (= 4.7 meters) for a time of 1 hour and then in another drift in one round (= 4.7 meters) for a time of 2 hours, due to rock conditions etc. the average operating rate for that machine will be 4.7 m / 1.5 h = 3.13 m / h. This will result in the software giving an operating time of 1.5 hours for one round for that machine, although it would actually be 1 hour on the first workface and 2 hours on the second workface. This could be however optimized manually by setting the workface workload higher for the second workface. This is however very difficult because it is hard to predict which workfaces will take longer even though they are the same in round length. To conclude, the assumption has to be made, that all workfaces (with one round being 4.7 meters) will take the same amount of time to be operated for a specific machine type.

The workface workload must be set to the value of what one round of excavation requires in meters. This is due to the fact that the software executes the total amount of workload for each workface in a row. For example, if there is a drift under development, with a plan of 50 meters excavated in the next month, the software has to be re-run after each round (e.g. 4.7 meter round) is completed. Otherwise if the workload is set to 50 meters, the software thinks it has to first scale 50 meters in a row, then shotcrete 50 meters in a row etc. However, this is not the case in a real mining operation, where you can only excavate a certain amount per round. It is therefore suggested to execute the software on a daily basis or whenever a failure occurs with any of the equipment. When running the software on a shift basis (daily basis), another benefit is the fact that there is not even enough time to excavate more than one round in a workface, so the workload of each workface won't exceed the one round limit.

3.3 Discrepancy of actual and planned equipment scheduling

Theoretically the software should be able to optimize the sequence of the underground mining machines. If the operating rates are assumed to be the same in theory and in practice, the time used at each workface is the same. Thus, only the distance travelled and the machine efficiency (minimum machine waiting time at workface) need to be optimized. The software will then find the shortest time span for the whole operation relative to the total distance travelled by the machines (shortest distance = shortest time spent) and the efficient use of the machines.

Multiple tests were performed on a shift basis at the Kittilä mine using the equipment scheduling optimization software. In addition to real time testing at the mine site, testing of ESOS on historical mine data was also performed, where the sequences of mining operations and their durations were known. Only the most significant scenarios produced by the scheduling optimization software are presented. These scenarios give an adequate overview of how the ESOS is performing in comparison with actual operation.

Scenario 1

The first scenario presents a day shift from Kittilä mine in 30.6.2015. During the day shift there were two scalers, one shotcreter, two bolters, one driller and one charger available for use. The specific input parameters used for Scenario 1 are presented in *Appendix 2* (*Table 21, Table 22* and *Table 23*). These machines were then allocated during the day shift to the prioritized workfaces set by site managers. The allocation of equipment is mainly done by intuition, experience and drift priority levels. The allocation of machines during the day shift is presented in *Table 10*.

***Table 10**. Scenario 1: Sequencing, distances, and duration of actual operation at Kittilä mine.*

Actual operation											
30.6.2015 Development (Scenario 1)		Initial location	Target workface				Operating time	Moving time [min]	Total time	Travel distance [km]	Total distance [km]
			#1	#2	#3	#4					
SCL01	Scaler	7	10	12	13		3h 30min	26	3h 56min	5.2	8.4
SCL02		7	3	5	7	14	4h 30min	16	4h 46min	3.2	
SHO01	Shotcreter	7	6	3	2		6h	2	6h 2min	0.4	0.4
BOL02	Bolter	7	15				3h	23	3h 23min	4.1	6.4
BOL04		7	11				2h	13	2h 13min	2.3	
JUM02	Driller	4	9	8	4		8h	15	8h 15min	2.5	2.5
CHA01	Charger	7	0	1			4h	5	4h 5min	0.9	0.9

The equipment scheduling optimization software was executed with the same initial data for the daily shift. The output of the equipment scheduling optimization software is presented in *Table 11*. The output of ESOS shows different sequences of workfaces for the actual operation and the planned operation. At this point the different sequencing of workfaces between the actual and planned operations doesn't necessarily mean that the solution provided by ESOS is better than that of the intuitively chosen solution by the site manager.

Table 11. *Scenario 1: Sequencing, distances, and duration of planned operation with ESOS at Kittilä mine.*

Planned operation											
30.6.2015 Development (Scenario 1)		Initial location	Target workface				Operating time	Moving time [min]	Total time	Travel distance [km]	Total distance [km]
			#1	#2	#3	#4					
SCL01	Scaler	**7**	**14**	**12**	**13**	**10**	4h 34min	27	5h 1min	5.4	5.7
SCL02		**7**	**7**	**5**	**3**		3h 26min	2	3h 28min	0.4	
SHO01	Shotcreter	**7**	**6**	**2**	**3**		6h	2	6h 2min	0.4	0.4
BOL02	Bolter	**7**	**15**				2h 30min	23	2h 53min	4.1	6.4
BOL04		**7**	**11**				2h 30min	13	2h 43min	2.3	
JUM02	Driller	**4**	**4**	**8**	**9**		8h	7	8h 7min	1.2	1.2
CHA01	Charger	**7**	**0**	**1**			4h	5	4h 5min	0.9	0.9

With the given data ESOS produced Gantt charts sorted by machine ID (*Figure 19*) and by workface number (*Figure 20*). The red marks in both Gantt charts indicate the time for moving from one workface to another, whereas the blue marks indicate the operating time for that machine in that specific workface. The dark color is an indication of a break, which in this case is a one hour lunch break.

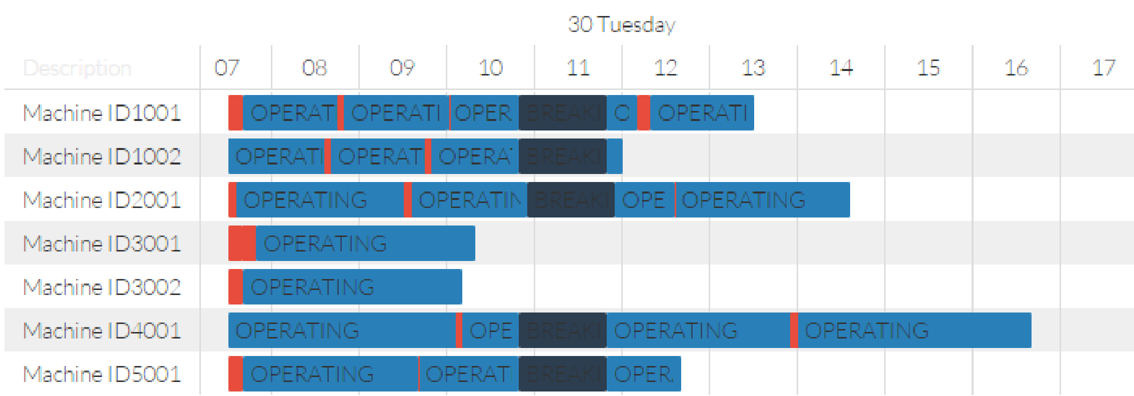

Figure 19. *Scenario 1: Produced Gantt chart by ESOS, sorted by machine ID.*

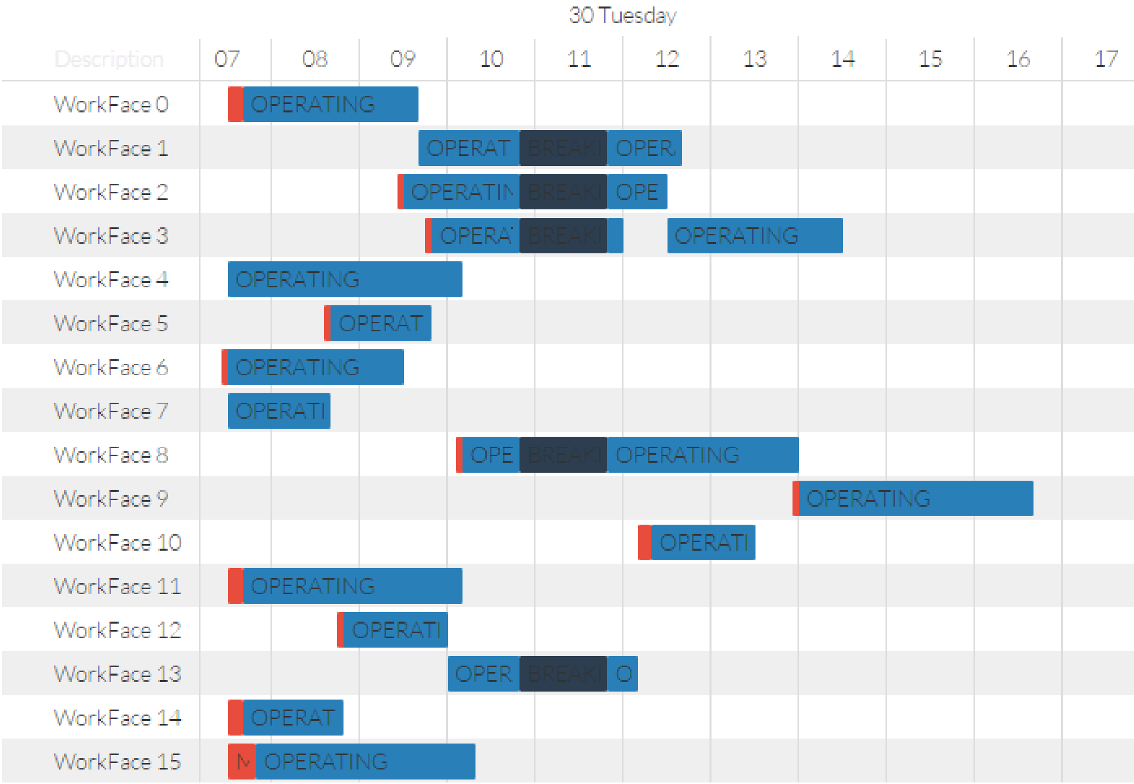

Figure 20. *Scenario 1: Produced Gantt chart by ESOS, sorted by workface number.*

The actual operation is compared with the planned operation produced by ESOS to find out the discrepancies in distance travelled and in time travelled. *Table 12* shows the comparison of actual and planned operation with discrepancies in distances travelled by different machine types.

Table 12. *Scenario 1: Comparison of actual and planned operation with discrepancies in distances travelled by machine types at Kittilä mine.*

30.6.2015 Development (Scenario 1)		**Actual operation**		**Planned operation**		**Discrepancies in distance**	
		Distance travelled	Total distance travelled	Distance travelled	Total distance travelled	Difference in distance travelled	Total difference in distance travelled
		[km]					
SCL01	Scaler	5.2	8.4	5.4	5.7	**0.2**	**-2.7**
SCL02		3.2		0.4		**-2.8**	
SHO01	Shotcreter	0.4	0.4	0.4	0.4	0.0	0.0
BOL02	Bolter	4.1	6.4	4.1	6.4	0.0	0.0
BOL03		2.3		2.3		0.0	
JUM01	Driller	2.5	2.5	1.2	1.2	**-1.2**	**-1.2**
CHA01	Charger	0.9	0.9	0.9	0.9	0.0	0.0

Since ESOS is not trying to optimize the operating efficiency of the machines, but the travelling and sequencing efficiency (minimal waiting time) of machines, the total distance travelled is very significant in the optimization. *Figure 21* presents the results of the total distance travelled for all machine types between actual operation and planned ESOS operation. In Scenario 1, the planned ESOS operation has a 3.9 km shorter total distance travelled than the actual operation.

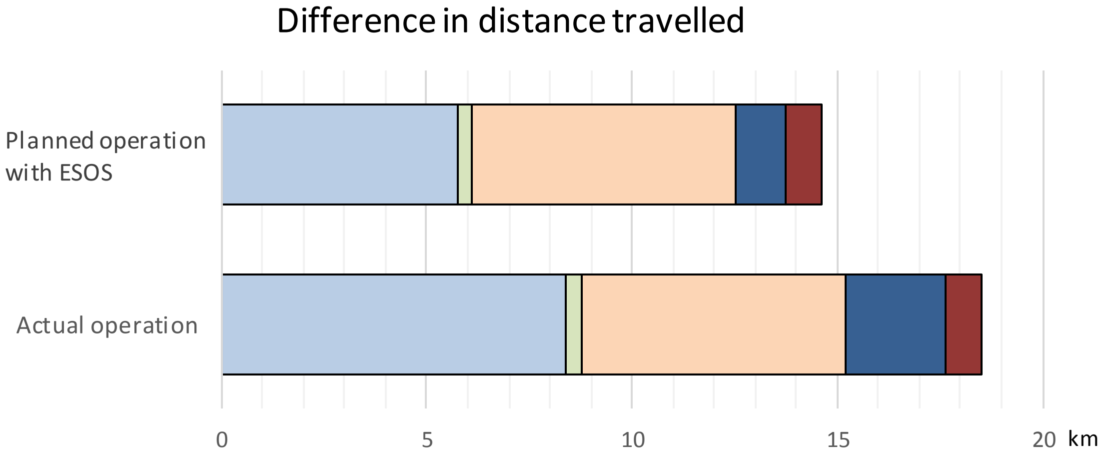

***Figure 21**. Scenario 1: Total difference in distance travelled for all machine types between actual operation and planned ESOS operation.*

The same comparison was performed for the actual and planned ESOS operation with discrepancies in the time travelled by each machine type. The discrepancies are presented in *Table 13*.

***Table 13**. Scenario 1: Comparison of actual and planned operation with discrepancies in time travelled by machine types at Kittilä mine.*

30.6.2015 Development (Scenario 1)		**Actual operation**		**Planned operation**		**Discrepancies in time**	
		Time travelled	Total time travelled	Time travelled	Total time travelled	Difference in time travelled	Total difference in time travelled
		[min]					
SCL01	Scaler	26	42	27	29	**1**	**-13**
SCL02		16		2		**-14**	
SHO01	Shotcreter	2	2	2	2	0	0
BOL02	Bolter	23	35	23	35	0	0
BOL03		13		13		0	
JUM01	Driller	15	15	7	7	**-7**	**-7**
CHA01	Charger	5	5	5	5	0	0

Figure 22 presents the results of the total time travelled for all machine types between actual operation and planned ESOS operation. In Scenario 1, the planned ESOS operation has a 20- 21 minute shorter total time travelled than the actual operation.

The results from Scenario 1 suggest that ESOS outperforms (in travel distance and travel time) the allocation of equipment in comparison to intuitively allocating the underground mining equipment. The results of Scenario 1, will be however evaluated with more depth in *chapter 5. Results*.

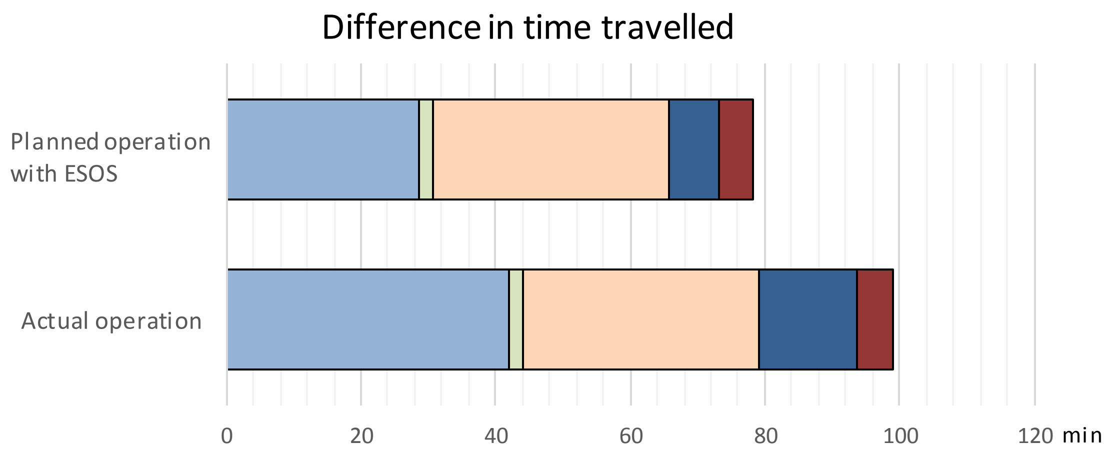

Figure 22*. Scenario 1: Total difference in time travelled for all machine types between actual operation and planned ESOS operation.*

Scenario 2

The second scenario presents a day shift from Kittilä mine in 1.2.2015. During the day shift there was one scaler, one shotcreter, three bolters, two drillers and two chargers available for use. The specific input parameters used for Scenario 1 are presented in *Appendix 2* (*Table 24, Table 25* and *Table 26*). These machines were then allocated during the day shift to the prioritized workfaces set by site managers. The allocation of machines during the day shift is presented in *Table 14*.

Table 14*. Scenario 2: Sequencing, distances, and duration of actual operation at Kittilä mine.*

Actual operation											
1.2.2015 Development (Scenario 2)		Initial location	Target workface				Operating time	Moving time [min]	Total time	Travel distance [km]	Total distance [km]
			#1	#2	#3	#4					
SCL01	Scaler	**5**	**5**	**26**	**12**	**11**	5h	38	5h 38min	7.5	7.5
SHO01	Shotcreter	**7**	**3**	**2**	**1**	**26**	6h	24	6h 24min	4.3	4.3
BOL01	Bolter	**6**	**24**				4h	17	4h 17min	3.1	6.0
BOL02		**18**	**21**	**20**	**22**	**23***	8h	15	8h 15min	2.8	
BOL03		**15**	**15**	**14**	**13**		7h 30min	1	7h 31min	0.2	
JUM01	Driller	**10**	**10**	**4**	**6**		7h 30min	2	7h 32min	0.4	3.0
JUM02		**19**	**19**	**25**			5h	16	5h 16min	2.6	
CHA01	Charger	**16**	**17**	**8**			4h	16	4h 16min	2.7	4.1
CHA02		**0**	**9**	**10**			4h 30min	8	4h 38min	1.4	
* **Workface 23**, only 20% of operation completed within the shift, 80% work left.											

ESOS was executed with the same initial data for the daily shift. The output of the equipment scheduling optimization software is presented in *Table 15*. The output of ESOS shows again different sequences of workfaces for the actual operation and the planned operation.

***Table 15**. Scenario 2: Sequencing, distances, and duration of planned operation with ESOS at Kittilä mine.*

Planned operation											
1.2.2015 Development (Scenario 2)		Initial location	Target workface				Operating time	Moving time [min]	Total time	Travel distance [km]	Total distance [km]
			#1	#2	#3	#4					
SCL01	Scaler	**5**	**12**	**11**	**5**	**26**	5h	28	5h 28min	5.5	5.5
SHO01	Shotcreter	**7**	**1**	**2**	**3**	**26**	6h	24	6h 24min	4.3	4.3
BOL01	Bolter	**6**	**13**	**22**			5h	31	5h 31min	5.7	9.8
BOL02		**18**	**20**	**21**	**23**		7h 30min	15	7h 45min	2.8	
BOL03		**15**	**15**	**14**	**24***		8h	8	8h 8min	1.4	
JUM01	Driller	**10**	**10**	**4**	**6**		7h 30min	2	7h 32min	0.4	3.0
JUM02		**19**	**19**	**25**			5h	16	5h 16min	2.6	
CHA01	Charger	**16**	**17**	**10**			4h 15min	17	4h 32min	2.9	4.3
CHA02		**0**	**8**	**9**			4h 15min	8	4h 23min	1.4	
* **Workface 24**, only 75% of operation completed within the shift, 25% work left.											

With the given data ESOS produced Gantt charts sorted by machine ID (*Figure 23*) and by workface number (*Figure 24*). In this scenario, additionally to the one hour lunch break which is indicated with the dark color (10:30-11:30), a shift change break is included and indicated with the same dark color (16:30-19:15).

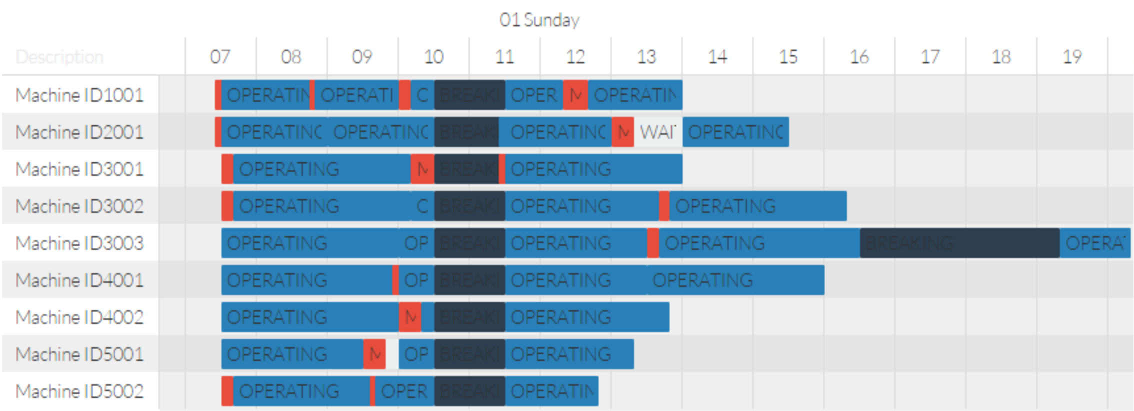

***Figure 23**. Scenario 2: Produced Gantt chart by ESOS, sorted by machine ID.*

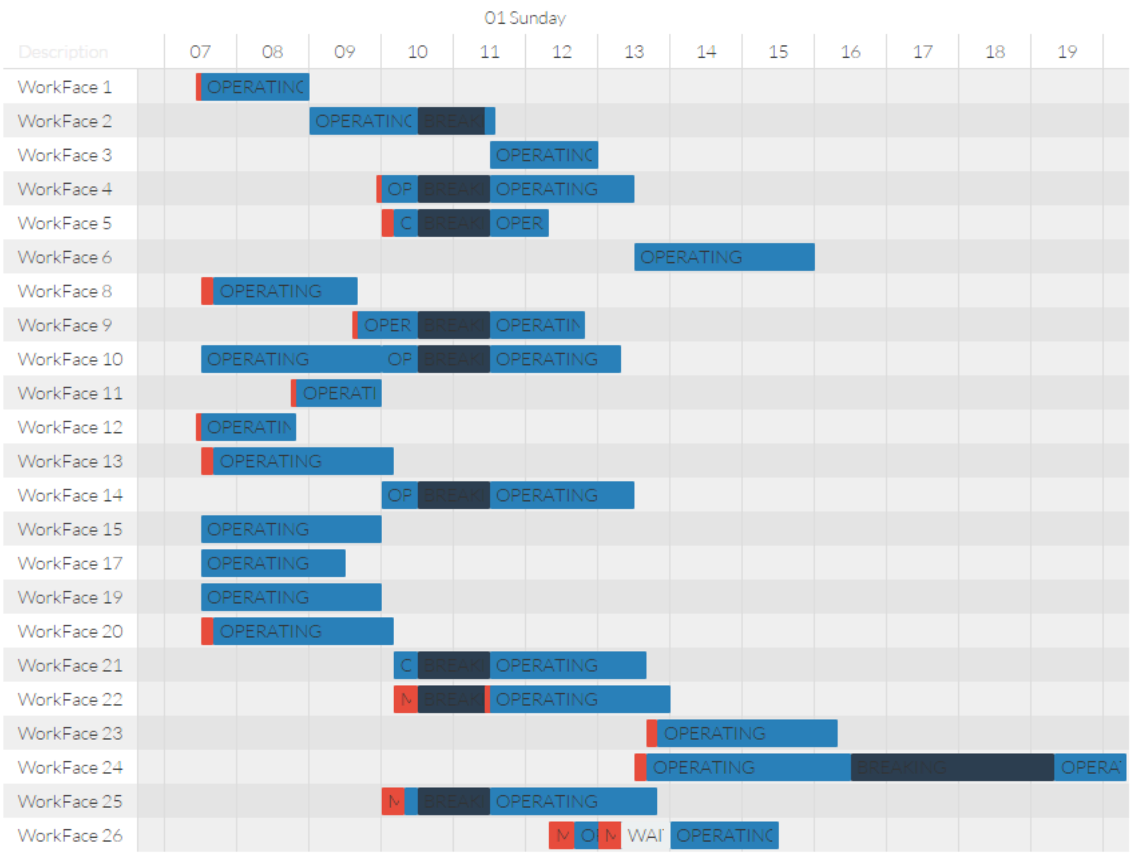

***Figure 24**. Scenario 2: Produced Gantt chart by ESOS, sorted by workface number.*

The actual operation is compared with the planned operation produced by ESOS to find out the discrepancies in distance travelled and in time travelled. *Table 16* shows the comparison of actual and planned operation with discrepancies in distances travelled by different machine types.

***Table 16**. Scenario 2: Comparison of actual and planned operation with discrepancies in distances travelled by machine types at Kittilä mine.*

1.2.2015 Development (Scenario 2)		**Actual operation**		**Planned operation**		**Discrepancies in distance**	
		Distance travelled	Total distance travelled	Distance travelled	Total distance travelled	Difference in distance travelled	Total difference in distance travelled
		[km]					
SCL01	Scaler	7.5	7.5	5.5	5.5	**-2.0**	**-2.0**
SHO01	Shotcreter	4.3	4.3	4.3	4.3	0.0	0.0
BOL01	Bolter	3.1	6.0	5.7	9.8	**2.6**	**3.8**
BOL02		2.8		2.8		0.0	
BOL03		0.2		1.4		**1.2**	
JUM01	Driller	0.4	3.0	0.4	3.0	0.0	0.0
JUM02		2.6		2.6		0.0	
CHA01	Charger	2.7	4.1	2.9	4.3	**0.2**	**0.2**
CHA02		1.4		1.4		0.0	

Figure 25 presents the results of the total distance travelled for all machine types between actual operation and planned ESOS operation. In Scenario 2, the planned ESOS operation has a 2.0 km longer total distance travelled than the actual operation.

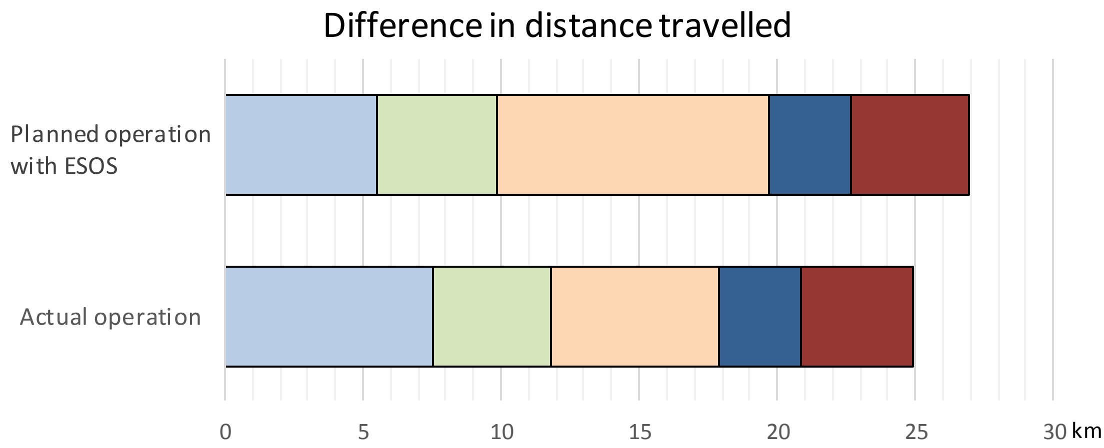

Figure 25*. Scenario 2: Total difference in distance travelled for all machine types between actual operation and planned ESOS operation.*

The same comparison was performed for the actual and planned ESOS operation with discrepancies in the time travelled by each machine type. The discrepancies are presented in *Table 17*.

Table 17*. Scenario 2: Comparison of actual and planned operation with discrepancies in time travelled by machine types at Kittilä mine.*

1.2.2015 Development (Scenario 2)		**Actual operation**		**Planned operation**		**Discrepancies in time**	
		Time travelled	Total time travelled	Time travelled	Total time travelled	Difference in time travelled	Total difference in time travelled
		[min]					
SCL01	Scaler	38	38	28	28	**-10**	**-10**
SHO01	Shotcreter	24	24	24	24	0	0
BOL01	Bolter	17	33	31	54	14	**21**
BOL02		15		15		0	
BOL03		1		8		7	
JUM01	Driller	2	18	2	18	0	0
JUM02		16		16		0	
CHA01	Charger	16	24	17	26	1	1
CHA02		8		8		0	

Figure 26 presents the results of the total time travelled for all machine types between actual operation and planned ESOS operation. In Scenario 2, the planned ESOS operation has a 12 minute longer total time travelled than the actual operation.

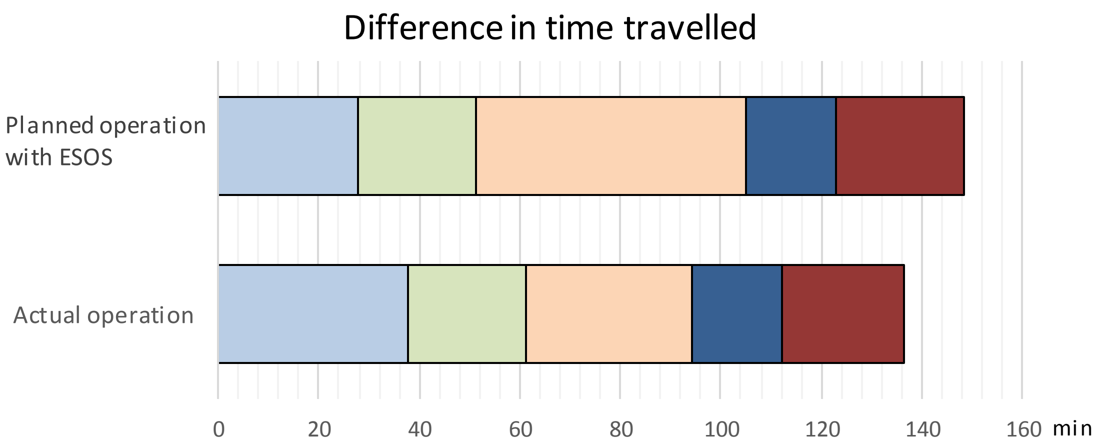

Figure 26*. Scenario 2: Total difference in time travelled for all machine types between actual operation and planned ESOS operation.*

The results from Scenario 2 suggest that ESOS underperforms (in travel distance and travel time) the allocation of equipment in comparison to intuitively allocating the underground mining equipment. Further examination of the results of Scenario 2 in *chapter 5. Results*, will however show that ESOS is not underperforming in comparison to intuitively allocating the underground mining equipment, even though having higher distances and duration.

3.4 User experience and bugs

The software was tested on a daily shift basis at the Kittilä mine for 10 consecutive days during the months of June and July 2015. The software was tested at the control room for the underground mine. At Kittilä mine, the control room operators together with the underground mine shift managers are responsible for the allocation of the mining equipment. For this thesis, the control room operators were asked to give feedback of the software interface, experience and usability of ESOS in a real mining operation.

The user interface and produced output Gantt charts received good feedback. The interface is kept simple and clear. Furthermore, the produced Gantt chart is easy to read and enables manual allocation of the operations if sudden priority changes within workfaces occur.

The software however needs a tedious amount of initial work which has to be done manually and which consumes a huge amount of time. This results in decreased efficiency and therefore allocating machines intuitively and with experience, seems like a more efficient solution. Another reason why ESOS doesn't create such significant benefit at Kittilä mine is that the underground mine is relatively small. Due to the relatively small and compact layout of Kittilä mine, the allocation of mining machines is relatively easy. Furthermore, the distances between different workfaces are also relatively short and thus the travelling time within different workfaces is marginal compared to the operating times at workfaces. Also, since there are less than a dozen mining machines working on the development of drifts, the allocation of machines, so that they don't have to wait at workfaces for the previous operation to finish, is pretty simple. Additionally, there are multiple possible workfaces that need to be operated, so the waiting time at workfaces can be easily avoided by experienced site managers.

One significant problem with the software may be that it is impossible to estimate the operating time at a workface accurately, even though the distance of one round is set to a standard length. As shown in Scenarios 1 and 2, the operating time at a workface might vary due to e.g. fluctuating rock conditions. The operating times at different workfaces are averaged out if the standard round length for a workface workload is used. It is almost impossible to predict a different workload for a workface with the standard round length. This will eventually cause discrepancies with actual and planned operations.

The software had initially quite a few bugs in the programmed code, all of which *Song et al.* were able to fix. Most of these bugs prevented the software from producing a Gantt chart of the optimized equipment scheduling. The difficulties faced with the software relate to the tedious amount of initial work that has to be carried out and processed before even getting to use the software. An assisting software in gathering the initial inputs might be of great benefit in order for the software to be of efficient assistance to the mine managers on a shift to shift basis.

4 Safety stock optimization

In the mining industry, safety stocks are utilized in order to avoid bottle necks in the production chain. The reason for wanting to avoid these bottle necks is the fact that they will cause a negative impact on the whole production chain. The negative impact can be regarded as poor operational efficiency and high working capital costs tied up in production. These can eventually lead to significant financial losses, however, all of which could be dealt by having optimal safety stock.

In this thesis, the optimization of safety stock focuses strictly at the bottle neck between supply from mine production and demand of processing plant. However, it can be mentioned that the same concept could be applied to other sections of the supply chain from mineral processing to end product sales. This would eliminate interruptions in the whole supply chain and thus maximize production efficiency.

This thesis makes the assumption that the supply of raw material (ore) has more of a stochastic than constant feature. This assumption is based on the fact that ore supply faces many uncertainties in the mining industry. These uncertainties stem from various geological conditions, such as, fluctuating ore grades and orebody geometry. Even though there have been many methods created to solve these problems to date, they cannot be predetermined with a 100% confidence level. Other significant factors that create uncertainty in the supply of raw material are the unforeseen equipment failures and logistical interruptions. These unexpected events can be dealt with by sophisticated mine management and preventive maintenance, but only up to a certain point.

The features of mill demand differ from those of the ore supply side. Mill demand can usually be considered as of having a constant feature, whereas the supply side has a stochastic feature. There are several reasons for mill demand to behave in a constant manner. The main reason for mill demand to follow a constant demand is the fact that mineral processing is a continuous process with a specific predetermined working capacity. Additionally, mining operations typically evaluate the NPV (Net Present Value) of the LOM (Life of Mine) with a predetermined constant feed for the processing plant. However, practical experience shows that this might not always be the case. The mill and the processing plant face their own problems more or less constantly. These problems and interruptions to the mill and processing plant affect the demand on a daily basis. Additionally, the problems and interruptions decrease the cumulative demand and thus also the average demand for a specific period of time.

Theoretically the mill demand can be thought to be constant, but since there are variations in the ore grade, the demand might differ over time. The mill demand is usually determined for a specific amount of specific ore grade. However, the ore grade supplied from the mine fluctuates around the cut-off grade and is therefore not at a constant value. This eventually affects the mill demand. In order for the processing plant to be able to meet the predetermined end product quantities, the mill demand needs to be increased if the ore grade decreases. Furthermore, the mill demand can be decreased if the ore grade increases. In order to achieve a constant mill demand, a blending strategy for different ore grade stockpiles should be implemented as shown in chapter *2.2.3 Ore blending strategies for mill feed*. This blending strategy will then provide a steady feed with the specific amount of specific ore grade for the mill demand. With the help of one blending strategy, the mill feed demand can be assumed to be constant as shown in *Figure 27*.

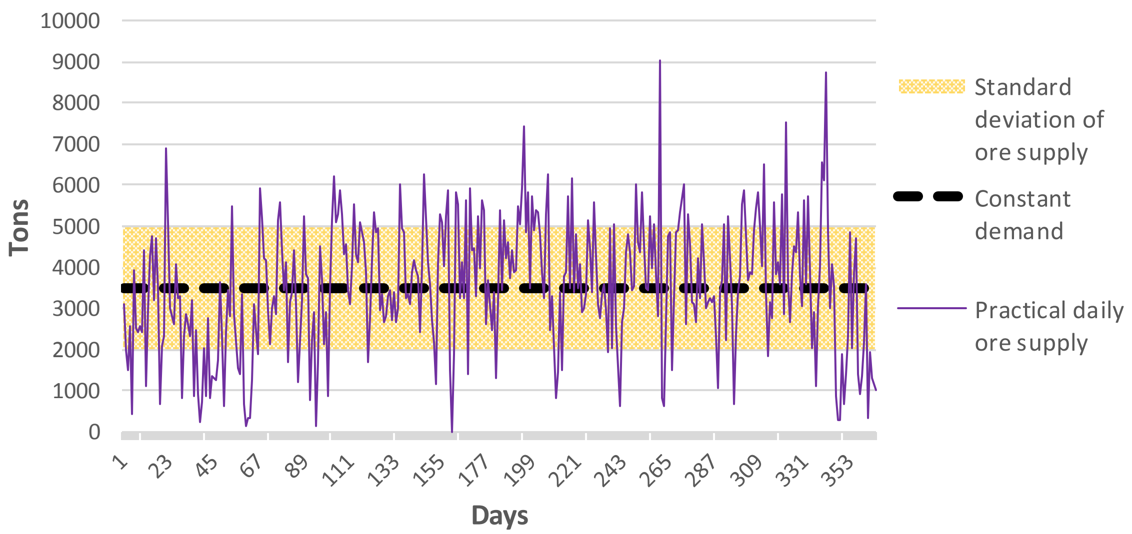

Figure 27*. A comparison of ore supply and constant (mean) mill demand of Kittilä mine for one year.*

In the mining industry, mining production and mineral processing can be either a combined integrated process or individual independent processes. From a managements perspective the combined integrated process simplifies the supply chain management where the mill feed target is defined to match the periodic ore supply qualities and quantities. In individual independent processes the need to have safety stock is increasingly high in order to avoid interruptions from stockouts to the process. In the case of individual independent processes, the amount of needed safety stock is further increased if the targeted supply of ore from the mine doesn't meet the targeted demand of the mill.

4.1 Comparison of mill demand target and ore supply target

There are numerous issues that have to be examined in mining production and mineral processing before being able to optimally determine the needed safety stock levels. There are a total of four scenarios (scenarios A-D) possible between the targeted values of ore supply and mill demand. The properties of these four scenarios are presented in *Table 18*. One of the first things that needs attention is whether or not the planned mill demand target meets with the planned ore supply target on a daily basis. In theory, the optimal scenario (Scenario A) is achieved when the mill demand is consistent with the ore supply target. This is because when constructing a mine plan at a pre-feasibility stage, you want to optimize your mill feed capacity to meet with the targeted ore supply capacity. These capacities will be smaller at the ramp up stage of the mine. However, the capacities should rise to the predetermined level simultaneously during the ramp up stage. This way the mill demand target would always meet the production target. Hypothetically, if the mill demand target is consistent with the ore supply target, there should be no need for having safety stock. The basic assumption behind this is the fact that ore supply will always fulfill the needed mill demand.

Table 18. *Properties of possible scenarios between targeted ore supply and mill demand.*

Scenario				Quality	Result	Safety stock required	Financial impact
A	Supply	=	Demand	Optimal	Optimal efficiency	No	Optimal
B	Supply	>	Demand	OK	Safety stock	No	Working capital tied in safety stock
C	Supply	<	Demand	Poor	Stockout	Yes	Mill shutdown, financial loss
D (Practical)	Supply	~	Demand	Fluctuates	Uncertainty	Initial safety stock	Working capital tied in safety stock

Scenario B demonstrates a situation where the supply of ore exceeds the targeted mill demand. This scenario can be useful for mining operations for short periods of time. Having the ore supply exceed mill demand, will naturally create safety stock for future use. However, having the ore supply exceed mill demand constantly, will also increase the amount of safety stock cumulatively. This will eventually result in an undesired situation where the safety stock quantities have grown to such size that they are no longer economical. The main reason for this not being economical is the fact that there is a large amount of money tied up in the working capital and also in the safety stock. This money could be invested in a more economical way, e.g. into a high interest rate account.

Scenario C demonstrates a situation where the supply of ore is less than the targeted mill demand. This scenario represents an undesired setting for ore supply and mill demand in a mining operation. Having a constant ore supply which is less than the mill demand is not only uneconomical but also a result of independent planning. This scenario is nonetheless feasible with certain methods. A solution to making this scenario economical is to create sufficient safety stock during scheduled mill maintenance periods. Unless sufficient quantities of safety stock are created during the scheduled mill maintenance periods, unscheduled mill shutdowns are inevitable due to stockouts.

Scenario D represents a practical scenario of ore supply and mill demand behavior. In this scenario ore supply and mill demand both fluctuate around individual mean values with individual standard deviations. An example of this is presented in *Figure 13*, showing the daily ore supply and mill demand of Kittilä mine for a time of one year.

4.2 A new method to optimize safety stock

A new method to optimize the safety stock of a mining operation was created. This method was created using real mine parameters from Agnico Eagle Kittilä mine. The ore supply and mill demand daily rates are presented in *Appendix 8*, *Figure 38* and *Figure 39*. The foundation of this method is based on creating safety stock naturally during scheduled mill maintenances multiple times during a year. This method is constructed around scenarios C and D which were presented in the previous chapter.

For the new method, average rates from the natural increase in safety stock during mill maintenance (*Figure 28*) and the natural decrease in safety stock during full mill capacity (*Figure 30*) were determined. These rates are indicated together with the coefficient of determination R^2, which indicates how well the trend line (equation y_x) represents the given curve. The closer the coefficient of determination (R^2 value) is to 1, the better it represents the given curve.

The rates for the natural increase in safety stock during mill maintenance were calculated from three time periods where the mill was shut down due to maintenance. These time periods are shown in *Appendix 9*, *Figure 40* and are labeled 1.st – 3.rd mill shutdown. The comparison of these rates gives an indication of how much ore is produced on a daily basis during the mill shutdown period. The duration of the mill shutdown determines the total quantity of safety stock produced naturally within that time period. The average natural increase in safety stock during mill maintenance for Kittilä mine is roughly 3500 tons per day (*Figure 28*).

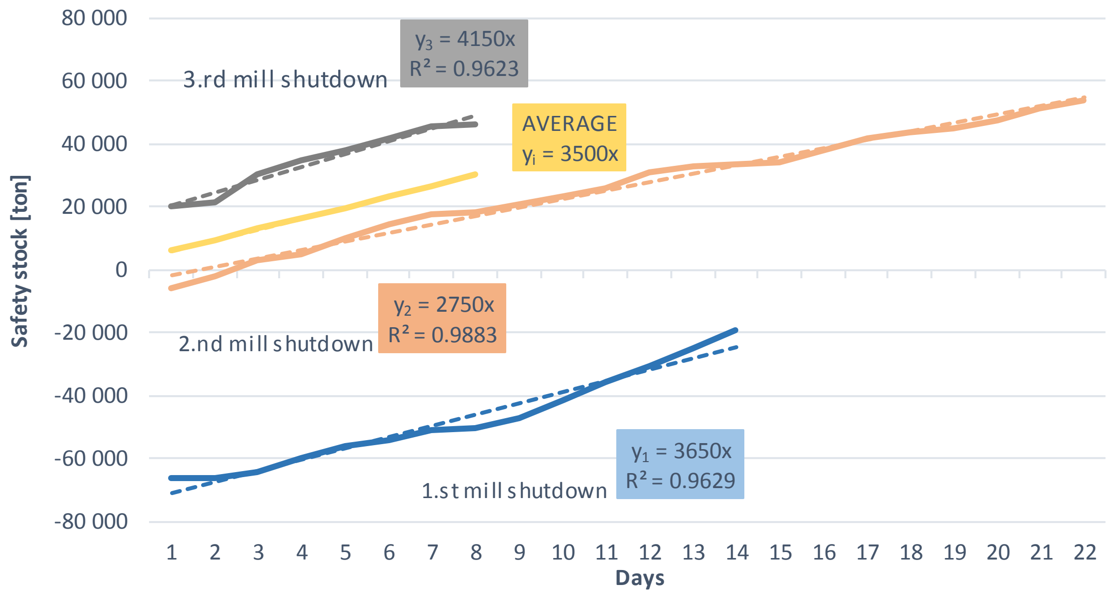

Figure 28. *Natural daily increase in safety stock during mill maintenance at Kittilä mine (Dotted lines represents linear estimations of real data presented by solid lines).*

The rates for the natural decrease in safety stock during full mill capacity were calculated from three time periods where the mill performed with full capacity after recovering from maintenance. These time periods are shown in *Appendix 9*, *Figure 40* and are labeled 1.st – 3.rd full mill capacity. The comparison of these rates gives an indication of how much more ore is consumed than produced during normal operation. The duration of the normal operation determines how much safety stock is needed in order to avoid stockouts.

Figure 11 shows the average difference between ore supply and mill demand on a yearly basis is roughly 400 tons per day, where demand will be greater than supply. However, Kittilä mine made upgrades to the mill and processing plant in the middle of the year, which increased the average demand towards the end of the examined time period. The upgraded mill demands an average 4 700 tons of ore during the end section of the examined time period. The increased mill demand towards the end of the year is presented in *Figure 29*. The upgraded mean demand is roughly 4 700 with a standard deviation of 1 050 tons.

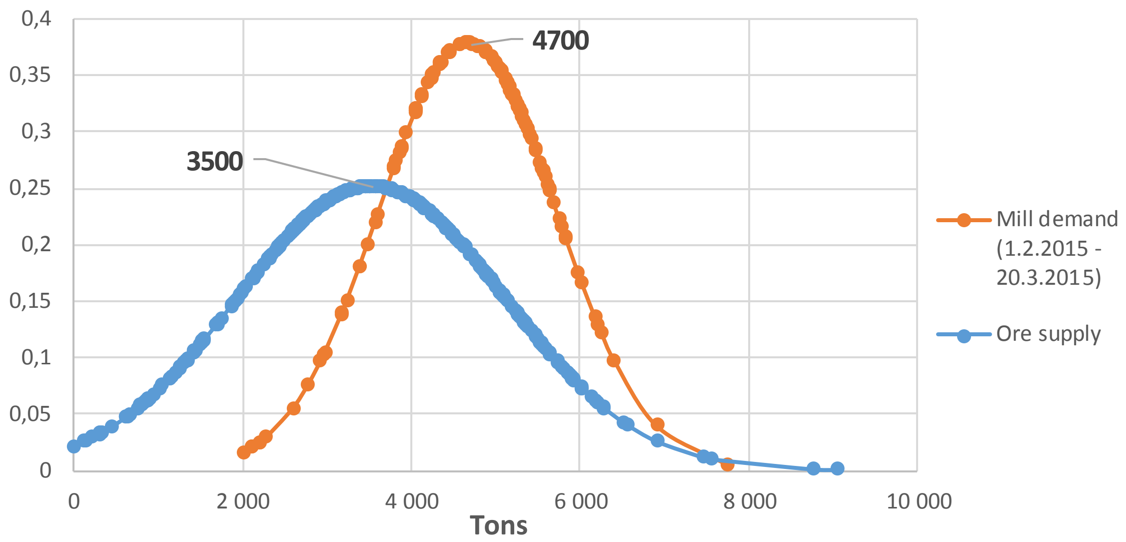

Figure 29. *Increased mill demand towards the end of the year at Kittilä mine.*

The average natural decrease in safety stock during full mill capacity is 1 200 tons per day (*Figure 30*). The natural decrease in safety stock during the first full mill capacity has a coefficient of determination value of 0.83, which is due to a short period of unexpected mill shutdown of three days, shown in *Figure 30*, during days 40-43. This scenario affects the outcome of the average decrease slightly, but is not considered to have significant impact on the results.

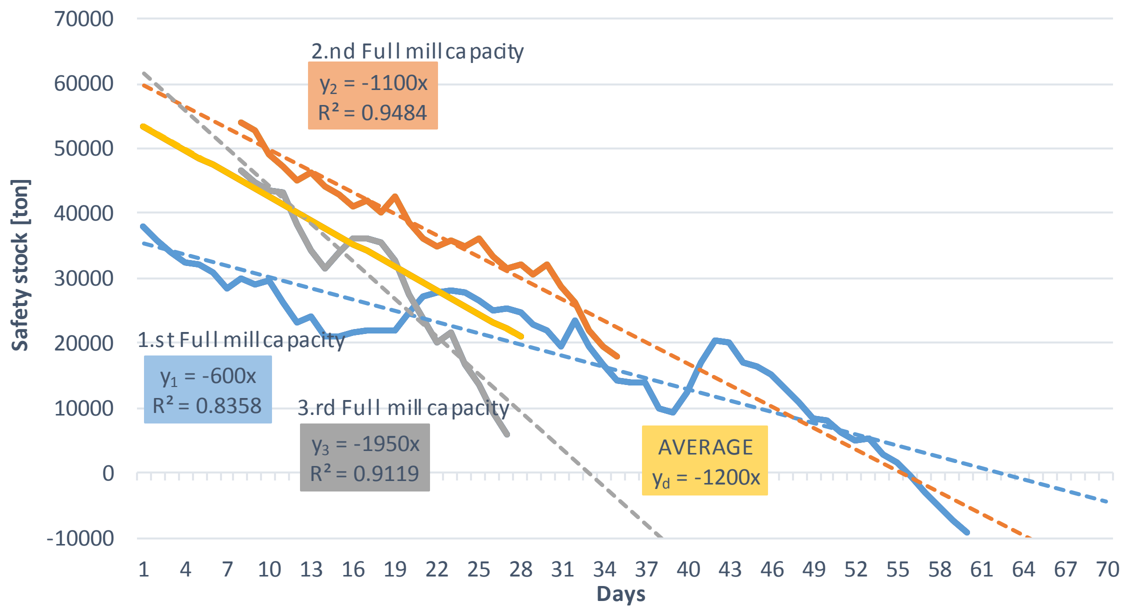

Figure 30. *Natural daily decrease in safety stock during full mill capacity at Kittilä mine (Dotted lines represents linear estimations of real data presented by solid lines).*

To determine safety stock capacities, the presented average equations for natural increase and natural decrease of safety stock were evaluated.

From historical data from Kittilä mine, there are around 50 days of mill maintenance during a year where the mill is shut down either from scheduled maintenance of unscheduled maintenance (*Appendix 9*, *Figure 40*). During these 50 days, an average of 175 000 tons (y_i) of safety stock is produced.

$$y_i = 3\ 500 * x_i \qquad ,where\ x_i = 50\ days$$

This leaves 315 days for normal operation. During the 315 days, the average needed safety stock is 378 000 tons (y_d).

$$y_d = -1\ 200 * x_d \quad ,where\ x_d = 315\ days$$

This leaves an amount of 203 000 tons of ore needed to have in stock additionally to the naturally created safety stock per year.

$$y_i - y_d = -203\ 000\ \frac{tons}{year}$$

There are a few possible scenarios in solving the problem of the missing 203 000 tons of ore on a yearly basis. One scenario is that the supply of ore can either be targeted higher by increasing the mining operations. Another scenario is that the mill demand is targeted lower to meet the targeted supply from the mine. Additionally, a scenario where the scheduled and unscheduled mill maintenance time on a yearly basis is set to match normal operation. An example of this is shown below, by solving the time needed for maintenance in order to create the needed safety stock for normal operation.

$$y_i = y_d$$

$$3\ 500 * x_i = 1\ 200 * x_d \qquad ,where\ x_i + x_d = 365\ days$$

$$3\ 500 * x_i = 1\ 200 * (365 - x_i)$$

$$x_i = 93\ days \quad and\ x_d = 272\ days$$

$$y_i - y_d = 325\ 500 - 326\ 400 = -900\frac{tonnes}{year}$$

The result shows that during the 93 day period of scheduled and unscheduled mill maintenance, the safety stock will increase with 325 500 tons of ore on average. Whereas during the 272 day period of normal operation the safety stock needed to ensure sufficient ore feed to the mill is 326 400 tons. This amount of safety stock covers roughly the needed ore capacity on a yearly basis.

4.3 Combination of new method and real options model

The proposed method was executed with imaginary ore supply and mill feed demand rates which represent the properties of the real Kittilä mine parameters which were analyzed in previous chapters. Since the natural increase and natural decrease rates fluctuate around the mean values on a daily basis, an initial safety stock is needed. This initial safety stock will ensure that even though natural decrease would exceed the mean value due to fluctuation, there would be enough stock to prevent stockouts. The safety stock will fluctuate as the increase and decrease fluctuate and will therefore be at times lower than the initial safety stock and at times higher than the initial safety stock.

Two scenarios (scenario X in *Figure 31* and scenario Y in *Figure 32*) are presented to show how the combination of the new method and the real options model can optimize safety stock quantities during a one year period. In this combined method the initial safety stock required is calculated by subtracting the created amount of ore supply during the scheduled mill shutdowns (325 500 tons) from the yearly sum of safety stock calculated with the real options model.

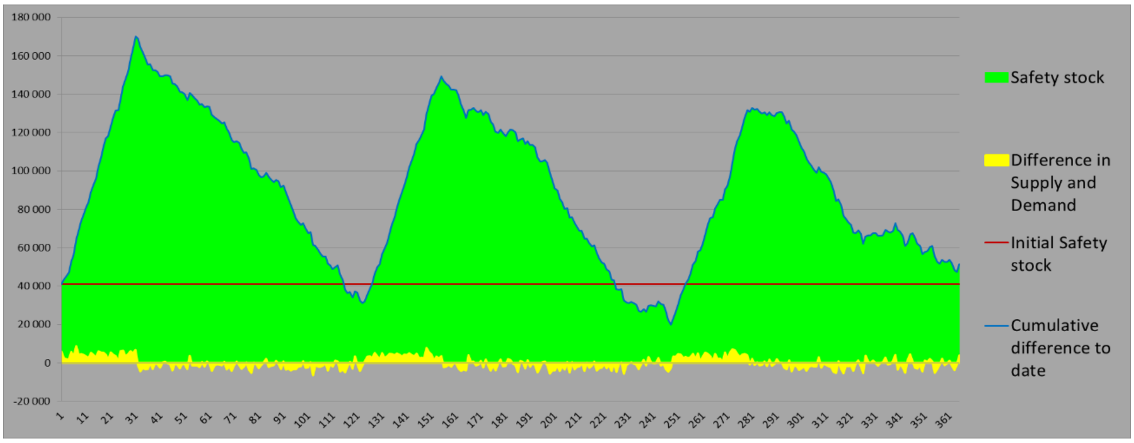

Figure 31*. Scenario X: Safety stock optimization with a combination of new method and real options model (Vertical axis: tons, Horizontal axis: days).*

Figure 31 shows scenario X, where the difference between the yearly real option safety stock and the calculated 325 500 tons created during the mill shutdown periods, resulted in an initial safety stock of roughly 40 000 tons. The initial 40 000 tons of safety stock calculated for scenario X allows for fluctuations to occur in supply and demand, without running into stockouts. In scenario X, there is a slight decreasing trend in safety stock on hand but this will balance out during the years.

Figure 32 shows scenario Y, where the difference between the yearly real option safety stock and the calculated 325 000 tons created during the mill shutdown periods, resulted in an initial safety stock of roughly 15 000 tons. The initial 15 000 tons of safety stock calculated for scenario Y, again allows for fluctuations to occur in supply and demand, without running into stockouts. The difference in scenario Y to scenario X is that there is a slight increasing trend in safety stock.

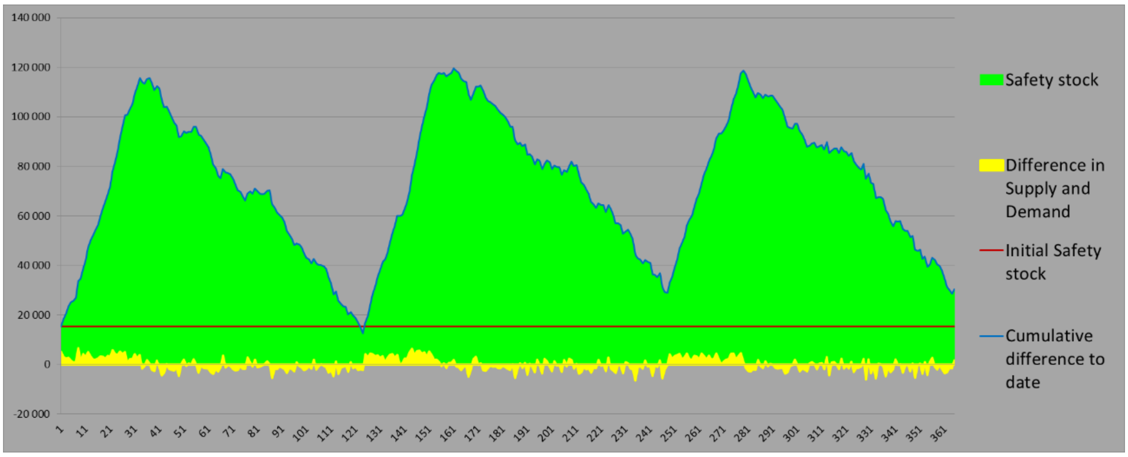

Figure 32. *Scenario Y: Safety stock optimization with a combination of new method and real options model (Vertical axis: tons, Horizontal axis: days).*

Even though the amount of safety stock on hand goes below the initial safety stock amount, at some points during the year, the initial safety stock will suffice to avoid stock-outs. The amounts of initial safety stock required in these two scenarios (scenario X and scenario Y) represent roughly a 12 day ore supply and a 5 day ore supply respectively.

5 Results

5.1 Efficiency of equipment scheduling optimization software

Scenario 1

The results from the equipment scheduling optimization software Scenario 1 and Scenario 2 (p. 50 - 58) are further analyzed. The planned sequencing produced by the equipment scheduling optimization software for Scenario 1 differs significantly from the actual sequencing carried out at the mine. The discrepancies occur with the two available scalers and with the one driller available for production during that time. The actual sequencing of the scalers was carried out by starting production from a workface closest to the surface and then continuing gradually deeper (*Table 10*). This resulted in scaler 2 (SCL02) having to travel upwards to workface 3 from the initial location on workface 7. Thereafter it proceeded to workfaces 5, again 7 and eventually workface 14. However, a logical assumption here would be to start the operation on workface 7 since the scaler is already initially located at that workface. This would presumably increase the efficiency of the sequencing. This actual sequencing can be however justified if the workface hasn't been mucked at the beginning of the shift. Therefor workface 7 couldn't be accessed at the beginning of the shift. The same scenario occurs with the one driller (JUM02). Here the actual sequencing starts with the initial location at workface 4, and proceeding to the deepest workface (workface 9) and gradually moving upwards through workfaces 8 and again workface 4. With the sequencing of the driller however, there is no previous work to be carried out at workface 4, so it is not justified to start the production at workface 9 instead of workface 4, since the machine is already initially located at workface 4.

The travel distances and the moving times for the actual operation in Scenario 1 are relatively small in comparison to the operating times. The planned sequencing of Scenario 1 was produced by the equipment scheduling optimization software and compared with the results of actual operation. ESOS produced the same sequences as were chosen for the actual operation of the shotcreter, two bolters, and the charger. The only difference here was that the bolting operations at workface 15 and workface 11 were averaged out in respect to operating time, since both workfaces had the same workload. As can be seen from comparing *Table 10* and *Table 11* the actual operation at workface 15 lasted for 3 hours and the actual operation at workface 11 lasted for 2 hours even though they had the same amount of workload. With the average operating rate for the two bolters being 1.88 meters per hour, the planned operation produced by ESOS gives a result of 2 hours 30 minutes for both workfaces. The operating time results produced by ESOS are thus averaged out in certain scenarios.

The equipment scheduling optimization software however shows significant benefits as well in Scenario 1. The planned sequencing for the two scalers and one driller in ESOS result in significantly smaller travelling distances and therefore also travelling times between workfaces compared to the actual operation. A further examination of the sequencing shows that the planned sequencing starts the operation from the initial workface of the machine if there's workload at that workface. This decreases the total travelling distance for the machines because of not having to go back and forth. This type of optimization is shown with scaler 2 (SCL02) and the driller (JUM02) in *Table 11*.

Another significant point to mention is that in the actual operation scaler 2 (SCL02) started operation at workface 3 because workface 3 was targeted to be shotcreted in the same shift. Therefore the intuition was to scale workface 3 first in order for the shotcreter to not have to wait for the workface to finish, and therefore avoid waiting time. This is a good assumption and intuitively sounds reasonable. ESOS however shows that this is not the only possible solution and might not be the most efficient in regarding to the whole operation. Since the shotcreting operations are significantly longer, the scaling of all three workfaces (7, 5, and 3) can be performed before the third targeted workface in shotcreting (workface 3) can be started the earliest (*Table 11*).

The planned operation with ESOS has optimized the sequencing of machines and therefore decreased the travelling distance and travelling time. The efficiency of the total operation has increased significantly in relation to what was possible to optimize, the moving of machines. In Scenario 1 the total efficiency was increased by saving almost 4 kilometers in travelling distances. This resulted in saving 21 minutes in total moving time. This might not sound like much but when the time used for moving in actual operation is reduced by 21% in planned operation it will increase the efficiency significantly in the long run. Additionally by planning with ESOS unnecessary waiting times at workfaces can be avoided with certainty contrary to intuitively planning. This is not recorded in the 21% decrease so the efficiency might increase even more.

Scenario 2

The planned sequencing produced by the equipment scheduling optimization software for Scenario 2 differs also significantly from the actual sequencing carried out at the mine.

The discrepancies occur with the available scaler, shotcreter, three bolters, and with the two chargers which were available for production during that time. The only operating sequence that stayed the same in both the actual and planned operations were the two drillers.

The actual sequencing of the scaler was carried out by starting production from workface 5 which is the same level as where the scaler was initially located (*Table 14*). Here after, it continued operation at to workfaces 26, 12 and 11. The reason for this sequence was to ensure that workface 26 was scaled with certainty before shotcreting was to be performed during the same shift. This ensured that there was be no waiting time for the shotcreter in workface 26. After workface 26 was scaled the operation gradually continued towards the surface (workfaces 12, 11).

The shotcreter was sequenced to start operation from workface 3 and continue with workfaces 2 and 1 before moving to workface 26. At this point workface 26 had already been scaled so no waiting was needed at that workface for the shotcreter. The basic logic that mine managers tend to use when sequencing the operations is to gradually move to the closest workfaces, unless workface priorities suggest differently. The same logic has been used for the rest of the actual sequencing of bolters, drillers and chargers shown in *Table 14*.

The scaling operations planned with ESOS were sequenced as follows, workface 12, 11, 5 and 26, with the initial workface being at 5. This sequence reduced the total traveling distance by 2.0 km in comparison to the actual operation. The total time reduced here was roughly 10 minutes as shown in *Figure 26*. However, with this sequence the shotcreting operation at workface 26 was forced to be the last operation for the shotcreter, since it had to be scaled first. The planned shotcreting operation was basically sequenced in the same way as the actual operation, even though the sequence changed from workface 3, 2, 1, 26 (actual operation) to workface 1, 2, 3, 26. This is because all the workfaces 1, 2, and 3 are located on the same level, so the distances between them are set to 0. Therefore the order of the workfaces doesn't affect the outcome. Since the shotcreting operation at workface 26 can be done earliest after the scaling of workface 26 has been completed, the planned operation suggests that the shotcreter must wait at that workface for 34 minutes (*Figure 24*). The software doesn't actually optimize single operations to their shortest possible time spans but instead it optimizes the whole operations critical path to its maximum efficiency. The waiting time of the shotcreter at workface 26 can be justified, since it's the last operation for shotcreting during that shift, and it doesn't exceed the time of the longest operation (critical path = BOL03 in this scenario). Another point to mention is that even though the waiting time for the scaler is 34 minutes at workface 26, the total increase in operating time is only 24 minutes because the scaling operations were reduced by 10 minutes due to optimized sequencing. It can be further analyzed that even though there is an additional 24 minutes of waiting time for the shotcreter, the amount of fuel saved with that 2.0 km decrease in the scaling sequence will be financially beneficial in a long run.

The sequencing of the bolters in this scenario is where it gets tricky. The first bolter (BOL01) in the actual operation was allocated to workface 24 with an initial location at workface 6. Only this workface was allocated for this bolter in actual operation. The total operating time at workface 24 is four hours which differs from all other workfaces planned for the bolters, which have an operating time of 2.5 hours. The reason for such a difference in operating time, even though all the workfaces have the same round length, is due to poor rock conditions in that workface. It could also be due to minor equipment failures, which unfortunately cannot be predicted with accuracy. The second bolter (BOL02) performed operations at workfaces 21, 20, 22, and 23. However, only 20% of bolting at workface 23 was completed before the shift was over. The third bolter (BOL03) operated first at its initial workface level 15, and after which it continued to workfaces 14, and 13.

The planned sequencing with ESOS has averaged out the workload for all three bolters. The difference in the sequences have increased the total distance travelled by 3.8 km (*Figure 25*) and thus the total time travelled by 21 minutes (*Figure 26*) for the planned operation in comparison to the actual operation. However, with this planned sequence the operating efficiency is increased. An indication of the increased efficiency can be best illustrated by comparing the amount of workload which hasn't been performed during the day shift. In the planned operation, workface 24 is the only workface which hasn't been finished completely during the shift. Only 75% of workface 24 has been bolted by the end of the shift. However, in the actual operation only 20% of workface 23 (the only workface not completed within the shift) had been bolted. This already results in a 5% increase in workload efficiency performed by all bolters. The 5% increase in workload efficiency corresponds to one hour of operating time in this scenario.

The two chargers in actual operation were both allocated with two workfaces during the day shift. The first charger (CHA01) was able to operate workfaces 17 and 8 with an operating rate of 2 hours per workface, whereas the second charger (CHA02) was able to operate workfaces 9 and 10 with an operating rate of 2 hours 15 minutes (*Table 14*). In the planned operating sequence by ESOS, the sequence of workfaces for the chargers has changed so that the total operating time for both machines is 4 hours and 15 minutes (*Table 15*). This extends the operating time on CHA01 by 15 minutes but at the same time decreases the operating time of CHA02 by 15 minutes. This way the maximum operating time for the charger (CHA02) is decreased and balanced out with the shorter operation (CHA01). Even though the travel distance increases (0.2 km increase = roughly 1 minute) due to this change in actual operation to planned operation, the benefit of the planned sequencing is much greater than the shorter travel distance of actual sequencing. To conclude, the total operating time (and critical path of the chargers) is optimized and the efficiency increased.

The efficiency of ESOS has proved to be beneficial in both scenarios. The efficiency of the whole operation can be improved with the following optimizations performed with ESOS.

- ✓ Optimizing travel distances of the machines between workfaces.
- ✓ Optimizing waiting time at workfaces.
- ✓ Increasing the total operating time during a shift.

5.1.1 Proposed improvements

The software doesn't account for the mucking operations, so that has to be accounted for manually or with a different software. When running the software, it is recommended to use a machine type sequence of; scaling, shotcreting, bolting, drilling, and charging. This way once a drift has been blasted and mucked, it can be updated into the input parameters and the software can be executed again. This is however slightly frustrating and can lead to different optimizations after updating the parameters within a shift. This would cause confusion and would eventually lead to inefficient planning.

Therefore a proposal for the equipment scheduling optimization software would be to include an option for mucking. Additionally, it would be user friendly to modify the interface in a way that the software could have drop down options for all input files where the user could choose and input single changes to the inputs and rerun them straight after.

The drop down options would consist of all up to date parameters of the mine and could be updated during operations progress. This way the mine layout with all its distances and workfaces etc., would have to be inputted only at the beginning and updated with ease during continuous operation. Another proposal would be to integrate all the needed mine parameters to update automatically into ESOS from the mines own documents. This would however have to be done as customized programming according to the mines documentation strategy.

5.2 Safety stock real options vs. conventional solution

Scenario X and scenario Y in chapter *4.3 Combination of new method and real options model* were also evaluated with the conventional method. The same method was used to evaluate the initial safety stock needed with a combination of the new method and the conventional method. The mean absolute percentage error of both methods and both scenarios are shown in *Table 19* and *Table 20*.

Table 19. *MAPE of integrated method with the new method and the conventional method for Scenario X and Scenario Y.*

μ_s, σ_s	New method with conventional method	
	Mean Absolute Percentage Error	*Initial Safety stock*
	Last 7-day (365 day period)	Yearly basis
Scenario **X**	11.03	154 000 tons
Scenario **Y**	13.25	56 000 tons

Table 20. *MAPE of integrated method with the new method and the real options method for Scenario X and Scenario Y.*

σ	New method with real options method	
	Mean Absolute Percentage Error	*Initial Safety stock*
	Last 7-day (365 day period)	Yearly basis
Scenario **X**	2.86	40 000 tons
Scenario **Y**	3.53	15 000 tons

The new method together with the real options method clearly results in a more realistic and optimized safety stock on a yearly basis. The last 7-day method was used for the comparison, since it clearly gives the most accurate estimate, as shown in chapter *2.2.2*. The results for the needed initial safety stock in *Table 20* are much more accurate than the results in *Table 19*. As mentioned, even though the amount of safety stock is smaller for the "new method with real options method" than the "new method with conventional method" it will clearly suffice without running into stockouts. Therefore, the extra safety stock proposed by the conventional method is unnecessary, and will not increase the guarantee of avoiding stockouts. The real options method will already suffice avoiding stockouts.

This combined method for evaluating safety stock should be however further studied and tested, in order to find out the real functionality and reliability of it. The created method creates nonetheless a solid foundation for a new approach towards optimizing safety stock for mining operations.

5.3 Inventory management decision model in practice

A flowchart of a practical decision model for safety stock management was created based on the studies conducted in this Master's thesis. The created flowchart, shown in *Figure 33* and in *Appendix 7 Figure 37* with greater size, indicates the decisions and steps to achieve the targeted mill demand with various supply quantities.

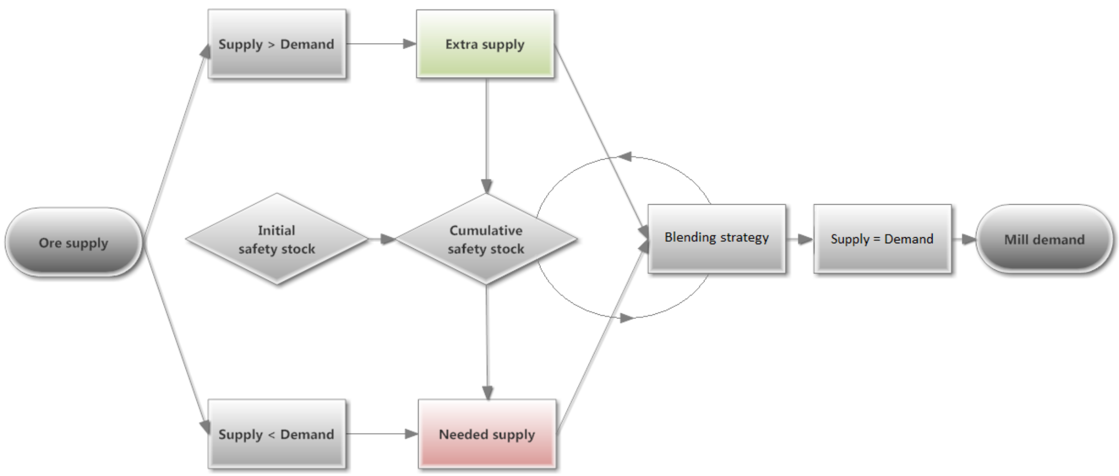

Figure 33. *Flowchart of a practical decision model for safety stock management.*

The flowchart starts with the daily ore supply, which can be either greater or less than the targeted daily mill demand. Daily ore supply can also be equal to the targeted daily mill demand, in which case you can directly jump to the blending strategy on the flow chart. In case the daily ore supply exceeds the targeted daily mill demand, the extra supply is added into the cumulative safety stock and the amount targeted for mill demand is carried into a blending strategy. This will increase the safety stock on hand. However, if the targeted daily ore supply is lower than the daily mill demand, there is a need to restock the amount of needed supply from the safety stock on hand and thereafter continue to the blending strategy.

The initial safety stock is included into the flow chart, because of the fluctuations in ore supply. If the mining operation starts off with a supply less than the demand, there needs to be initial safety stock. In due course the ore supply will also fluctuate higher than the demand which will again balance the amount of safety stock on hand.

5.4 Linking ESOS with safety stock optimization

Linking equipment scheduling optimization software with safety stock optimization could provide a tool for even more efficient planning in the mining industry. The equipment scheduling optimization software decreases the total time on average for a planned operation and additionally improves the probability of achieving the targeted ore supply on a daily basis. This will reduce the fluctuation on the supply side and therefore decrease the needed safety stock level to fulfill the demand.

With the use of the equipment scheduling optimization software the reliability of achieving the targeted ore supply on a daily basis is significantly increased. With the use of the equipment scheduling optimization software there shouldn’t be a need to constantly develop large amounts of safety stock. This will result in not having to store large amounts of money into the unnecessarily large safety stocks. The financial benefit from this seems to be quite significant since the money allocated into working capital will be processed rather quickly with smaller safety stocks and therefor revenue will be created more constantly and with a shorter lead time.

The financial effect of the size of the safety stocks should be further analyzed for individual mining operations. As mentioned, by using ESOS the requirement for safety stock in the long run is slightly decreased. The decrease of safety stock will furthermore be beneficial to mines that produce ore which is vulnerable to weathering, dilution, oxidization etc., because of a rather fast turnover in stock.

6 Conclusion

The results of the equipment scheduling optimization software indicate that the software is capable of improving the efficiency of underground mining operations on a daily shift basis. However, the software needs a tedious amount of initial work before it can be executed and this creates a negative impact. In order to reduce the amount of initial work needed for the input files, an integrated solution with the mine is needed. This would remove the negative impact and sustain the increase in efficiency. At the moment, the efficiency of the software is lost due to the inefficiency of the initial preparations needed for the software to work properly.

The software is still however not waterproof. As an example, the priority of a certain workface before another workface cannot be implemented into the software. Therefore these operations have to be manually appointed and thereafter run the software for a further optimized operating sequence. As a generalization, the software works significantly well in optimizing the operating sequence of mining machines as long as there are no restrictions or priorities in the sequencing on a shift basis.

The equipment scheduling optimization software should be further developed and integrated to match the specific features of each mine. To conclude, the software shows potential in improving the operational efficiency of underground mining operations.

Safety stock optimization in the mining industry has received significantly little attention to date. The methods presented in this Master's thesis are just single solutions in solving and optimizing safety stock for mining operations. Safety stock optimization may need customized solutions at individual mine sites but the presented methods give a solid foundation of what needs to be considered and how to approach the problems. The real options method for determining safety stock, created by Song (2015a), seems very promising and clearly outperforms the conventional method.

With the help of the presented safety stock management methods and models, mining operations can reduce the amount of safety stock without encountering stockouts. This would be more economical for the mines, since the money isn't tied up in the large safety stocks. The relevance of optimizing safety stock in mining operations has received much too little attention from a research point of view and should therefore be studied further.

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List of Appendices

Appendix 1. *Produced Gantt chart by ESOS, sorted by machine ID and by workface number.*

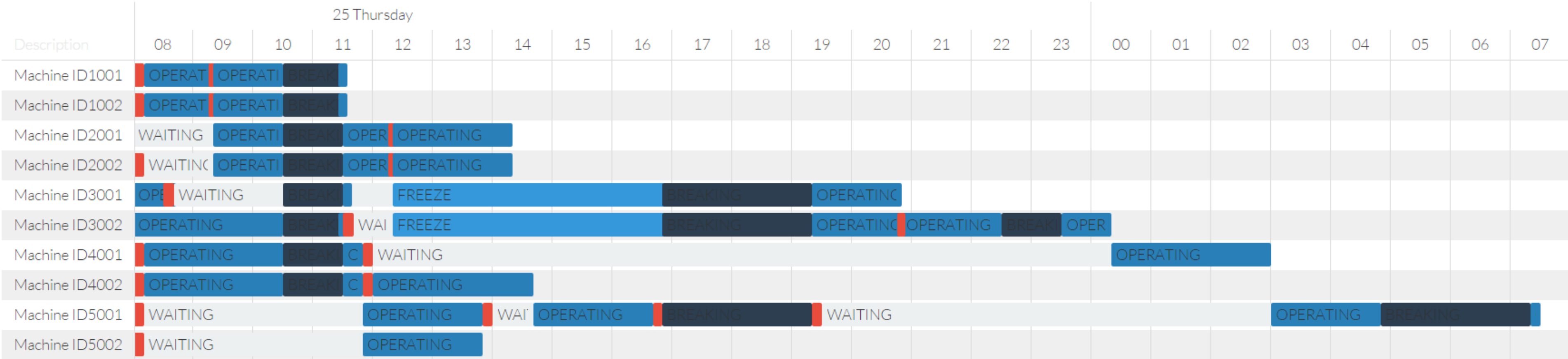

Figure 34. *Gantt chart sorted by machine ID.*

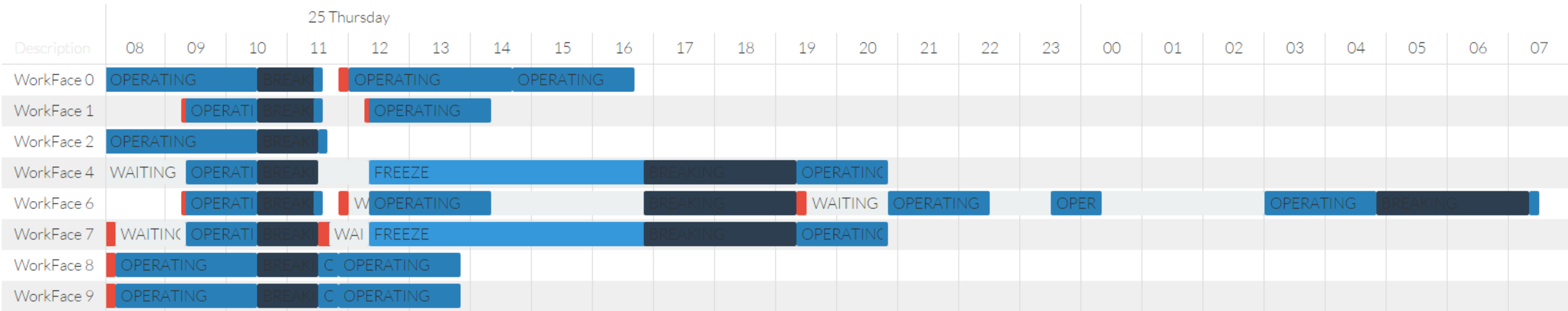

Figure 35. *Gantt chart sorted by workface number.*

Appendix 2*. Input files used for scenarios 1 and 2 for the testing of ESOS at Kittilä mine.*

Table 21*. Equipment operating input for Scenario 1 in ESOS.*

Operating Info	Operating rate [m/h]	Moving speed [m/h]	Operating rate Min. [m/h]	Operating rate Max. [m/h]	Operating rate Mode. [m/h]	Moving speed Min. [m/h]	Moving speed Max. [m/h]	Moving speed Mode [m/h]
machine1(scaler)	4.11	12 000	3.29	4.32	3.80	7 000	17 000	13 200
machine2(shotcreter)	2.35	11 000	1.88	2.47	2.17	5 000	17 000	12 100
machine3(bolter)	1.88	11 000	1.50	1.97	1.74	5 000	17 000	12 100
machine4(driller)	1.76	10 000	1.41	1.85	1.63	5 000	15 000	11 000
machine5(charger)	2.35	10 000	1.88	2.47	2.17	5 000	15 000	11 000

Table 22*. Workface-workface distance input for Scenario 1 in ESOS.*

	WF0	WF1	WF2	WF3	WF4	WF5	WF6	WF7	WF8	WF9	WF10	WF11	WF12	WF13	WF14	WF15
WF0	0.00	0.00	0.53	0.53	0.70	0.70	0.70	0.88	1.40	1.93	2.32	3.29	3.26	3.26	3.58	5.13
WF1	0.00	0.00	0.53	0.53	0.70	0.70	0.70	0.88	1.40	1.93	2.32	3.29	3.26	3.26	3.58	5.13
WF2	0.35	0.35	0.00	0.00	0.35	0.35	0.35	0.53	1.05	1.58	2.36	2.63	2.60	2.60	2.85	4.47
WF3	0.53	0.53	0.00	0.00	0.18	0.18	0.18	0.35	0.88	1.40	2.58	2.50	2.47	2.47	2.72	4.34
WF4	0.70	0.70	0.18	0.18	0.00	0.00	0.00	0.18	0.70	1.23	2.81	2.38	2.35	2.35	2.58	4.22
WF5	0.70	0.70	0.18	0.18	0.00	0.00	0.00	0.18	0.70	1.23	2.81	2.38	2.35	2.35	2.58	4.22
WF6	0.70	0.70	0.18	0.18	0.00	0.00	0.00	0.18	0.70	1.23	2.81	2.38	2.35	2.35	2.58	4.22
WF7	0.88	0.88	0.35	0.35	0.18	0.18	0.18	0.00	0.53	1.05	2.82	2.30	2.27	2.27	2.50	4.14
WF8	1.40	1.40	0.88	0.88	0.70	0.70	0.70	0.53	0.00	0.53	3.20	2.68	2.65	2.65	2.91	3.47
WF9	1.93	1.93	1.40	1.40	1.23	1.23	1.23	1.05	0.53	0.00	3.69	3.17	3.14	3.14	3.45	2.91
WF10	2.32	2.32	2.58	2.58	2.81	2.81	2.81	2.82	3.20	3.69	0.00	1.86	2.39	2.39	2.63	5.66
WF11	3.29	3.29	2.50	2.50	2.38	2.38	2.38	2.30	2.68	3.17	1.86	0.00	1.87	1.87	2.06	5.14
WF12	3.26	3.26	2.47	2.47	2.35	2.35	2.35	2.27	2.65	3.14	2.39	1.87	0.00	0.00	0.50	5.11
WF13	3.26	3.26	2.47	2.47	2.35	2.35	2.35	2.27	2.65	3.14	2.39	1.87	0.00	0.00	0.50	5.11
WF14	3.58	3.58	2.72	2.72	2.58	2.58	2.58	2.50	2.91	3.45	2.63	2.06	0.50	0.50	0.00	2.00
WF15	5.13	5.13	4.34	4.34	4.22	4.22	4.22	4.14	3.47	2.91	5.66	5.14	5.11	5.11	2.00	0.00

Table 23*. Workload input for Scenario 1 in ESOS.*

WF	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Sca.	0.0	0.0	0.0	4.7	0.0	4.7	0.0	4.7	0.0	0.0	4.7	0.0	4.7	4.7	4.7	0.0
Sho.	0.0	0.0	4.7	4.7	0.0	0.0	4.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Bol.	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	4.7	0.0	0.0	0.0	4.7
Dri.	0.0	0.0	0.0	0.0	4.7	0.0	0.0	0.0	4.7	4.7	0.0	0.0	0.0	0.0	0.0	0.0
Cha.	4.7	4.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

***Table 24**. Equipment operating input for Scenario 2 in ESOS.*

Operating Info	Operating rate [m/h]	Moving speed [m/h]	Operating rate Min. [m/h]	Operating rate Max. [m/h]	Operating rate Mode. [m/h]	Moving speed Min. [m/h]	Moving speed Max. [m/h]	Moving speed Mode [m/h]
machine1(scaler)	3.76	12 000	3.01	3.95	3.48	7 000	17 000	13 200
machine2(shotcreter)	3.13	11 000	2.51	3.29	2.90	5 000	17 000	12 100
machine3(bolter)	1.88	11 000	1.50	1.97	1.74	5 000	17 000	12 100
machine4(driller)	1.88	10 000	1.50	1.97	1.74	5 000	15 000	11 000
machine5(charger)	2.35	10 000	1.88	2.47	2.17	5 000	15 000	11 000

***Table 25**. Workface-workface distance input for Scenario 2 in ESOS.*

WF	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
0	0.0	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.4	1.4	1.6	1.8	2.8	3.0	3.0	1.8	1.8	2.2	2.2	2.2	2.6	3.4	3.4	4.1	3.9	5.4
1	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
2	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
3	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
4	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
5	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
6	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.4	0.5	0.7	1.8	1.9	1.9	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	3.1	2.5	4.1
7	1.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.2	0.2	0.4	0.5	1.6	1.8	1.8	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	2.8	2.5	3.8
8	1.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.2	0.2	0.4	0.5	1.6	1.8	1.8	2.7	2.7	2.8	2.8	2.6	2.6	2.3	2.3	2.8	2.5	3.8
9	1.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.0	0.0	0.2	0.4	1.4	1.6	1.6	2.9	2.9	3.0	3.0	2.8	2.8	2.5	2.5	2.7	2.7	3.6
10	1.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.0	0.0	0.2	0.4	1.4	1.6	1.6	2.9	2.9	3.0	3.0	2.8	2.8	2.5	2.5	2.7	2.7	3.6
11	1.6	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.2	0.2	0.0	0.2	1.2	1.4	1.4	3.0	3.0	3.2	3.2	3.0	3.0	2.7	2.7	2.6	2.9	3.5
12	1.8	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.4	0.4	0.2	0.0	1.1	1.2	1.2	3.1	3.1	3.3	3.3	3.1	3.1	2.8	2.8	2.4	3.0	3.2
13	2.8	1.8	1.8	1.8	1.8	1.8	1.8	1.6	1.6	1.4	1.4	1.2	1.1	0.0	0.2	0.2	4.3	4.3	4.4	4.4	4.3	4.3	3.9	3.9	1.7	4.3	2.3
14	3.0	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.6	1.6	1.4	1.2	0.2	0.0	0.0	4.3	4.3	4.4	4.4	4.2	4.2	3.9	3.9	1.4	4.3	1.9
15	3.0	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.6	1.6	1.4	1.2	0.2	0.0	0.0	4.3	4.3	4.4	4.4	4.2	4.2	3.9	3.9	1.4	4.3	1.9
16	1.8	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.9	2.9	3.0	3.1	4.3	4.3	4.3	0.0	0.0	0.9	0.9	1.2	1.2	2.0	2.0	4.1	2.8	5.5
17	1.8	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.9	2.9	3.0	3.1	4.3	4.3	4.3	0.0	0.0	0.9	0.9	1.2	1.2	2.0	2.0	4.1	2.8	5.5
18	2.2	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	3.0	3.0	3.2	3.3	4.4	4.4	4.4	0.9	0.9	0.0	0.0	1.1	1.1	1.9	1.9	4.2	2.6	5.7
19	2.2	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	3.0	3.0	3.2	3.3	4.4	4.4	4.4	0.9	0.9	0.0	0.0	1.1	1.1	1.9	1.9	4.2	2.6	5.7
20	2.2	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.8	2.8	3.0	3.1	4.3	4.2	4.2	1.2	1.2	1.1	1.1	0.0	0.0	1.7	1.7	4.1	2.4	5.5
21	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.8	2.8	3.0	3.1	4.3	4.2	4.2	1.2	1.2	1.1	1.1	0.0	0.0	1.7	1.7	4.1	2.4	5.5
22	3.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.5	2.5	2.7	2.8	3.9	3.9	3.9	2.0	2.0	1.9	1.9	1.7	1.7	0.0	0.0	3.9	2.1	5.1
23	3.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.5	2.5	2.7	2.8	3.9	3.9	3.9	2.0	2.0	1.9	1.9	1.7	1.7	0.0	0.0	3.9	2.1	5.1
24	4.1	3.1	3.1	3.1	3.1	3.1	3.1	2.8	2.8	2.7	2.7	2.6	2.4	1.7	1.4	1.4	4.1	4.1	4.2	4.2	4.1	4.1	3.9	3.9	0.0	4.2	1.0
25	3.9	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	2.9	3.0	4.3	4.3	4.3	2.8	2.8	2.6	2.6	2.4	2.4	2.1	2.1	4.2	0.0	2.0
26	5.4	4.1	4.1	4.1	4.1	4.1	4.1	3.8	3.8	3.6	3.6	3.5	3.2	2.3	1.9	1.9	5.5	5.5	5.7	5.7	5.5	5.5	5.1	5.1	1.0	2.0	0.0

***Table 26**. Workload input for Scenario 2 in ESOS.*

WF	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Sca.	0	0	0	0	0	4.7	0	0	0	0	0	4.7	4.7	0	0	0	0	0	0	0	0	0	0	0	0	0	4.7
Sho.	0	4.7	4.7	4.7	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4.7
Bol.	0	0	0	0	0	0	0	0	0	0	0	0	0	4.7	4.7	4.7	0	0	0	0	4.7	4.7	4.7	4.7	7.5	0	0
Dri.	0	0	0	0	4.7	0	4.7	0	0	0	4.7	0	0	0	0	0	0	0	0	4.7	0	0	0	0	0	4.7	0
Cha.	0	0	0	0	0	0	0	0	4.7	5.3	5.3	0	0	0	0	0	0	4.7	0	0	0	0	0	0	0	0	0

Appendix 3. *Equations for evaluating fixed-order quantity and fixed-time period with safety stock (Jacobs & Chase, 2013).*

Fixed-order quantity:

$$\boldsymbol{R = \bar{d}L + z\sigma_L}$$

$R =$ Reorder point in units

$\bar{d} =$ Average daily demand

$L =$ Lead time in days (time between placing an order and recieving the items)

$z =$ Number of standard deviations for a specified service probability

$\sigma_L =$ Standard deviation of usage during lead time

$$\boldsymbol{SS = z\sigma_L}$$

$SS =$ The amount of safety stock

Fixed-time period:

$$\boldsymbol{q = \bar{d}(T + L) + z\sigma_{T+L} - I}$$

q = Quantity to be ordered

$\bar{d} =$ Forecast average daily demand

$T =$ The number of days between reviews

$z =$ Number of standard deviations for a specified service probability

$\sigma_{T+L} =$ Standard deviation of demand over the review and lead time

$L =$ Lead time in days (time between placing an order and recieving it)

I = Current inventory level (includes items on order)

$$\boldsymbol{SS = z\sigma_{T+L}}$$

$SS =$ The amount of safety stock

Appendix 4. *Layout of Kittilä underground mine with workfaces.*

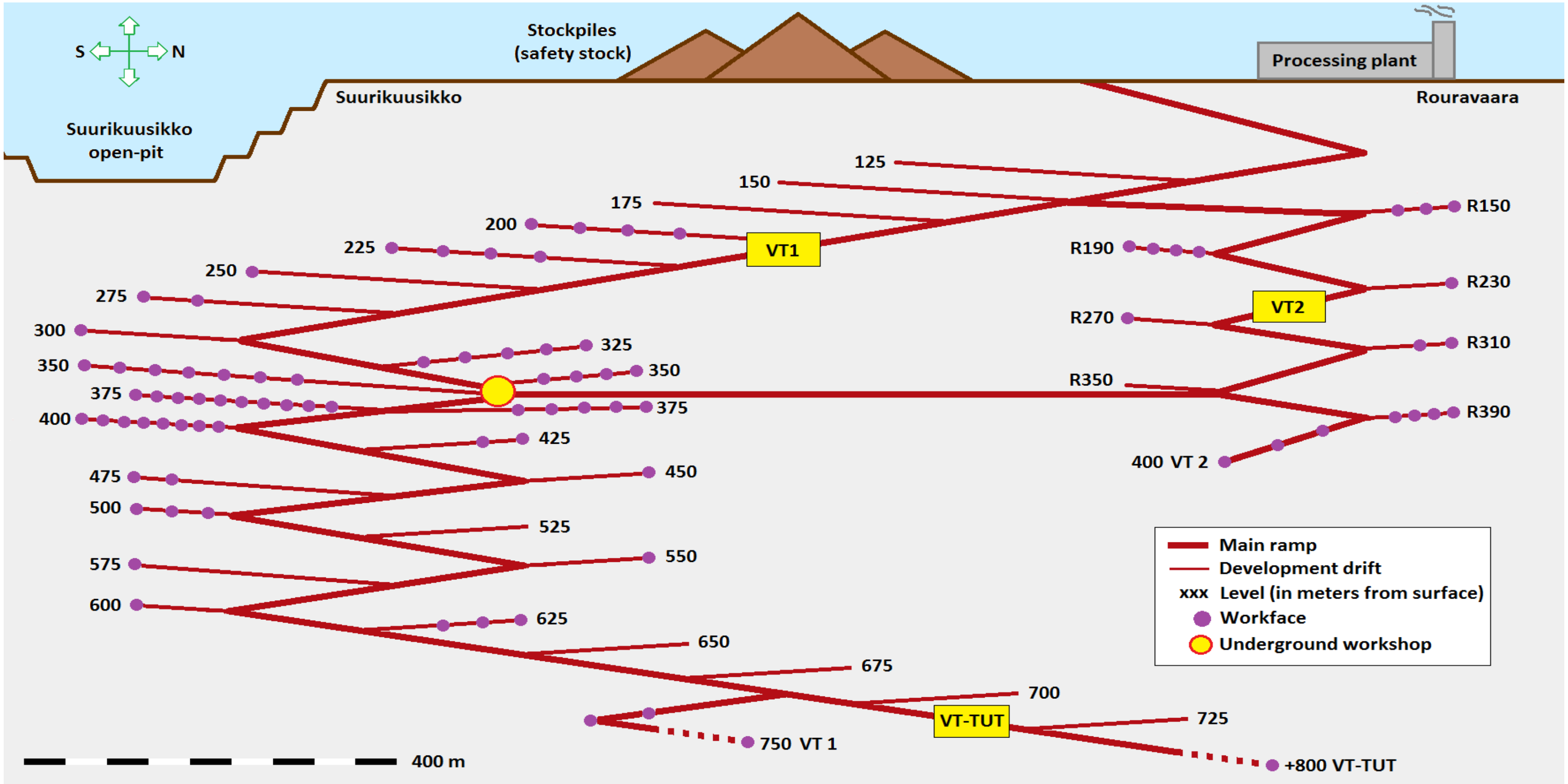

Figure 36. *Specified layout of Kittilä underground mine.*

Appendix 5. *Kittilä mine workface distribution.*

Table 27. *Workface numbering for equipment scheduling optimization software.*

Workface number	Real workface
WF0	200 P128
WF1	200 P165
WF2	200 PP127
WF3	200 TP2
WF4	225 P129
WF5	225 P167
WF6	225 PP118S
WF7	225 TP1
WF8	250 PP140S
WF9	275 P159
WF10	275 TP1
WF11	300 TP1
WF12	325 P137
WF13	325 P160
WF14	325 P163
WF15	325 TP1
WF16	325 TP2
WF17	350 P135
WF18	350 P136
WF19	350 P137
WF20	350 P139
WF21	350 P140
WF22	350 P141
WF23	350 P147
WF24	350 P159
WF25	350 PP146S
WF26	350 TP2
WF27	350 TP3
WF28	375 P129
WF29	375 P131
WF30	375 P132
WF31	375 P134
WF32	375 P139
WF33	375 P156
WF34	375 P158
WF35	375 P161
WF36	375 P163
WF37	375 P164
WF38	375 P165
WF39	375 P167
WF40	375 P169
WF41	375 PP147N
WF42	375 TP2
WF43	400 P144
WF44	400 P163
WF45	400 P165
WF46	400 P166
WF47	400 P167
WF48	400 P168
WF49	400 P169
WF50	400 PP162
WF51	425 P163
WF52	425 P164
WF53	450 P165
WF54	475 P165
WF55	475 RIP
WF56	500 P165
WF57	500 P167
WF58	500 PP168N
WF59	550 YP
WF60	575 YP
WF61	600 TP2
WF62	625 LIP
WF63	625 TP2
WF64	625 YP
WF65	738 MANWAY
WF66	738 RIP
WF67	R150 P211
WF68	R150 P212
WF69	R150 TP1
WF70	R190 P212
WF71	R190 P215
WF72	R190 P217
WF73	R190 TP1
WF74	R230 TP1
WF75	R270 P219
WF76	R310 P221
WF77	R310 P222
WF78	R390 P216
WF79	R390 TP1
WF80	R390 TP2
WF81	R394 LIP
WF82	R400 MUP
WF83	R400 PUP
WF84	VT1
WF85	VT2
WF86	VT-TUT

Appendix 6. *Kittilä mine workface-workface distances.*

Table 28. *Kittilä mine workface-workface distances of all active development drifts.*

Kittilä workfaces		WF0 200P128	WF1 200P165	WF2 200PP127	WF3 200TP2	WF4 225P129	WF5 225P167	WF6 225PP118S	WF7 225TP1	WF8 250PP140S	WF9 275P159	WF10 275TP1	WF11 300TP1	WF12 325P137	WF13 325P160	WF14 325P163	WF15 325TP1	WF16 325TP2	WF17 350P135	WF18 350P136	WF19 350P137	WF20 350P139	WF21 350P140	WF22 350P141	WF23 350P147	WF24 350P159	WF25 350PP146S	WF26 350TP2	WF27 350TP3	WF28 375P129	WF29 375P131	WF30 375P132	WF31 375P134	WF32 375P139	WF33 375P156	WF34 375P158	WF35 375P161	WF36 375P163	WF37 375P164	WF38 375P165	WF39 375P167	WF40 375P169	WF41 375PP147N	WF42 375TP2
WF0	200P128	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.4	0.5	0.5	0.7	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
WF1	200P165	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.4	0.5	0.5	0.7	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
WF2	200PP127	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.4	0.5	0.5	0.7	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
WF3	200TP2	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.4	0.5	0.5	0.7	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
WF4	225P129	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.2	0.4	0.4	0.5	0.7	0.7	0.7	0.7	0.7	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
WF5	225P167	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.2	0.4	0.4	0.5	0.7	0.7	0.7	0.7	0.7	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
WF6	225PP118S	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.2	0.4	0.4	0.5	0.7	0.7	0.7	0.7	0.7	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
WF7	225TP1	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.2	0.4	0.4	0.5	0.7	0.7	0.7	0.7	0.7	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
WF8	250PP140S	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.0	0.2	0.2	0.4	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
WF9	275P159	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.2	0.0	0.0	0.2	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
WF10	275TP1	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.2	0.0	0.0	0.2	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
WF11	300TP1	0.7	0.7	0.7	0.7	0.5	0.5	0.5	0.5	0.4	0.2	0.2	0.0	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
WF12	325P137	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.5	0.4	0.4	0.2	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF13	325P160	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.5	0.4	0.4	0.2	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF14	325P163	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.5	0.4	0.4	0.2	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF15	325TP1	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.5	0.4	0.4	0.2	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF16	325TP2	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.5	0.4	0.4	0.2	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF17	350P135	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF18	350P136	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF19	350P137	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF20	350P139	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF21	350P140	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF22	350P141	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF23	350P147	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF24	350P159	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF25	350PP146S	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF26	350TP2	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF27	350TP3	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.7	0.5	0.5	0.4	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF28	375P129	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF29	375P131	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF30	375P132	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF31	375P134	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF32	375P139	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF33	375P156	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF34	375P158	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF35	375P161	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF36	375P163	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF37	375P164	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF38	375P165	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF39	375P167	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF40	375P169	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF41	375PP147N	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF42	375TP2	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.9	0.7	0.7	0.5	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
WF43	400P144	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF44	400P163	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF45	400P165	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF46	400P166	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF47	400P167	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF48	400P168	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF49	400P169	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF50	400PP162	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.1	0.9	0.9	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
WF51	425P163	1.6	1.6	1.6	1.6	1.4	1.4	1.4	1.4	1.2	1.1	1.1	0.9	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF52	425P164	1.6	1.6	1.6	1.6	1.4	1.4	1.4	1.4	1.2	1.1	1.1	0.9	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
WF53	450P165	1.8	1.8	1.8	1.8	1.6	1.6	1.6	1.6	1.4	1.2	1.2	1.1	0.9	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
WF54	475P165	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.6	1.4	1.4	1.2	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
WF55	475RIP	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.6	1.4	1.4	1.2	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
WF56	500P165	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.8	1.6	1.6	1.4	1.2	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
WF57	500P167	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.8	1.6	1.6	1.4	1.2	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
WF58	500PP168N	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.8	1.6	1.6	1.4	1.2	1.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
WF59	550YP	2.5	2.5	2.5	2.5	2.3	2.3	2.3	2.3	2.1	1.9	1.9	1.8	1.6	1.6	1.6	1.6	1.6	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
WF60	575YP	2.6	2.6	2.6	2.6	2.5	2.5	2.5	2.5	2.3	2.1	2.1	1.9	1.8	1.8	1.8	1.8	1.8	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4
WF61	600TP2	2.8	2.8	2.8	2.8	2.6	2.6	2.6	2.6	2.5	2.3	2.3	2.1	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6
WF62	625LIP	3.0	3.0	3.0	3.0	2.8	2.8	2.8	2.8	2.6	2.5	2.5	2.3	2.1	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8
WF63	625TP2	3.0	3.0	3.0	3.0	2.8	2.8	2.8	2.8	2.6	2.5	2.5	2.3	2.1	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8
WF64	625YP	3.0	3.0	3.0	3.0	2.8	2.8	2.8	2.8	2.6	2.5	2.5	2.3	2.1	2.1	2.1	2.1	2.1	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8
WF65	738MANWAY	3.9	3.9	3.9	3.9	3.7	3.7	3.7	3.7	3.5	3.3	3.3	3.2	3.0	3.0	3.0	3.0	3.0	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
WF66	738RIP	3.9	3.9	3.9	3.9	3.7	3.7	3.7	3.7	3.5	3.3	3.3	3.2	3.0	3.0	3.0	3.0	3.0	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
WF67	R150P211	1.8	1.8	1.8	1.8	1.9	1.9	1.9	1.9	1.7	1.9	1.9	2.2	2.4	2.4	2.4	2.4	2.4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7
WF68	R150P212	1.8	1.8	1.8	1.8	1.9	1.9	1.9	1.9	1.7	1.9	1.9	2.2	2.4	2.4	2.4	2.4	2.4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7
WF69	R150TP1	1.8	1.8	1.8	1.8	1.9	1.9	1.9	1.9	1.7	1.9	1.9	2.2	2.4	2.4	2.4	2.4	2.4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7
WF70	R190P212	2.2	2.2	2.2	2.2	2.3	2.3	2.3	2.3	2.1	2.4	2.4	2.6	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
WF71	R190P215	2.2	2.2	2.2	2.2	2.3	2.3	2.3	2.3	2.1	2.4	2.4	2.6	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
WF72	R190P217	2.2	2.2	2.2	2.2	2.3	2.3	2.3	2.3	2.1	2.4	2.4	2.6	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
WF73	R190TP1	2.2	2.2	2.2	2.2	2.3	2.3	2.3	2.3	2.1	2.4	2.4	2.6	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
WF74	R230TP1	2.6	2.6	2.6	2.6	2.7	2.7	2.7	2.7	2.5	2.7	2.7	2.8	2.7	2.7	2.7	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
WF75	R270P219	2.9	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.7	2.7	2.6	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF76	R310P221	3.4	3.4	3.4	3.4	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.4	2.4	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF77	R310P222	3.4	3.4	3.4	3.4	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.4	2.4	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF78	R390P216	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF79	R390TP1	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF80	R390TP2	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF81	R394LIP	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF82	R400MUP	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF83	R400PUP	3.5	3.5	3.5	3.5	3.3	3.3	3.3	3.3	2.7	2.6	2.6	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WF84	VT1	4.1	4.1	4.1	4.1	3.8	3.8	3.8	3.8	3.4	3.3	3.3	3.3	3.2	3.2	3.2	3.2	3.2	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
WF85	VT2	3.9	3.9	3.9	3.9	3.6	3.6	3.6	3.6	3.0	2.9	2.9	2.7	2.6	2.6	2.6	2.6	2.6	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
WF86	VT-TUT	5.4	5.4	5.4	5.4	5.1	5.1	5.1	5.1	4.6	4.5	4.5	4.3	4.2	4.2	4.2	4.2	4.2	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8

***Table 29**. Kittilä mine workface-workface distances of all active development drifts.*

Kittilä workfaces		WF43 400P144	WF44 400P163	WF45 400P165	WF46 400P166	WF47 400P167	WF48 400P168	WF49 400P169	WF50 400PP162	WF51 425P163	WF52 425P164	WF53 450P165	WF54 475P165	WF55 475RIP	WF56 500P165	WF57 500P167	WF58 500PP168N	WF59 550YP	WF60 575YP	WF61 600TP2	WF62 625LIP	WF63 625TP2	WF64 625YP	WF65 738MANWAY	WF66 738RIP	WF67 R150P211	WF68 R150P212	WF69 R150TP1	WF70 R190P212	WF71 R190P215	WF72 R190P217	WF73 R190TP1	WF74 R230TP1	WF75 R270P219	WF76 R310P221	WF77 R310P222	WF78 R390P216	WF79 R390TP1	WF80 R390TP2	WF81 R394LIP	WF82 R400MUP	WF83 R400PUP	WF84 VT1	WF85 VT2	WF86 VT-TUT
WF0	200P128	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.6	1.6	1.8	1.9	1.9	2.1	2.1	2.1	2.5	2.6	2.8	3.0	3.0	3.0	3.9	3.9	1.8	1.8	1.8	2.2	2.2	2.2	2.2	2.6	2.9	3.4	3.4	3.5	3.5	3.5	3.5	3.5	3.5	4.1	3.9	5.4
WF1	200P165	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.6	1.6	1.8	1.9	1.9	2.1	2.1	2.1	2.5	2.6	2.8	3.0	3.0	3.0	3.9	3.9	1.8	1.8	1.8	2.2	2.2	2.2	2.2	2.6	2.9	3.4	3.4	3.5	3.5	3.5	3.5	3.5	3.5	4.1	3.9	5.4
WF2	200PP127	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.6	1.6	1.8	1.9	1.9	2.1	2.1	2.1	2.5	2.6	2.8	3.0	3.0	3.0	3.9	3.9	1.8	1.8	1.8	2.2	2.2	2.2	2.2	2.6	2.9	3.4	3.4	3.5	3.5	3.5	3.5	3.5	3.5	4.1	3.9	5.4
WF3	200TP2	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.6	1.6	1.8	1.9	1.9	2.1	2.1	2.1	2.5	2.6	2.8	3.0	3.0	3.0	3.9	3.9	1.8	1.8	1.8	2.2	2.2	2.2	2.2	2.6	2.9	3.4	3.4	3.5	3.5	3.5	3.5	3.5	3.5	4.1	3.9	5.4
WF4	225P129	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.4	1.4	1.6	1.8	1.8	1.9	1.9	1.9	2.3	2.5	2.6	2.8	2.8	2.8	3.7	3.7	1.9	1.9	1.9	2.3	2.3	2.3	2.3	2.7	3.0	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.8	3.6	5.1
WF5	225P167	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.4	1.4	1.6	1.8	1.8	1.9	1.9	1.9	2.3	2.5	2.6	2.8	2.8	2.8	3.7	3.7	1.9	1.9	1.9	2.3	2.3	2.3	2.3	2.7	3.0	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.8	3.6	5.1
WF6	225PP118S	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.4	1.4	1.6	1.8	1.8	1.9	1.9	1.9	2.3	2.5	2.6	2.8	2.8	2.8	3.7	3.7	1.9	1.9	1.9	2.3	2.3	2.3	2.3	2.7	3.0	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.8	3.6	5.1
WF7	225TP1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.4	1.4	1.6	1.8	1.8	1.9	1.9	1.9	2.3	2.5	2.6	2.8	2.8	2.8	3.7	3.7	1.9	1.9	1.9	2.3	2.3	2.3	2.3	2.7	3.0	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.8	3.6	5.1
WF8	250PP140S	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.4	1.6	1.6	1.8	1.8	1.8	2.1	2.3	2.5	2.6	2.6	2.6	3.5	3.5	1.7	1.7	1.7	2.1	2.1	2.1	2.1	2.5	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	3.4	3.0	4.6
WF9	275P159	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.2	1.4	1.4	1.6	1.6	1.6	1.9	2.1	2.3	2.5	2.5	2.5	3.3	3.3	1.9	1.9	1.9	2.4	2.4	2.4	2.4	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	3.3	2.9	4.5
WF10	275TP1	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.1	1.2	1.4	1.4	1.6	1.6	1.6	1.9	2.1	2.3	2.5	2.5	2.5	3.3	3.3	1.9	1.9	1.9	2.4	2.4	2.4	2.4	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	3.3	2.9	4.5
WF11	300TP1	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.9	0.9	1.1	1.2	1.2	1.4	1.4	1.4	1.8	1.9	2.1	2.3	2.3	2.3	3.2	3.2	2.2	2.2	2.2	2.6	2.6	2.6	2.6	2.8	2.6	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	3.3	2.7	4.3
WF12	325P137	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.9	1.1	1.1	1.2	1.2	1.2	1.6	1.8	1.9	2.1	2.1	2.1	3.0	3.0	2.4	2.4	2.4	2.8	2.8	2.8	2.8	2.7	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	3.2	2.6	4.2
WF13	325P160	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.9	1.1	1.1	1.2	1.2	1.2	1.6	1.8	1.9	2.1	2.1	2.1	3.0	3.0	2.4	2.4	2.4	2.8	2.8	2.8	2.8	2.7	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	3.2	2.6	4.2
WF14	325P163	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.9	1.1	1.1	1.2	1.2	1.2	1.6	1.8	1.9	2.1	2.1	2.1	3.0	3.0	2.4	2.4	2.4	2.8	2.8	2.8	2.8	2.7	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	3.2	2.6	4.2
WF15	325TP1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.9	1.1	1.1	1.2	1.2	1.2	1.6	1.8	1.9	2.1	2.1	2.1	3.0	3.0	2.4	2.4	2.4	2.8	2.8	2.8	2.8	2.7	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	3.2	2.6	4.2
WF16	325TP2	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.7	0.7	0.9	1.1	1.1	1.2	1.2	1.2	1.6	1.8	1.9	2.1	2.1	2.1	3.0	3.0	2.4	2.4	2.4	2.8	2.8	2.8	2.8	2.7	2.4	2.4	2.4	2.3	2.3	2.3	2.3	2.3	2.3	3.2	2.6	4.2
WF17	350P135	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF18	350P136	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF19	350P137	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF20	350P139	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF21	350P140	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF22	350P141	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF23	350P147	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF24	350P159	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF25	350PP146S	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF26	350TP2	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF27	350TP3	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.7	0.9	0.9	1.1	1.1	1.1	1.4	1.6	1.8	1.9	1.9	1.9	2.8	2.8	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	3.1	2.5	4.1
WF28	375P129	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF29	375P131	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF30	375P132	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF31	375P134	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF32	375P139	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF33	375P156	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF34	375P158	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF35	375P161	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF36	375P163	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF37	375P164	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF38	375P165	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF39	375P167	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF40	375P169	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF41	375PP147N	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF42	375TP2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4	0.4	0.5	0.7	0.7	0.9	0.9	0.9	1.2	1.4	1.6	1.8	1.8	1.8	2.6	2.6	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.8	2.5	3.8
WF43	400P144	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF44	400P163	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF45	400P165	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF46	400P166	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF47	400P167	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF48	400P168	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF49	400P169	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF50	400PP162	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.4	0.5	0.5	0.7	0.7	0.7	1.1	1.2	1.4	1.6	1.6	1.6	2.5	2.5	2.9	2.9	2.9	3.0	3.0	3.0	3.0	2.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	3.6
WF51	425P163	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.2	0.4	0.4	0.5	0.5	0.5	0.9	1.1	1.2	1.4	1.4	1.4	2.3	2.3	3.0	3.0	3.0	3.2	3.2	3.2	3.2	3.0	2.7	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.9	3.5
WF52	425P164	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0	0.2	0.4	0.4	0.5	0.5	0.5	0.9	1.1	1.2	1.4	1.4	1.4	2.3	2.3	3.0	3.0	3.0	3.2	3.2	3.2	3.2	3.0	2.7	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.9	3.5
WF53	450P165	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	0.2	0.0	0.2	0.2	0.4	0.4	0.4	0.7	0.9	1.1	1.2	1.2	1.2	2.1	2.1	3.1	3.1	3.1	3.3	3.3	3.3	3.3	3.1	2.8	2.8	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.4	3.0	3.2
WF54	475P165	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.2	0.0	0.0	0.2	0.2	0.2	0.5	0.7	0.9	1.1	1.1	1.1	1.9	1.9	3.3	3.3	3.3	3.5	3.5	3.5	3.5	3.3	3.0	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.3	3.2	3.0
WF55	475RIP	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.4	0.2	0.0	0.0	0.2	0.2	0.2	0.5	0.7	0.9	1.1	1.1	1.1	1.9	1.9	3.3	3.3	3.3	3.5	3.5	3.5	3.5	3.3	3.0	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.3	3.2	3.0
WF56	500P165	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.4	0.2	0.2	0.0	0.0	0.0	0.4	0.5	0.7	0.9	0.9	0.9	1.8	1.8	3.5	3.5	3.5	3.7	3.7	3.7	3.7	3.5	3.2	3.2	3.2	3.1	3.1	3.1	3.1	3.1	3.1	2.2	3.5	2.9
WF57	500P167	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.4	0.2	0.2	0.0	0.0	0.0	0.4	0.5	0.7	0.9	0.9	0.9	1.8	1.8	3.5	3.5	3.5	3.7	3.7	3.7	3.7	3.5	3.2	3.2	3.2	3.1	3.1	3.1	3.1	3.1	3.1	2.2	3.5	2.9
WF58	500PP168N	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.4	0.2	0.2	0.0	0.0	0.0	0.4	0.5	0.7	0.9	0.9	0.9	1.8	1.8	3.5	3.5	3.5	3.7	3.7	3.7	3.7	3.5	3.2	3.2	3.2	3.1	3.1	3.1	3.1	3.1	3.1	2.2	3.5	2.9
WF59	550YP	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.9	0.9	0.7	0.5	0.5	0.4	0.4	0.4	0.0	0.2	0.4	0.5	0.5	0.5	1.4	1.4	4.0	4.0	4.0	4.2	4.2	4.2	4.2	4.0	3.7	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	2.0	4.0	2.7
WF60	575YP	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.1	1.1	0.9	0.7	0.7	0.5	0.5	0.5	0.2	0.0	0.2	0.4	0.4	0.4	1.2	1.2	4.1	4.1	4.1	4.0	4.0	4.0	4.0	3.8	3.5	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	1.6	3.8	2.1
WF61	600TP2	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.2	1.2	1.1	0.9	0.9	0.7	0.7	0.7	0.4	0.2	0.0	0.2	0.2	0.2	1.1	1.1	4.3	4.3	4.3	4.4	4.4	4.4	4.4	4.3	4.0	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	1.7	4.3	2.3
WF62	625LIP	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.4	1.4	1.2	1.1	1.1	0.9	0.9	0.9	0.5	0.4	0.2	0.0	0.0	0.0	0.9	0.9	4.3	4.3	4.3	4.4	4.4	4.4	4.4	4.2	4.0	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	1.4	4.3	1.9
WF63	625TP2	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.4	1.4	1.2	1.1	1.1	0.9	0.9	0.9	0.5	0.4	0.2	0.0	0.0	0.0	0.9	0.9	4.3	4.3	4.3	4.4	4.4	4.4	4.4	4.2	4.0	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	1.4	4.3	1.9
WF64	625YP	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.4	1.4	1.2	1.1	1.1	0.9	0.9	0.9	0.5	0.4	0.2	0.0	0.0	0.0	0.9	0.9	4.3	4.3	4.3	4.4	4.4	4.4	4.4	4.2	4.0	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	1.4	4.3	1.9
WF65	738MANWAY	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.3	2.3	2.1	1.9	1.9	1.8	1.8	1.8	1.4	1.2	1.1	0.9	0.9	0.9	0.0	0.0	5.2	5.2	5.2	5.4	5.4	5.4	5.4	5.2	4.9	4.9	4.9	4.8	4.8	4.8	4.8	4.8	4.8	0.8	5.3	1.1
WF66	738RIP	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.3	2.3	2.1	1.9	1.9	1.8	1.8	1.8	1.4	1.2	1.1	0.9	0.9	0.9	0.0	0.0	5.2	5.2	5.2	5.4	5.4	5.4	5.4	5.2	4.9	4.9	4.9	4.8	4.8	4.8	4.8	4.8	4.8	0.8	5.3	1.1
WF67	R150P211	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	3.0	3.0	3.1	3.3	3.3	3.5	3.5	3.5	4.0	4.1	4.3	4.3	4.3	4.3	5.2	5.2	0.0	0.0	0.0	0.9	0.9	0.9	0.9	1.2	1.5	2.0	2.0	2.5	2.5	2.5	2.5	2.5	2.5	4.1	2.8	5.5
WF68	R150P212	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	3.0	3.0	3.1	3.3	3.3	3.5	3.5	3.5	4.0	4.1	4.3	4.3	4.3	4.3	5.2	5.2	0.0	0.0	0.0	0.9	0.9	0.9	0.9	1.2	1.5	2.0	2.0	2.5	2.5	2.5	2.5	2.5	2.5	4.1	2.8	5.5
WF69	R150TP1	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	3.0	3.0	3.1	3.3	3.3	3.5	3.5	3.5	4.0	4.1	4.3	4.3	4.3	4.3	5.2	5.2	0.0	0.0	0.0	0.9	0.9	0.9	0.9	1.2	1.5	2.0	2.0	2.5	2.5	2.5	2.5	2.5	2.5	4.1	2.8	5.5
WF70	R190P212	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.2	3.2	3.3	3.5	3.5	3.7	3.7	3.7	4.2	4.0	4.4	4.4	4.4	4.4	5.4	5.4	0.9	0.9	0.9	0.0	0.0	0.0	0.0	1.1	1.4	1.9	1.9	2.4	2.4	2.4	2.4	2.4	2.4	4.2	2.6	5.7
WF71	R190P215	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.2	3.2	3.3	3.5	3.5	3.7	3.7	3.7	4.2	4.0	4.4	4.4	4.4	4.4	5.4	5.4	0.9	0.9	0.9	0.0	0.0	0.0	0.0	1.1	1.4	1.9	1.9	2.4	2.4	2.4	2.4	2.4	2.4	4.2	2.6	5.7
WF72	R190P217	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.2	3.2	3.3	3.5	3.5	3.7	3.7	3.7	4.2	4.0	4.4	4.4	4.4	4.4	5.4	5.4	0.9	0.9	0.9	0.0	0.0	0.0	0.0	1.1	1.4	1.9	1.9	2.4	2.4	2.4	2.4	2.4	2.4	4.2	2.6	5.7
WF73	R190TP1	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.2	3.2	3.3	3.5	3.5	3.7	3.7	3.7	4.2	4.0	4.4	4.4	4.4	4.4	5.4	5.4	0.9	0.9	0.9	0.0	0.0	0.0	0.0	1.1	1.4	1.9	1.9	2.4	2.4	2.4	2.4	2.4	2.4	4.2	2.6	5.7
WF74	R230TP1	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	3.0	3.0	3.1	3.3	3.3	3.5	3.5	3.5	4.0	3.8	4.3	4.2	4.2	4.2	5.2	5.2	1.2	1.2	1.2	1.1	1.1	1.1	1.1	0.0	1.2	1.7	1.7	2.2	2.2	2.2	2.2	2.2	2.2	4.1	2.4	5.5
WF75	R270P219	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	2.8	3.0	3.0	3.2	3.2	3.2	3.7	3.5	4.0	4.0	4.0	4.0	4.9	4.9	1.5	1.5	1.5	1.4	1.4	1.4	1.4	1.2	0.0	1.4	1.4	1.9	1.9	1.9	1.9	1.9	1.9	3.9	2.1	5.2
WF76	R310P221	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	2.8	2.9	2.9	3.2	3.2	3.2	3.6	3.4	3.9	3.9	3.9	3.9	4.9	4.9	2.0	2.0	2.0	1.9	1.9	1.9	1.9	1.7	1.4	0.0	0.0	1.9	1.9	1.9	1.9	1.9	1.9	3.9	2.1	5.1
WF77	R310P222	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.7	2.7	2.8	2.9	2.9	3.2	3.2	3.2	3.6	3.4	3.9	3.9	3.9	3.9	4.9	4.9	2.0	2.0	2.0	1.9	1.9	1.9	1.9	1.7	1.4	0.0	0.0	1.9	1.9	1.9	1.9	1.9	1.9	3.9	2.1	5.1
WF78	R390P216	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF79	R390TP1	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF80	R390TP2	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF81	R394LIP	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF82	R400MUP	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF83	R400PUP	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6	2.6	2.7	2.9	2.9	3.1	3.1	3.1	3.6	3.4	3.9	3.9	3.9	3.9	4.8	4.8	2.5	2.5	2.5	2.4	2.4	2.4	2.4	2.2	1.9	1.9	1.9	0.0	0.0	0.0	0.0	0.0	0.0	3.8	0.5	5.1
WF84	VT1	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.6	2.6	2.4	2.3	2.3	2.2	2.2	2.2	2.0	1.6	1.7	1.4	1.4	1.4	0.8	0.8	4.1	4.1	4.1	4.2	4.2	4.2	4.2	4.1	3.9	3.9	3.9	3.8	3.8	3.8	3.8	3.8	3.8	0.0	4.2	1.0
WF85	VT2	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.9	2.9	3.0	3.2	3.2	3.5	3.5	3.5	4.0	3.8	4.3	4.3	4.3	4.3	5.3	5.3	2.8	2.8	2.8	2.6	2.6	2.6	2.6	2.4	2.1	2.1	2.1	0.5	0.5	0.5	0.5	0.5	0.5	4.2	0.0	2.0
WF86	VT-TUT	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.5	3.5	3.2	3.0	3.0	2.9	2.9	2.9	2.7	2.1	2.3	1.9	1.9	1.9	1.1	1.1	5.5	5.5	5.5	5.7	5.7	5.7	5.7	5.5	5.2	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	1.0	2.0	0.0

***Appendix 7**. A practical decision model for inventory management.*

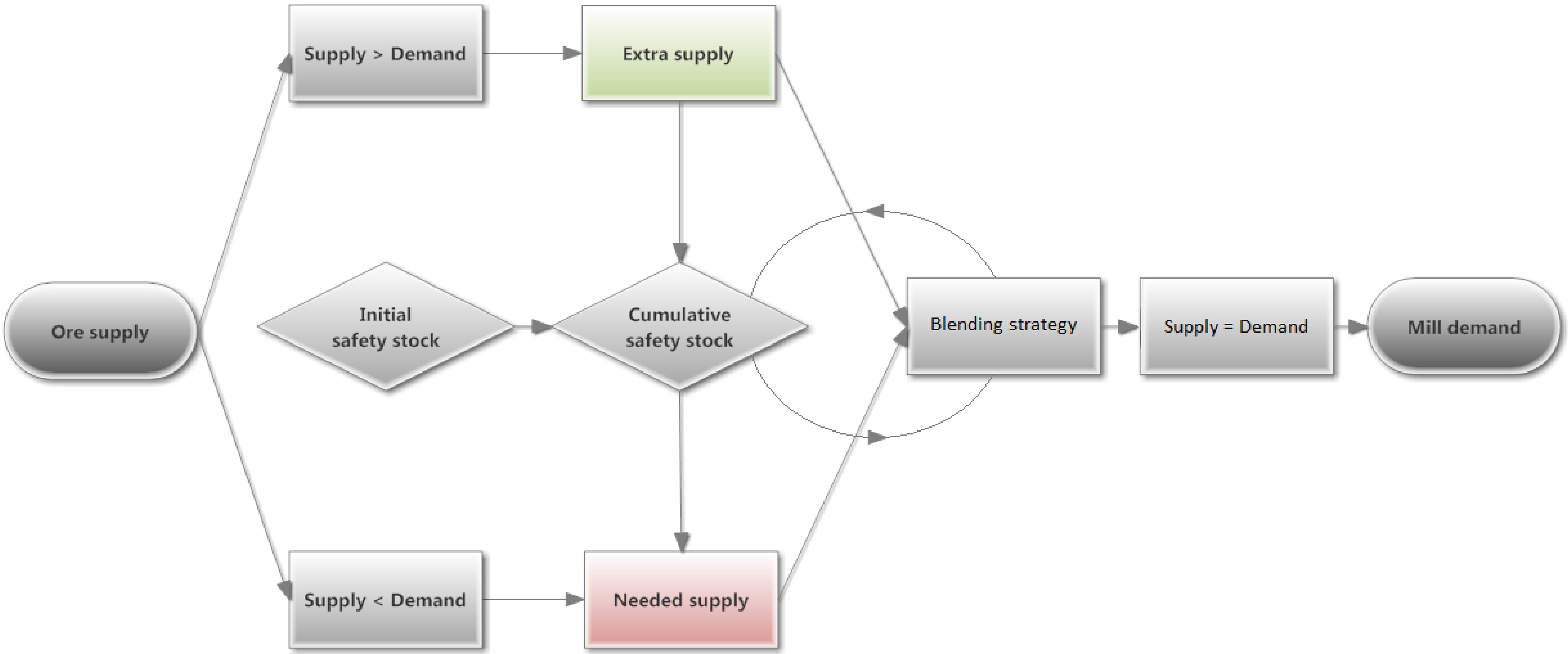

***Figure 37**. Flowchart of a practical decision model for safety stock management.*

Appendix 8. *Ore supply and mill demand rates of Kittilä mine for one year.*

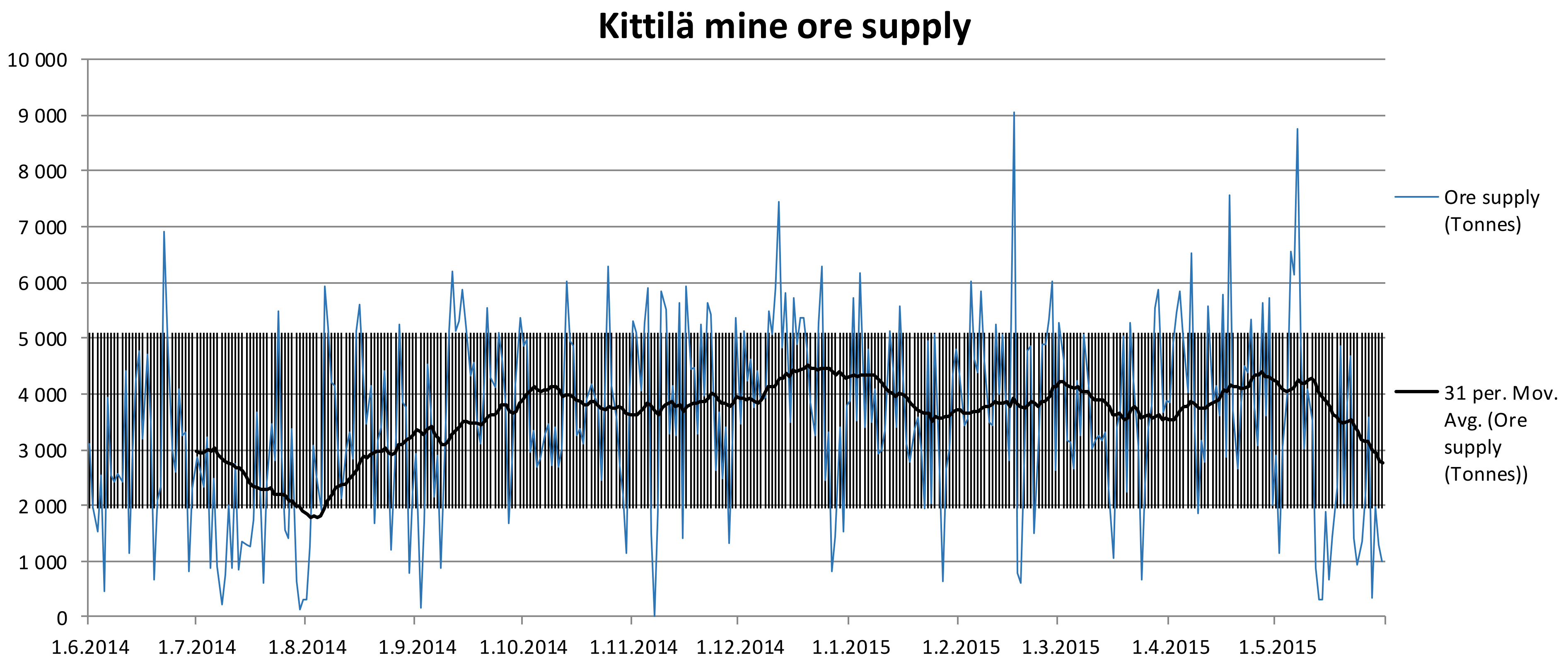

Figure 38. *Kittilä mine ore supply for one year with standard deviation.*

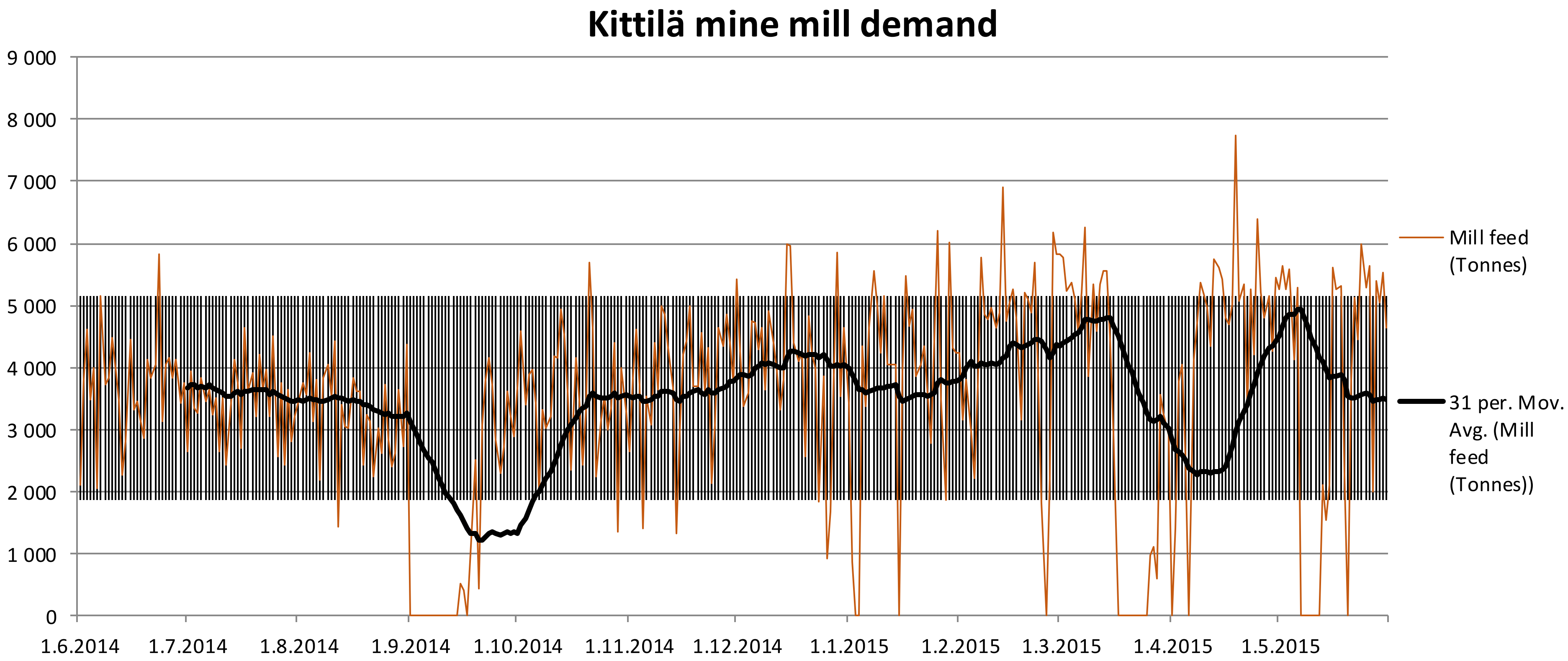

Figure 39. *Kittilä mine mill demand for one year with standard deviation.*

Appendix 9. *Kittilä mine safety stock behavior for one year.*

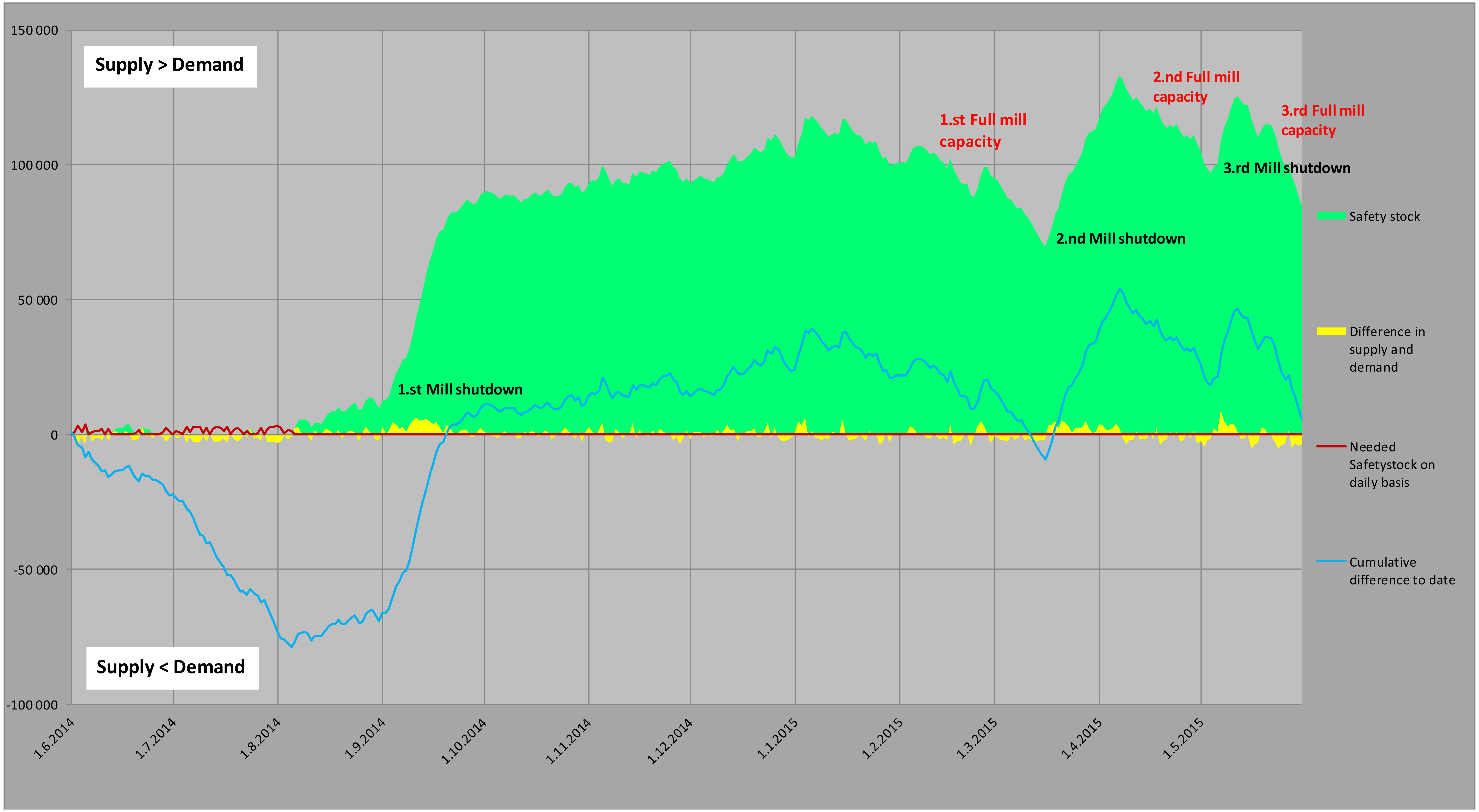

Figure 40. *Properties of Kittilä mine safety stock, including the difference in daily ore supply and demand, needed safety stock on daily basis, and the cumulative difference between supply and demand to date.*

***Appendix 10**. Interview with Agnico Eagle Kittilä mine's geologist Koppström (2015).*

Safety stock analysis at Agnico Eagle Kittilä mine. The following questions were presented to Kittilä mine geologist Kimmo Koppström 27.6.2015.

1. **How is uncertainty in ore supply considered?**

   Uncertainty of ore supply is mainly considered in the LOM plan (life of mine) and other shorter plans (monthly- and weekly-plan). The daily ore supply is set to a target value, which in the case of Kittilä mine isn't optimized at the moment with the demand target of Kittilä mine's processing plant due to the expansion of the processing plant.

2. **How to calculate needed ore safety stock?**
3. **How is ore safety stock optimized?**

   At the moment the safety stock amount isn't optimized in any way. The only restriction is the maximum amount of safety stock due to environmental restrictions in quantity. The environmental restrictions allow an amount of 450 000 tons to be stored on ore safety stock stockpiles. This is the equivalent of around three months of ore supply for the processing plant. The only optimization for the safety stock ore stockpiles is that ore is used constantly for the prevailing blending strategy and also so that the location of the ore stockpile doesn't extend to a significantly far away position from the crusher. So in order to keep the ore stockpiles near the crusher, the stockpiles are gradually removed and replaced.

4. **What is the level of confidence for stockout?**
   - What does it take to achieve 100% service level?
   - Are stockouts tolerated?
   - What is the financial impact of stockout?

   To achieve 100% service level, there needs to be perfect synchronization between processes. Additionally, safety stocks should be implemented between all processes in order to avoid stockouts. Stockouts are not tolerated from the supply side of the operation. However, the processing plant goes through multiple scheduled and unscheduled maintenances which affect the downstream processes in stockouts. During the maintenance periods, the production of ore continues to run which results in gaining safety stock. However, once the processing plant works with full power, the demand for ore exceeds the supply side, and safety stock is required. The financial impact of stockouts is significant and can't really be compensated, and therefore should be avoided.

5. How is safety stock sustained?

Safety stock is built during maintenance breaks. Additionally, multiple small open-pits are under development to ensure a larger supply during crucial moments. By increasing production, the ore supply will meet the targeted value set for mill demand.

6. Is downstream (mill feed / processing plant) demand static?

The mill demand is not static. This is due to failures in the processing chain and also due to the fact that the processing plant cannot process different types of blended material with the same constant speed. However, the demand will be more constant with specific blending strategies.

7. Reasons for multiple ore stockpiles?

Multiple ore stockpile are needed because of large variations within the mined ore grade. The ore at Kittilä mine varies in not only the gold grade, but furthermore in sulphur- and carbon-content. These inevitable and unwanted elements cause problems in the processing chain and therefore have to be blended to meet specific restrictions. Thus multiple ore stockpiles (five stockpiles) are created. This allows optimal blends with specific element proportions to be created.

8. Blending strategies?

The blending strategy is updated on a daily basis. It uses a specific percentage amount of ore from each of the five ore stockpiles and results in 100%. The blending strategy is constructed in a way that the gold content has to be over the targeted cut-off and furthermore the sulphur and carbon content have to be under specific restricted values.

9. How to monitor grade distribution?

Ore grade distribution is monitored with specimens from bore holes and scoop specimens from LHD's (load haul dump machine). The scoop specimens are taken from the LHD's with a 100 ton interval. These scoops are further marked with electronic tags which track the ore for specified information once stacked onto safety stock stockpiles.

Appendix 11. *Executable input files for the equipment scheduling optimization software.*

(*Operating_Info.txt*)

```
machine1(scaler)       4.11    12000   3.29    4.32    3.80    7000    17000   13200
machine2(shotcreter)   2.35    11000   1.88    2.47    2.17    5000    17000   12100
machine3(bolter)       1.88    11000   1.50    1.97    1.74    5000    17000   12100
machine4(driller)      1.76    10000   1.41    1.85    1.63    5000    15000   11000
machine5(charger)      2.35    10000   1.88    2.47    2.17    5000    15000   11000
```

(*Distance.txt*)

```
0.00    0.00    0.35    0.53    0.70    0.70    0.70    0.88    1.40    1.93
0.00    0.00    0.35    0.53    0.70    0.70    0.70    0.88    1.40    1.93
0.35    0.35    0.00    0.00    0.35    0.35    0.35    0.53    1.05    1.58
0.53    0.53    0.00    0.00    0.18    0.18    0.18    0.35    0.88    1.40
0.70    0.70    0.35    0.18    0.00    0.00    0.00    0.18    0.70    1.23
0.70    0.70    0.35    0.18    0.00    0.00    0.00    0.18    0.70    1.23
0.70    0.70    0.35    0.18    0.00    0.00    0.00    0.18    0.70    1.23
0.88    0.88    0.53    0.35    0.18    0.18    0.18    0.00    0.53    1.05
1.40    1.40    1.05    0.88    0.70    0.70    0.70    0.53    0.00    0.53
1.93    1.93    1.58    1.40    1.23    1.23    1.23    1.05    0.53    0.00
```

(*Workload.txt*)

```
0.0     4.7     0.0     0.0     4.7     0.0     4.7     4.7     0.0     0.0
0.0     4.7     0.0     0.0     4.7     0.0     4.7     4.7     0.0     0.0
4.7     0.0     4.7     0.0     4.7     0.0     4.7     4.7     0.0     0.0
4.7     0.0     0.0     0.0     0.0     0.0     4.7     0.0     4.7     4.7
4.7     0.0     0.0     0.0     0.0     0.0     4.7     0.0     4.7     4.7
```

(*Machine_Location.txt*)

```
0           3           1001
0           5           1002
1           1           2001
2           7           3001
2           0           3002
3           4           4001
4           7           5001
```

(*Machine_Number.txt*)

```
0           2
1           1
2           2
3           1
4           1
```

(*Break_Time.txt*)

```
04:45:00
07:15:00
10:45:00
11:45:00
16:45:00
19:15:00
22:45:00
23:45:00
```