



Aalto University
School of Engineering

Pierre Mahieu

Evaluation and Optimization of an Underground Haulage System using Discrete Event Simulation

Master's thesis

Aachen, 6th September 2017
Supervisor: Prof. Mikael Rinne
Instructor: M.Sc. Lars Barnewold

Author Pierre Mahieu (547026)

Title of thesis Evaluation and Optimization of an Underground Haulage System using Discrete Event Simulation

Master programme European Mining, Minerals and Environmental Programme – European Mining Course

Code R3008

Thesis supervisor Prof. Mikael Rinne

Thesis advisor(s) M.Sc. Lars Barnewold

Date 06.09.2017

Number of pages 43

Language English

Abstract

Against the background of a gradual increase of underground mining operations in production depth potentially results in lower productivity due to factors like longer haulage distances, the objective of this thesis was to investigate the productivity of a current truck haulage system and its performance on deeper production levels with the aim to determine its maximum range and to compare the option of relocating the crusher to a deeper mine level. As alternative haulage system, vertical belt conveyor Pocketlift was determined as the system most suitable for the connection between the crusher in its new location and the established conveyor belt connecting the mine with the processing plant. Simulink and its discrete event simulation tool Simevents was applied for the investigation of the production rates of the current and the new system. Results showed that the production rate of the current truck-loader system decreases on each lower production level, so that it cannot be used for future production. A cost evaluation of the alternative system resulted in significantly lower cost per ton compared to the current system

Keywords underground mining, haulage system optimization, discrete event simulation, vertical belt conveyor, Pocketlift

Content

- 1. Introduction 1
 - 1.1 Case Study Mine 2
 - 1.2 Problem Statement..... 3
- 2. State of the Art 4
 - 2.1 Underground Haulage Systems..... 4
 - 2.1.1 Shaft Hoisting..... 5
 - 2.1.2 Vertical Conveyor..... 6
 - 2.1.3 Rail..... 8
 - 2.1.4 Truck..... 9
 - 2.1.5 Belt Conveyor 9
 - 2.2 Crusher 10
 - 2.3 Discrete Event Simulation 11
 - 2.4 Current Haulage System 12
- 3. Haulage System Selection 15
 - 3.1 Equipment Selection..... 15
 - 3.2 Crusher Selection 16
- 4. Method and Data (Simulation approach)..... 18
 - 4.1 Data Collection 18
 - 4.1.1 Truck Cycle Time 18
 - 4.1.2 LHD Cycle Time..... 18
 - 4.2 Input Data and Assumptions 19
 - 4.2.1 Equipment Capacity and Speed..... 19
 - 4.2.2 Ramp Configuration and Travel Distances 20
 - 4.2.3 Loading and Dumping Times 20
 - 4.2.4 Ramp Traffic 21
 - 4.3 Verification and Validation 21
 - 4.4 The Simulation Models 22
 - 4.4.1 Truck Model 25
 - 4.4.2 LHD Model 27
- 5. Simulation and Results 29

5.1	Truck Model Variations.....	29
5.1.1	Results.....	29
5.2	LHD Model Variations.....	30
5.2.1	Results.....	31
6.	Cost Evaluation.....	32
7.	Discussion.....	35
7.1	Practical Implications.....	36
7.2	Strengths and Limitations.....	37
8.	Summary.....	39
	References.....	40

List of Figures, Tables and Equations

<i>Figure 1: Lower Mine Mode</i>	13
<i>Figure 2: Shaft Location In Current Mine Model (550 Level)</i>	25
<i>Figure 3: Truck Model</i>	26
<i>Figure 4: LHD Model</i>	27
<i>Figure 5: Waiting Times Truck Model</i>	30
<i>Figure 6: Operational Cost Truck System</i>	32
<i>Figure 7: Operational Cost LHD System</i>	34
<i>Table 1: Equipment specification as used in the simulation model</i>	20
<i>Table 2: Production Rates Truck System</i>	29
<i>Table 3: Production Rates LHD Model</i>	31
<i>Table 4: Equipment Cost LHD System</i>	33
<i>Equation 1: Euclidean length</i>	21

1. Introduction

The mining industry is developing towards automatization, cleaner and safer working environments, increased productivity and lower labor and energy costs. Additionally, underground mining operations are gradually increasing in production depth, potentially resulting in lower productivity due to increasing challenges of operational factors such as longer haulage distances. Longer haulage distances mean that the haulage equipment will need to have higher capacity to stay efficient, while the emerging geotechnical constraints in deeper mining environment will demand for smaller equipment. Especially mine operations using truck haulage, which have been widely used for material handling in underground hard rock mines (Salama, Greberg & Schunnesson, 2014), will have to deal with this problem. Therefore, the investigation of the topic of optimizing haulage systems to make them more efficient and less harmful to the environment, is highly important.

As the production moves downdip for dipping deposits over time, operation haulage systems will need to be improved somehow. This can be realized either by optimizing the current system or by introducing a new one. The transition from a current haulage system to a different one is a difficult task and needs profound investigation. In most cases, it is not possible to have actual testing of new haulage systems in a running mining operation as costs and time effort are extremely high and most likely affect the current production. In order to get a relatively fast and realistic determination of the performance of a new or changed haulage system, simulation can be applied to approach such a task without the need of high capital and time investments. In the past, the interest and application of simulation for investigating and designing new mining processes or modifying and optimizing established systems has rapidly grown. (Panagiotou, 1999) Processes like the material extraction and transportation are of dynamic nature. While analytical methods often fail to represent these dynamic and random aspects, simulation modeling is a highly precise tool to predict the behavior of future mining processes. (Lu, Lau & Chan; 2007). For such complex systems, discrete event simulation (DES) is an especially suitable tool as it can perform complex system modelling in detail. (Salama, Nehring & Greberg, 2014)

The objective of this thesis is to provide an insight into the application of discrete event simulation in mining and especially for investigating and optimizing underground haulage systems. In this work, DES will be applied to a case study dealing with the

relocation of an underground crusher to a deeper mine level. Simulink and its DES tool SimEvents will be used to investigate the productivity of the current truck haulage system and its performance on deeper production levels. The aim is to determine its maximum range and to compare the option of relocating the crusher or not. For this comparison, underground haulage systems will be investigated to select a suitable connecting system between the crusher in its new location and the established conveyor belt connecting the mine with the processing plant. A cost evaluation of the alternative haulage system and the current one will assess the results on the question of relocating the crusher. The conduction of this research will yield results on the capabilities and flexibility of the discrete event simulation tool SimEvents and whether it is suitable to be used for such haulage system optimization.

1.1 Case Study Mine

The case study presented in this thesis was carried out in a hard rock underground mine in Austria. The mine is currently producing around 500,000 t/a operating at depths of up to 525 m, but the orebody has been explored until a depth of 675m and the deposit geology infers that the mineralization extends even deeper. The levels in the mine are referred to with their elevation above sea level starting at 1175 m.a.s.l (1175 level) where the main access opening (adit) was driven into the mountain. A second access opening is a 2.7 km long incline hosting a belt conveyor which connects the mine level 815 with the processing plant. The production levels are connected by a spiral ramp. As the orebody extends above the 1175 level, the ramp was driven upwards to extract the upper orebody parts. Nowadays production is only taking place beneath the 1175 level down to the 650 level. A large fault zone divides the mineralized zone. Whilst up to seven elongated WNW-plunging ore lenses are developed in the upper deposit part, some of them merge downdip to a single large orebody. Down to the 850 level, the ore lenses are still developed, but below that, they merge into 3 orebodies that are generally divided in orebody K7, orebody A and orebody K2/K2'. This division is mainly due to the orebody limitations set by the cut-off grade. Between the mine access level and the deepest production level there is only one single lane ramp connecting all production levels. Currently, the deepest point of the ramp is on level 625. Recently, a second ramp has been developed between the levels 815 and 736 to allow access to the K7 orebody.

The mine is operating an underground crusher on level 850 which is fed by a surge bin reaching up to the 910 level with a capacity of roughly 5000 t. The ore is crushed in two steps (primary jaw crusher and secondary cone crusher) to a discharge size of 60 mm and then transported to the processing plant by the belt conveyor.

1.2 Problem Statement

As the mine is supposed to produce ore with a constant grade of 0.27% WO_3 , the run of mine (ROM) delivery on the surge bin must be mixed from different orebodies. Therefore, the production needs constant sampling and monitoring of the produced ore grade as well as a flexible mining system. The mine is operating with two eight hours shifts. The production from the levels below 910 occur only during one shift. A weekly production capacity of 12,000 t needs to be supplied to the crusher for constant operation. Therefore, the minimum effective hourly production that must be delivered by the truck-loader system is 150 t/h.

Currently, the amount of ore produced from the levels above the surge bin and down to the level of 850 (crusher level) is still high and represents around 50 % of the yearly production. Therefore, the truck haulage only needs to cope with half of the production requirement. Due to the flexibility of the truck system and the sufficient availability of material near the crusher plant, skilled dispatching of the production equipment satisfies the production requirements. However, production is moving downwards and will focus on the lower levels only in the near future.

The decreasing efficiency and production rate of the truck system raises the question if relocating the crusher to a deeper point in the mine is a feasible and beneficial option. The relocation of the crusher would require the acquisition of a connecting haulage system between the new crusher location and the existing belt conveyor which is connected to the processing plant. This study will compare the selected options regarding their productivity and costs.

2. State of the Art

Literature has been reviewed to obtain an overview of different haulage systems and equipment for underground hard rock mines. Furthermore, discrete event simulation was studied to gain knowledge of this method. The information was used to identify suitable haulage systems for implementation in the case study and as a fundament for the investigation of the simulation modeling. For the research on transportation systems, only transportation of ore was relevant; the transportation of supply material and personnel were excluded. The review was conducted focusing on the application possibilities and performances of the different systems and equipment while technical specifications about the different equipment components were not considered. These do not contribute to the intended use of the information. The selection criteria were operational features, capital expenditure and operational expenditure. Furthermore, two crusher types mainly used for primary crushing are presented comprising rules for the selection of the suitable crusher type.

2.1 Underground Haulage Systems

The haulage system is one of the most important components in an underground mining operation and its efficiency can affect the profitability of the whole operation. (Salama, Greberg & Gustafson, 2014). The underground haulage process starts with the transportation of the mined material from the production faces to the loading area, where it is loaded on a different haulage equipment or dumped through orepasses to a loading bay or crusher. From this location, it is transported to the surface or to its intended destination. Depending on the lump sizes of the ROM material and the transportation equipment, the material is either directly transported to the surface or first handled with a crusher. Different equipment is available to handle the various possibilities of material transportation and commonly require additional equipment to complete the full haulage cycle, e.g. load-haul-dump (LHD) equipment. The available haulage system categories include truck haulage, shaft systems, conveyor belts and rail. (Bloss, Harvey, Grant & Routley, 2011) Material loaded at the draw point (production face) can be transported in vertical shafts (skip hoisting, vertical conveyors), in inclined shafts (steep incline conveyors, vertical conveyors), declines and ramps (rail, truck, conveyor belts) or horizontally in drifts (truck, conveyor belt, rail). (Salama, Greberg & Gustafson; 2014).

The selection of the haulage system needs careful investigation and planning. It is mainly driven by following factors: ore reserve, production capacity, underground mining method, ground conditions, orebody characteristics, planned mine life, price of the machinery, amortization and discount rate. In most cases, especially the production rate and mine depth are decisive factors. (Gonen, Malli & Kose, 2012) In general, the distance that the ore must be transported is the crucial factor influencing the selection of the haulage system, still with respect to other relevant factors like rock stability, capital budget and mine access purpose. From that point of view, vertical and inclined shafts are the best option, as vertical or near vertical transportation distances are the shortest. Therefore, the past decades produced significant innovations in vertical and high angle transportation equipment. However, from all haulage systems and equipment available, mostly truck haulage, shaft systems and conveyor belts are feasible considerations for haulage over large vertical distances. (Greberg, Salama, Gustafson & Skawina, 2015) It is worthwhile to consider all haulage systems in the decision-making process. The following chapter intends to summarize most of the transportation system and equipment and to present their application range as well as advantages and disadvantages.

2.1.1 Shaft Hoisting

A shaft is usually sunk from the surface where the minable ore lies too deep underground to be extracted economically from an open pit or with decline haulage. Shaft hoisting is used for operations with deep, steeply dipping ore bodies and high productivity. Shafts are also sunk to connect flat lying orebodies in great depth to the surface. According to De La Vergne (2003), shaft hoisting is suitable for orebodies with a vertical depth of >500 m and a production capacity of >2500 t/d. Where the overburden is shallow, but the rock conditions are bad, a vertical shaft may be a suitable solution, too. (Vergne de la, 2003) Pratt (2005) gathered operating ranges for different haulage systems from underground showing that shaft hoisting has a wide application range. According to the information provided by him, shaft hoisting dominates before truck and conveyor haulage systems for operations from depths of >600 m and capacities starting at 500,000 t/a. Ore transportation in a shaft is typically done with skips. Skip hoisting systems are usually automated and vary in the discharge method, number and size of the skips and type of hoist. (Tiley, 2011)

Shaft sinking is a highly cost and time intensive process and requires good planning of the cross section and especially of the shaft location. Usually, only main shafts are equipped with skip hoisting systems, as they are predominantly established for direct haulage of the ore to the surface. The dimensions of the shaft must be capable to bear the necessary ventilation capacity, the supply installations (pipes, air ducts etc.) and hoisting equipment. The possible throughput capacity for air and material must be designed sufficiently for the production to reach out several kilometers. (Vergne de la, 2003)

Even though shaft hoisting systems have very high investment costs, the operating costs are low and the difference of capital expenditure to alternative haulage systems decreases as depth increases. (Bloss et al., 2011) Skips are only used to hoist the ore vertically from the underground to the surface so that there is always the need of another transportation system in the underground to reach for the production faces. This can be an LHD tramming the ore to orepasses, trucks, locomotives or conveyor belts. Changing from ramp to shaft haulage is a critical task which is depending on many factors, mostly the same as during the planning phase. Production rate, rock stresses, ore reserves, ventilation and investment budget still dominate the decision. For example, the optimum changeover depth from decline/ramp haulage to shaft hoisting lies shallower if the production rate is high. Also, technological innovations for trucks increased the overall changeover depth. (Gonen et al., 2012)

Some research has been conducted on the changeover depth, yielding different results in a range of 300 m to 500 m. (McCarthy, 1999) As every mining operation has different operational constraints, it is difficult to predict a universal rule. Most investigations compared the option of shafts with skip hoisting and decline trucking. Less information is available on the comparison of shafts with vertical conveyors and decline trucking. However, recent developments in this technology push these equipment into the focus.

2.1.2 Vertical Conveyor

The continuous haulage of material through a vertical or inclined shaft can be realized with vertical or high angle conveyors. This equipment category includes bucket elevators, Flexowell conveyor belts, Pocketlifts, “sandwich” belt conveyors and pipe conveyors. As it is derivable from the names, the material is conveyed in buckets, between cleats and sidewalls or pockets which are installed on belts. The sandwich high angle conveyor “sandwiches” the ore by hugging the material between two

smooth surfaced rubber belts over the carrying path (Dos Santos, 2007) and a pipe conveyor rolls up the belt and enclose the material. The newest technology is the Flexowell technology based Pocketlift. It is a combination of the Flexowell conveyor and bucket elevator belt. While the Flexowell conveyors were restricted to a lift height of 250 m (Gonen et al., 2012) the Pocketlift allows ore transportation over vertical distances of 700 m with any inclination between 0° and 90° and capacities of up to 5,000 t/h. (Zuchowski, 2013) However, practical realizations are beneath these limits. For example, the current performance of 1,818 t/h (2000 m³/h) of ROM coal over 273 m vertical height is realized with the Pocketlift in the Patikki Mine 2 (Alspaugh, 2004) and a Flexowell conveyor belt lifts copper ore at a height of 140 m at the Asikoy Kure underground copper mine (Gonen et al., 2012) The Pocketlift booklet by ContiTech mentions another installation in Novomskovsk, Russia, where a Pocketlift lifts gypsum up 138 m with a capacity of 900 t/h (643 m³/h). (ContiTech AG, 2014) The Pocketlift consists of two narrow steel cord belts which are connected with rigid triangular cross bars. The material is loaded into fabric reinforced rubber pockets. These pockets are bolted at the center of the cross bars. All elements are connected detachable from each other. Thus, the pockets can be replaced without changing the belts or any other parts of the conveyor. This reduces maintenance efforts. As no mechanical parts are in the shaft, this also reduces complexity, time and thus costs of maintenance.

Vertical conveyor belts can be applied in any operation using vertical or inclined shafts as haulage opening. They are restricted by the maximum tensile strength of the belts (up to 10,000 N/mm), maximum capacity and maximum lump sizes (max. 150 mm to 400 mm) of the material. (Neumann, 2017) Compared to skip hoisting, the possible vertical distances that can be overcome are shorter, but concerning lift heights of up to 700 m, vertical conveyors may be a better choice. The main advantages of using vertical conveyors are the reduced shaft diameter – not more than 2,5m (Neumann, 2017) –, high availability, high energy efficiency (Bloss et al., 2011), low maintenance and long service life. (ContiTech AG, 2013) Especially the small shaft diameter and the low energy consumption benefit the consideration of vertical conveyors for shaft operations. While skips must accelerate the material to high speeds after being loaded, vertical conveyors realize a continuous mass flow and thus, reduce power peaks resulting in less driving energy and a reduction of investment for the energy supply. (Töpfer, 2014)

Gonen, Malli and Kose (2012) compared the unit transportation costs of shaft hoisting, ramp haulage and vertical belt conveyor for different exemplary production rates and depths. As a result, changeover depth from decline trucking to vertical conveyor is at 160 m at a production rate of 360,000 t/a. In comparison, when changing from decline trucking to shaft hoisting, the changeover depth lies deeper at a higher yearly production rate (425 m, 400,000 t/a). Even though this investigation only considered the Flexowell technology up to 250 m lift height, it is derivable that vertical conveyors have the lowest unit transportation costs at production rates above 400,000 t/a in any technically feasible depth. This may in particular be due to the smaller required shaft diameter, the continuous material flow and the low energy cost of not more than 0,3 kWh per ton and 100 m lift (Roland, 2014).

2.1.3 Rail

In the past, rail haulage systems used to be the common transportation method until they got replaced by truck and conveyor belt systems due to their investment and flexibility advantages. (Dammers, Brudek, Pütz & Merchiers, 2016) Rails usually work where long distances must be overcome, high and constant production is maintained and where deep shafts are used to haul the material to the surface (Bloss et al., 2011). That is why most high capacity hard rock operations are still applying this technology with capacities reaching up to 160,000 t/d (Dammers et al., 2016).

Rail systems are proven to be energy efficient with high capacity and quick travel speeds and are either floor or roof mounted. The first can transport high tonnage per trip but is sensitive to inclination leaving their applicability to operations with low drift gradients and high production rates. The roof mounted equipment is more suitable to cope with inclination and has reduced initial costs and building effort. (Salama, Greberg & Gustafson, 2014) The main drawbacks for floor mounted rails are the cost and time intensive installation of the system and its restricted flexibility. In terms of a monorail system (roof mounted), the main drawback is the relatively low maximum payload (30 t) and its advantages are the lower initial costs and smaller dimensions. (Salama, Greberg & Gustafson, 2014). As electrical systems are preferred, rail haulage, besides having high capacity, low operational costs and high reliability, is also benefiting the ventilation. (Dammers et al., 2016).

2.1.4 Truck

The truck haulage system is widely used in underground mines. Transportation of ore by trucks on declines seems to be the most suitable and economic method for small and medium sized mines. (Gonen et al., 2012) According to De La Vergne (2003), ramp haulage with trucks is well suitable for orebodies with shallow overburden, vertical haulage depth up to 300 m and production capacity <2500 t/d. When considering the mining reserves, declining may be the most suitable method for medium sized deposits of around 4,000,000 t. In Pratt's (2005) it is stated that truck haulage dominates operations ranging from <200 m to 400 m depth and annual capacities of <1,000,000 t.

Depending on the lump sizes of the mined material, trucks are either used to transport the ore from a loading bay near the production front to an underground crusher, to the shaft station or directly to the surface. As the truck is driving through the development openings (declines and drifts on the production levels), the dimensions (width, height, curve/turning radius) of both the truck and the openings must be selected mutually. Support, ventilation ducts, service pipes and clearance to the sidewalls also need to be considered. (Bloss et al., 2011) Wrong dimensioning will slow down the truck and decrease the productivity while the demanding handling of the vehicle and the restricted view in the decline will decrease safety as well.

Diesel trucks are highly flexible in routing and capacity (fleet size) and require relatively low initial investment. Furthermore, they do not have electrical hazards and provide high productivity (Paraszczyk, Svedlund, Fytas & Laflamme, 2014). On the other hand, the disadvantages are the high operating costs due to low diesel efficiency (around 40%) (Vergne de la, 2003) and high maintenance as well as high noise, heat and toxic gas emissions. Another disadvantage is the ramp traffic. With increasing depth, the required number of haulage trucks will increase and therefore more frequent interferences of each other on the ramp may result in congestion and productivity losses (Wilson, Willis & du Plessis, 2004). Depending on the travel distance, this may be solved by good operator experience, optimized waiting bay selection, the application of dispatching systems or a second decline.

2.1.5 Belt Conveyor

Traditional belt conveyor systems are commonly applied for mass mining operations with long-life duration. (Bloss et al., 2011) Looking at De La Vergne's (2003) flowchart for preliminary haulage system selection, inclined belt conveyors are suitable for

operations not deeper than 500m and with productions >5,000t/d. Furthermore, Pratt (2005) showed that conveyer systems mostly perform in operations ranging from <200m up to 400m depth with capacities of >500,000t/a.

Like rail systems, belt conveyors have a high capital expenditure, a large footprint and low flexibility. (Bloss et al., 2011) Furthermore, conventional belts usually operate most economically at angles of 12° to 18° (Dos Santos, 2007), which excludes an application for high vertical lifts and leaves this equipment a good choice only for high production rates and long (horizontal or little inclined) distances. Additionally, as curves are not possible with belt conveyors, the investment cost will rise if complex system layouts must be established with transition stations. Compared to truck and rail haulage and similar to haulage systems accommodated in shafts, another disadvantage is that system breakdowns will usually cause the operation to stop if there is no backup system. However, the implementation of a second conveyor system is unlikely due to the high investment costs and footprint. (Merchiers, Brudek & Dammers, 2016) The advantages of belt conveyor systems are the high automation, low labor requirements, low operating costs and high availability. (Bloss et al., 2011)

When large vertical distances must be overcome, inclined conveyor belts (<18°) require a large area to be installed into. The belts can be installed in several declines which are either matched straightly together or which are positioned like steps in a vertical direction. The first requires a large distance from the belt feeding point to its discharge point and thus a larger footprint, but the second option requires more transition stations and therefore higher investment. Pratt (2005) summarized some operations which use inclined conveyors for lifts from 185 m up to 750 m over a length of 1,500 m to 4,000 m. However, the high lift operations are run with capacities of 5,600,000 t/a and 6,000,000 t/a and thus shows that the application of conveyor belts for such high lifts is restricted to high production capacities as well as long life.

2.2 Crusher

The investment in a crusher is a significant one. Therefore, it is essential to select the equipment so that it can fulfil the needs of the production system. Several types of crushers are available from which two are usually used for primary crushing, namely jaw and gyratory crushers (Vergne de la, 2003). As primary crushers are the first equipment the ore will be fed to, especially operations using bulk mining methods must consider the top feed size for dimensioning of the crusher. Scalping equipment like

grizzlies can be used to control the feed to the crusher, but the additional work to break the oversized lumps could reduce the productivity and enhance personnel costs. The required production rate for the operation is the second important parameter to be considered in the selection process. Generally, once the suitable crusher type has been chosen, the equipment selection is a simple comparison of the capital costs. Both the jaw and the gyratory crusher perform similar if applied to hard rock. The choice will be primarily made based on the required production rate, the capital investment and the possibility to transport and install the equipment in the underground.

Usually, jaw crushers are used for lower capacities whereas gyratory crushers are used for large capacities. The capacity range for which jaw crushers are the preferred choice range from less than 800 to 1200t/h and gyratory crushers for capacities >1200t/h. (Metso Minerals Processing Handbook, 2015) However, it is indicated that gyratory crushers will be more economical for high capacity operations of >1000 t/h or >8000 t/d. (Mosher, 2011; Vergne de la, 2003)

In comparison, the main advantages of jaw crushers are a lower profile compared to gyratory crushers, the large feed openings in relation to their capacity, smaller components that are easier to transport underground and the lower capital costs. The advantages of gyratory crushers are higher capacities at lower energy consumption per ton and the release of products which are more uniform and benefits following transportation via belt conveyors. (Vergne de la, 2003) Jaw crushers will mostly be applied when space requirements, capital expenditure and throughput are low whereas gyratory crushers are preferred when high throughput is required.

2.3 Discrete Event Simulation

Simulations intend to reproduce system behavior as realistically as possible. It is generally accepted that simulations cannot fully recreate the real state of any processes, but can mostly yield knowledge about the behavior of the system variables and therefore allow theoretical experiments within the systems parameters in order to suggest possible system modifications (Salama & Greberg, 2012). The simulation of mining systems is an effective approach to investigate system problems and bottlenecks or to decide about the suitability of a designed process or system for a running mining operation. Since minimization of haulage costs is one of the main tasks that are tackled in the process of mine optimization (Basu & Baafi, 1999), simulation is often applied to such problems. Its ability to conduct “what if” scenarios for real or

designed systems makes simulation an indispensable tool for both identification and solution of mining problems. (Banks, 1999)

DES is based on discrete, dynamic and stochastic simulation models. While stochastic models involve the use of random numbers to simulate an actual system, dynamic models represent a system as it evolves over a period of time. Discrete models are characterized by the instantaneous change of state variables at separated points in time (Law & Kelton, 1991). With regard to these characteristics, DES can be described as a simulation that is time based, includes logical details, randomness and is usually applied to gain knowledge about the interactions and behavior of objects within a system (e. g. queueing problems). DES is used for the modelling of systems in which the simulated entities are subject to changes at discrete points in time within the simulated duration (activities). (Brunner, 2001) These changes are induced by defined operations (e. g. loading, dumping) that are built in the model and discrete changes only occur in fulfilment of such operations. It is possible to feed random variables to the model which are based on appropriate probability distributions and thusly, can better capture the dynamic and random nature of haulage systems compared to analytical methods. (Greberg et al., 2015)

DES was first applied in 1960 and took some time to get attention in the mining sector. (Sturgul, 2001) However, since then DES has been applied in some mining operation research. Simulation can be applied to different disciplines in mining and predominantly focus on transportation systems, mine planning and production scheduling. (Vagenas, 1999)

2.4 Current Haulage System

The ore is extracted with 17-ton LHD equipment from the production face (drawing point) and hauled to a loading bay located on the production level. Due to the mining methods in use, remote controlled loading is required in the stopes. In the loading bay, the material is dumped on trucks which drive to the 910 level and dump it into the surge bin. Within the truck-loader system, extraction can only take place in one stope or production face at a time. In the upper production levels above the 830 level, LHDs extract the ore from the drawing points (production faces) and haul it to gravity shafts or dump it directly on the surge bin. The gravity shafts are either directly connected to the surge bin or end in a loading bay closer to the surge bin. From there, another LHD

takes and dumps the material to another gravity shaft connected to the surge bin or directly in the surge bin.

As the scope of this study excludes the levels above the current crusher location, only the three orebodies below the 910m level are relevant. They can be seen in Figure 1. Three different mining methods are applied for the extraction of the different orebodies. Those are overhead cut and fill in the K7, sublevel stoping in the K2/K2' and sublevel caving in the massive orebody A. The red ramp sections are currently under development. It will take up to six years to develop the ramp down to the 525 level.

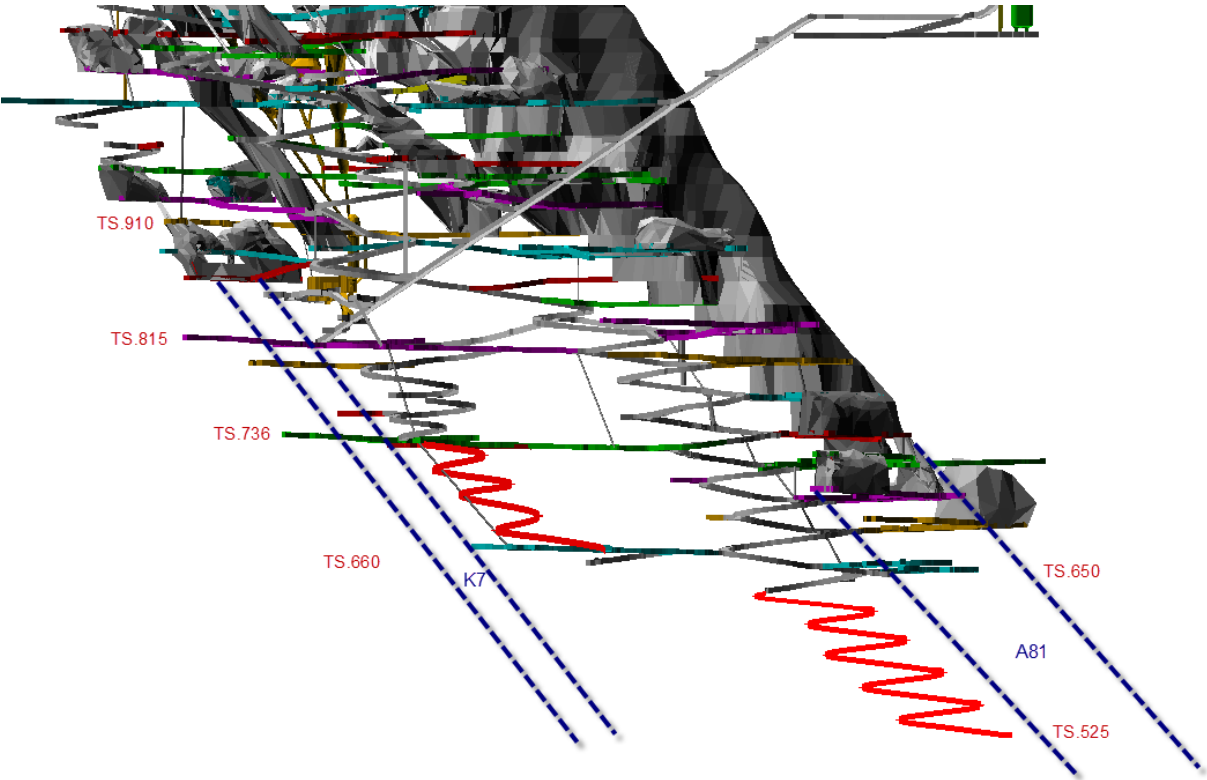


Figure 1: Lower Mine Model

The trucking is realized with three trucks, comprising of two Volvo A30G type with a capacity of 30t each and one Sandvik TH551 with 51t capacity. At shift start, the trucks drive to the designated production level and line on the ramp waiting in waiting bays as close together as possible. The Sandvik truck is always the first to be loaded at shift start. The loading of the Sandvik truck takes three scoops, and the Volvos are only loaded with two. When one truck is loaded, it travels up the ramp to the surge bin to dump the material. As soon as the loaded truck drives past the first waiting truck, the next one drives down to the loading bay and is loaded. As the ramp dimensions allow

only one equipment driving on it, the trucks must drive to waiting bays in order to pass each other. Since no routing system (e.g. block systems) is used the drivers cannot estimate at which point they will encounter an upcoming truck. Therefore, they predefine the waiting bays based on experience and wait until the next truck passes by their position. However, this is an iterative process for new production levels and can cause extended waiting times.

3. Haulage System Selection

Based on the selection “rules”, typical application range, limits and performances found in literature, this chapter intends to determine the most suitable system for implementation in the case study mine. The basic parameters that are considered for the selection are the production depth, the required production rate, the capital costs, the operational costs and the suitability of the equipment to function with the setup of an underground primary crusher.

3.1 Equipment Selection

Based on the information described in the previous chapter, a preliminary investigation of the available haulage systems and equipment was conducted. Regarding the mine production rate and the operating depth, following systems are of interest for this case study: vertical conveyors and haulage trucks. Trucks provide the highest operational flexibility. Since flexibility is a crucial operating requirement for this operation, the option to optimize the current truck system in order to provide the required production rate should be considered. This could be realized by improving the traffic situation or/and by adding additional trucks. The vertical conveyor equipment Pocketlift is chosen as the best connecting haulage system if the crusher is relocated to a deeper mine level. This is due to its unique performance of low energy consumption, small shaft diameter and continuous flow of material over intensive lift heights. It can be applied either vertically or for inclined transportation.

Rail and traditional belt conveyors cannot be considered for this case study because of the narrow area in which the ore must be transported. Such equipment is predominantly suitable for large capacities and long horizontal haulage distances, but not for overcoming high vertical distances in a confined space. Also, the development cost and effort for inclined drifts is significantly higher compared to the establishment of raises. With these constraints, the haulage systems using rail or conventional belt conveyors can be excluded from the equipment pool.

The application of shaft systems is favorable due to the medium steep dip of the orebodies of around 55°. With a company owned raise boring equipment, shafts with diameters of up to 2.1 m can be established at low cost and within short time. The mine has already created several orepasses and ventilation raises with this method within different parts of the deposit and rock so that the stability of the shaft and orepasses is

guaranteed in almost every possible location. These conditions highly favor the selection of a shaft accommodated haulage systems. As the maximum diameter of the raises is limited to 2 m (after the installation of support), the installation of a skip system is difficult due to its required dimensions and may not be able to hold the required production rate. Also, skips cannot continuously transport material and therefore require further surge bins after the crusher and before the secondary cone crusher, each with a feeding equipment. Consequently, vertical conveyors are preferred to skips.

In general, the direct vertical connection is the best choice, as haulage distance is the shortest. Also, the mine's geology only allows vertical shafts with respect the required length of the shafts. Two vertical shafts have been operating for many years without failure. Therefore, the best choice to connect the new primary crusher with the existing cone crusher is a vertical shaft. Hence, the vertical conveyor Pocketlift is considered to be the most suitable equipment for this operation.

3.2 Crusher Selection

The current size reduction is realized in two steps. Since the production from the levels above 910 is still high and will last for the upcoming years, the current crusher cannot be relocated, hence, a new one needs to be installed. In order to reduce the extraction work for establishing the crusher room, it was considered to find a crusher that can perform the required size reduction in one step. No such equipment could be determined. However, it was decided to only establish a primary crusher on the lower level and to connect it to the existing facility. From there, the sized material flows back into the secondary sizing cycle. As the production from the lower and the upper levels is run in separate shifts, the secondary crusher will not have to cope with additional capacity. Hereby, excavation costs as well as capital expenditure for an additional secondary crusher system can be saved. Hence, only a primary crusher need to be selected.

As the mine uses caving and stoping methods, the ROM ore comprises of large lump sizes. This is predominantly due to the over break from the orebodies. With a feed opening of 1800 x 1400 [mm], the current crusher is capable to cope with the large lumps. During loading, the LHDs remove large rocks from the drawing point and dump them to a remote bay. But often, the production in a specific stope must stop due to rocks that are too large for the LHD to handle. Secondary blasting is then applied to

size the rocks. However, large ore lumps are still loaded onto the trucks and dumped on the crusher. Hence, a crusher with a smaller feed opening would only be operable with the use of a grizzly. Since the number of large lumps is high, scalping the feed material would result in high auxiliary work and productivity reduction. A second jaw crusher – the same equipment as currently installed in the underground – is available at the mine. Even though some reconditioning, a new drive unit and new bearings are required, the selection of this crusher seems to be the best choice. The equipment has proven to be suitable and efficient for the operation and even offers the possibility to increase the throughput capacity. Gyratory crushers seem to be inadequate for this operation. The throughput rate is way under the recommended range of >800 t/h and thus, a gyratory crusher would work inefficiently and most likely result in higher costs per ton. However, such gyratory crushers have higher reduction ratios and could therefore be used to directly and properly size the material for the processing plant (60 mm) thus potentially justifying the higher investment costs. Nevertheless, research on the minimum production size of gyratory crushers from several known manufacturers (Sandvik, Metso, ThyssenKrupp, FLSmidth) yield that the limit is 125 mm.

Since a suitable stationary jaw crusher is available for lower capital costs and most likely with the best performance for this operation, this equipment is chosen to be considered for the installation underground.

4. Method and Data (Simulation approach)

To investigate a process and simulate its current situation, real system data needed to be collected. Cycle times for both trucks and LHDs were measured in the underground mine in a full production environment. This was intended to yield the current state of the haulage system in order to have references for the simulation model and to determine its productivity and weak points. Two models were built using Simulink and its DES tool SimEvents. After verification and validation of the models, simulation experiments were conducted to determine the effective production rate of the haulage systems in scope. A cost evaluation was conducted based on the results of the simulation.

4.1 Data Collection

The cycle times were measured for the loading and haulage equipment and were conducted on three different production levels (750, 700, 650) within the A and K2' orebodies. In total, 61 cycle times were recorded for the trucks from which 38 were gained by the Volvos and 23 by the Sandvik truck. 57 loading cycles were measured for the LHDs. Due to the flexible dispatching of the production between the production levels, truck measurements could not be done continuously, but only over short time periods resulting in data sets of 2 to 10 cycles.

4.1.1 Truck Cycle Time

Truck cycle times were measured while sitting in the truck and by taking notes during the route. The survey was carried out separately for each equipment. The survey included the truck maneuvering at the loading bay, duration of the loading process, travelling time when loaded (ramp up), maneuvering at the surge bin, duration of dumping the ore to the surge bin and travelling time when empty (ramp down). Waiting times in the different waiting bays were recorded as well. The cycle times were subject to some variations. This was a result of the additional traffic on the ramp which caused delays in the cycles, but also due to drivers' varying performance. Variations in other factors like payload, ramp grade and ramp conditions contributed to these time differences as well.

4.1.2 LHD Cycle Time

The LHD cycle times were recorded by observing the loading process from different places in the production drifts. The survey includes loading ore at the drawing point,

travelling to the loading bay with the loaded bucket, dumping to the truck and travelling back to the drawing point with empty bucket. The times for dumping ore in orepasses were measured as well. Here, cycle times differed from each other as well. This was due to the additional work which the LHDs conducted during measurement, like road maintenance (driving through the drift with the bucket on the ground to level the floor) or handling of large rocks being too big to be loaded on the truck. These actions were excluded from the measurements. Again, varying performance of the operators and lump sizes affected the measurements.

4.2 Input Data and Assumptions

The data measured from the mine are used to feed the model. Beside the conceptual information used to constrain and condition the model behavior, basic input data are required to drive the simulation. The equipment performance and specifications as well as the ramp configuration and the distances in the production levels are accepted to stay the same. The following components and data are used to feed the model.

4.2.1 Equipment Capacity and Speed

There is only one LHD working in the production level and loading material to the trucks, thus one LHD serves three trucks. The loading of the Sandvik truck takes three scoops and only two for the Volvo trucks only two. As the effective bucket capacity has been accepted to be 15 t of ore, the full capacity of the Sandvik truck is not used. The travel speeds were modeled based on the observations in the mine and the manufacturers information. It was decided to restrict the ramp down speed to 20 km/h for both equipment type. Due to the reduced payload on the Sandvik truck and its stronger engine, the travel speed for ramp up resulted in around 12 km/h. The speed for the Volvo was selected to be less at around 11 km/h. For the LHDs, the selected travel speed on level ground was based on the observation and manufacturers information, too. The speed was set to 10 km/h. Since it was not possible to measure real traveling times of the LHD on the ramp, the speeds were based on information from the equipment operators. Following, the ramp up speed was chosen to be around 8 km/h and ramp down around 16 km/h. A summary of the equipment specifications can be found in the following table:

Table 1: Equipment specification as used in the simulation model

Equipment Specifications					
Equipment	Amount	Payload	Speed level	Speed up	Speed down
Sandvik	1	45	-	3,3	5,6
Volvo	2	30	-	3	5,6
LHD	4	15	2,8	2,2	4,4

4.2.2 Ramp Configuration and Travel Distances

The dimensions and conditions of the ramp were not implemented in detail into the model. Thus, the only required information concerning the ramp are its inclination and length. Also, it is established that the ramp width can only accommodate one truck. The average ramp inclination was set to 12% and used to model the ramp length. The travel distances on the ramp were estimated with the standard Euclidean length:

$$L = d_v * \sqrt{\left(1 + \frac{1}{m^2}\right)} \quad [1]$$

$$L = \text{Length}, d_v = \text{vertical distance}, m = \text{inclination } [\%]$$

Since the trucking distance does not change a lot for the different orebodies, an average representative tramming distance for the LHD was selected for each orebody. Accordingly, a LHD tramming distance of 120 m was selected for the simulation of the lower production levels. The travel distances for the LHDs for the lower levels in the alternative systems were estimated by geometric calculations.

4.2.3 Loading and Dumping Times

The loading and dumping time implemented into the model are composed of the spotting time and the loading or dumping time respectively. The time required for both activities can affect the waiting time for the following trucks and thus affect the total cycle times. Furthermore, the loading times were simulated for remote control loading and for normal loading. If the model is fed with static values, the traffic flow converts to a steady state shortly after being generated. Thus, the trucks repeat all activities in the same predictable order and time. To allow the model to simulate varying performances in loading and dumping, as it occurs in the real system, these times were generated in the model following a normal distribution computed with the mean and standard deviation of the measured data. The dumping times used for the alternative model are

different from the truck model but also follow a normal distribution. The normal distribution of the data was verified by conducting the Lilliefors-test at a confidence interval of 1 %.

4.2.4 Ramp Traffic

Only one truck can operate at the time in the loading bay or in the dumping bay. Therefore, the other trucks must queue and wait in predefined waiting bays on other mine levels until the specific bay is free. Also, the trucks must drive to waiting bays during their trip on the single lane ramp to allow them to pass each other. In the model, only one truck can use any of the ramp section between the waiting bays as well as any of the waiting bays, the loading and the dumping bay. Hence, if another truck wants to use the specific operational spot or ramp section, it must wait until the other truck is done with the activity. Nevertheless, even if two trucks compete for the use of the same spot or ramp section, the third truck can still operate in other spots or sections as long as these are free. Data about additional traffic on the ramp (e. g. service vehicles, development equipment) was not available and therefore not implemented in the model.

4.3 Verification and Validation

Verification of a simulation model is the determination of its logical accuracy in representing real systems and should be performed as a continuous process during the development of the model. (Banks, 1999) To complete the verification, it was checked whether the model performed the activities intended by the modeler. During the data collection process at the mine, the conceptual characteristics of the operational system were monitored and recreated in the models. Furthermore, frequent discussions have been held with the mines' engineers and shift leaders concerning the correctness of the assumptions which were included in the model developing process. The software allows to monitor some parameters that are important to verify the logical correctness of the model activities (e. g. number of entities in block, avg. waiting time in block). The model was checked for errors by using such monitoring components.

A validation of simulation models is conducted to determine if the model outputs are representative for the real system and can be used for predictions. The ideal way to complete the validation is to compare the model outputs with the real system (Banks, 1999). The validation of a simulation model almost never yields a perfect fit with the

measured data. Since assumptions are made in a model, the goal is not to recreate a perfect model of the real system, but to have a model that is good enough to perform the intended analysis. (Kleijnen, 1995) With the verification being conducted and accepted, the conceptual and logical behavior of the simulation model was accepted to be close enough to the real system. Therefore, the cycle times generated from the model could be compared with the measured cycle times to validate the truck model. As some assumptions have been made for the input data (see chapter 4.2.), variations of the simulated cycle times were expected and accepted in comparison to the measured data. However, the simulated cycle times differ from the measured ones by 3 % to 8 % when fed with the measured data. This is mainly due to the additional traffic on the ramp, random delays caused by the driver, and variances in speed and payload. During the measurement, the delays induced by encountering other vehicles than trucks on the ramp was included in the total cycle time, but not documented separately. The actual average travel speeds of the trucks were fed to the model and the simulation results were compared to the measured data. The simulated cycle times from the 750 level only differ from the average measured cycle time by 2%. For the 700 level, the difference is 3 %. The highest difference was detected for the 650 level. During the measurements in the mine it was recognized that a water truck was travelling intensively between the levels 736 and 625. As no other intensive and repetitive interference on the ramp was observed during the survey, the movement of the water truck can be used to explain the higher differences between the simulated and measured cycle times from the lower production level. Thus, this difference is accepted to be reasonable due to assumptions included in the model.

4.4 The Simulation Models

The simulation models were created with Simulink and its DES tool SimEvents. SimEvents offers tools to create DES models to simulate entities which pass through a system of queues (resources), servers and switches for routing. Predefined modelling blocks provide the possibility to model event driven systems and to analyze routing, queueing and prioritization problems.

The production schedule follows short term planning which is determined by the shift leaders. They base the decision to produce ore from a specific stope according to the grade requirement as well as the filling level of the surge bin of the crusher. As caving and stoping methods are applied, ore is available for long time from several draw points

without having to drill and blast a new stope/section. Therefore, the production can be scheduled following requirements on ore grade, tonnage and stope availability. Due to these operational conditions, it was not possible to create a fixed schedule for implementation in the simulation. Hence, the simulation was used to estimate the effective production rates of the equipment in the different systems in order to determine the average effective production rate from specific levels. The model outputs are cycle times, waiting times and effective production rate. The performances of the equipment were influenced by varying durations for LHD activities (loading, dumping), truck spotting (at loading and dumping bay) and dumping times and include the waiting time in the waiting bays. The models were run several times with the simulated duration set to the effective working time in one shift. The effective working time was calculated with the equipment availability and time losses due to shift change and breaks. The truck availability was 87 % and the shift time availability was 81.25 % (90 minutes losses).

The models do not include the crusher or the conveyors. As this equipment work continuously and are not dependent on the trucks or LHDs, their simulation is out of scope. The main goal of the simulation was to retrieve the effective hourly production rates to see if it meets the 150 t/h requirement. The scope of the model ends with the surge bin. Material can be stored to provide enough capacity for the crusher. With the known production rates from the different levels, the required amount of ore can be delivered by short term planning of the production levels.

The measurements conducted in the mine as well as the simulation of the truck system resulted in sufficiently high effective production rates down to the 650 level (currently deepest production level). Therefore, it was decided to not consider the levels above 650 level for the alternative haulage system. The deepest level in scope was limited to the 450 level. This was mainly due to the reason that the only orebody currently explored beneath the 650 level is the A orebody which continues down to the 500 level. The other orebodies are expected to continue further downdip as well, but none of them has been explored yet. Another reason is that in order to position the crusher on a deeper level, the ramp must be extended first. Currently, the advance rate for ramp development is 25 m vertical distance per two years. This means that the crusher could only be installed on the 450 level in around 6 years (current mine life is 8 years). Assuming that further exploration will increase the mine life and enhanced ramp

development would reduce the development time, the 450 level is a reasonable scope limit. Thus, the study of both systems focuses on the mine levels 650 to 450. Furthermore, it is already known that the truck system can reach the production rate up to a vertical distance of 260 m from the crusher. Hence, after reaching the maximum performance limit of the new haulage system, the truck system could theoretically be used again to ensure efficient production up to the 250 level.

The truck model was built following the real mine layout while the LHD model required a new conceptual layout. The trucks are usually loaded by the LHD in the access point of the ramp to the drift. From there, the trucks drive up the ramp up to the 910 level and wait in the waiting bays for the upcoming trucks to pass them. The alternative system aims to replace the trucks for the levels in scope. Therefore, only LHDs are used. Basically, a similar layout is considered as it is applied in the levels above 910 except that the material must be lifted again after being crushed. A suitable location from where the connecting shaft could be lowered was found on the 815 level near the main ramp (Figure 2). The location of the shaft can be seen in the picture below (vertical red line). From there, the shaft head can be connected to the secondary crusher via a 190 m long conveyor belt (yellow horizontal line). The shaft will be lowered down to the 450 level. The new crusher chamber is around 10 m long and the stope must have a minimum distance to the shaft of 20 m and to the ramp of 32 m. Main levels connect the orebodies in the hanging wall with the foot wall and are established every 100 m. The dip of the orebodies K2/K2' and A move the drawpoints away from the shaft. On the 660 level, the distance from the shaft to the main ramp is around 120 m. Down to the 450 level, the distance will be around 230 m. Theoretically, the crusher can be placed on any level in scope. However, it is the best choice to locate the new crusher as deep as possible to allow the gravity helped movement of the ore. Therefore, the crusher was located on the 450 level. Due to the surge bin height, the feeding point would be on the 500 level. In order to dump the material to the surge bin, orepasses must be established between the main levels and crosscuts must be driven from the ramp to the orepasses every 50 m in vertical distance. The first orepass reaches from the 660 level to the 560 level and is located close to the main ramp. This way, the travel distance for the LHDs can be minimized. A second orepass reaches from the 560 level to the surge bin on the 510 level. This orepass is located above the surge bin. Thus, the orepass head is placed 160 m away from the main ramp. This will

enlarge the traveling distance for the LHDs, but it reduces the required amount of LHDs for the ore haulage by one. Six LHDs are available for the operation. Assuming that four LHDs are available at the same time, the alternative system could use all of them to achieve the required production rate. However, the LHDs are also required for production in the upper levels. In order to avoid time losses due to frequent relocation and intensive travel routes of the LHDs between the levels above and below 910. The system is established so that the least amount of LHDs must be combined to reach the required production rate of 150 t/h.

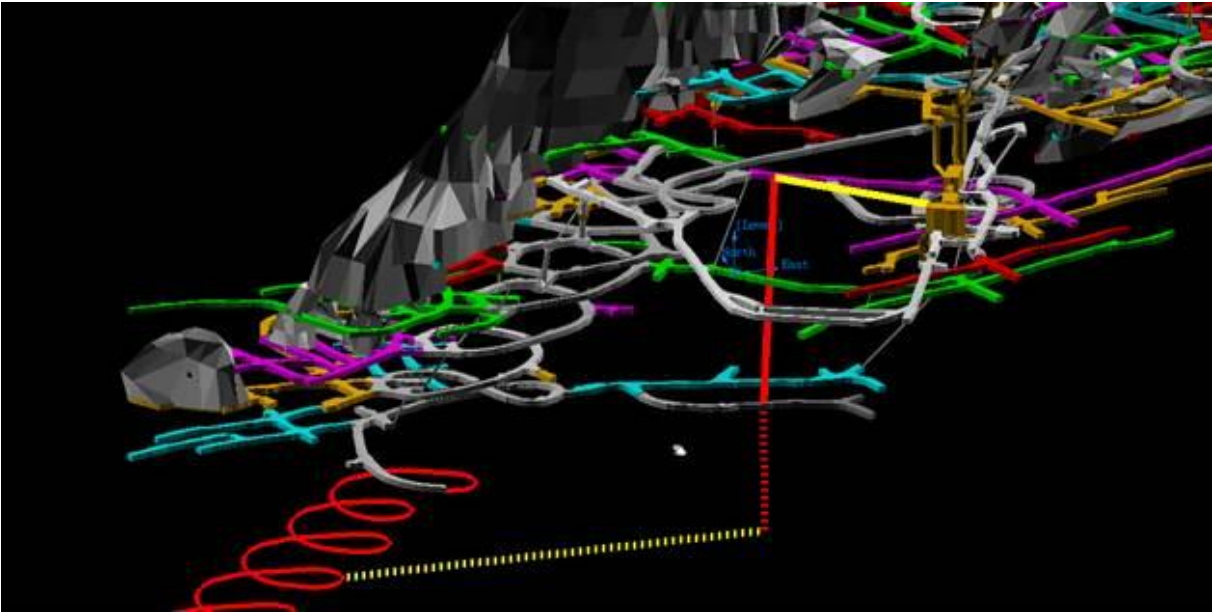


Figure 2: Shaft Location In The Current Mine Model (550 Level)

4.4.1 Truck Model

The model tries to reproduce the real system at a level of detail which is reasonable for the intended investigation. Therefore, the model mainly consists of the ramp with two waiting bays, a loading bay “system” (production level) and a dumping bay (surge bin at crusher). Also, priorities for driving on a ramp section were computed following the observed policy (see 4.2.) The picture below shows the model setup (insert picture) The output of the model is an array containing the effective production rates of the three trucks and the total system production capacity. The number of dumps to the surge bin are cumulated, multiplied with the truck capacity and divided by the theoretical shift length of eight hours. Further the cycle times are measured by

retrieving the times when the truck completes the dumping process and when it enters the first block after dumping bay. By subtracting both times from each other and assigning the duration to the specific truck, several times can be collected so that the average cycle time can be retrieved. Additionally, the same time measuring procedure is applied for both waiting bays (4 locations in the model). This yields the waiting times for each truck in each waiting bay.

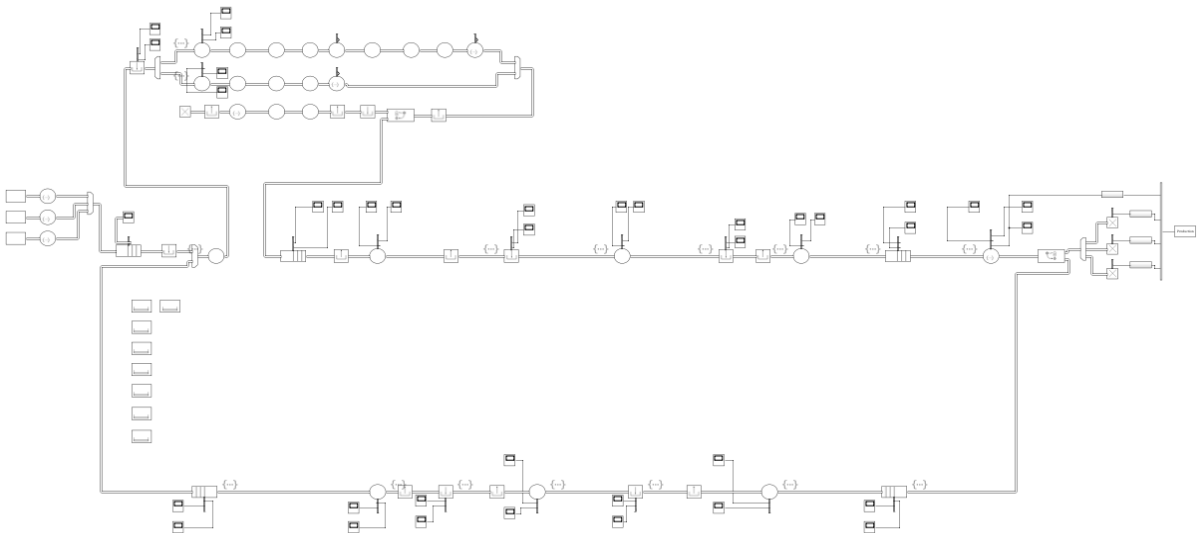


Figure 3: Truck Model

Three entities are generated and move through the model. Each of them is defined by a set of attributes which is used in the different simulation steps of the model to define the duration and type of the activity. The loading bay and the dumping bay are represented by servers in which the entities are retained for the duration of the loading or dumping process respectively. Depending on which entity (Sandvik or Volvo) enters the server, the duration of the activity is different. When the process is complete, the entity moves to the next server in the model. The loading cycle is modeled with several servers which simulate the LHD loading, tramming and dumping. When a truck arrives, the LHD is ready to dump a full bucket on the truck. The truck will “leave” the loading bay “system” after having received the second (Volvo) or third scoop (Sandvik). While the truck continues its journey, the LHD drives back to the loading bay, loads a scoop and drives back to the loading bay to wait for the next truck. The LHD is only available after conducting the last loading cycle

The following three servers simulate the ramp sections. The activity durations are defined by the selected distance between the waiting bays divided by the entity speed. The duration of the travel time is also different depending on which entity type is in the server and if the entity is “driving” down or up the ramp. Since the entities must move in both directions on the ramp (up and down) the model is constructed as a cycle. The ramp is modeled with two streams but represent a single lane spiral ramp. In order to make the entities move like on a real single lane ramp, the usage of the ramp sections was constrained. The trucks take up different resources which are restricted to a specific amount. The resources restrict the use of a ramp section or the loading or dumping bay. When an entity has completed the activity (loading or dumping), it releases the resources so that the next truck can use it.

4.4.2 LHD Model

The LHD model mainly consists of the loading bays, dumping bays, drifts and orepasses. Here, the outputs are also the production rates for the different LHDs. The production rates are calculated like in the truck model and delivered in an array. The

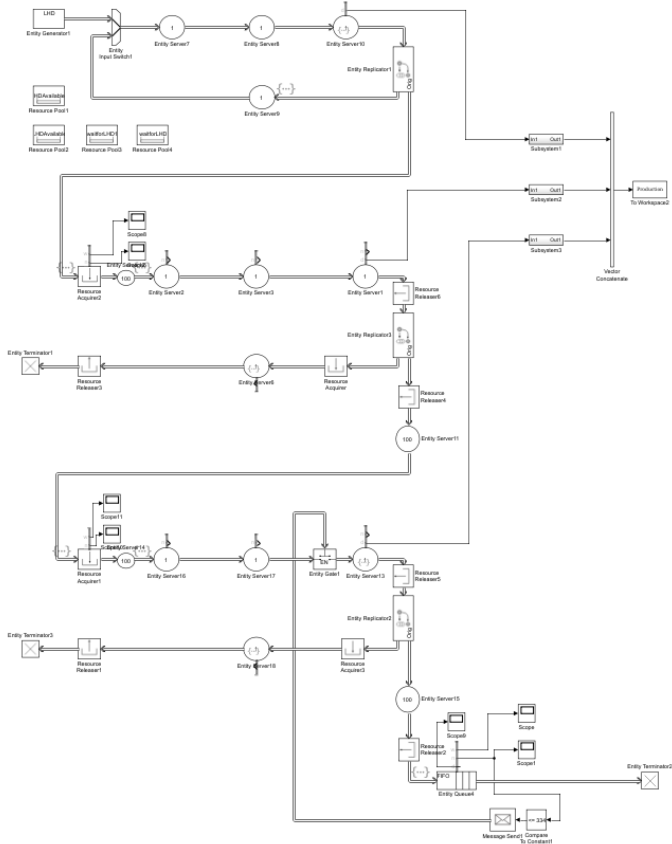


Figure 4: LHD Model

total production rate of the system was not retrieved, as the production rate achieved by the last LHD is the effective production rate of the system.

One entity is generated at simulation start and represent the LHD on the production level. This entity is advancing through a loop of activities (loading cycle). It first loads the material from a drawpoint and then drives it to the orepass. There, it dumps the material to the orepass (or in a drift) and then drives back to the loading bay. After the dumping process, the entity is duplicated and continues to the deeper level. From there, the entity passes through another loading cycle consisting of four servers (draw point, travel to the orepass and travel from the orepass). The entity is loaded from the orepass drawpoint and then trammed to the surge bin or the next orepass which is connected to the surge bin. After dumping the material, the entity is duplicated a second time whereby one entity continues to the surge bin -where it is stored- and the other one drives back to the draw point of the orepass (or drift). The availability of the LHD to load the material from the draw point of the orepass is constrained with a resource. The first entity that is dumped to the first orepass acquires the resource and only releases it after returning to the draw point. If the cycle of the first LHD is shorter than the cycle of the second LHD, the ore (duplicated entities) are stored in the draw point of the orepass and will be retrieved one by one. The activity duration for tramping is defined by the selected distance divided by the entity speed. The loading times are the same as used in the truck model, and are generated randomly following a normal distribution. The dumping times are generated randomly as well.

5. Simulation and Results

The models described above are base models which can be modified and varied. Both models can be modified by changing the travel distances of the equipment. In order to predict the production rates of the systems for the yet undeveloped levels, several simulation tests were conducted.

5.1 Truck Model Variations

For the truck haulage cycle the truck drivers communicate two predefined waiting bays and stop in these bays to wait for the oncoming truck. Even though there are possible waiting bays on each sublevel spaced from each other in 25 m vertical distance, the drivers cannot decide on when to continue and when to wait. This would require some kind of tracking tool or block signals for each ramp section between the sublevels. Therefore, two sublevels are selected as waiting bays. As the waiting bays for the active production levels are selected according to the drivers' experience, there is no fix pattern or formula to choose the waiting bays for the undeveloped levels. Hence, all possible combinations of ramp sections between the waiting bays were simulated. The first waiting bay was always the first one above the production level. The last possible waiting bay (before the surge bin) was the 830 level. Apart from the section between the 800 and 830 level, each waiting bay was spaced 25 m vertically from each other. The combination resulting in the lowest cycle times and the highest production rate was selected for the evaluation.

5.1.1 Results

This procedure described above was repeated for each production level in scope. The best results are presented in the table below.

Table 2: Production Rates Truck System

Simulation							
<i>81,25% shift availability; 87% equipment availability = 70% eff. working time</i>							
Truck System	Production Level						
Production Rate [t/h]	750	700	650	600	550	500	450
Total	219	206	162	136	127	114	101
Sandvik	96	90	73	62	56	51	45
Volvo 1	64	60	45	38	38	34	30
Volvo 2	60	56	45	38	34	30	26
Waiting Bays	775 - 830	750 - 830	700 - 800	675 - 800	625 - 775	600 - 750	575 - 725

The waiting bay are in order of passing when driving ramp up. The possible effective production rate decreases constantly with increasing depth. The simulation results show that the current system cannot deliver the required production rate beneath the 650 level. Further, it can be seen that the distance between the waiting bays increases with depth. The waiting times in the different waiting bays were monitored in order to gain more information on the truck cycles. The chart below shows the average total waiting times for the different trucks and the total waiting time of all trucks in one cycle.

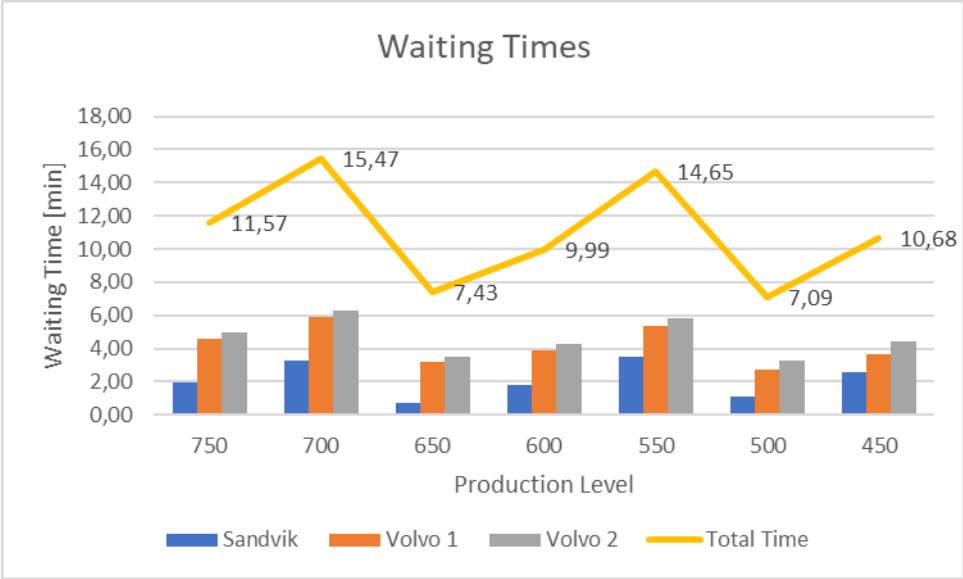


Figure 5: Waiting Times Truck Model

The total time loss due to waiting in the waiting bay varies. The variation seems to follow a saw tooth pattern. The waiting times peak on the 700 level and the 550 level and have the lowest values on the 650 level and the 500 level. Generally, the total waiting time seem to slightly decrease with depth even though they fluctuate. When looking at the distance from the loading bay to the first waiting bay a slight pattern can be identified. The results indicate that the distance to the optimal location of the first waiting bay increases by 25 m in vertical distance for every deeper production level. After two steps, the location stalls for one further step and then continues with the same pattern. It can be observed that the total waiting time drops in these steps.

5.2 LHD Model Variations

Depending on the production level the LHD must dump the material to orepasses, drifts or directly on the surge bin and thus, drive different distances from the stope to the surge bin. In lower production levels, the distance become larger from the draw point

to the surge bin or orepasses. Therefore, the best combination of tramming distances for the different LHDs must be examined. The stope layout as well as the ramp sections and the connecting drift to the orepass or surge bin are known. There are some bays in which the material can be dumped to and from where another LHD can load the material. The different possible dump bays were identified and the tramming distances were varied for each production level. The simulation runs were always started with only two LHDs. If the required production rate could not be reached, one LHD was added to the system. This iteration yields the best combination of tramming distances while using the least possible amount of LHDs.

5.2.1 Results

The results of the procedure mentioned above are presented in the table below.

Table 3: Production Rates LHD Model

Simulation							
<i>81,25% shift availability; 100% equipment availability = 81,25% eff. working time</i>							
LHD System	Production Level						
Production Rate [t/h]	750	700	650	600	550	500	450
LHD 1	-	-	238	238	173	163	173
LHD 2	-	-	214	189	171	161	171
LHD 3	-	-	212	189	-	-	169
LHD 4	-	-	-	-	-	-	169
Tramming Distances [m]	-	-	140/185/90	140/220/90	240/245	260/260	240/210/210/195

The tramming distances are showed in the order for the LHD 1, LHD 2, LHD 3 and LHD 4. The results show that the required production rate can be achieved at all levels in scope. However, the system requires 3 LHDs for the levels 650 and 600, 2 LHDs for the production in the level 550 and 500 and even 4 LHDs for the production from 450 level. As can be seen from the production rates of the separate LHDs in use, the deliverable tonnage is always dependent on the slowest LHD. This is especially a disadvantage for the system with 3 LHDs. The delivered production rate is only 189 t/h whereby the possible production rate could be higher. However, the ore that is produced more from the faster LHDs is stored in the orepasses and thus closer to the surge bin. Therefore, the ore gains some more value. The LHD 2 and LHD 3 on the 450 level drive up the ramp and therefore have higher cycle times.

6. Cost Evaluation

The simulation showed that the current truck system is not able to deliver the required production rate and thereby already confirms that the system must be changed anyways. Nevertheless, the costs can be presented in order to have reference values for the costs which will occur during the construction phase of the new system. The costs of the current haulage systems were calculated based on the simulated production rates. The hourly costs for the trucks were determined with the historic data from 2014 to 2017 provided by the company. The Sandvik truck is a company owned equipment while the Volvos are provided and operated by a contractor. Therefore, maintenance is conducted outside the mine workshop. The costs are calculated based on the fuel consumption, hourly cost for operator and the cost factor for maintenance. The Sandvik truck costs are calculated the same way but include depreciation costs and maintenance costs. The model outputs the effective hourly production rate and thus, the effective cost per ton can be directly calculated. Also, the truck system costs include the LHD cost which is required to load the material to the trucks. The LHD costs are also available in hourly costs. The costs were determined for each equipment separately and were added together for the total system costs. The LHD hourly costs were divided by the total hourly production of the system. The following diagram shows the total system production costs along with the production rate and the separate costs for each equipment (Sandvik, Volvo 1, Volvo 2, LHD).

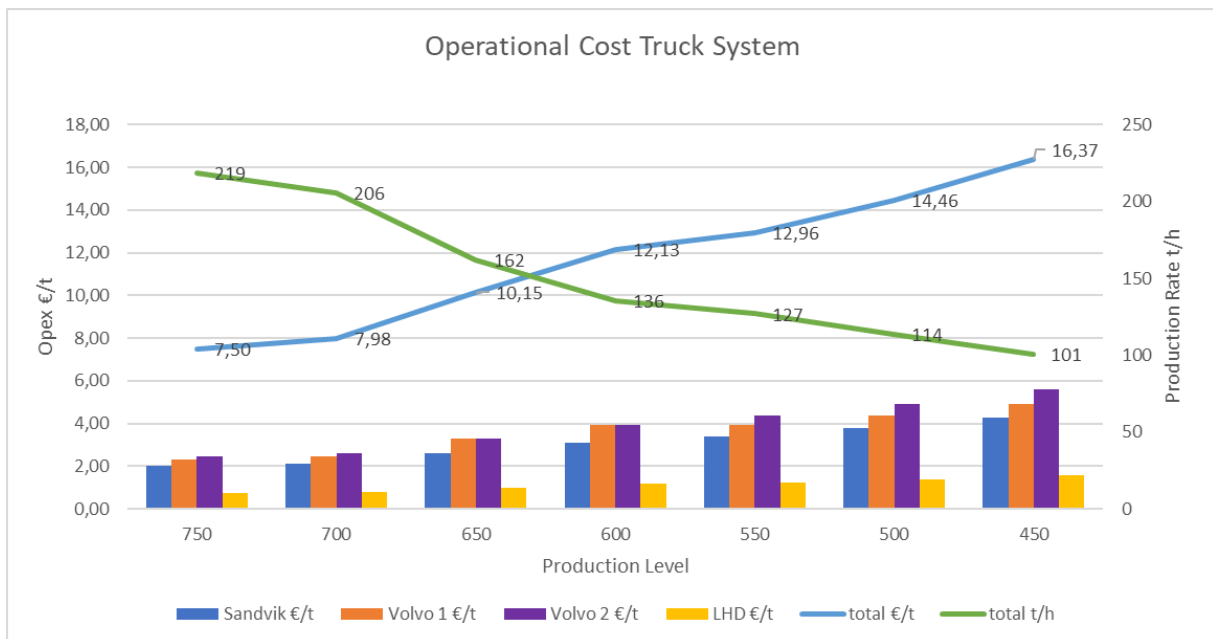


Figure 6: Operational Cost Truck System

The data shows that the Sandvik truck costs less than the Volvos when calculated over the production rate and the cost of the Volvos increase more rapidly as well. The development of the total costs is almost a mirror image of the production rate. It can be seen from the diagram that the production costs more than double from the 750 level down to the 450 level. The development is fairly linear and thus, a change of roughly 1.48 € per meter can be retrieved.

For the LHD system costs, the information was retrieved from different sources. The costs provided in this work are values based on estimations and information from conceptual studies. Nevertheless, they were selected with the best knowledge of the author to be representative for this level of study. Those sources included cost estimators, manufacturer information and information from other case studies and papers. The costs for the new systems were divided in operational costs and capital costs. However, the operating costs of the vertical conveyor and the belt conveyor also include the depreciation costs. By this, they can be better compared to the truck costs. The new system requires new equipment, rework of the old crusher, the establishment of orepasses, crosscuts and a shaft. Also, the crusher chamber must be established as well as a surge bin. The required equipment and mine buildings are listed in the following table. The required crusher room has a volume of 2100m³ and the surge bin have the same dimensions as on the level 910. By this, the second crusher facility is also capable to store material for two shifts. All costs are based on an hourly production rate of 150 t. The operational life of the Pocketlift, the belt conveyor and the trucks are set to 15000 h. The required construction and installation effort was not included into the study since the level of scope did not allow proper determination of such costs.

Table 4: Equipment Cost LHD System

Equipment Cost LHD System				
<i>wihtout LHD costs</i>				
Equipment	Investment [€]	Power [€/t]	Maintenance (€/t)	Opex [€/t]
Pocketlift	1861500	0,08	0,46	1,36
Conveyor Belt	229825	-	-	0,21
Shaft	346750	-	-	-
Raises	105000	-	-	-
Crusher	55107	-	-	-
Crusher Facilities	209344	-	-	-
Drifts	210000	-	-	-

The selected crusher requires some reworking. Thus, the investment costs are relatively low. The highest investment is due to the vertical conveyor (costs derived from Gonen et al., 2012; currency transferred to € with 0,85 \$/€). The costs for the crusher parts as well as vibrating feeder and the conveyor belt were retrieved from the Mine and Mill Equipment Costs Guide (InfoMine, 2012). The costs for the establishment of the mine buildings were calculated based on information provided by the company. The maintenance costs vertical conveyor was estimated to be 5 % of the investment costs per year and the power consumption (0,3 kWh/ 100 m and ton) was retrieved from the World Coal article. (Rowland, 2014) The operating costs for the crusher and its surrounding equipment were not included in the calculation. Since the new crusher station is operating only one shift and the old crusher in the other shift, the total cost for the operation stays the same. The operating costs for the belt conveyor are based on the Mine and Mill Equipment Cost Guide as well. The production costs for the LHD systems were plotted along with the production rates.

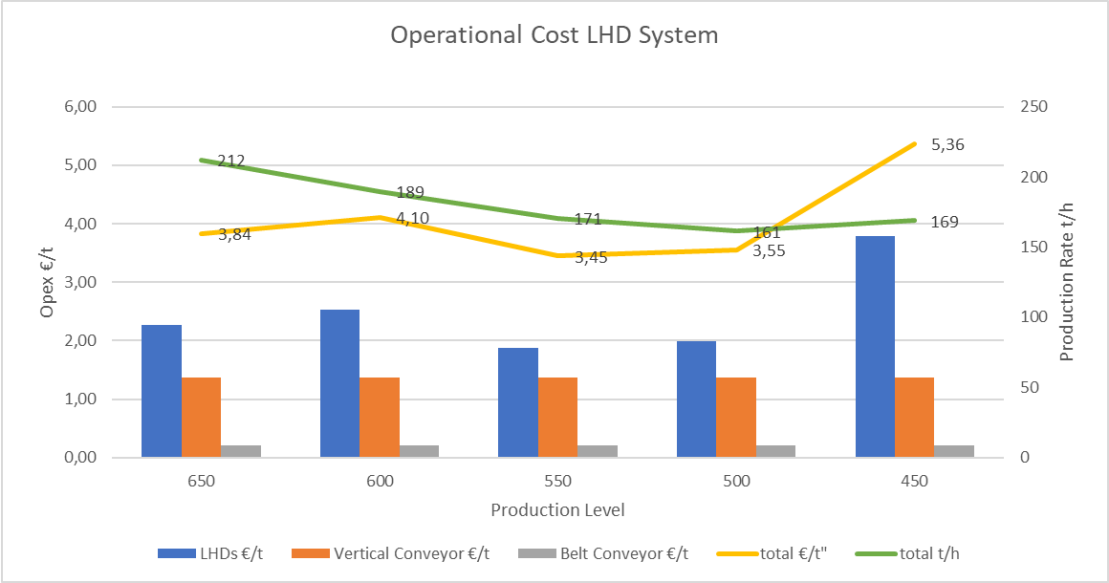


Figure 7: Operational Cost LHD System

The costs for the LHD system are varying. This is mainly due to the fluctuating production rates on the different levels and the amount of LHDs that are necessary to deliver the required production rate. The costs for the levels 650 to 500 are fairly constant while the costs on the 450 level increase of roughly 30 % compared to the average costs on the other levels. That is because four LHDs are required to deliver a production rate roughly satisfying the requirement.

7. Discussion

The present study was carried out to determine the possibility and benefits of relocating the crusher facility of the investigated mine to a deeper production level. For that purpose, a suitable alternative haulage system was selected to connect the main transportation level with the crusher in its new location. The conceptual selection process for the new system identified the vertical conveyor Pocketlift as the most suitable equipment for the operation. DES was chosen to be used in this study because of its capability to model the dynamic nature of haulage systems. Two models were built to simulate the production rates on the yet undeveloped production levels.

The simulation of the current truck system yields that the production rate decreases on each lower production level. The simulation showed that the cycle times increase with the depth and thus reduce the capacity of the system. The simulation proved the incapability of the current system to meet the required production rate in order to maintain the current yearly production. Thus, the truck system cannot be used for future production. The current presence of enough resources in the upper production levels leaves operational options to balance the production from low productive levels and highly productive levels. Consequently, as long as enough material is available from the levels above 910 and down to 700, the current system can be applied. But due to the current development to the deeper production levels, the alternative system becomes of high interest. Further investigation showed that the waiting times during each trip amounts up to more than 15 minutes. The shortest total waiting time that was measured was 7 minutes. The longest delay for one truck was measured to be 6.27 minutes, what is 25 % of the total cycle. This shows that a significant delay results from the truck traffic on the ramp and introduces a bottleneck to the system. Therefore, the current system cannot deliver the required production rate beneath the level of 650.

In order to improve the system capacity, additional trucks could be introduced to the current system. However, as the traffic congestion is already high with three trucks, the specific costs for a single truck would increase with a higher number of trucks in the system. Thus, the production costs as well. As already recognized by Salama and Greberg (2012) the production rate increases with an increasing number of trucks, but not significantly. Furthermore, increased traffic does not only affect the production, but can also reduce the operations safety (Haviland, 2014) and increases ventilation costs (Paraszczak, 2014). Therefore, an alternative haulage system was designed that

excludes the usage of trucks. The vertical conveyor is energy efficient and has low emissions to its environment. Also, the use of automated and continuous equipment reduces personnel and maintenance cost. The simulation of the alternative system resulted in significantly lower cost per ton compared to the current system. It showed its capability to deliver the required production rate from all levels in scope. In terms of operational costs, the system even performs at only a third of the costs of the current system when running at the system limit (450 level). However, this benefit comes along with high investment resulting in more than three million euros without considering the construction and installation effort and costs. Compared to the unit cost of 900,000 € for a Sandvik truck, this investment would seem reasonable when considering the ventilation and safety benefits as well as the lower operating costs. However, the bottleneck of the LHD system can be found in the dependency of the system productivity from the one slowest LHD.

These results can be used to show the potential benefits of placing the crusher to a deeper level and changing the truck system to a LHD based system with a vertical conveyor. The technology of vertical conveyors was found to be very promising and its consideration for such operations should be spread. Furthermore, the remaining flexibility of the system also satisfy the operations need for quick changes from production levels.

7.1 Practical Implications

DES proved to be an effective tool in the investigation of the dynamic systems. Its capability to trace the equipment in the cycle and monitor their interferences enhanced the understanding of the system and yields information on the weak points of the current system. However, the capability of this simulation to model highly detailed system requires a high amount of data and detailed measurements from the real system. Attention should be payed to the required detail of the simulation to meet the intended use. The availability of more detailed and structured information on the production schedule could have enhanced the detail in the model. Also, knowledge about the actual delays due to additional ramp traffic would have yield a much more detailed and reality near model.

The investigation which was conducted with the models is of basic nature and could most likely have been performed with common analytical methods. Nevertheless, the models have implemented randomness which could not have been included by

analytical methods (Greberg et al., 2015). The degree of benefit of having used DES for this investigation may be low, but the potential of this simulation method was recognized and the basic advantages of DES could be highlighted, namely the potential of concerning randomness and the reality near interaction of the simulated equipment.

SimEvents was used to model the systems and proved to be a suitable tool. The user-friendly interface and the block based model construction delivered an easy to use tool. However, the software fails to help for the verification process. Even though there some data scope tools, the observation of the entity flow demands a lot of effort and intensive simulation runs. The available animation function is not very useful for the verification of the entity behavior. Improvements of the animation function would pay great benefit to the software.

7.2 Strengths and Limitations

A strength of the present study was the comparison of the two systems in scope. The current system is a constellation which is widely used in different mining operations. Therefore, it was a valuable contribution to propose an alternative system that proved to be efficient for such operations. Hence, this consideration could be applied in different operations which seek to improve their haulage system in an innovative and efficient way. Furthermore, the results encourage to question existing standards and to consider new methods. Even though the transition to a new haulage system might be imply mayor investments and efforts, the benefits could pay back for both the operation and the environment.

As it was already recognized by Tako and Robinson (2009), the greatest benefit of DES - its capability to simulate great complexity and detail- is also its disadvantage as it requires high expertise. Since the study was time limited, it was not possible to fully familiarize with all aspects of DES. Consequently, the full potential of the simulation could not be used.

Further limitations were given due to equipment breakdowns during the mine visits. Thus, only a few measurements of the truck cycles could be conducted. In order to generate more representative data for implementation into the model, more measurements would have been required.

The cost evaluation is based on assumptions and exemplary data. While the presented data was sufficient to conduct superficial calculations and to present a representative comparison of the operational costs, more detailed information could enhance the results.

The model was constructed with respect to the traffic policy that is applied in the considered mine. However, it does not consider the additional delays resulting from auxiliary equipment and vehicles on the ramp, neither does it implement equipment breakdowns or stope availability. Consequently, the limited availability of data on the equipment performance and the simplification of the model produced an uncertainty of up to 10 % in the model.

8. Summary

The gradual increase of underground mining operations in production depth potentially results in lower productivity due to factors like longer haulage distances. This means that the haulage equipment will need to have higher capacity to stay efficient, while the emerging geotechnical constraints in deeper mining environment will demand for smaller equipment. Especially mine operations using truck haulage, which have been widely used for material handling in underground hard rock mines, will have to deal with this problem. Against this background, the objective of this thesis was to investigate and optimize an underground haulage system of a case study mine using discrete event simulation. In this context, the productivity of a current truck-loader system and its performance on deeper production levels was investigated with the aim to determine its maximum range and to compare the option of relocating the crusher to a deeper mine level or not. After thorough consideration of different haulage systems, the vertical conveyor Pocketlift was determined as the system most suitable for the connection between the crusher in its new location and the established conveyor belt connecting the mine with the processing plant. Simulink and its discrete event simulation tool SimEvents was applied for the investigation of the production rates of the current and the new system. The advantage of simulation is its use without the need of high capital and time investments. Discrete event simulation was chosen because it better captures the dynamic and random nature of haulage systems compared to analytical methods. Results showed that the production rate of the current truck-loader system decreases on each lower production level, so that it cannot be used to meet the required production in future. A cost evaluation of the alternative system resulted in significantly lower cost per ton compared to the current system. It showed its capability to deliver the required production rate from all levels in scope. Even though the investment costs for this new system are high, cost evaluation revealed that the new system performs at only a third of the costs of the current system when running at the system limit (450 level), offering an innovative optimization to the case study mine by relocating the crusher. Finally, discrete event simulation proved to be an effective tool for the investigation of the dynamic systems. Its capability to trace the equipment in the cycle and monitor their interferences enhanced the understanding of the system and yields information on the problems of the current system.

References

- Alspaugh MA (2004) Latest developments in belt conveyor technology. In: MINExpo 2004. Las Vegas, pp. 275-281.
- Banks J (1999) Introduction to Simulation. In: Farrington AP, Nembhard HB, Sturrock DT & Evans GW (eds.), 1999 Winter Simulation Conference Proceedings. Phoenix, pp. 7-13.
- Basu AJ & Baafi EY (1999) Discrete event simulation of mining systems: current practice in Australia. *International Journal of Surface Mining, Reclamation and Environment*, 13, 79-84.
- Bloss M, Harvey P, Grant Don & Routley C (2011) Underground Ore Movement. In: Darling P (ed.), *SME Mining Engineering Handbook (Vol. 2)*. SME, pp. 1271-1294.
- Brunner DT (2001) Simulation of Underground Mining Operations. In: Hustrulid WA, Bullock RL (eds.), *Underground Mining Methods. Engineering Fundamentals and International Case Studies*. SME, Littleton, pp. 705-712.
- ContiTech AG (2013) Versatile and Robust Conveyor Belts for Industrial Applications. *Advanced Mining Solutions Online*, 94-96.
- ContiTech AG (2014) Pocketlift-The innovation in highend vertical conveying <https://www.contitech.de/getmedia/5e46271d-d893-4da7-ad94-7b33afc17f74/CBG7836-En-Pocketlift.aspx> (last access on August 20, 2017).
- InfoMine USA, Inc. (2012) *Mine and Mill Equipment Costs – An Estimators Guide*.
- Dammers M, Brudek G, Pütz M & Merchiers A (2016) Opportunities and Challenges of Rail Haulage Systems in Current Mining Operations, *Engineering and Mining Journal*, 217, 42-47.
- Dos Santos JA (2007) Sandwich Belt High Angle Conveyors 2007-Progress to Date. <http://www.beltcon.org.za/docs/b1416.pdf> (last access on July 28, 2017).
- Gonen A, Malli T & Kose H (2012) Selection of ore transport system for a metalliferous underground mine, *Archives of Mining Sciences*, Vol. 57, 779–785.

- Greberg J, Salama A, Gustafson A & Skawina B (2016) Alternative Process Flow for Underground Mining Operations: Analysis of Conceptual Transport Methods Using Discrete Event Simulation. *Minerals*, 6, 65.
- Haviland D & Marshall J (2015) Fundamental behaviours of production traffic in underground mine haulage ramps. *International Journal of Mining Science and Technology*, 25, 7-14.
- Kleijnen JP (1995) Statistical validation of simulation models. *European journal of operational research*, 87, 21-34.
- Law AM & Kelton W D (1991) *Simulation modeling analysis* (2nd ed.). New York, McGraw-Hill.
- Lu M, Lau SC & Chan EK (2007) Combined simulation modeling using simplified discrete event simulation approach: A mining case study. In: *Proceedings of the 2007 Summer Computer Simulation Conference*. Society for Computer Simulation International, pp. 421-428.
- McCarthy PL. (1999) Selection of shaft hoisting or decline trucking for underground mines. *Driving Down Haulage Costs*. AMC Reference Library, Kalgoorlie, Australia.
- Merchiers A, Brudek G & Dammers M (2015) Advantages and Applications of Rail Haulage Systems in Underground Hard Rock Mining Operations. *Mining Report* 151, 400-416.
- Metso Minerals Processing Handbook (2015) Basics in Minerals Processing. [http://www.metso.com/miningandconstruction/MaTobox7.nsf/DocsByID/EAE6CA3B8E216295C2257E4B003FBBA6/\\$File/Basics-in-minerals-processing.pdf](http://www.metso.com/miningandconstruction/MaTobox7.nsf/DocsByID/EAE6CA3B8E216295C2257E4B003FBBA6/$File/Basics-in-minerals-processing.pdf) (last access on July 29, 2017).
- Mosher J (2011) Crushing, Milling and Grinding. In: Darling P (ed.), *SME Mining Engineering Handbook* (Vol. 2). SME, pp. 1461-1480.
- Neumann T (2017) Innovative transport technologies for steep incline conveying in open pit and underground mining. In: *Mining Forum Conference Transcript*. DMT GmbH & Co. KG, Berlin, pp. 79-80.

- Panagiotou GN (1999) Discrete mine system simulation in Europe. *International Journal of Surface Mining, Reclamation and Environment*, 13, 43-46, DOI: 10.1080/09208119908944208.
- Paraszczak J, Svedlund E, Fytas K & Laflamme M (2014) Electrification of loaders and trucks: A step towards more sustainable underground mining. In: *International Conference on Renewable Energies and Power Quality*. Cordoba.
- Pratt AGL (2005) Application of conveyors for underground haulage. In: *Proceedings of the Ninth Underground Operators Conference*. Australasian Institute of Mining and Metallurgy, Melbourne, pp. 179-188.
- Rowland J (2014) POCKETLIFT vertical conveyor: a case study. https://www.worldcoal.com/handling/13052014/pocketlift_vertical_conveyor_a_coal_mining_case_study_coal830/ (latest access on August 21, 2017).
- Salama A & Greberg J (2012) Optimization of truck-loader haulage system in an underground mine: A simulation approach using SimMine. In: *International Conference & Exhibition on Mass Mining*. Canadian Institute of Mining, Metallurgy and Petroleum, Sudbury.
- Salama A, Gerberg J & Gustafson A (2014) *Conceptual Models Regarding Mine Layout/Infrastructure and Process Flow at Greater Depths: A Literature Review on Haulage Systems*. Luleå: Luleå University of Technology.
- Salama A, Greberg J & Schunnesson H (2014) The use of discrete event simulation for underground haulage mining equipment selection. *International Journal Mining and Mineral Engineering*, Vol. 5, 256–271.
- Salama A, Nehring M & Greberg J (2014) Operating value optimisation using simulation and mixed integer programming. *International Journal of Mining, Reclamation and Environment*, 28, 25-46. DOI: 10.1080/17480930.2013.768019.
- Sturgul JR (2001) Modeling and Simulation in Mining-Its Time Has Finally Arrived. *Simulation*, 76, 286-288.
- Tako AA & Robinson S (2009) Comparing discrete-event simulation and system dynamics: users' perceptions. *Journal of the operational research society*, 60, 296-312.

- Tiley P (2011) Hoisting Systems. In: Darling P (Ed.), SME Mining Engineering Handbook (Vol. 2). SME, pp. 1295-1323.
- Töpfer M (2014) Going Up-Convey Vertical. bulk solids handling <http://www.bulk-solids-handling.com/?q=topics/conveying-transportation/going-convey-vertical> (last access on August 25, 2017).
- Vagenas N (1999) Applications of discrete-event simulation in Canadian mining operations in the nineties. International Journal of Surface Mining, Reclamation and Environment, 13, 77-78, DOI: 10.1080/09208119908944213.
- Vergne de la J (2003) Hard Rock Miner's Handbook. North Bay, Ontario, McIntosh Engineering.
- Wilson RB, Willis RPH, & Plessis du AG (2004) Considerations in the choice of primary access and transportation options in platinum mines. In: First International Platinum Conference 'Platinum Adding Value'. The South African Institute of Mining and Metallurgy, Sun City, pp. 3-7.
- Zuchowski J (2013) Industry. World of Mining - Surface & Underground, 65, 74-77.