STUDY ON THE OPTIMIZATION OF DUMPING DESIGN FOR OPEN PIT METAL MINES IN MONGOLIA: A CASE STUDY OF THE ERDENET MINE

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出版情報：Kyushu University, 2018, 博士（工学）, 課程博士
バージョン：
権利関係：

STUDY ON OPTIMIZATION OF DUMPING DESIGN FOR OPEN PIT METAL MINES IN MONGOLIA: A CASE STUDY OF THE ERDENET MINE

A DOCTORAL DISSERTATION

Submitted to the Department of Earth Resources Engineering
Graduated School of Engineering
Kyushu University

As a partial fulfillment of the requirement for the degree of Doctor of Engineering

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Abstract

The mining sector in Mongolia is one of the largest financial contributors to the state economy. The share of mineral exports is 88% out of total exports that amounts to almost one-third of the government revenues and contributes 20% to the country's GDP. The primary outputs of the mining industry are copper and gold, iron ore, molybdenum, fluorspar and coal. In Mongolia, the projects of mineral resources are operating by open pit mining method since 1960s. A relative deeper and large scale open pit mines developed in last 10 years and which are mainly operated in copper ore and hard coal mines. The depth and scale of open pit mines has been increasing with the mining capacity, mineral products and high economic growth in major projects. However, geotechnical investigations, data and practical experience are insufficient in many active deeper open pit mining. It is the main problems in Mongolia for mine planning such as open pit optimizing, slope stability studies and appropriate large waste dump design etc. at most large scale open pit mines. In order to understand of this problems, a case study is conducted at the Erdenet Cu-Mo open pit mine in northern Mongolia, which currently faces the challenge of designing overall slope angles for current pit depth around 300 meters. The design problem and lack of knowlegde for optimzation of open pit and dumping area was the starting point for this work. The purpose of this research is to develop an innovative method for optimization of appropriate design of open pit and dumping using geotechnical, geochemical and economic data at one of major mining project, Erdenet Cu-Mo porphyry deposit. The dissertation consists of seven chapters and the main contents in each chapter are as follows:

Chapter 1: This chapter introduce the information about the country and mining industry in Mongolia, problem descriptions of large scale open pit mines and the general background of studies including the processes of optimization procedures of open pit mine and dumping, problem description and literature review and survey of previous studies, objective and outline of dissertation are proposed.

Chapter 2: This chapter describes the general information of the Erdenetyn-Ovoo Cu-Mo deposit mine site, geological conditions, mining activity and mine site problems

related to the objectives of the research. The total area of the Erdenetyn-Ovoo Cu-Mo deposit mine project is 5,500 km^2. On the basis of exploration surrounding the mine and feasibility study, ore reserves in the Erdenet Central deposit and the Erdenet Southeast (Oyut) deposit were calculated to be 1,250,000 tons (0.43 % Cu, 0.018 % Mo) and 41,890,000 tons (0.40 % Cu, 0.007 % Mo) respectively. The open pit mine currently covers an area of 2.5x1.5 km^2. The geological explorations in the deposit have been studied for the purpose of estimating and increasing the reserves and there are very insufficient studies for geotechnical research. Due to the insufficient of geotechnical investigation and researches, the stability angle, the dimensions of the design of the open pit mine and the underlying dimensions of the open pit mine are justified. Based on these real problems, the study is subsequently focused to develop an innovative method to optimization of appropriate design of open pit and dumping area using geotechnical, geochemical and economic data.

Chapter 3: This chapter discusses about the determination of the nested pit shells, pushbacks and ultimate pit limit of the Erdenet Cu-Mo deposit, as well as open pit optimization study through geological model, and rock mass characterization and space factors to optimize the design of mining and dumping. In the research, the Geovia Surpac software is utilized for generating a 3D geological and deposit block model, the Rocscience Dips software is for kinematic analyses and the Geovia Whittle software is for establishing the final pit limit in terms of the maximum Net Present Value (NPV) and associated pushbacks to produce a best case mining scenario were used. From the results, Pit shell-34 with Revenue Factor=100% covers the maximum net present value (NPV). And the result differences between the Pit shell-34 (RF=1.0) and Pit shell-84 (RF=2.0) are 79 mil. $ of NPV (2294 mil.$ undiscounted cash flows) and 2155.8 mil.t. of waste rock. From the sensitive analysis, resource in open pit mine is the most sensitive to metal prices. When the metal price drops to 30%, while the sulfide ore decreases to 935 million tonnes and increases by 30% to 497 million tonnes. Increasing the overall slope angle of the open pit by 4°, amount of the waste rock decreased very low as 3.3%. However, decreasing the overall slope angle of the open pit by 4°, amount of the waste rock increased quite high as 21.5%. The current concept of the Erdenet Mining Corporation has a total of 950 mil.t of ore at the open pit mine depth of 905m.

The results of pit optimization analysis show the possibility of open pit mine depth considering stability condition reach to the elevation of 780m which allows 125m more depth and to allow more than 550 mil.t of ore reserve to be exploited by the current Concept of Erdenet mining. Determining the location of the waste dumping and surface infrastructure and constructions based on the established open pit boundaries is quite risky. Open pit mining boundary is quite dynamic and is constantly changing from the beginning of the mine life to the end. The size, location and final shape of open pit should be optimized based on prospective production prices and open pit revenue factors are important in planning the location of waste dumps, stock piles; processing plant, access roads and other surface constructions, facilities and infrastructures.

Chapter 4: This chapter discussed the formation mechanism of benches on stability of dumping area to optimize waste dump design. In open pit mine, providing a proper dump is crucial to mine's successful operation. The improper waste dump result in stability issues which may affect safety and production of the mine. The geological overview, dumping operation, waste particle distribution, and stable problems were investigated at the Erdenet open pit mine. Then, a series of the experiments was conducted in the laboratory to simulate the formation process of single bench, multiple benches, and the efficiency of dumping operation's design. Finally, the relationship between safety factor of dumping area and bench height, bench angle, bulk factor of waste rock, and truck transport were simulated by using numerical simulation. From the results, two methods are proposed to increase the stability of dumping areas. Firstly, the loose earth and all vegetation need to be removed to make the floor strong seam. Secondly, floor surface of dumping area becomes rough by blasting, which can prevent the floor to be slide surface. Design of the dumping operation must consider the total efficiency of ground leveling operation work and forming dumping area work. Height of bench can be as high as possible, up to the allowed safety values of workers and equipment working. Angle of bench is not important to dumping operation. Bulk factor of waste rock should be as small as possible to improve dumping operation stability. The activity of transport truck in dumping area has a beneficial effect on stability of dumping area.

Chapter 5: This chapter discussed about the effects of different wide of buffer zone for different pit wal as well as was dump's bench configuration that fit a given stability of the slope. Creating waste dump near to the pit is one of the solusions when the waste rock contain low grade of valuable minerals that planned to be extracted in future, as adopted by Erdenet open pit mine. Waste dump alongside the pit gives advantage in regards to waste hauling cost. However, from geotechnical point of view, constructing a waste dump alongside the pit should be planned well thus satisfy the stability criteria by adopting buffer zone, the distance between crest of pit wall and toe of waste dump slope. This chapter also discussed about the influence of cohesion and friction angle on pit wall stability. According to simulation result, when the configuration is without buffer zone, the SRF will be strongly influenced by pit wall height and slightly influenced by waste dump height. However, when the configuration considers the buffer zone, it would be strongly influeced by length of buffer zone besides the pit wall height. By having a buffer zone on the configuration is able to reduce gravity loading on the pit wall thus able to reduce the shear stress along the wall. When the pit wall angle is increased, the tensile stress around toe of pit wall due to increase in self-load of the wall. The study also shows that the stability is changed when the quality of friction angle and cohesion is changed. However, a changing of friction angle will give more impact than that of cohesion. Based on the study result, a proper configuration can be designed for any conditions of rock quality particularly friction angle and cohesion.

Chapter 6: This chapter discussed about optimization the fleet efficiency and utilization through a detailed haulage analysis, and to identify any potentioal cost savings available within the dumping operation of Erdent open pit mine. The haulage and dumping aspect of peint pit mining operations is one of the largest cost components of the mining cost constituing approximaterly 50-60% of mining operational costs. The overall aim of correct dump design is to plan a series of wste disposal stages that will effectively mininmize the vertical and horizontal distances (buffer zone) between the pit and potential waste dump site that has been discussed at Chapter 5. Two haulage distance i.e. 100 m and 200 m buffer length are considered. The study shows that the estimation of dumping cost analyses 'S to All' direction of d200 scenario is covers the maximum Net present cost (NPC) and the both directions of d100 scenario are getting minimum NPC and DCPT values. And the result differences between the d200, ‘S to All’ and d100, ‘N to All’ is totally 233.7 mil.$ of NPC (846.4 mil.$ of undiscounted cash flows) and 0.2$ of DCPT.

Chapter 7: This chapter concludes the results of this research.

Acknowledgements

Deep appreciation and sincere gratitude are expressed to Associate Professor Dr. Takashi SASAOKA, my supervisor from the Laboratory of Rock Engineering and Mining Machinery, Kyushu University, for his invaluable guidance, encouragement and support throughout the research. His expertise on mining and geological engineering has been invaluable and many inspiring discussions with him have been of great value to me. Fortunately, this dissertation is achieved with his strong supports and valuable comments. His willingness to give his time has been very much appreciated.

I would like to extend my deep gratitude to Professor Dr. Hideki SHIMADA, Head of the Laboratory of Rock Engineering and Mining Machinery, Kyushu University, for his guidance, precious and constructive suggestions to improve this research. His encouragement is an important motivation for me to pursue the academic study.

Sincere gratitude also goes to the member of my examination committee, Professor Dr. Yasuhiro MITANI, Department of Civil Engineering, Kyushu University, for raising a number of interesting points for discussion. His valuable comments toward improving my work are very much appreciated.

Special appreciation and gratitude are given to Mr. Suichi Fujita, Assistant Professor Dr. Sugeng WAHYUDI, Assistant Professor Dr. Akihiro HAMANAKA and all friends from the Laboratory of Rock Engineering and Mining Machinery, Kyushu University, for their enthusiastic and grateful help, encouragements and friendships.

Sincerely appreciation is also given to my father, Professor Dr. Sodnom TSEDENDORJ, Head of Mining Research and Design Center, Mongolian University of Science and Technology, for his boundless love, guidance and support for me. He has walked me through all the stages of the writing of this thesis.

Invaluable supports and help during my research and dissertation writing from all of you will keep in my mind.

Finally, special and deepest thanks go to my family for their endless love, support, great patience and encouragement throughout my whole life.

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August 2018

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CHAPTER I. INTRODUCTION

1.1. Mining industry of Mongolia

Mongolia is a Central Asian country neighboring with the Russian Federation and the People's Republic of China. The country has a total territory of 1,565,600 km^2. The country shares 4,673 km border with China on its east, west and south, and 3,485 km border with Russia in the north. The population of Mongolia is about 3 million people, one of the countries with the lowest population density in the world. Almost half of the population lives in Ulaanbaatar, the capital of Mongolia. Other major cities include Darkhan, an industrial center near the Russian border in the north, and Erdenet, known with its large copper plant, also in the north. Some 40% of the population lives in the countryside, primarily subsisting as nomadic livestock herders, while the rest lives in the cities or small settlements spread throughout the country.

Figure 1-1 Geographical location of Mongolia

The latitude of Mongolia, between 42 and 52 degrees north, is roughly the same as that of Central Europe or the northern United States and southern Canada. However, because the country is landlocked and has a relatively high median altitude, the climate is characterized as extreme continental with large temperature fluctuations and low precipitation. Total annual rainfall in Ulaanbaatar averages 220 mm. Much of the precipitation falls during the short summer, while winter is generally dry and extremely cold. The average summer temperature is about 25°C, while winter temperature is average at -25°C (MRPAM 2009).

The mining sector in Mongolia is one of the largest financial contributors to the economy. The primary outputs of the industry are copper and gold, along with iron ore, molybdenum and fluorspar. Mongolia is also estimated to have potential reserves of 125 billion metric tons of coal reserves, and the current coal production is over 20 million metric tons per year, mostly exported to China. Mining sector accounts for approximately 20% of gross domestic product (GDP), approximately 65% to the industrial output and more than 85% of exports. Between 1996 and 2006, the share of GDP produced by the industrial sector, including mining, went up from 20.6% to 40.3%. Since the 1970s the Erdenet mine and concentration, a joint Mongolian-Russian venture, has been the major source of Mongolian export income accounting for more than 10% of GDP.

The Mongolian government has promoted a range of policies to attract foreign investment in the minerals sector, especially since the late 1990s, which resulted in a significant expansion of mineral exploration and mining. The giant Oyu Tolgoi copper and gold deposit was discovered in the Mongolian Gobi in the early 2000s, offering the country a window of opportunity to attract investors and developers. The Oyu Tolgoi project is three to four times bigger than Erdenet and expected to account for more than one third of GDP by 2020.

As of 2012, the total 3,208 explorations and mining licences are owned by approximately 1,000 mining companies. However, only about 30 of them have a total annual revenue more than USD 4 million. The number of people working in the formal mining sector has increased three times since 2001, and currently, there are 48,000 Mongolian and foreign employees.

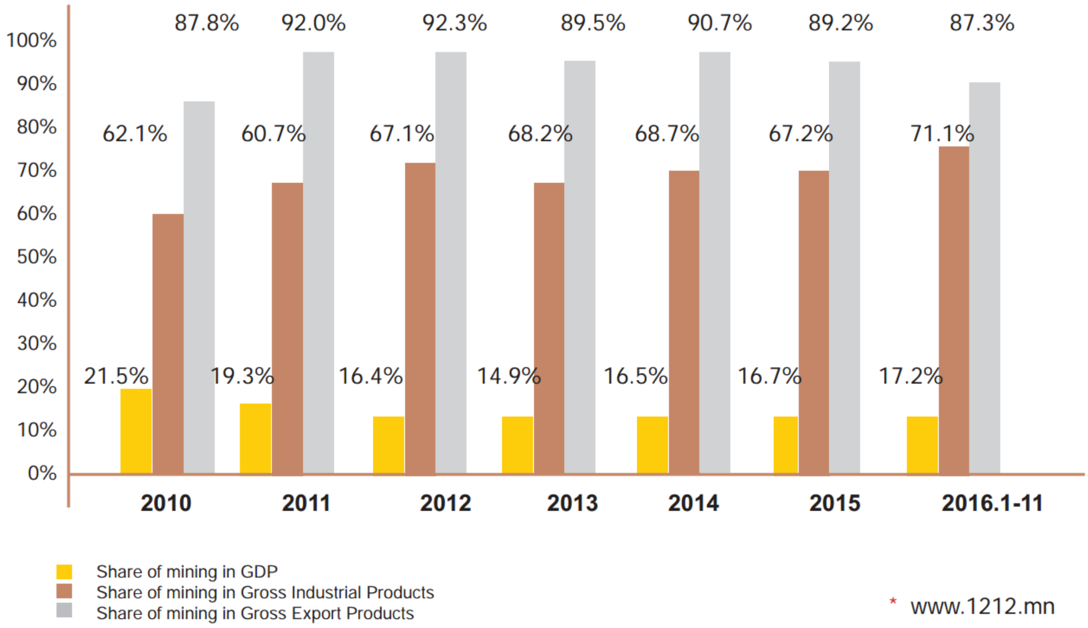

Figure 1-2 Share of mining sector in country's GDP, Industrial and Export Product (Mongolian Statistical Information Center, www.1212.mn, 2016)

Mongolian economy is heavily dependent on the mineral exports. The share of mineral exports is 88% out of total exports that amounts to almost one-third of the government revenues and contributes 20% to the country's GDP.

The exchange rate volatility is highly affected by the world's copper and gold price movement. When copper price reached USD 10,000 per ton and copper price USD 300 in 2011, Mongolian economic growth reached 17.3 percent a year, and was considered the fastest growing economy in the world. However, the economy plummeted in 2015 at 2.3 percent, following the drop in minerals price and it is estimated that the economic growth will be around 0.3% at the end of 2016. (MRPAM 2016).

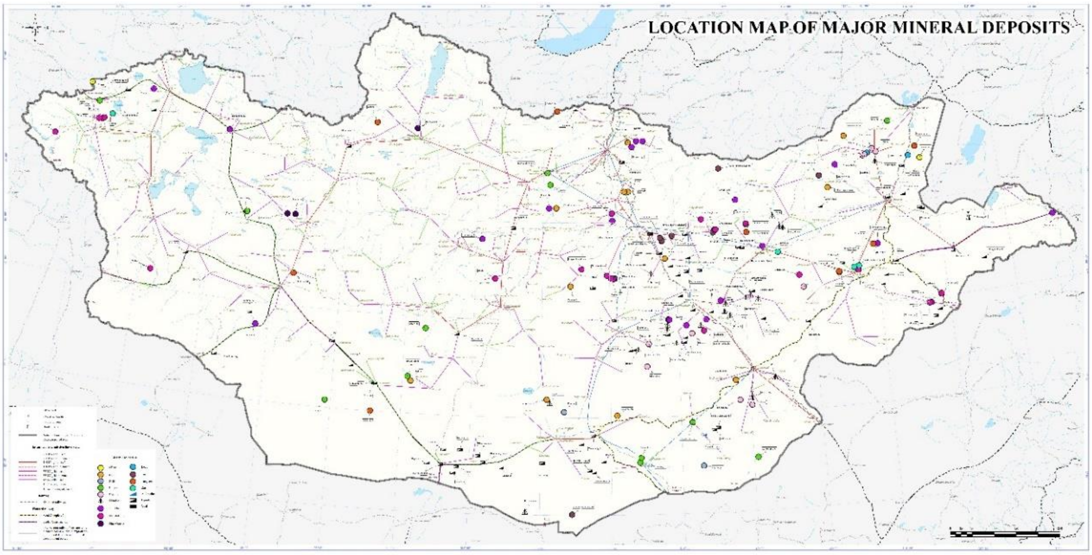

Figure 1-3 Location map of major mineral deposits in Mongolia (MRAM 2016)

In 1924, coal mining was the sole mining industry in Mongolia, which is the main supplier to electric power plants of Ulaanbaatar-4. In the 1961 a major iron ore metallurgical plant was established in Darkhan province and in 1978, a copper and molybdenum processing factory was established in Erdenet province. In the late 1980s, several coal mining sites were founded to supply the Power Plants of Ulaanbaatar. Meantime, limestone factories started their processing activities, becoming a major cement production center. Along with this, gold mining has been growing steady (Batchuluun and Lin, 2010).

Prior to 1992 and the shift to a market economy, the mining industry was dominated by state owned, Mongolian-Soviet joint ventures with Russia, Bulgaria, Czechoslovakia, East Germany and Hungary. These ventures focused mostly on coal, fluorspar, gold, tin, copper, and molybdenum. Since 1992, the number of local Mongolian companies, joint companies with Canadian, British, Australian, Russian, and Chinese companies involved in exploration and mining has increased (The World Bank, 2006). At the end of 2015, about 23 major mines were in operation. Apart from the major mines, about 423 small scale mines operating as partnerships were active. (MPRAM 2017)

Table 1.1. Production and export of major mining products (NSO, 2016).

Commodity	unit	Year	Production volume	Export volume
Copper concentrate	thous.t	2012	517.9	574.3
		2013	803.0	679.8
		2014	1,080.4	1,378.1
		2015	1,334.7	1,477.8
Molybdenum concentrate	thous.t	2012	4.0	4.3
		2013	3.7	4.0
		2014	4.0	4.0
		2015	5.2	5.0
Coal (thermal and cocking)	thous.t	2012	28,561.0	20,915.5
		2013	29,163.6	18,373.1
		2014	24,927.1	19,499.0
		2015	24,148.9	14,472.7
Iron ore and concentrate	thous.t	2012	12,112.1	6,415.9
		2013	11,135.9	6,724.5
		2014	10,260.5	6,324.4
		2015	6,173.4	5,065.1
Gold	kg	2012	5,995.0	2.8
		2013	8,904.4	7.6
		2014	11,503.8	10
		2015	14,556.2	11.3
Zinc ore and concentrate	thous.t	2012	119.1	140.9
		2013	104.1	130.9
		2014	93.2	99.4
		2015	89.6	84.1

1.2. Problem descriptions of large-scale open pit mines in Mongolia

The projects of mineral resources operating by open pit mining method in Mongolia since 1960s. A relative deeper and large scale open pit mines developed in last 10 years and which are mainly operated in copper ore and hard coal mines. The depth of open pit mines was mainly less than 150m before 2000’s. However, the depth and scale of open pit mines has been increasing with the mining capacity, mineral products and high economic growth in major projects. For example, Oyutolgoi open pit copper mine with depth of 400m (in further 760m), Erdenet open pit copper-molybdenum mine with depth of 495m (in further 700m) and Narynsukhait and Tavantolgoi hard-coal mines with depth of 150m-250m (in further 350m-400m) in mining operation currently.

The most of deposits were explored and the feasibility studies were developed before 1990s. The main problems occurred in large-scale open pit mines in Mongolia which established before the 1990s, are optimization of open pit mine design, locations of waste dump and other surface structures due to feasibility studies and other studies developed at different politics and economic situations, the estimations of price increases of mineral products and future growth of resources are not proper to current economic and technical conditions. Furthermore, geotechnical and other base investigations and data are insufficient in many deposits it is the main problems for mining planning such as open pit optimizing, slope stability studies and appropriate large waste dump design etc. at most large scale open pit mines.

In order to develop an appropriate open pit mine and waste dump design, these investigations have been conducted. In addition, the geotechnical and economic data has to be collected and optimizing to stable and economical design of open pit and waste dumping area are urgent issues for further large-scale open pit mine development were discussed.

1.3. General background

Mining is the extraction of valuable minerals or other geological materials from the earth from an orebody, vein, seam, or reef, which forms the mineralized package of economic interest to the miner. Mineral deposits are generally extracted from the either by underground or surface mining methods with the objective of extracting the ore at a profit.

Open pit mining is one of the most important methods of surface mining in which any waste material or overburden is stripped and transported to a waste dump prior to, and sometimes during, mining in order to uncover, and gain access to the mineral deposit. In general, mining proceeds from the top to the bottom of the orebody. Both stripping and mining are carried out in a series of horizontal layers, usually of uniform thickness, called benches. The choice between an underground mining and a surface mining method depends on the depth, grade and tonnage of the orebody and consequently on technical and economic criteria. Surface mining methods in the form of quarries or open pits are extensively used throughout the world to extract ore at or near the surface. Mineral deposits at depth may be extracted by underground mining methods but in general, significantly higher grades are required for profitability as the depth increases.

The development and extraction of ore by open pit mining is a complex operation that may extend over several decades and require very large investments. Before extraction, it is necessary to determine the size and final shape of the pit and the waste dump at the end of its life. This final shape, or ultimate pit limit and the final dump shape represents the volume beyond which further extraction of ore and waste and dumping using current or assumed economic and technical parameters, is uneconomic.

The ultimate pit limit determines minable reserves and the total amount of waste to be removed. It is also used to determine locations for the waste dump, and surface infrastructure (such as, processing plants and access roads) and to develop a production program. Determination of the ultimate pit limit is one of the most important design considerations in open pit mining and it may be recalculated many times during the life of the mine as production prices, costs, technical considerations and geology change. There are a number of factors which affect the size and shape of the ultimate pit. These include, geology, grade, topography, production rate, bench height, pit slopes, mining and processing costs, metal recovery, marketing and cut-off grade. Some of these factors are discussed below:

Cut-off grade. The open pit mining method is usually used for low-grade deposits in which the ore is not contained within well-defined geological boundaries. In such deposits ore and waste are defined by a cut-off grade as opposed to a geological boundary. This cut-off grade is a very important factor in mine planning as it determines the overall ore reserve and the physical location of ore as well as the amount of waste to, be removed. It is a complex function of many variables such as grade, price, pit slopes, size of mining (selection) equipment, and mining and processing costs. As the cut-off grade increases, the tonnage of ore above the cut-off decreases and its average grade increases. Up to a certain point the quantity of metal contained in the ore above the cut-off grade (product of tonnage and average grade) will remain constant. Beyond this point (and certainly for cut-off grades at or above the mean grade of the orebody) the quantity of contained metal, and hence the profit, will decline.

There are several commonly used cut-off grades. The break-even cut-off grade is one of the simplest and most widely used. This is the grade at which the, recoverable revenue is exactly equal to the cost of mining, processing and marketing. Cut-off grades may be classified as either planning or operating cut-off grades depending on the time scales to which they refer. Planning cut-off grades are usually used to define geological or minable reserves before the start of operations or for long periods of time. Operating cut-off grades are usually used during the operation to make short-term to medium-term decisions, e.g. to mine or not, to stockpile or process.

Stripping ratios. The stripping ratio is another important factor in open pit mining as it has a major bearing on profitability, scheduling and pit design. The stripping ratio is ratio of the amount of waste that must be removed in order to mine a unit quantity of ore and is usually expressed as:

$$SR = \frac{Waste\ (amount)}{Ore\ (amount)}$$

The break-even stripping ratio (BESR) is calculated from the following equation:

$$BESR = \frac{R - C}{W}$$

where:

R is the revenue per tons of ore

C is the production cost of per tons of ore

W is the stripping cost per tons of waste

Bench geometry. In open pit mining, the extraction of ore and the stripping of waste material are done in a series of layers called benches. Each bench has an upper and lower horizontal level separated by a distance equal to the bench height. The exposed subvertical surfaces are called the bench faces. The bench faces described by the toe, the crest and the face angle. The bench height depends on the types of extract materials, the manner in which it is dispersed in the host rock, the size and type of equipment used to extract, the blasting method, the production rate and the geotechnical characteristics of the rock. The height is usually set as high as possible with regard to the size and type of equipment selected for the operation.

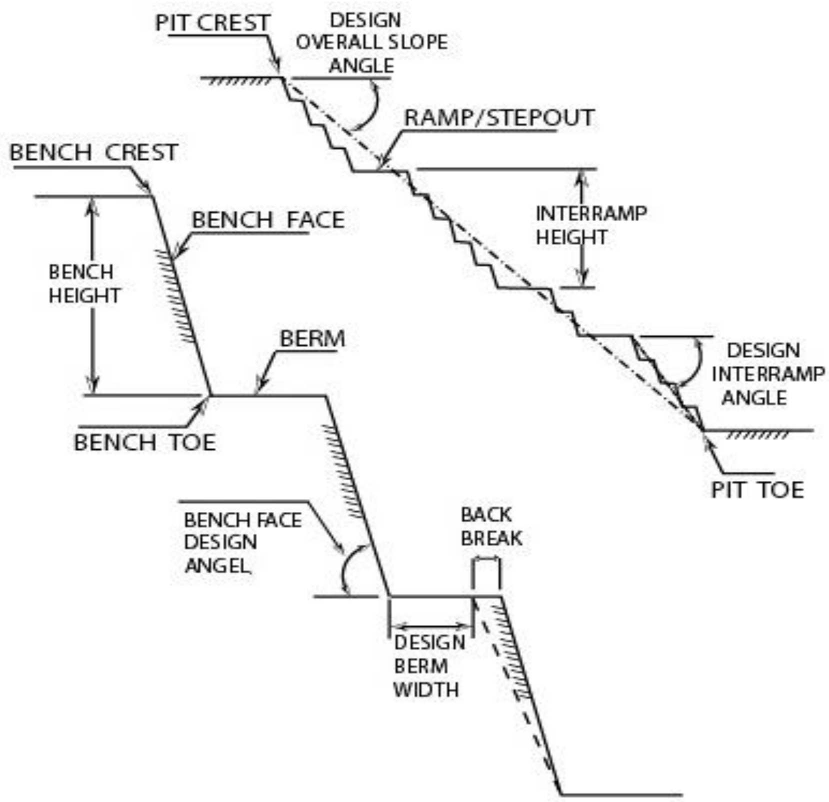

Figure 1-4 Open pit slope geometry parameters

Pit slopes. The pit slope is a major factor affecting the size and shape of an open pit. Stability is an over-riding consideration as pit walls must remain stable during the life of the mine. The steeper the final slope can be designed the smaller the amount of waste that has to be removed. However, as slopes become steeper the probability of failure increases. An optimum mine plan should, therefore, have the steepest final pit limit commensurate with stability throughout the period of mining activity.

Pit slope angle, ultimate pit slope or overall slope angle which is the angle between the horizontal and the line connecting the toe of the lowest bench to the crest of the uppermost bench. This slope makes no allowance for safety berms and haul roads.

Average pit slope angle or average ultimate pit slope which is the angle between the horizontal and the line connecting the toe of the lowest bench to the crest of the uppermost bench allowing for the haul road or access ramp and safety berms. Working pit slope angle which is the slope of the pit wall during the mining operation. This is usually smaller than the ultimate pit slope angle so as to ensure stability and provide a wider space for operation. The working pit slope increases during the mining operation until it reaches the pit slope angle at the end of mining.

In general, the overall slope is designed to be as steep as possible in order to reduce the stripping ratio. Pit slope angles may vary through the orebody and may vary with direction due to changes in geological structure and stability requirements. They may also vary with elevation. Any realistic method of pit design, pit slopes may vary though a deposit due to changes in lithology and geological structure take into account variable slope angles. The determination of pit slopes is essential and must be done before planning the pit limit. These are determined mainly by slope stability methods from geotechnical information gathered during the site investigation.

Dumping management. The largescale open pit mines move thousands of tons of material daily from the loading sources to the destination zones, whether these are massive mine dumps or, to a lesser extent, to the grinding mills. Mine dumps can be classified as leach or waste dumps, depending upon their economic viability to be processed in-place, a condition that has experienced great progress in the last decades and has reconfigured the open pit haulage network with an increase in the number of dumps.

In designing the dump, there are many ways to assign values and combine the different geometric and size parameters while respecting the safety, economic and environmental constraints. The total tonnage capacity required can have as many geometrical representations as its limitations allow. In this situation, building a mathematical optimization model is the best option to interrelate certain key variables and the first approach to calculating the values that seek to maximize the satisfaction of a linear programming objective. As most of the dumps are emplaced on irregular topographies, a second approach has to contrast the values got by the generalized model and correct them, if necessary, by a series of successive iterations and projections to the field.

The truck-shovel mining operation generally constitutes approximately 50% to 60% of the total open pit mining cost (Nel, *et al.*, 2011). The typical truck-shovel operation encompasses a loading unit which extracts/handles material, and a haulage unit for material transport. The truck-shovel mining method is popular due to increased flexibility and is capable of consistent high productivity in surface mining operations. The continual improvements and technological development of truck-shovel operations has resulted in increased utilization of this extraction method within the mining industry. The main benefit of running a truck-shovel operation is the improved flexibility of the mining system, resulting in better suitability and selective mining of complex ore deposits, varying ore depths, varying overburden thicknesses and is not restricted to ore deposit size. An additional benefit is that the initial capital investment required to employ a truck-shovel operation can be lower compared to other methods depending on operational scale, however truck-shovel includes higher operating costs per bank cubic m (Mitra and Saydam, 2012).

The transport and dumping of waste material is a substantial cost component of truck-shovel operations. Therefore a significant amount of research has been conducted regarding the optimization of waste dump design standards and implementation aspects. However, the time constraints associated with the rapid expansion of open pit mining operations does not allow for target waste dump design optimization at specific locations. Hence a risk factor or compromise is generally associated with the development and costs of waste dump implementation. The ideal mining operation consists of optimal fleet productivity, utilization and availability, along with the lowest attainable mining cost.

Unfortunately, systematic studies on interration between appropirate design, optimization of open pit and waste dumping were limited especially in large scale and long lifetime mining projects in Mongolia. The code for design of open pit and dumping of mining industry in Mongolia mentioned only about how to select dumping area, how to set up the parameters of dumping operation, and for stability of dumping area.

However, open pit coal mines do not pay enough attention to development and planning for dumping area. For example, in Erdenet Cu-Mo open pit mining, the development of waste dumping were dumped on the further's pit boundary and then, last 5 years re-located over 15 M.m3 of waste with 40 M.$ of costs and this is not only example in Mongolian minings. Therefore, the systematic research on the optimization of large scale open pit mining and appropriate design of waste dumping is valuable and urgent.

1.4. Literature review

The procedure, open pit mine designing and planning that can be started just after ultimate pit determination and cut-off grade calculation which both of them directly depend on final product price of the mine. The ultimate limits of an open pit define its size and shape at the end of the mine's life. In addition to defining total minable reserves and determining total profitability, these limits are needed to locate the waste dump, processing plant and other facilities. They are also required for the design of overall production schedules within the planned pit shape. There are numbers of algorithms have been developed to determine the optimal pit shape/boundary all with a common objective: to maximize the overall mining profit within the designed pit limit. This chapter presents a literature review and survey of the previous work including an assessment of the methods for optimal pit design together with the methods of slope design used in open pit mining.

Optimization is a scientific approach to decision making through the application of mathematical methods and the use of modern computing technology. It concerns the maximization or minimization of an objective function, e.g. maximization of profit or minimization of cost, subject to a set of constraints being imposed by the nature of the problem under study (Francisco, 2010).

The objective of optimal open pit design methods/algorithms is to determine the ultimate pit shape/boundary for an ore body together with the associated grade and tonnage that optimize some specified economic and/or technical criteria whilst satisfying practical operational constraints. The most common criteria used in optimization are: maximum net profit, maximum net present value, maximum metal/mineral content and optimal mine life. Many attempts have been made to devise a general theory of cut-off grades within the context of which an optimal sequence of cut-off grades can be defined and, in practice, determined, for the life of a mine. The most advanced approach is that of Lane (1964 and 1988) which is based on the assumption that there are three stages in the mining operation comprising mining, concentrating or processing, and refinery and/or marketing. Each stage has its own associated costs and a certain capacity. (Khalokakaie, 1999) The effective optimum cut-off grade is, the middle value of the three optimum cut-off grades. Lane's method is regarded as a landmark in the determination of optimum cut-off grades. His method, however, relies on the assumption that prices, costs and recovery remain constant throughout the operation. Dowd (1973), Dowd and Xu (1995) and Whittle and Wharton (1995) have coded Lane's method into a computer program.

Optimizers generate optimal solutions assuming that the data given as input is correct, which is not the case in mining due to the uncertainty around the economic value of a block. Conventional approaches to the optimal design of open pit mines, do not incorporate uncertainty into the process because they make use of a single estimated orebody model generated through kriging (Goovaerts, 1997) as input to the optimization model. Past efforts in the area of conventional approaches are Johnson (1968), Dagadalen and Johnson (1986) and Hochbaum (2001). Dimitrakopoulos (1998) highlights that due to the smoothing effect present in any estimated type orebody model, as in the case of a kriged model, the histogram and variogram show lower variability than the actual data which leads to not meeting production targets and NPV forecasts. Dimitrakopoulos et al. (2002) discuss the effect of estimated orebody models on non-linear transfer functions used to schedule production throughout the whole life of mine and the risk that arises from not accounting for geological uncertainty.

The Lerch and Grossman algorithm. Lerch and Grossmann (1965) introduced an efficient algorithm to find the optimal pit limits algorithm, which is based on three-dimensional graph theory, is the most commonly used optimization algorithm which takes into account the influence of a grade block model, operating costs, product prices, slope geometry, etc. Then, within this framework scheduling is carried out by breaking the pit space into pushbacks. Pushbacks are represented by a set of connected blocks that facilitate the mining operation in terms of safety slope requirements, minimum working width required by mining equipment and maximization of the NPV of the project through the adequate management of stripping ratios.

Since then many algorithms have been developed for determining optimal pit outlines. Some authors, namely, Kim (1978), Dowd and Onur (1993), Gill, Robey and Caelli (1996) have provided surveys and comparative studies of these methods. Kim classified the various methods of optimal design as "rigorous" and "heuristic" techniques. He used the word "rigorous" for the methods that have mathematical proofs such as graph theory and dynamic programming (Khalokakaie, 1999).

The Lerch-Grossman algorithm, it is also used in mining optimization software as the industry standard, for example in Gemcom's Whittle software, to find the optimal pit and pushbacks. The algorithm uses different revenue factors to generate a value-based mining sequence strategy to design pit shells. Early pit shells are constructed using high-grade blocks and a low stripping ratio.

The results also consider practical considerations such as haul road access, cut-off grades and processing, etc. To maximize the use of block modelling functions and optimize the pit design process, block modelling and slope stability analysis have to be fully integrated. This is a logical extension to assign mines rock types and grades to every block. This process will be further optimized by defining every block location especially those blocks with high value of NPV. The Lerch and Grossmann algorithm is based on two theorems:

1. The maximum closure of a normalized tree is the set of that tree's strong vertices
2. A normalized tree can be found such that the set of strong vertices in this tree constitutes a closure of the graph so the set of strong vertices is the maximum closure of the graph with the highest NPV.

This method converts the revenue block model of the deposit into a directed graph which is a simple diagram consisting of a set of small circles, called nodes or vertices, and a set of connecting arcs (lines with direction) used to indicate the relationship between the vertices. A vertex represents each block. Each vertex is assigned a mass that is equal to the net value of the corresponding block. Vertices are connected by arcs in such way as to represent the mining constraints. These arcs indicate which blocks should be removed before a particular block can be mined.

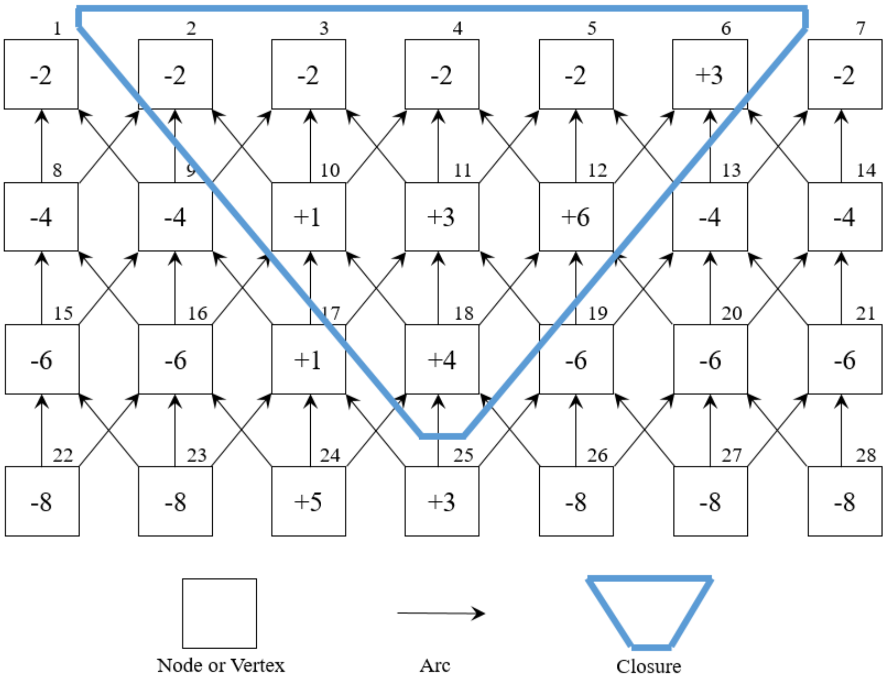

Figure 1-5 Directed graph Lerch-Grossman

Computer based methods. A 3D program called GEOVIA Whittle™, introduced by Whittle (1985), was a computer based implementation of the Lerch and Grossmann method which used a block model, whose blocks have economic values representing the net cash flow that result from mining the block in isolation. However, the resulting optimal pit did not use discounted cash flows.

The Floating Cone method, which is the simplest and fastest technique to determine optimum ultimate pit limits to which variable slope angle can be easily applied, repeatedly searches for and checks the total value of block groups forming inverted cones. Total cones are identified for mining if their total value was positive. This procedure is iterated until no more positive cones are recognised. However, this method cannot guarantee the final pit is optimum. Other block groups (as mentioned above) also implemented a two and-a-half dimensional Lerch and Grossmann algorithm (Dimitrakopoulos et al., 2002; Osanloo et al., 2008a; Asad and Dimitrakopoulos, 2013).

The 4D (and subsequently Four-X) programs also use the same Lerch and Grossmann technique to generate a set of nested optimal pits. Each pit that is optimal is used to guide different mining schedules. Financial analysis of these programs which consider discounted cash flows allows selection and sensitivity analysis of the best pit (Dowd, 1994; NPV – Scheduler, 2001; Osanloo et al., 2008a; Askari-Nasab et al., 2011)

Dump design methodology. Three major destination groups, characterized by a cut-off grade criteria and ore type, represent the places in the mine where the material receives specific treatment after its delivery from the pit: leach dumps, waste dumps and mill (Hustrulid, Kuchta, & Martin, 2013). Dump leaching facilities are built to receive and treat low-grade ore by the use of solution agents, while waste rock dumps store uneconomic material. Dump leaching technologies have developed over the last decades, allowing the mining industry to build larger and higher dumps faster than ever (Smith, 2002), since they have proven to be an efficient method of treating oxide and sulfide ores, an attractive way to treat large low-grade deposits (Dorey, Van Zyl, & Kiel, 1988). As a result, an increase in the number of dumps, which are the most visual landforms left after mining (Hekmat, Osanloo, & Shirazi, 2008) has reconfigured the open pit mines network organization and landscape.

1.5. The objective of research project

A relative deeper open pit mines developed in last 10 years in Mongolia. However, geotechnical investigations, data and practical experience are insufficient in many activity deeper open pit mining and this is main cause of geotechnical, planning, operating and economic problems occurred at some open pit mines. A case point is the Erdenet Cu-Mo open pit mine in northern Mongolia, which currently faces the design of overall slope angles for current pit depth around 300 meters and to final mining depth around another 300 meters. The design problem and the lack knowlegde regarding the behavior of relative deeper pit design for optimzation of open pit and dumping area was the starting point for this work.

The objective of the work presented in this thesis was attempt to develop an innovative method to optimization of appropriate design of open pit and dumping using geotechnical, geochemical and economic data and its at one of major mining project, Erdenet Cu-Mo porphyry deposit.

1.6. Study approach

The steps of studies to approach the solution for the objective of study were commenced from the difficulties of planning, stability, operational and economics of large scale open pit mine and development of waste rock dumping site to understand their optimization relations. Subsequently, the optimization and determining ultimate pit limit and further prospects pit limits in large scale open pit mine and their roles in development and planning of waste dumping were conducted, following by the engineering application of the findings. Technically, the study was conducted by performing data processing i.e. geological, geotechnical, economical and mining operational, and by numerical analyzing of the data at obtained from the field.

1.7. Organization of the thesis

This dissertation consists of 7 (seven) chapters in which the contents are described as follow:

Chapter 1 explains the information about the country and mining sector in Mongolia, problem descriptions of large scale open pit mines and the general background of studies including the processes of optimization procedures of open pit mine and dumping, problem description and literature review and survey of previous studies, objective and outline of dissertation are proposed.

Chapter 2 discusses the information about the country and mining sector in Mongolia, general description of literature review and survey of previous research, objective and outline of dissertation are proposed.

Chapter 3 discusses the optimization procedures for dynamic open pit boundary shells based at case investigation of Erdenet mining and numerical simulation by Surpac, Whittle, Slide and Dips software using engineering, geotechnical properties and economic data.

Chapter 4 discusses the formation mechanism of benches on dumping area. It consists of a field investigation at Erdenet open pit mine, experiment methods and results, and simulation for the effect of dumping operation on stability of dumping area

Chapter 5 discusses the optimization and consideration of buffer zone distance between the large scale open pit and dumping area consists of failure effects and stability consideration.

Chapter 6 presents a methodology to model and optimize the design of a mine dump by minimizing the total haulage costs. The methodology consists on optimization of a dump model based on the multiple relevant parameters and solves by minimizing the total cost using programming and determines a preliminary dumping design.

Chapter 7 concludes the results of this study, including the recommendations for dumping management in largescale open pit mining optimization and planning.

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CHAPTER II. SITE DESCRIPTION: NORTHWEST SECTION OF ERDENETYN-OVOO CU-MO DEPOSIT

2.1. Location and regional information

The western Erdenet area is located in the Bulgan District and Erdenet city of the northern-central part of Mongolia, about 350 km northwest of the capital city Ulaanbaatar. The total area of the project area is 5,500 km^2.

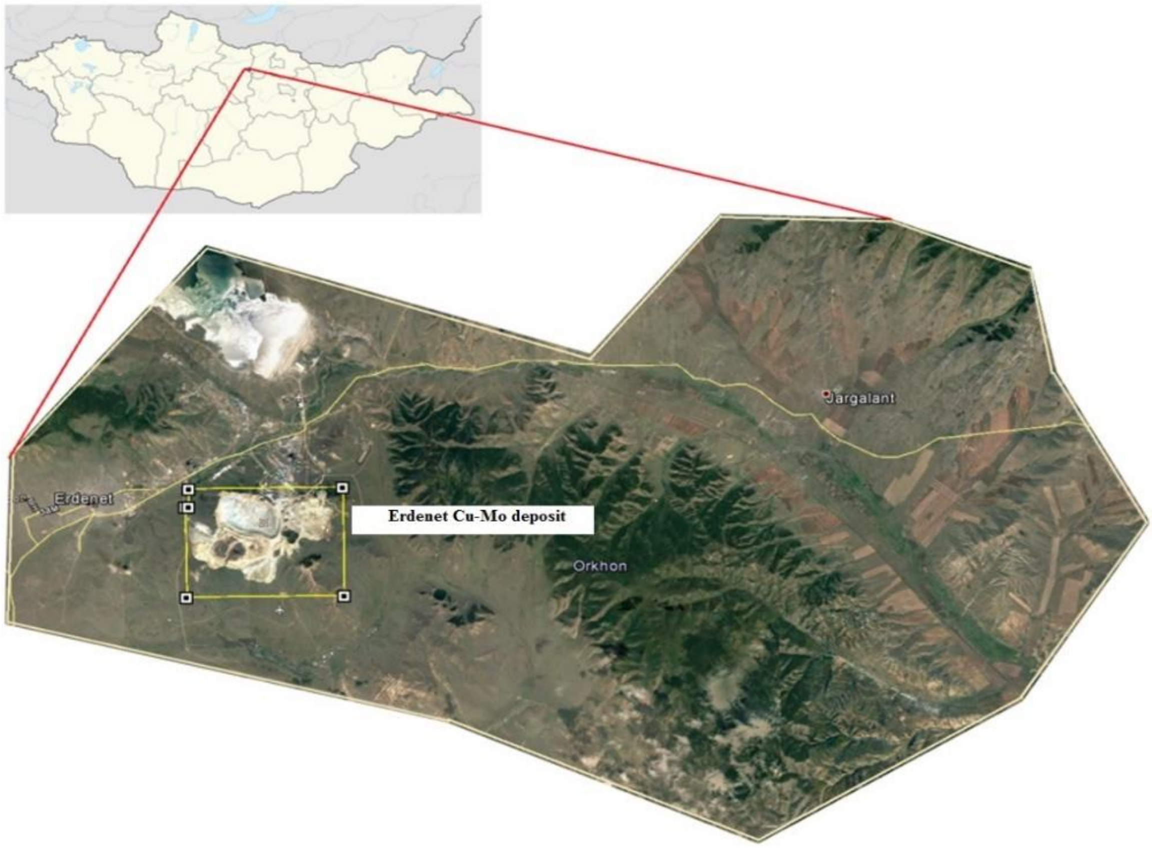

Figure 2-1 Location map of Erdenetyn Ovoo Cu-Mo deposit

2.1.1. Information of Erdenet city, region.

Erdenet city was founded for the Erdenet mining company (EMC) with 7,800 population, today it is over 100,000 and this number is the second place after Ulaanbaatar city. In comparison with the situation in Mongolia, the region is considered to be economically more developed. EMC of has been exploited in NW section of Erdenetyn-Ovoo deposit in 1978 and produces copper and molybdenum concentrate. Oxidized ore developer Erdmin factory was established in 1997 to produce cathode copper and the factory producing 3,600 tons of catode pure copper annually.

EMC and other industries in this region have fully provided infrastructure facilities and connected to Ulaanbaatar, Irkutsk and Darkhan stations by railway. There is a good supply of labor. Engineers and technicians of EMC and other industries of the region are trained and studied at the Erdenet Technology School and the Mining Engineering School at the Mongolian University of Science and Technology. Workers are trained at the training center of the Erdenet Technology School. There are two power plants operating in the city and the plant, while there are high voltage transmission lines connected to Erdenet-Darkhan and Erdenet-Gusiniozersk and the area's energy base is in full compliance with the needs of the region.

2.1.2. Climate

Local climate is extreme continental climate of motherland, average January temperature -14^0 to -16^0, July temperatures $+15^0$ to $+18^0$. During the last 15 years, annual precipitation is 241-599 mm, annual average is 370 mm, of which 86% falls during summer. The average thickness of the snow cover is 5cm and the maximum is 25cm. Seasonal frostbacks are relatively high, especially in the backdrop of elevated areas, depth of 4 m to 10m. Directions and varies of the wind are different depending on seasons, but mostly wind from the NW.

2.1.3. Topography and Drainage System.

The topography is generally gentle hillside and flat grassland with the elevation of 1,000m to 2,000m. The land is covered by plain and forest and outcrops of rocks are very few. As main rivers in the area, there is the Okhon River running northeast in the southern part, and the Selenge River running east in the northern part. The prominent ridges and valleys trending NS to NW-SE develop with crossing these rivers.

2.2. Geological conditions of Erdenetyn-Ovoo deposit

2.2.1. Regional geology

The major part of Mongolian territory stretches over Central Asia Orogenic Belt (CAOB) bounded by three cratons formed at the end of the Archean eon. Siberian craton in the north and Tarim; Sino-Korean cratons in the south. Conventionally (from Amantov et al., 1970, cited in Badarch et al., 2002), the territory of Mongolia is subdivided into two domains: *I)* the northern Precambrian- ower Paleozoic domain of dispersed outcrops of metamorphic rocks, ophiolites, volcanic and volcanoclastic sediments, *II)* the southern Lower-Middle Paleozoic domain with rocks related to volcanic activity along the active marginal arc. A regional topographic and structural border between these two domains is defined by Main Mongolian Lineament separating prevailing Precambrian and Lower-Paleozoic rocks to the north from mainly Upper Paleozoic rocks to the south. To the east of both domains several granite plutons intruded and were overlapped by Jurassic and Cretaceous terrestrial volcanic and sedimentary formations (Badarch et al., 2002)

The Middle Triassic Erdenet deposit is related to a granodiorite porphyry that is either a part of or intruded the Late Permian to Early Triassic Selenge Intrusive Complex. The Selenge Intrusive Complex comprises medium grain size and equigranular plutons, characterized by biotite, hornblende and accessory zircon, apatite, magnetite, and titanite. Whereas Mikhalov and Shabalovskii (1971) regarded the Erdenet granodiorite porphyry as part of the Selenge Intrusive Complex, later authors (Matreniskii, 1977, 1981; Yashina and Matrenitskii, 1978; Gavrilova et al., 1984; Sotnikov et al., 1984, 1985; Gerel and Munkhtsengel, 2005), removed stage 3 from the Selenge Intrusive Complex and renamed the porphyry-related intrusions as the Erdenet Complex, or Porphyry association.

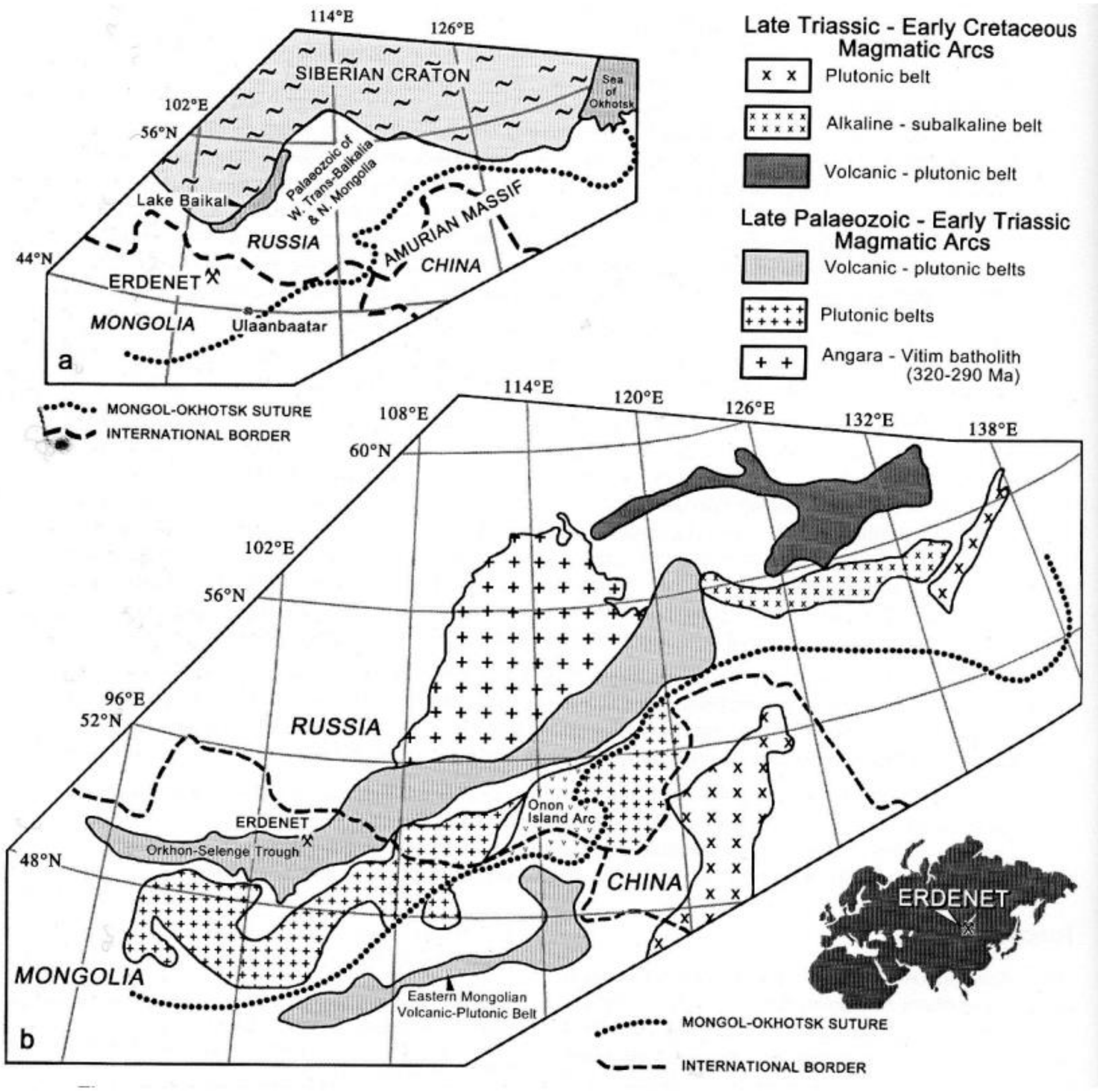

Figure 2-2 Schematic map of the Mongol-Okhotsk fold belt and the setting of the Erdenet ore deposit. a. Location and regional setting. b. Tectono-magmatic elements of the Mongol Okhotsk fold belt, northern Mongolia, northern China and southern Siberia (Gerel and Munkhtsengel – 2005).

The Porphyry association is defined by the above-mentioned authors as comprising 5 phases: diorite porphyry and microdiorite, granodiorite porphyry, plagiogranite porphyry, fine-grained granodiorite, and aplite. Although this range of rock types may be present, for simplicity, we describe the Porphyry association as mainly comprising granodiorite porphyry and porphyritic dacite dikes.

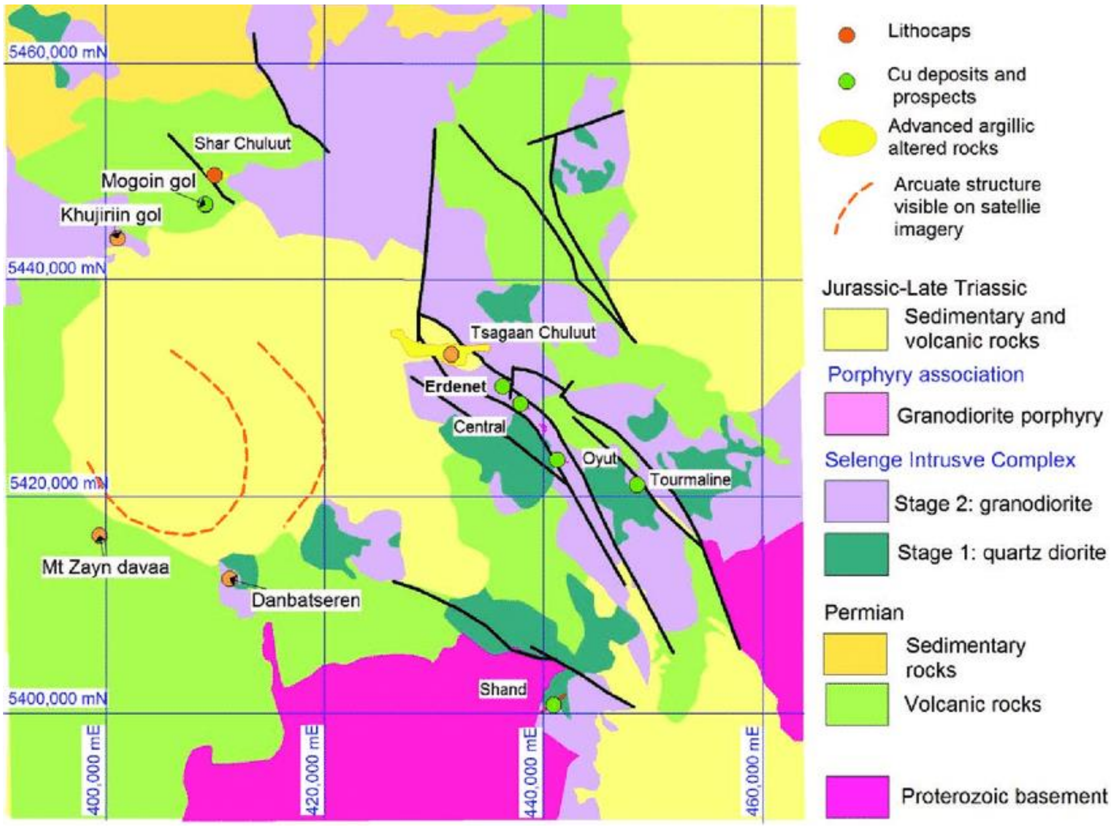

Figure 2-3 Simplified geology of the Erdenet district (Kavalieris et al.,2017)

The Selenge Complex occupies an area of 2,800 km^2 (Koval et al., 1982). It intruded volcanics in the Orkhon-Selenge trough, which extends 200 km east-northeast and is 30 to 40 km wide (Gerel and Munkhtsengel, 2005). Lower to Upper Permian bimodal, trachyandesite-rhyolite volcanics (Khanui Group) flled the Orkhon-Selenge trough, and are estimated to be up to 10 km thick. The alkalic geochemistry and bimodal character (Kepejinskas and Luchitskii, 1973, 1974) of this volcanic sequence suggest that it is rift-related.

2.2.2. Erdenet deposit geology

In the Erdenet mine district, the Selenge Intrusive Complex and major faults trend NW. Late Triassic to Jurassic volcanic and sedimentary rocks form younger basins that may cover earlier porphyry systems of similar age to Erdenet. Although Erdenet is the only known economic Cu-Mo porphyry deposit in northern Mongolia, four Cu-Mo prospects with potassic alteration (Central, Wedge, Oyut, and Tourmaline) are aligned over about 10 km strike, to the southeast of Erdenet. Porphyry Cu-Mo alteration and mineralization is also found at Shand, 24 km south of Erdenet and there are numerous zones of advanced argillic alteration in the district.

Figure 2-4 Geology map of Erdenet deposit NW section (Undrakhtamir et al., 2017)

Early interpretations of the Erdenet deposit by Gavrilova et al. (1989) envisaged a multistage intrusion with a complex geometry. The mineralized zone was referred to as a cupola of the Selenge Intrusive Complex that has been intruded by stocks with narrow root zones and dikes.

Two granodiorite porphyry stocks were described in the northwest and southeast parts of the mineralized zone, followed by porphyritic granodiorite dikes, and finally trachyandesite dikes that trend north-northeast. The southeast stock has a bulb shape, is multiphase, and was concentrically intruded by successive phases. The northwest stock is horseshoe-shaped at the surface with subvertical boundaries. The upper 300 to 400 m of the northwest stock was described as flat lying, and is connected at depth by two subvertical dikes. Several varieties of breccia have been described, comprising granodiorite porphyry clasts cemented by dacite, and granodiorite clasts cemented by plagioclase porphyry and dacite.

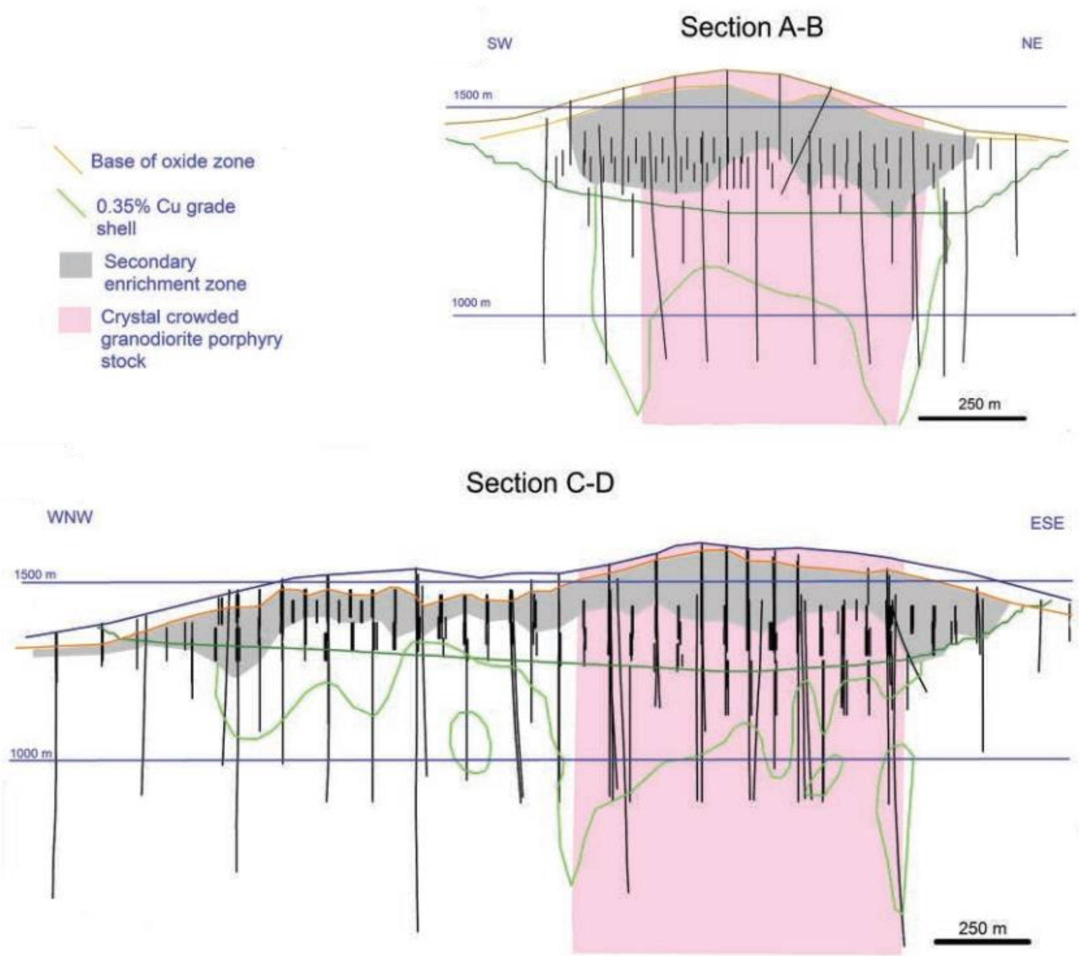

Figure 2-5 Geology sections of Erdenet deposit NW section (Undrakhtamir et al., 2017)

The wall rocks comprise medium-grained equigranular biotite granodiorite, assigned to the Selenge Intrusive Complex. The granodiorite porphyry stock is interpreted to have subvertical sides, based on the Cu grade distribution, which is geometrically similar to a molar tooth. Narrow intervals (meters) of breccia were observed in drill core, but no large zones of breccia were found in the current pit. The northwest porphyry center, which was described by Gavrilova et al. (1989), may have been based on the abundance of porphyritic micro granodiorite dikes (Khavalieris et al, 2017).

The development of the magmatic and mineralization-hydrothermal processes in the history of Erdenetyn-Ovoo deposit is evident in a number of times and many cycles, and its formulation separates the following several stages.

Pre-porphyry stages have been shaped by the North-West block structures by the intermediate breaks of western, north-west, north-west, northeast and longitudinal directions. The intrusive rocks of the Erdenet porphyry intrusion into the north-west cracked zone. It is more convenient to deep penetration of porphyry graniteite porphyry and granitei of Erdenet copper group in the "West-North" area, which is part of the central zones of the latitude, north-west and longitude and north-east corridors. A unique subwyvance structure was formed with internal structural integrations within the tectonic continuum movement.

Table 2.2. Geological exploration, reserve categories and results in Northwest section of Erdenetyn-Ovoo deposit (FS of Erdenet mine, 2017).

Exploration phases	**Res.Categories- Drilling grid elevation/cut-off grade**	**Drilling count Drilling length l/m**	**Reserve, mil.t Cu%/Mo%**
1965-1968 **Regional exploration**	C_2–500x500m -100% 1200м/0.40%	$\frac{16}{2613.6} \cdot \frac{16}{2616.6}$	$\frac{512.0}{0.72/0.008}$
1969-1970 **Pre-prospecting**	C_2–500x500m – 40% C_2–500x250m – 60% 1200m/0.40%	$\frac{10}{1992.1} \cdot \frac{26}{4605.7}$	$\frac{512.0}{0.72/0.018}$
1971-1972 **Pre-exploration**	B – 125x125 – 20% C_1–250x125 – 80% 1200m/0.40%	$\frac{44}{27113.5} \cdot \frac{70}{31719.2}$	$\frac{521.7}{0.81/0.017}$
1980-1987 **Detailed exploration**	B – 125x125 – 25% C_1–250x125 – 75% 905m 0.25%	$\frac{166}{74750.9} \cdot \frac{236}{106470.1}$	$\frac{1672.1^4}{0.55/0.015}$

2.3. Mining activity

The biggest porphyry copper-molybdenum deposit in eastern Asia exists is the Erdenet copper deposit. This deposit is composed of the Erdenet NW ore deposit, which is being mined as an open pit, the Erdenet Central deposit, the Erdenet Intermediate deposit and the Erdenet SE (Oyut) deposit, from north to south direction. The Erdenet Central deposit, the Erdenet Intermediate deposit and the Erdenet SE (Oyut) deposit already finished exploration including feasibility studies.

In 1941, the Erdenet deposits were firstly reported when the area was geological mapped by a USSR team. During 1964 and 1969 the exploration program were intensively performed by a cooperative work between the Czechslovakia and Mongolian Governments.

In 1972, it was decided that the Erdenet mine be developed in cooperation with the USSR.

In 1978, the Erdenet mine started operations with a production of about 4,000,000 tons per year. After that the production was increased to 16,000,000 tons per year in 1983 and 20,000,000 tons per year in 1989. To 1990 the copper concentrate of 30% to 32% Cu was produced 350,000 tons.

In 1995, the ores were extracted 20,900,000 tons grading 0.73% Cu and 0.02% Mo which were equivalent to 152,570 tons copper metal and 4,180 tons molybdenum metal. Cu concentrate produced was 346,300 tons Cu (Cu grade in Cu concentration is about 40%) and 3,900 tons Mo.

The deposit has been mined since its first producing 760.7 million tons of ore and 15.44 million tons of copper concentrate and 120.8 thousand tons of molybdenum concentrate and sales of $10.4 billion for the last 40 years and open pit mine currently covers an area of 2.5km x 1.5km.

From its historical momentum, the majority of the state budget was created as the main pillar of the country economic and from this time it has been allocated 5.1 trillion MNT to the state and local budgets.

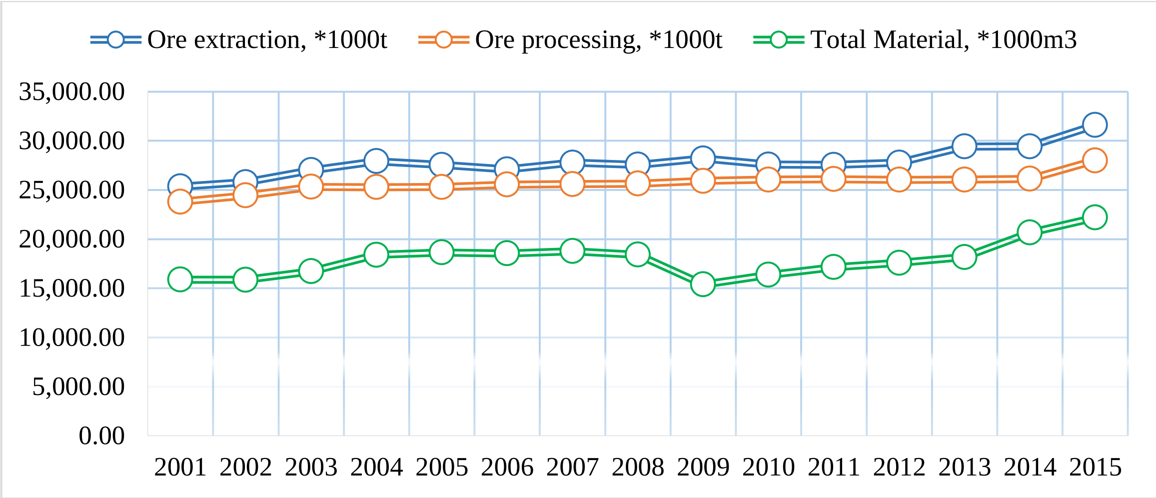

Figure 2-6 Total material movement in last 15 years (EMC, 2016)

On the basis of exploration surrounding the mine and feasibility study, ore reserves in the Erdenet Central deposit and the Erdenet Southeast (Oyut) deposit were calculated 1,250,000 tons (0.43%Cu, 0.018 %Mo) and 41,890,000 tons (0.40%Cu, 0.007%Mo) respectively.

The tenement of the Erdenet mine is limited in the vicinity of the Erdenet NW only which is currently mined and does not include other deposits such as the Central deposits etc.

2.4. Site problems related to the objectives of the research

The Erdenet Cu-Mo ore deposit is composed of a mountain-deeper structure of the stockwork. Open pit mine was developed from the mountain top at the elevation of 1,605m. At the elevation of 1,400m, the mountain part of the deposit has been completed and the mining operation of the deeper section of the mine began. At the present, the bottom of the open pit mine is 1,235m.

The first mining project in the Erdenet Cu-Mo deposit established at 1976 and the first phase of the mine operation since 1978. The project reflects the future prospects for the open pit mine to reach the elevation of 905m. However, it has been a relatively long time to pay for investment over 20 years since its use. During the same period, implemented a policy to place the waste rock, low grade and oxidized ore dumps close to the open pit to minimize the costs of production.

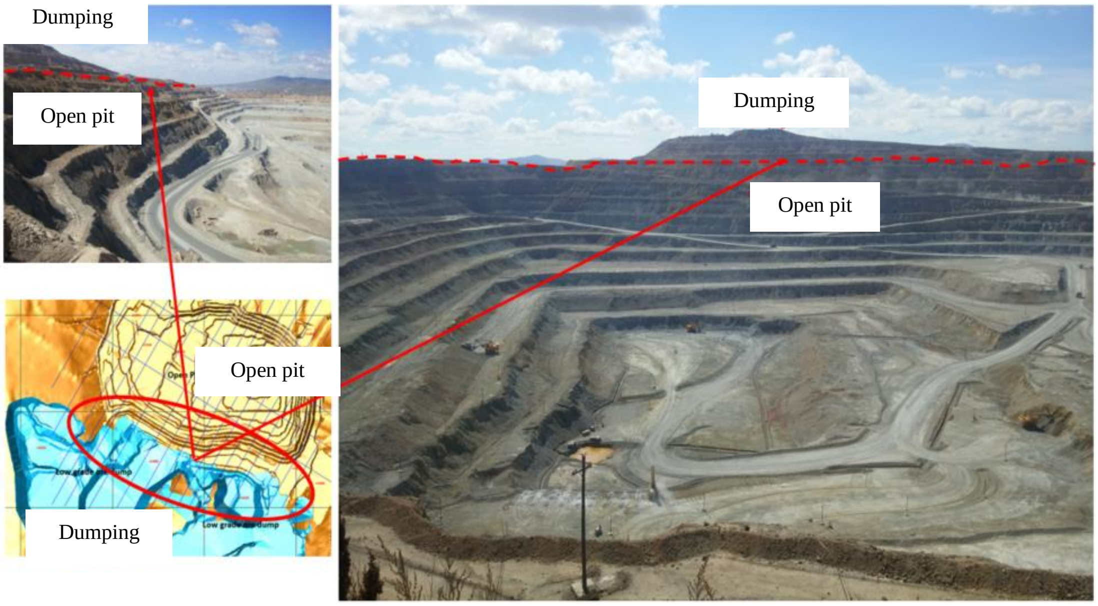

Figure 2-7 Intersection between open pit and dumping area

In this connection, the development of waste dumping were dumped on the further's pit boundary and then, last 5 years re-located over 15 M.m3 of waste with 40 M.$ of costs. In the near future, it is necessary to re-locate additional substantial waste dumping to the implementation of the concept to operation at the 905m level in the mining prospectus with increasing of the stripping and consequently increases of the operating costs.

Figure 2-8 Re-located dumping area

Another problem in this mine site is the waste dumping which near and on the side of the open pit mine will adversely affect the stability of the open pit stability. The geological explorations in the deposit have been studied for the mainly purpose of estimating and increasing the reserves and there are very insufficient studies for geotechnical research. Furthermore, operational exploration for over 40 years was also focused on resource enhancement.

Due to the insufficient of geotechnical investigation and researches, the stability angle, the dimensions of the design of the open pit mine, the underlying dimensions of the open pit mine are justified.

Figure 2-9 Faults in south side of open pit

The optimization of design and location of dumping area especially in south side of open pit is current sentimental problem for further's mine development and stability of open pit and dumping. Based on these real problems, this study is subsequently focused to develop an innovative method to optimization of appropriate design of open pit and dumping area using geotechnical and economic data.

Summary

Mining industry in Mongolia plays an important role in national economic growth. Especially, the Erdenet Cu-Mo depostit is the biggest porphyry copper-molybdenum deposit in eastern Asia exists and the majority of the state budget was created as the main pillar of the country economy.

Based on the current world price of mineral product and conditions of economic situations, it is necessary to optimize the Erdenet open pit mine boundaries, to explore the possibility of mining operation below 905m and optimize the location and design of dumping area and surface structures in relation to the large open pit mine boundaries. Having taken insufficient geotechnical and operational investigation and economic data analyses, the planning and stability problems of open pit mine and waste dump area at Erdenet mine are considered from mining activity. Reviewing current planning, management and developing some corrective actions are required in order to design and optimize the appropriate open pit and waste dumping area.

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CHAPTER III. DETERMINATION OF APPROPIRATE OPEN PIT MINE DESIGN

3.1. Introduction

The current strategic planning methodology to the Erdenet open pit mine is a relatively statically resolved characteristic approach belong to the centrally planned economy. In the centrally planned economy, the prices of raw materials are determined regardless of the world market, and the price and cost parameters are relatively constant and some indicators are normative. Therefore, the technical and economical main parameters for the strategic planning are equal attribution and the final result based on the data is also equable in concept. For example, the main parameters such as final pit design and cut-off grade are defined in the feasibility phase, the mine life cycle period will not change over time. These methodologies are often hand-made, so the time and manpower are high.

However, in today's market conditions, the production price and cost parameters are relative variable and constantly changing over time, so these methodologies are not suitable in current economic situations. Therefore, it is important to use methodologies of variable features to identify main parameters of mine strategic planning that are relevant to these variable situations.

The optimization of open pit mine design consists primarily of defining the ultimate pit limits which, in their turn, define what will eventually be removed from the ground, and dividing up the pit into manageable volumes of materials often referred to as pushbacks, cutbacks, or phases. Pushbacks, as they are referred to herein, can be seen as individual pit units with their own working front and mining dynamics while allowing the mine designer to develop short-term planning. They also contribute to the yearly production schedules so one can apply an economic discount rate when calculating the net present value (NPV) of the mine. Typically, an ore body model of what is predicted to be in the ground is produced through one of various techniques. From the ore body model, optimization techniques are used to produce the ultimate pit.

The ultimate pit is the maximum valued pit possible that obeys slope and physical constraints. Pushbacks are produced from the sections of the ore body model that remain within the ultimate pit limits. As a result, ultimate pit recognition in each period of time is a function of financial affairs, which is well defined by stripping ratio. The calculation of economic elements of a deposit therefore has to be performed according to the final exploration information and economical regime of the country in which the project is being carried out (Johnson, 1968).

To maximize the use of block modelling functions in order to optimize the pit design process is to fully integrate block modelling and slope stability analysis. This is because it is believed that optimized slope stability results in a lower amount of waste material removed which reduces mining cost and correspondingly raises the NPV of the whole project (Lerch and Grossmann, 1965; Koenigsberg, 1982; Hustrulid and Kuchta, 2006; Yasrebi et al., 2011 and 2014; Marcotte and Caron, 2013). In addition, there is a logical action where one identifies different rock types (ore or waste) in terms of the grades of each block. This process can be further enhanced by defining at every block location an identified cut-off grade. Before performing any of the computerized optimization processes, a range of basic information was required for the study. Technical data and economic information are crucial within the optimization process.

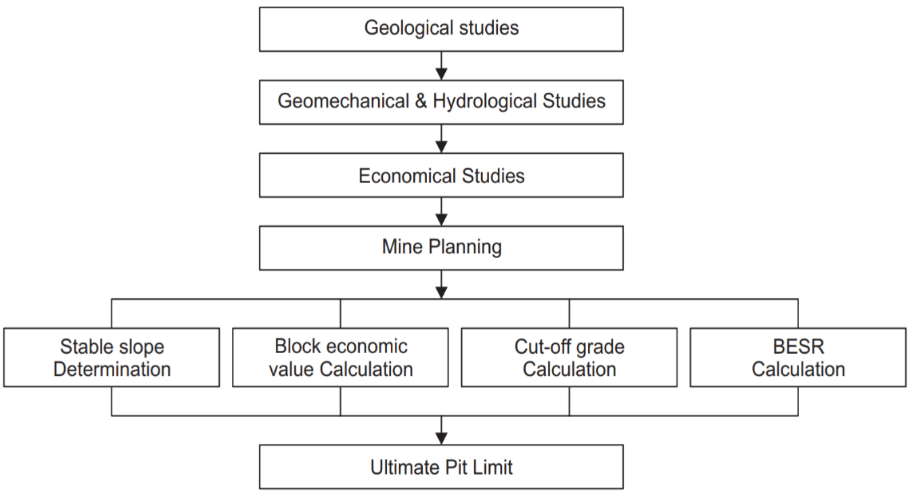

Figure 3-1 Design procedure in an open pit mine with regard to ultimate pit limit determination (Akbari et al., 2008)

In the previous Chapter, the current situations and problem descriptions of planning and operational of Erdenet mine's open pit and dumping area has been discussed. As description of site problems, it can be seen that the stability and available height of dumping area decreases and stripping of dumping re-location increases with the expansion of the open pit boundary dramatically. Therefore, this Chapter focuses on determination of the nested pit shells, pushbacks and appropriate open pit design of the NW section of Erdenet deposit and open pit optimization study through geological data and rock mass characterization which the achieved results can be used for the solution of variable open pit boundaries and required dumping volume to optimize the design of dumping area in next Chapters of this study.

3.2. Methodology

Initially, the dataset obtained to the block model for this study was exported from the geological database of the Erdenetyn-Ovoo Cu-Mo and basic economic indicators of several copper mining companies of Mongolia. Before this, validation of all data analyzed from boreholes was conducted, which is an important preliminary action before generating a block model. The Geovia Surpac software enables us to generate a 3D geological and deposit block model which includes ore grade, volume, lithology, structures and rock types. Following this, the Rocscience Dips software was used for kinematic analyses in order to set the appropriate slope design for pit shells belonging different rock type domains and the Geovia Whittle software was used in order to establish the final pit limit in terms of the maximum Net Present Value (NPV) and associated pushbacks to produce a best case mining scenario. To do this, all the required data such as grade, density and rock type and other similar data were entered as numerical values into each of the deposit's block models.

3.3. 3D block modelling of Cu-Mo deposit for open pit optimization

The open pit mining in Erdenet Cu-Mo deposit is currently 2.5km x 1.5km in area. About 400 m of material have been removed from the original surface at an elevation of 1,650 m to the present mine level at 1,250 m.

3.3.1. Block model summary

The 3D geological block model of the NW section of Erdenetyn-Ovoo deposit was generated using the Geovia Surpac software using data from previous geological exploration data of the deposit.

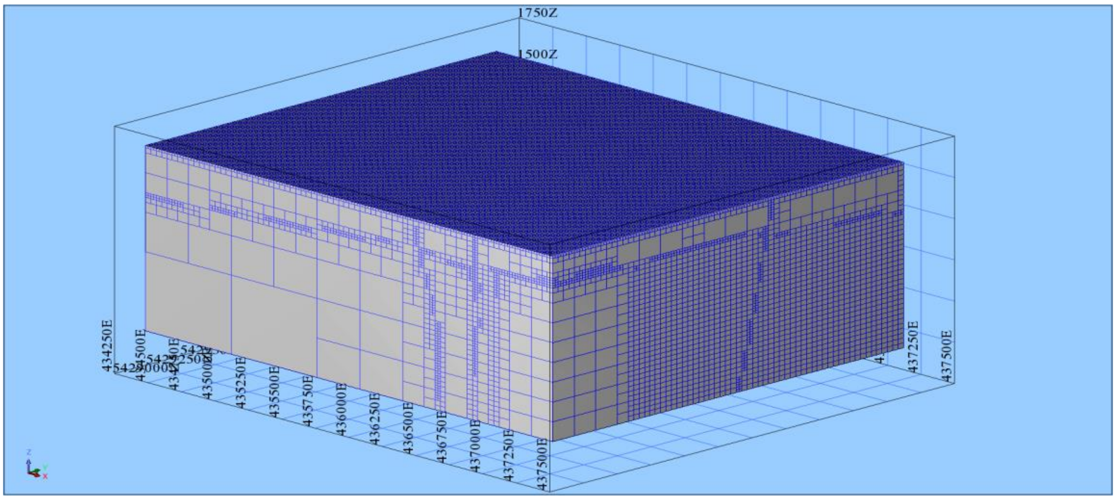

Figure 3-2 3D block model of NW section, Erdenetyn-Ovoo Cu-Mo deposit

The block model for this study estimated basic size of the each block in the model is 20m x 20m x 15m, total count of blocks are 430,117 and the block model conducted by several attributes. The three dimensional block modeling and attribution sequence applied to open pit optimization and planning is shown in the following subcategories.

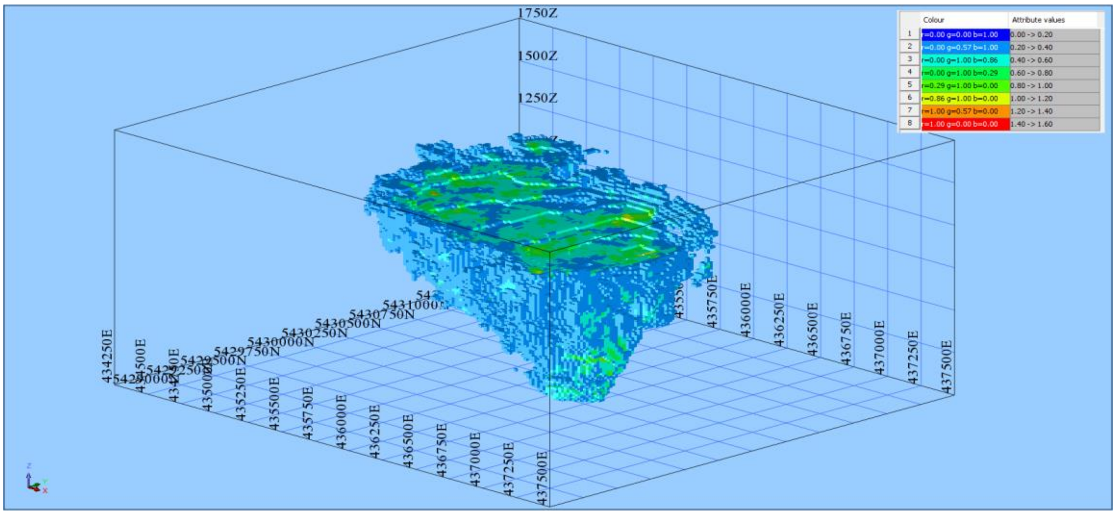

Figure 3-3 Orebody block model, Cu≥0.2 wt %

The mineralization data of the block model was entered into the optimization algorithms. The mineralized zones with Cu less than 0.2 wt % are assumed to be waste materials which decrease NPV by increasing stripping ratio. The block model was generated at Cu ≥ 0.2 resulting in lower stripping and the weakly mineralized zones with less than 0.2 wt %.

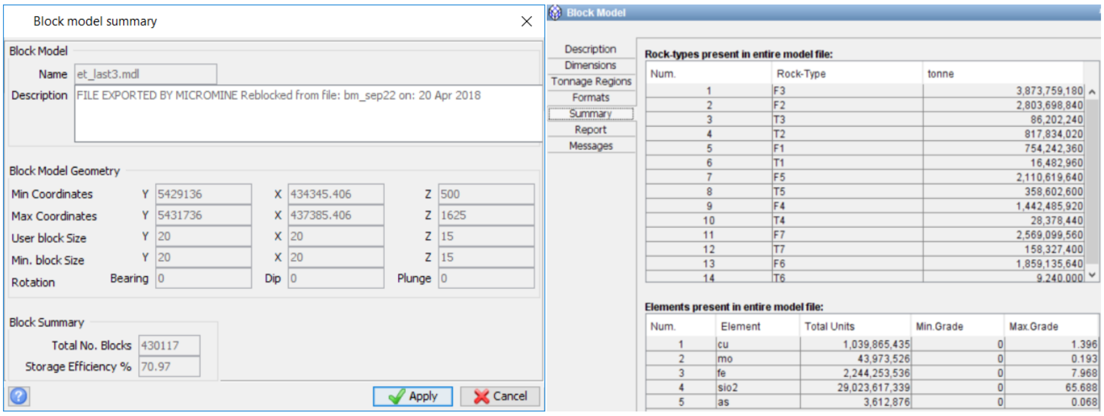

Figure 3-4 Block model summary a.Surpac geological block model summary, b.Whittle optimization block model summary

3.3.2. Topographical features

Topographical features of the deposit land surface as well as other related data, are presented in a 3D block model entered in the optimization software, prepared using the original topographical and current mining level topographical maps. It is clear that blocks located between topographical surfaces and deposit surfaces are considered as waste blocks and are entered in the economic model as blocks of negative significance.

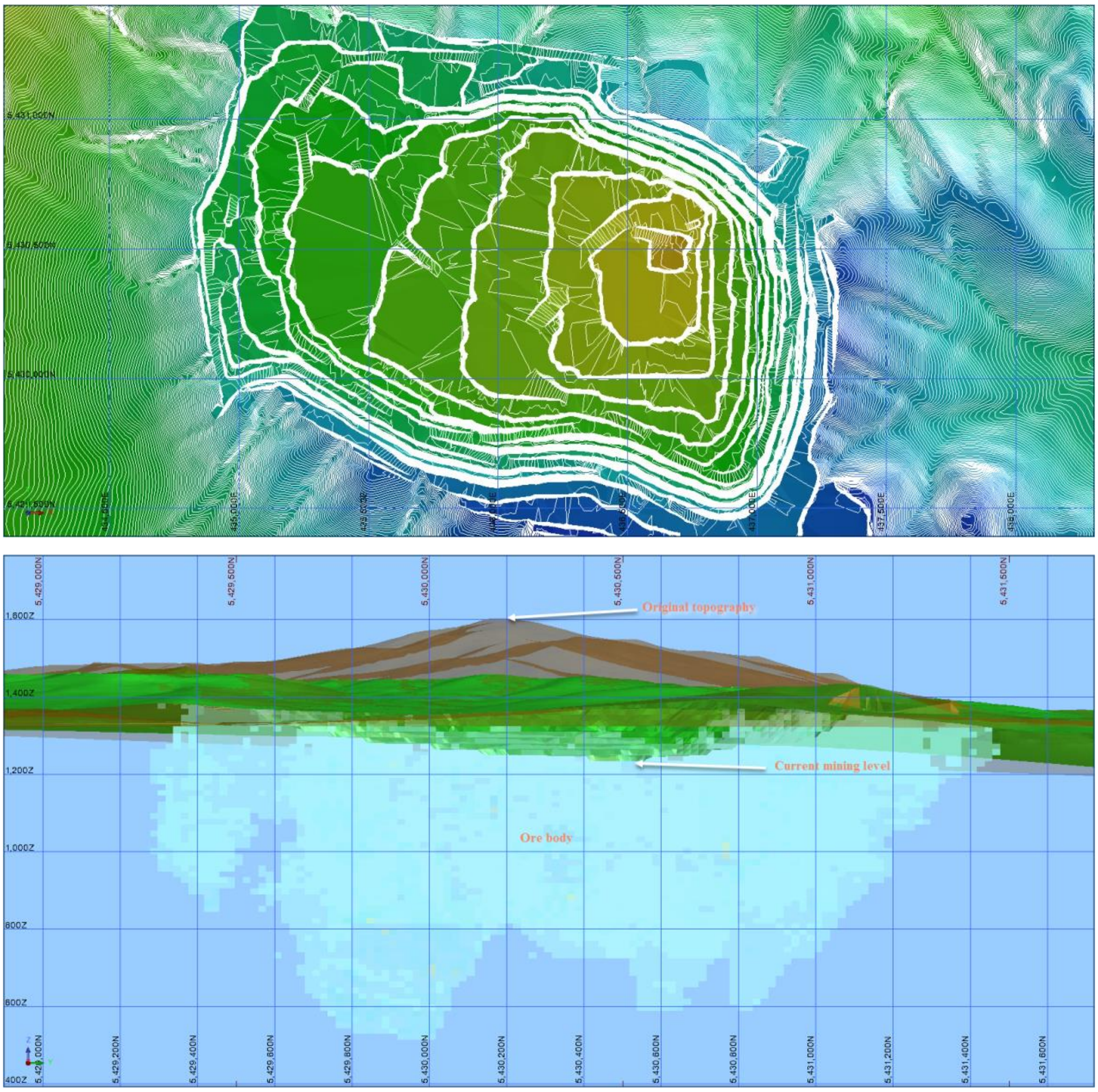

Figure 3-5 Original topographical surface (before 1978) and the current mining level of the deposit for optimization study

3.3.3. Lithological model.

The ore field incorporates four sectors: Northwestern, Central, Intermediate, and Southeastern. Major rock types in the NW section of the Erdenetyn-Ovoo deposit are sub-volcanic units such as andesite, dacite, diorite, granodiorite, quartz andesite, quartz andesite-riolite, porphyric quartz diorite and tuff. Granodiorite and granite host ores in the NW part of the study area.

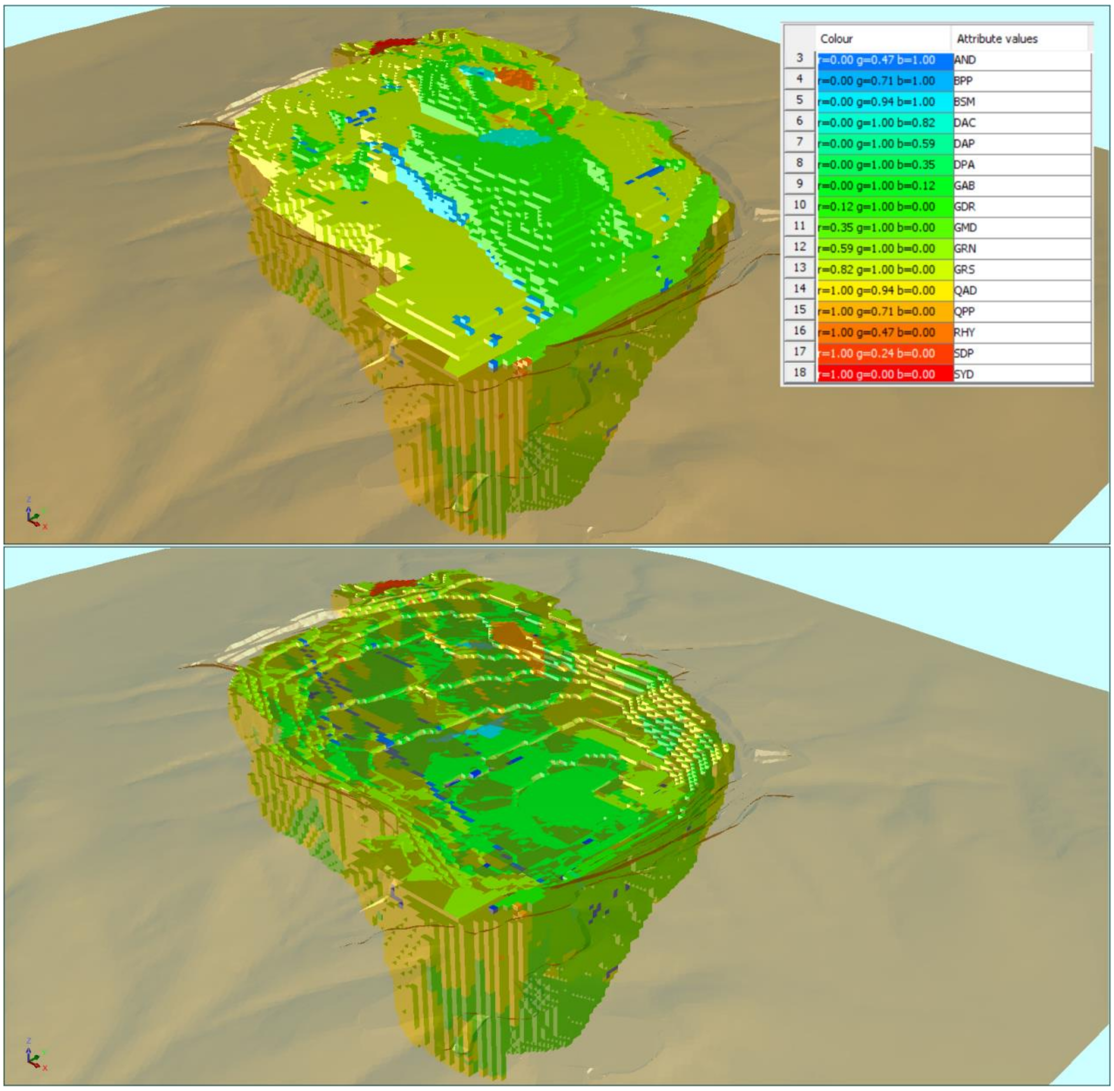

Figure 3-6 Lithology model, a. original topography, b. current mining level

Rock types are separated by geological age, complex and qualities following 5 different lithology domain and groups:

- Middle Mesozoic (J23)
- Selenge complex (P2S)
- Erdenet porphyry complex (P2T1)
- Early Mesozoic (T2J1)

Table 3.1. Lithology Domains, Code, Rock types

Domain	Rock type	Block model Litho.Code
J23	AND	1
P2S	GRS	2
	GMD	
	GAB	
	GRN	
P2T1	GDR	3
	DAC	
	BPP	
	QPP	
	DAP	
	QAD	
	PPD	
T2J1	RHY	4
	SDP	
	DPA	
	TAL	
	SYD	

The rocks such as RHY-ryolite, SDP-siennite diorite porphyry, DPA-diorite porphyry, andesite, TAL-trachyandesite lathite, SYD-sienite diorite are taken as rock structure and spatial locations and geological age into T2J1 domain and variation coefficient is relatively high.

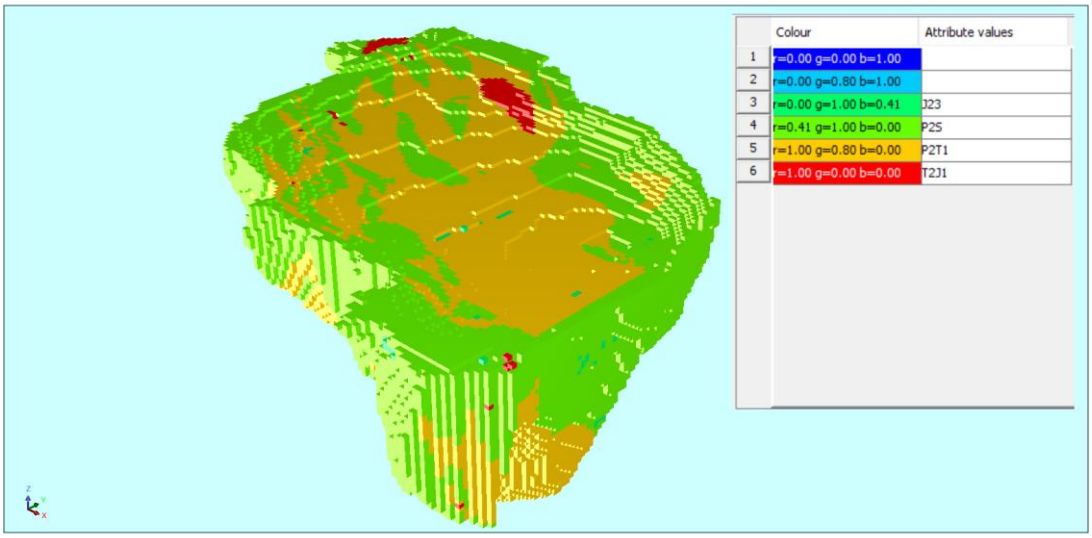

Figure 3-7 Lithology Domains

AND-andesite is a post-mineralization dykes, which is associated with sample and partial sampling in the simulations of specimens for pollution. Some specimens have been added to this domain, which are modeled as part of the content and used as a pollution model. The rocks such as GDR-granadorite porphyry, DAC-datsite, BPP-biotite, plagioclase porphyry, QPP-quartz-plagioclase porphyry, DAP-datsite porphyry, QAD-quartz andesite datsite, and PPD-plagioporphere datsite are taken as spatial locations and statistical values into P2T1 domain. This domain is contents high grade, variation coefficient is relatively low and smooth distribution compared with other domains. GAB-gabbro, GRN-granite, GMD-granite, monzonodiorite and GRS-selenge granite were taken into P2S domain with spatial position and geological age.

3.3.4. Structural model

There are four major fault systems in the open pit mining area, north, south, northwest and central. The fault system of the north with 175^0 - 190^0 azimuth and fractures is predominantly up to 65^0 - 87^0 dips. The central of the mining almost 70% is joint and cracks with 250^0-290^0 azimuth and 55^0 - 90^0 dips. The joint and cracks below are all vertical. The fault systems with 50^0-70^0 and 170^0-220^0 azimuth in the south of the mining are more pronounced than others. There are 60^0-90^0 and 180^0-225^0 azimuth fault systems in the northwest with 60^0 - 90^0 dips.

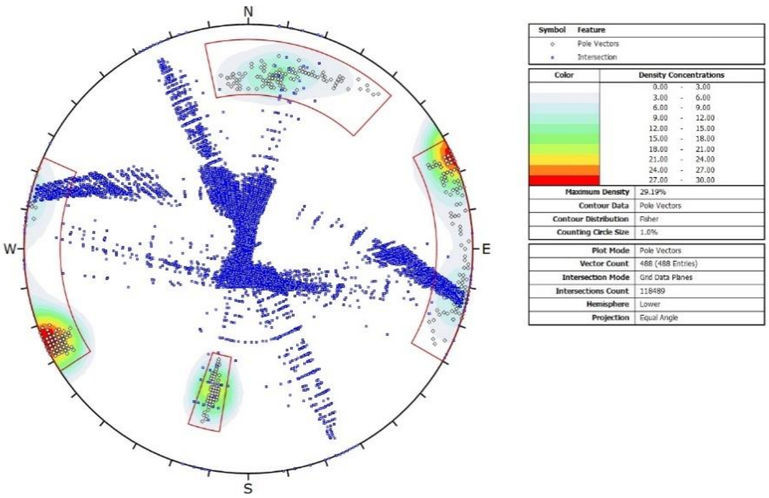

Figure 3-8 Stereonet plots of major faults

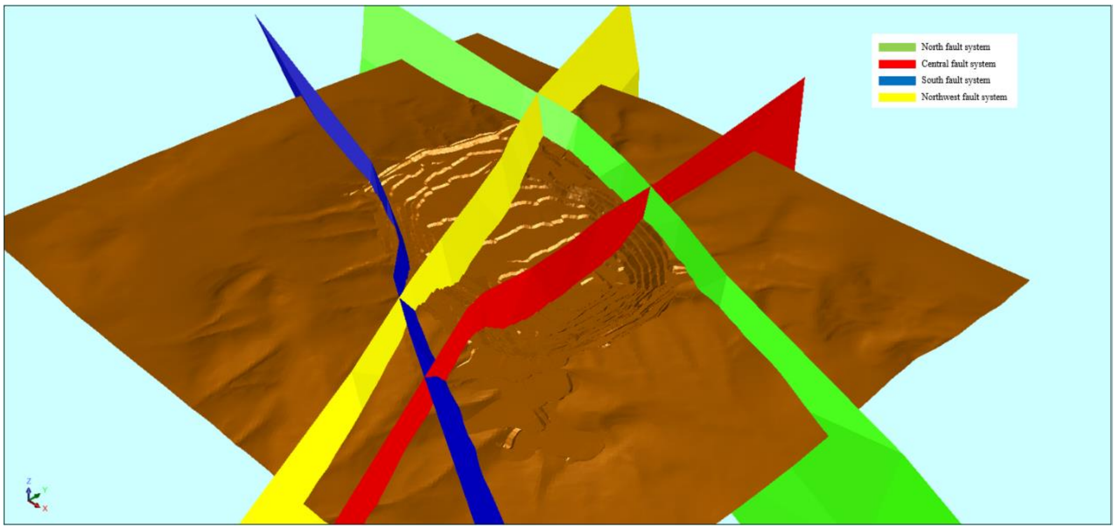

Figure 3-9 3D surface plane of faults

The fault systems divided the mining area into 7 general structural domain sections and the sections are enclosed in the block model of the deposit.

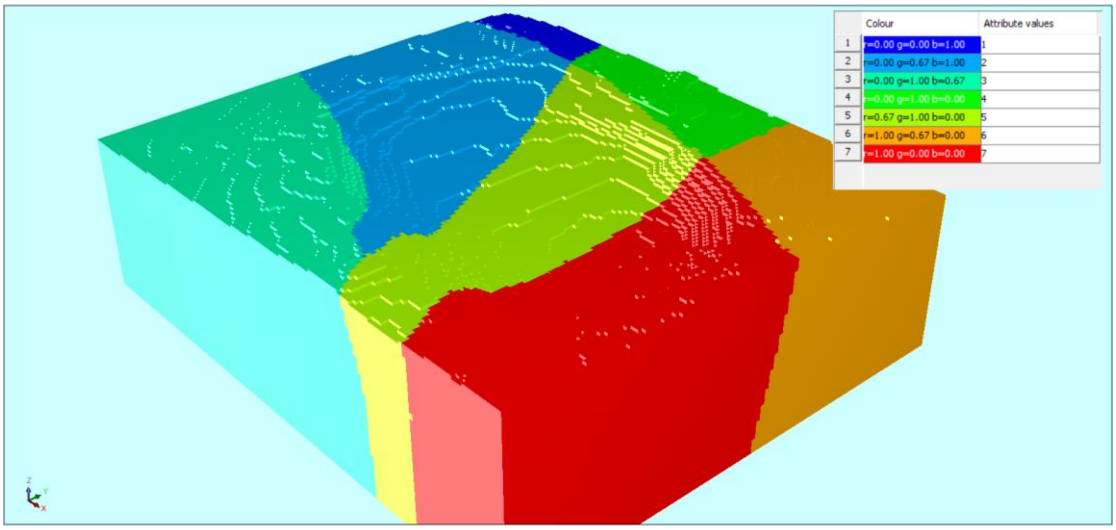

Figure 3-10 Structural domain

3.3.5. Weathering zonation model

Oxidation and alkalinity zone above the sulphidite ore body. The thickness of this zone is the highest in the lower part of the surface and the periphery of the ore body.

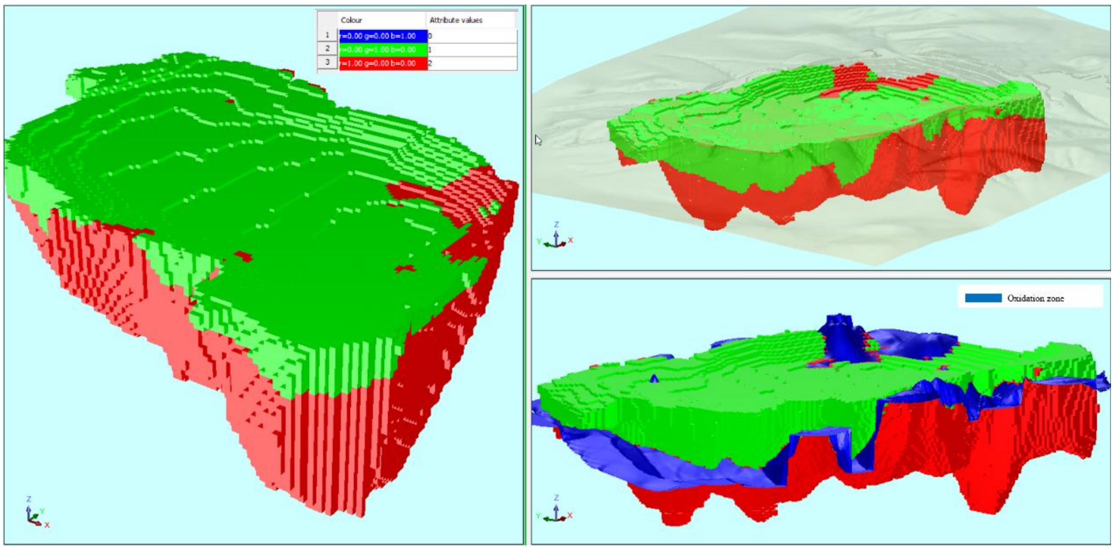

Figure 3-11 Weathering model

Sometimes the lower boundary of the oxidation zone decreases by 40-60 m in the active spacing region. The oxidation process is widely distributed, such as ferrous oxide, manganese oxide, molybdenum, and copper oxide minerals. The oxidation alkalinity zone divided the block model into 2 general weathering zones and the zones are enclosed in the 3D block model of the deposit.

2.3.3. Rock type model

The fault systems divided the mining area into 7 general structural domain sections by vertically and the weathering zones divided mining area into 2 general zones by horizontally. In order to classified rock type by weathering zonation in each structural domains, 14 rock type domain are enclosed in the 3D block model of the deposit.

Table 3.2. Rock type domain

	Structural zone	1	2	3	4	5	6	7
Weathering zone	Transition (oxidation, alkalinity)	T1	T2	T3	T4	T5	T6	T7
	Fresh zone	F1	F2	F3	F4	F5	F6	F7

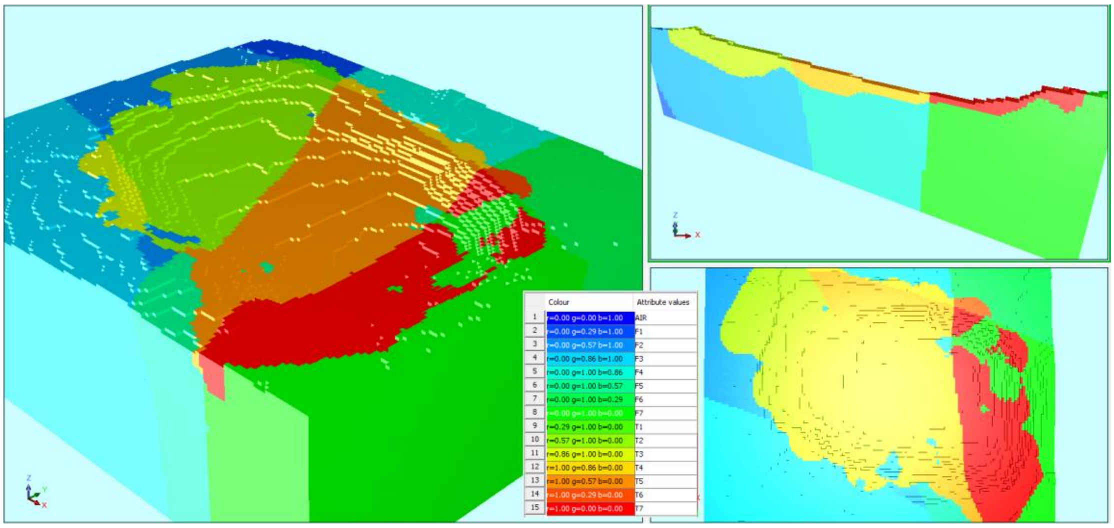

Figure 3-12 Rock type model

From the result of block model attributing and coding, it seems rather obvious that there are likely to be multiple populations, presumably related to geology, e.g. lithology. Certainly from a slope stability point of view, it would be expected that anyone examining this data would consider at least multiple domains for slope stability assessments, hence several final pit slopes would be selected in determining an ultimate pit limit. Further geotechnical characterization will, however, be necessary to establish any potential influence of the 3D fracture network and presence of any major discontinuity controlled instability.

3.4. Stability assessment for open pit

Any truly optimal pit design methods must, therefore, take into account variable slope angles. Incorporating variable slope constraints into the Lerch-Grossmann algorithm will make it much more flexible, practical and reliable. Optimization of software algorithms have been designed in such a way that the blocks of different levels are extracted given final pit slope. In optimization softwares such as Gemcom Whittle, a slope region is a physical volume to which a particular group of overall slope angles and corresponding azimuths are defined. The purpose of these analyses was to identify the kinematically possible failure modes within each domain sector and to set appropriate overall slope angle for the Lerch-Grossman open pit optimization using the stereographic technique.

3.4.1. Modes of Failure

Kinematically possible failure modes in rock slopes typically include planar, wedge and toppling failures. These failure modes will occur if discontinuities are pervasive at bench scale or greater, if weak infilling is present along the discontinuities, or if the geometry of the discontinuities is conducive to failure. Stereographic analyses of peak pole concentrations of the discontinuity data can be used to identify potential modes of failure. A brief introduction on each mode of failure is provided below:

***Planar Failure* –** This failure mode is kinematically possible where a discontinuity plane sits at a shallower inclination than the slope face (daylights) and at an angle steeper than the friction angle.

***Wedge Failure* –** This failure mode is kinematically possible where the plunge of the intersection of two planes (sliding vector) is inclined less than the slope face (daylights) and at an greater than the combined friction angle which is determined from the characteristics of each plane that forms the wedge. Where kinematics are the controlling factor, the recommended pit slope angles have been adjusted to reduce the potential for large-scale, multiple bench wedge failures.

***Toppling Failure* –** This failure mode is kinematically possible due to interlayer slip along discontinuity surfaces where sub-vertical jointing dips into the slope at a steep angle β. The condition for toppling to occur is when $\beta > (\phi j + (90 - \Psi))$, where Ψ is the slope face angle and ϕj is the friction angle (Goodman, 1989).

Table 3.3. Description of major faults

Faults	NW		N		S		C	
	dip	direct	dip	direct	dip	direct	dip	direct
mean	65.00	83.00	77.00	185.00	60.00	16.00	83.00	245.00

3.4.2. Stereographic Analysis

Stereographic analyses have been carried out for each design sector using the Rocscience Dips software. The analysis assumes that failures will occur as the results of sliding blocks or wedges along the defects of rock mass. This type of analysis does not consider slope failure within the rock mass. The drillhole rock mass structural data were grouped by design sectors for further kinematic stability analyses.

Table 3.4. Rock mass characteristic of weathering zones

Material	Unit weight, MN/m^3	UCS, MPa	Young's Modulus, GPa	Poisson's Ratio	GSI	mi	Ave. RQD, %	Fric. Angle, deg
Oxidation	0.023	100	45	0.254	28	29	40	35
Transition	0.026	200	50	0.254	46	29	68	35
Fresh	0.027	250	60	0.332	56	32	75	37

The overall slope angle of 45° to 35° have been applied for each domains for kinematic analyses and optimization calculations. Multiple stereographic plots were created for each design sector in order to account for the variation of pit wall orientations within each sector.

Table 3.5. Orientation of pit slopes

Slopes	East		West		North		South	
	dip	direct	dip	direct	dip	direct	dip	direct
min	35	275	35	95	35	185	35	10
max	45	275	45	95	45	185	45	10

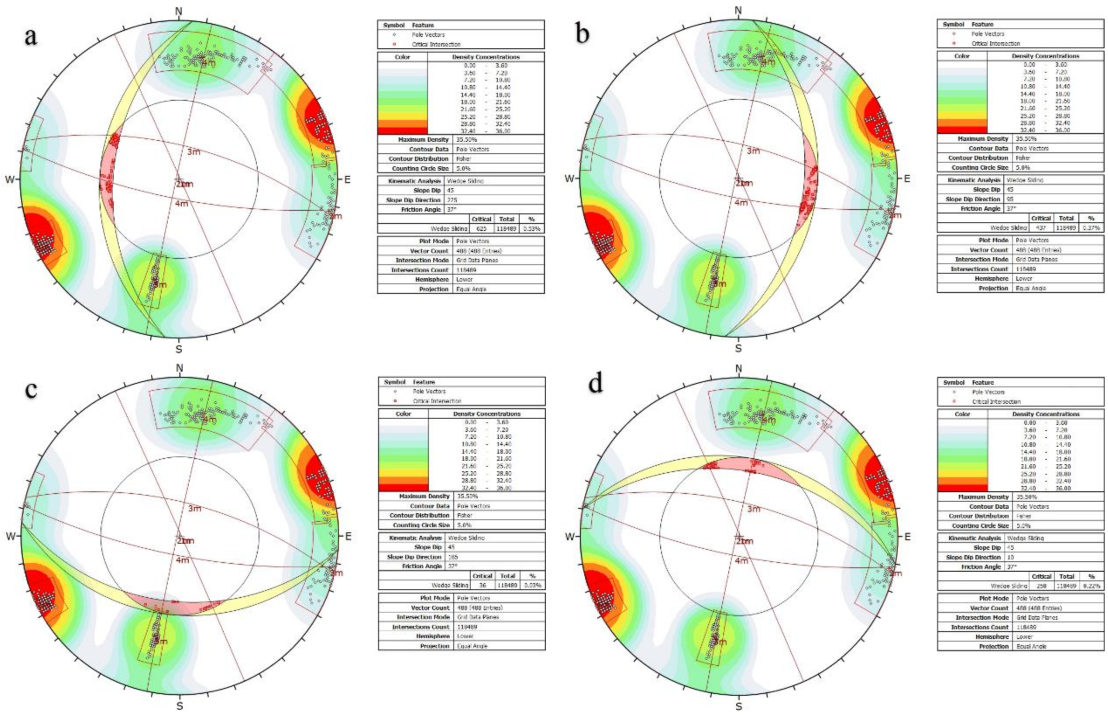

Figure 3-13 Stereographic results for overall slope angle of 45° wedge failure analyses in fresh zone for each wall orientations; a.east, b.west, c.north and d.south

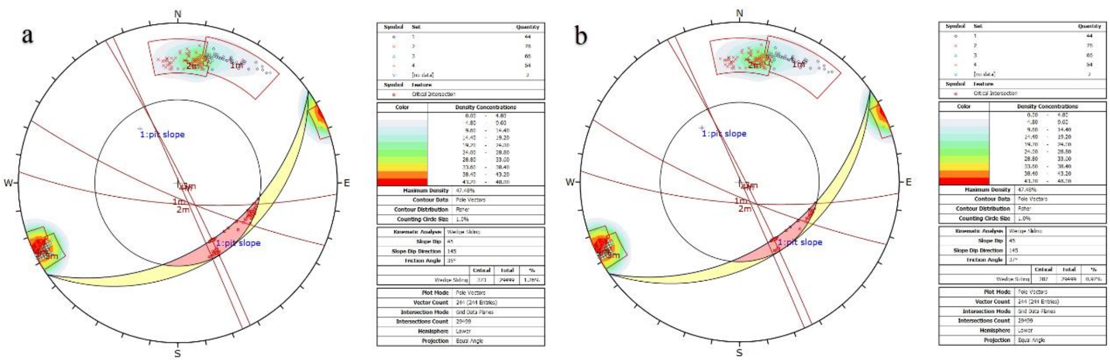

Figure 3-14 Stereographic results for overall slope angle of 45° wedge failure analyses in Stuctural domain-1; a.rock type T-1, b.rock type F-1

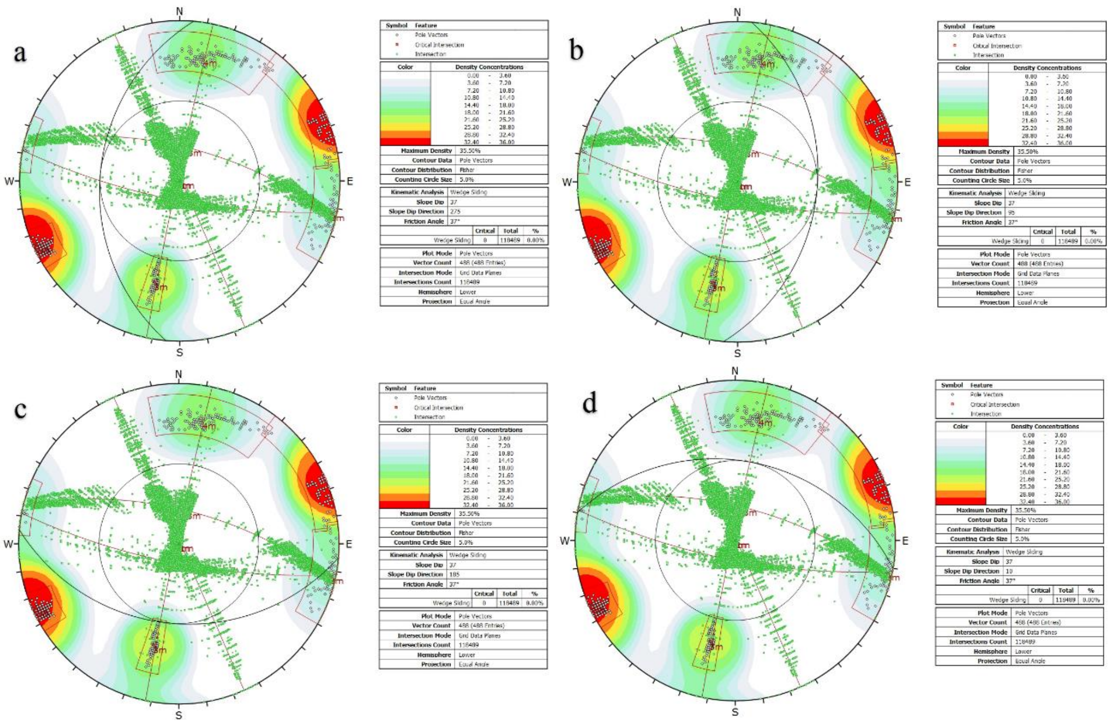

Figure 3-15 Stereographic results for overall slope angle of 37° wedge failure analyses in fresh zone for each wall orientations; a.east, b.west, c.north and d.south

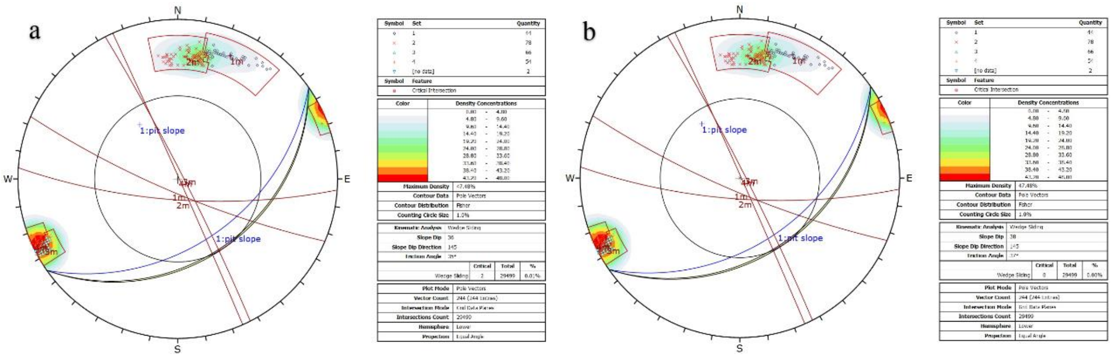

Figure 3-16 Stereographic results for overall slope angle of 36° and 38° wedge failure analyses in Stuctural domain-1; a.rock type T-1, b.rock type F-1

In the present study four major joint systems were examined using a slope mass rating classification scheme which were further investigated through kinematic analysis. The major four fault sets were observed at a particular location have generally same dip direction of vertically 60° to 90°. Pit wall orientations were measured from a preliminary current pit design in 2016. Where the pit walls contain a pronounced curve, multiple analyses were conducted to account for the changing pit wall dip direction within the sector. The friction angle of 35° and 37° was used for discontinuities in oxidation zone and fresh zones, respectively. This value is based on the results of direct shear testing performed on samples collected during site investigation in 2014. Multiple stereographic plots were created for each structural domains including rock types in order to account for the variation of pit wall orientations within each domains.

From the results of kinematic analyses, probability of toppling and planar failures was relatively low in applied overall slope angles and wedge failures are kinematically possible in higher overall slope angles of 40° for most sections. For long-term stability and its sustainability, the slope requires immediate attention to prevent and mitigate chances of failure in order to enhance the productivity of the mine. The results of the appropriate overall slope angle using kinematic stability analyses are summarized on Table 3.6.

Table 3.6. Kinematic analyses results of appropriate overall pit slope angle for each rock type zones

Structural zone	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Rock type zone / **Overall slope angle, deg**	***T1***	***T2***	***T3***	***T4***	***T5***	***T6***	***T7***
	36	35	45	36	36	37	35
	F1	***F2***	***F3***	***F4***	***F5***	***F6***	***F7***
	38	37	45	38	38	39	38

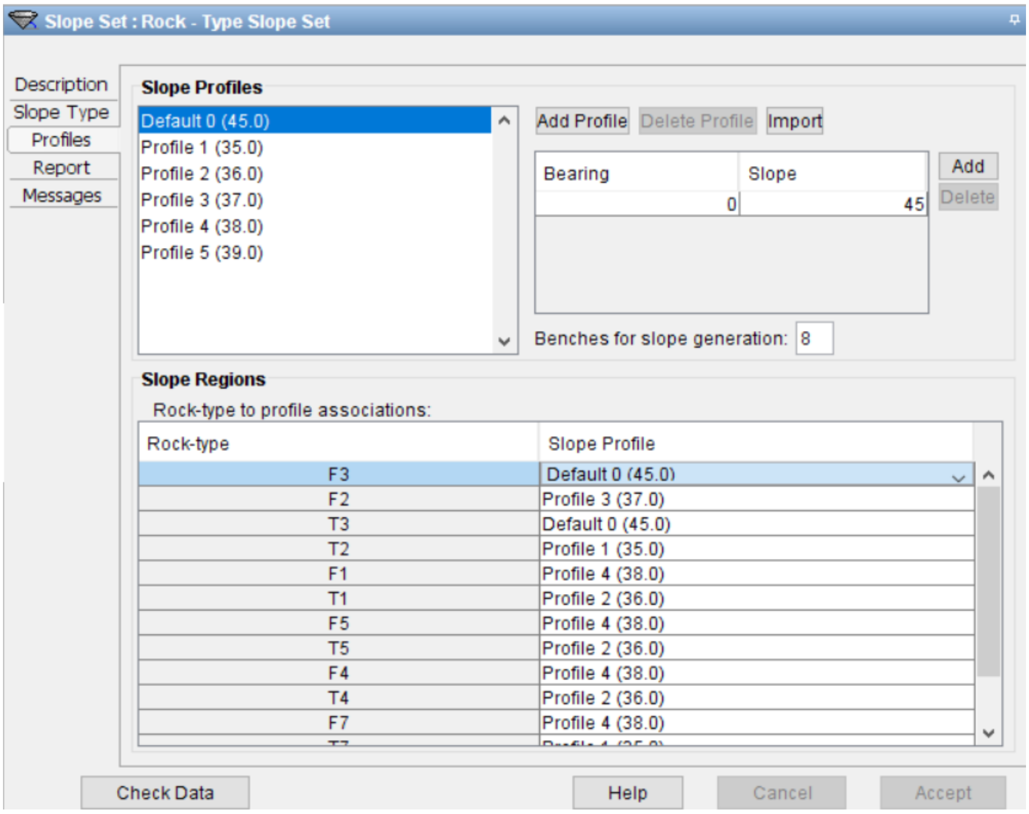

Figure 3-17 Slope set input data for pit optimization

3.5. Economic principles

3.5.1. Metal recovery

The block model was calculated based on the net smelter return (NSR) value of each blocks in the block design of a copper and molybdenum content. The calculations include the block content, expected melting/refining contracts (e.g payment terms, fines, fines), metal recovery, market prices for each metal (Cu, Mo and Ag prices on LME) and net US \$. The processing plant metal recoveries from Erdenet mining company, as provided for long-term planning of 2016-2025. Metal recovery in processing plant is relative different depending on the ore content of the feedstock from open pit. In general, high-grade ore recovery is relatively higher than the low grade ore. The metal recovery percentage used for the calculations of Lerch-Grossman are shown in the Table 3.7.

Table 3.7. Metal recovery percentage used for Lerch-Grossman calculation

Metal	Oxidized ore	Sulphide ore
Copper (Cu)	60.4 %	86.84 %
Molybdenum (Mo)	-	40.07 %
Silver (Ag)	-	80 %

The value of the NSR is calculated based on the parameters of Table 3.7 and Table 3.8 and the formula for primary and oxidized ores is as follows:

$$\text{NSR, \$/t.} \quad (Cu_{np} \times Cu_g \times Cu_r)$$
$$+ (Mo_{np} \times Mo_g \times Mo_r)$$
$$+ (Ag_{np} \times Ag_g \times Ag_r)]$$

Where:

Cu_{np}	=	Cu net price, \$/%
Mo_{np}	=	Mo net price, \$/%
Ag_{np}	=	Ag net price, \$/%
Cu_g	=	Cu grade content in each block, %
Mo_g	=	Mo grade content in each block, %
Ag_g	=	Ag grade content in each block, oz/t
Cu_r	=	Cu metal recovery
Mo_r	=	Mo metal recovery
Ag_r	=	Ag metal recovery

Metal prices for oxidized ore are calculated as NSR (oxidized) formula:

$$\text{NSR(oxidized), \$/t.} = (Cu_{ox\ np} \times Cu_g \times Cu_r)$$

3.5.2. Prices and expenses

The results obtained from optimization studies are significantly affected by the price of the product and operational costs of production. These parameters must therefore be determined more precisely. However, due to the importance of economic principles, and the large influence they have on the results, the above-mentioned parameters are discussed in detail in the subsequent sections. The product prices and operational costs including ore and waste exploitation costs and milling costs parameters as reflected in the long-term planning of the Erdenet Mining Corporation 2016-2025. The average cost of extraction costs at each level is $3.80/m^3. The cost of mine mining will not exceed $1.61/m^3 from the elevation of 1,250 m to the bottom level of the mine.

The net worth of the unit block is also calculated based on the mining, processing, general and administrative costs. The block model was used for mine optimization calculations and net smelting of each block was estimated as a revenue to be concentrated on "mining gate" conditions. In the NSR calculation have not been included in mining, sputum, general and management costs. In the calculation of the NSR, the following parameters are included:

- Ore processing recovery
- Loss of fatigue and purification
- Concentrate transport costs
- Clearing and refining charges
- Royalty fee

The value of the NSR estimated for each block in the module is $/t. The net profit value is for Lerch-Grossman analyses calculated as excludes operational, processing and administration costs in each blocks. The economic parameters used for the calculations of Lerch-Grossman are shown in the Table 3.8.

Table 3.8. Economic parameters belonging to the main calculation of Lerch-Grossman

Metal price:	
Copper (Cu)	$ 6,000 / t Cu
Molybdenum (Mo)	$ 25,000 / t Mo
Silver (Ag)	$ 25.00 / oz Ag
Operational cost:	
Mining operation cost	$ 3.80 / m^3
Increase in each level (down below 1,250 m elevation)	$ 0.07 / m^3 / bench
Processing sulphide ore cost	$ 7.08 / t ore
General and administrative costs	$ 0.68 / t ore
Other operational costs	$ 0.75 / t ore
Processing oxidized ore cost	$ 10.01 / t ore
Processing Cu concentrate:	
Cu content in concentrate	22.36 %
Smelting reduction Cu in concentrate	96.50 %
Concentrate moisture content	9.80 %
Concentrate transportation cost	$ 22.36 / t concentrate
Concentrate smelting and refining costs	$ 149.00 / t concentrate
Ag content in concentrate	70 g / t concentrate
Ag refining losses in concentrate	90.0 %
Ag refining cost	$ 0.91 / t concentrate
Processing Mo concentrate:	
Mo content in concentrate	47.00 %
Smelting reduction Mo in concentrate	96.70 %
Concentrate moisture content	9.00 %
Concentrate transportation cost	$ 78.00 / t concentrate

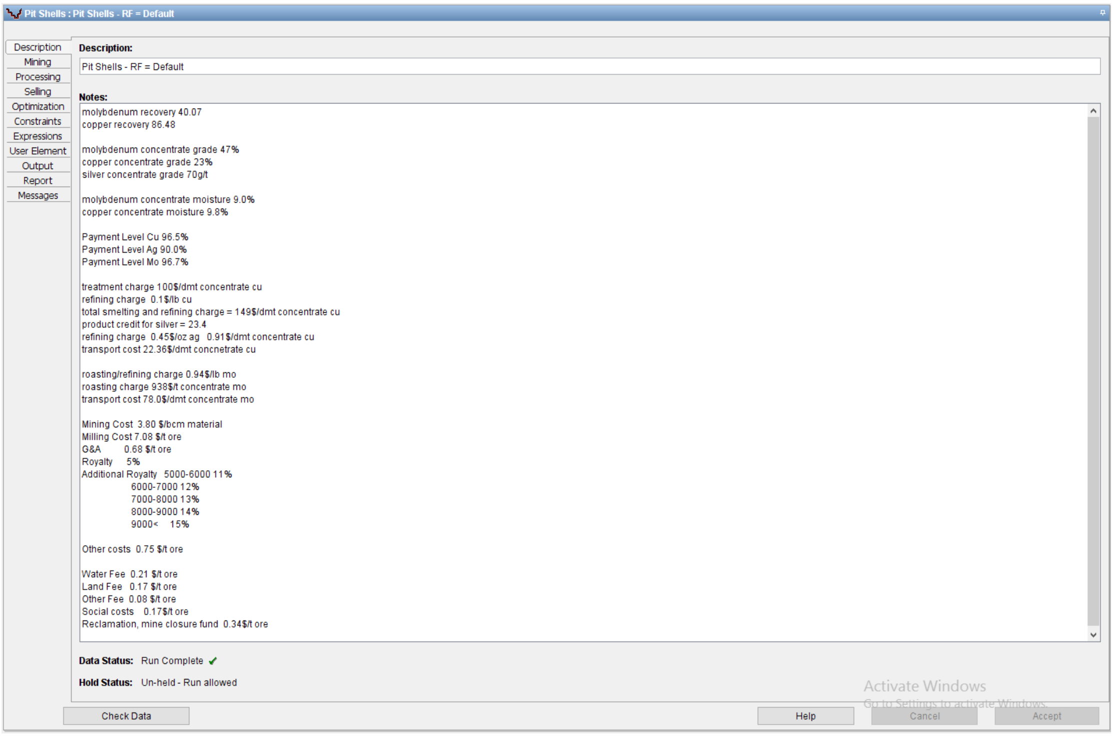

Figure 3-18 Economical data summary for Whittle model

3.6. The Lerch-Grossman's analysis

The Lerch–Grossman algorithm is well known for being the only method that can be proved, rigorously, always to yield the true optimum pit. However, when the algorithm was first introduced it was based on a fixed slope angle governed by the block dimensions. The methods presented here have been incorporated into the algorithm to overcome this limitation and to take account of variable slope angles. As demonstrated by a case study, the algorithm is able to generate an optimal open pit with variable slopes. The method can be used for both cubic and rectangular block models. Slope angles can vary in different parts of the orebody without change to the block dimensions, which are independent of the slope angles. It is also used in mining optimization software as the industry standard, for example in Gemcom's Whittle software, to find the optimal pit and pushbacks. The algorithm uses different revenue factors to generate a value-based mining sequence strategy to design pit shells. Early pit shells are constructed using high-grade blocks and a low stripping ratio.

For this study, the open pit mining boundaries which relates to the specific case of the Lerch-Grossman's analyses based on technical and economic parameters as shown in Table 3.7 and 3.8. The price of metal was calculated in the long-term planning of Erdenet Mining Corporation from 2016-2025, which was calculated on average over the last five years; $6,000/t. Cu, $25,000/t. Mo and $25.00/oz. Ag. The optimized pit shell contains approximately $8.5/t. of NSR cut-off grade of approximately 1,469.09 M.t. sulphide ore. The average content of metals in the sulphide ore is 0.425% Cu, 0.018% Mo and average stripping ratio is 0.67 t./t.

3.6.1. Nested pit shell optimization

Changes in product prices are reflected in nested pit shells optimization calculations using revenue factors. The base price used for calculations is defined as income factor or "Revenue Factor = 1.0". Price sensitivity analysis was calculated from 30% to 200% of the base price. The results of optimization of open pit are shown in Table 3.9. In this case, the specific case (Revenue Factor = 1.0) pit shell is a pit case of 34th.

Table 3.9. Optimization results of the Lerch-Grossman analyses

Pit shells	Revenue Factor	Rock	Ore	Strip. Ratio	Total benches		Cu	Mo
		t.	t.		Max	Min	%	%
1	0.34	46,080.00	46,080.00	0.00	56.00	50.00	1.16	0.07
2	0.36	154,260.00	154,260.00	0.00	56.00	50.00	1.18	0.04
3	0.38	416,520.00	416,520.00	0.00	63.00	50.00	1.12	0.03
4	0.40	801,000.00	801,000.00	0.00	63.00	49.00	1.06	0.03
5	0.42	1,692,720.00	1,615,680.00	0.05	63.00	49.00	1.00	0.03
6	0.44	3,615,660.00	3,246,300.00	0.11	63.00	48.00	0.94	0.03
7	0.46	7,220,220.00	6,063,300.00	0.19	63.00	47.00	0.89	0.03
8	0.48	10,914,540.00	9,248,040.00	0.18	63.00	47.00	0.85	0.03
9	0.50	16,958,460.00	14,212,620.00	0.19	63.00	46.00	0.81	0.03
10	0.52	27,687,420.00	22,503,300.00	0.23	63.00	44.00	0.77	0.03
11	0.54	39,873,720.00	32,595,780.00	0.22	63.00	43.00	0.73	0.03
12	0.56	71,696,160.00	53,419,800.00	0.34	63.00	40.00	0.70	0.03
13	0.58	108,676,260.00	78,837,900.00	0.38	63.00	38.00	0.67	0.03
14	0.60	139,907,220.00	103,408,800.00	0.35	63.00	37.00	0.65	0.02
15	0.62	179,879,280.00	133,939,980.00	0.34	63.00	37.00	0.63	0.02
16	0.64	250,165,200.00	180,838,560.00	0.38	63.00	37.00	0.60	0.02
17	0.66	318,662,940.00	226,900,380.00	0.40	63.00	36.00	0.59	0.02
18	0.68	399,883,260.00	274,632,780.00	0.46	64.00	35.00	0.57	0.02
19	0.70	487,454,520.00	327,131,160.00	0.49	64.00	34.00	0.56	0.02
20	0.72	551,458,440.00	370,121,880.00	0.49	64.00	34.00	0.55	0.02
21	0.74	676,000,140.00	443,226,900.00	0.53	64.00	33.00	0.53	0.02
22	0.76	875,976,780.00	542,556,180.00	0.61	65.00	28.00	0.52	0.02
23	0.78	992,280,660.00	613,180,920.00	0.62	65.00	27.00	0.51	0.02
24	0.80	1,161,418,020.00	707,579,880.00	0.64	65.00	25.00	0.50	0.02
25	0.82	1,305,749,580.00	790,870,440.00	0.65	66.00	25.00	0.49	0.02
26	0.84	1,398,607,800.00	851,648,340.00	0.64	66.00	25.00	0.48	0.02
27	0.86	1,496,990,280.00	911,571,540.00	0.64	66.00	24.00	0.47	0.02
28	0.88	1,612,875,000.00	992,628,600.00	0.62	66.00	23.00	0.46	0.02
29	0.90	1,768,100,400.00	1,079,632,080.00	0.64	66.00	22.00	0.46	0.02

30	0.92	1,841,784,300.00	1,133,637,060.00	0.62	66.00	22.00	0.45	0.02
31	0.94	1,934,214,240.00	1,198,548,240.00	0.61	66.00	22.00	0.44	0.02
32	0.96	2,033,541,840.00	1,261,723,740.00	0.61	66.00	22.00	0.44	0.02
33	0.98	2,379,019,440.00	1,413,547,920.00	0.68	66.00	19.00	0.43	0.02
34	**1.00**	**2,455,493,280.00**	**1,469,093,700.00**	**0.67**	**66.00**	**19.00**	**0.42**	**0.02**
35	1.02	2,574,691,860.00	1,536,057,540.00	0.68	66.00	19.00	0.42	0.02
36	1.04	2,652,961,260.00	1,590,880,200.00	0.67	66.00	18.00	0.41	0.02
37	1.06	2,810,688,060.00	1,668,167,160.00	0.68	66.00	18.00	0.41	0.02
38	1.08	2,920,258,260.00	1,741,300,920.00	0.68	66.00	18.00	0.40	0.02
39	1.10	3,028,470,360.00	1,808,090,580.00	0.67	66.00	17.00	0.40	0.02
40	1.12	3,202,422,720.00	1,876,292,280.00	0.71	66.00	16.00	0.40	0.02
41	1.14	3,327,439,380.00	1,940,906,340.00	0.71	66.00	14.00	0.39	0.02
42	1.16	3,363,271,200.00	1,984,445,220.00	0.69	66.00	14.00	0.39	0.02
43	1.18	3,423,891,180.00	2,026,088,940.00	0.69	66.00	14.00	0.39	0.02
44	1.20	3,467,808,780.00	2,069,601,300.00	0.68	66.00	14.00	0.38	0.02
45	1.22	3,561,045,960.00	2,118,293,340.00	0.68	66.00	14.00	0.38	0.02
46	1.24	3,620,094,420.00	2,159,061,600.00	0.68	66.00	14.00	0.38	0.02
47	1.26	3,670,673,160.00	2,202,700,740.00	0.67	66.00	13.00	0.38	0.02
48	1.28	3,697,030,860.00	2,235,596,580.00	0.65	66.00	13.00	0.37	0.02
49	1.30	3,775,394,760.00	2,280,563,280.00	0.66	66.00	13.00	0.37	0.02
50	1.32	3,836,621,340.00	2,318,629,320.00	0.65	66.00	12.00	0.37	0.02
51	1.34	3,869,787,720.00	2,354,428,920.00	0.64	66.00	12.00	0.37	0.02
52	1.36	3,925,019,580.00	2,391,836,580.00	0.64	66.00	12.00	0.36	0.02
53	1.38	3,970,213,560.00	2,422,225,380.00	0.64	66.00	12.00	0.36	0.02
54	1.40	4,057,630,980.00	2,467,406,220.00	0.64	66.00	12.00	0.36	0.02
55	1.42	4,187,229,240.00	2,519,522,520.00	0.66	66.00	12.00	0.36	0.02
56	1.44	4,218,024,660.00	2,545,467,300.00	0.66	66.00	11.00	0.36	0.02
57	1.46	4,288,999,140.00	2,576,631,000.00	0.66	66.00	11.00	0.35	0.02
58	1.48	4,306,388,280.00	2,599,126,920.00	0.66	66.00	11.00	0.35	0.02
59	1.50	4,367,218,620.00	2,629,070,880.00	0.66	66.00	11.00	0.35	0.02
60	1.52	4,404,346,440.00	2,648,703,000.00	0.66	66.00	11.00	0.35	0.02
61	1.54	4,474,803,720.00	2,680,932,960.00	0.67	66.00	10.00	0.35	0.02
62	1.56	4,527,581,640.00	2,701,913,160.00	0.68	66.00	10.00	0.35	0.02
63	1.58	4,550,804,340.00	2,725,125,660.00	0.67	66.00	10.00	0.35	0.02
64	1.60	4,603,865,700.00	2,746,975,260.00	0.68	66.00	10.00	0.34	0.02
65	1.62	4,614,483,000.00	2,761,413,600.00	0.67	66.00	10.00	0.34	0.02
66	1.64	4,646,090,640.00	2,778,886,080.00	0.67	66.00	10.00	0.34	0.02
67	1.66	4,684,588,140.00	2,803,595,100.00	0.67	66.00	10.00	0.34	0.01
68	1.68	4,741,298,040.00	2,824,517,160.00	0.68	66.00	9.00	0.34	0.01
69	1.70	4,767,852,240.00	2,840,613,900.00	0.68	66.00	9.00	0.34	0.01
70	1.72	4,771,771,500.00	2,851,273,020.00	0.67	66.00	9.00	0.34	0.01
71	1.74	4,820,883,660.00	2,869,044,660.00	0.68	66.00	9.00	0.34	0.01
72	1.76	4,829,619,240.00	2,880,805,800.00	0.68	66.00	9.00	0.34	0.01
73	1.78	4,871,816,040.00	2,894,122,020.00	0.68	66.00	8.00	0.34	0.01
74	1.80	4,882,044,600.00	2,906,587,020.00	0.68	66.00	8.00	0.34	0.01
75	1.82	4,887,719,100.00	2,918,169,120.00	0.67	66.00	8.00	0.34	0.01
76	1.84	4,906,277,280.00	2,931,178,260.00	0.67	66.00	8.00	0.33	0.01
77	1.86	4,913,790,480.00	2,938,407,780.00	0.67	66.00	8.00	0.33	0.01
78	1.88	4,948,946,880.00	2,954,128,860.00	0.68	66.00	8.00	0.33	0.01
79	1.90	4,960,187,460.00	2,966,468,160.00	0.67	66.00	8.00	0.33	0.01
80	1.92	4,971,077,340.00	2,974,254,120.00	0.67	66.00	8.00	0.33	0.01
81	1.94	4,992,843,540.00	2,988,470,760.00	0.67	66.00	8.00	0.33	0.01
82	1.96	5,009,055,840.00	2,997,736,260.00	0.67	66.00	8.00	0.33	0.01
83	1.98	5,011,681,020.00	3,008,642,640.00	0.67	66.00	8.00	0.33	0.01
84	2.00	5,015,555,640.00	3,014,613,540.00	0.66	66.00	8.00	0.33	0.01

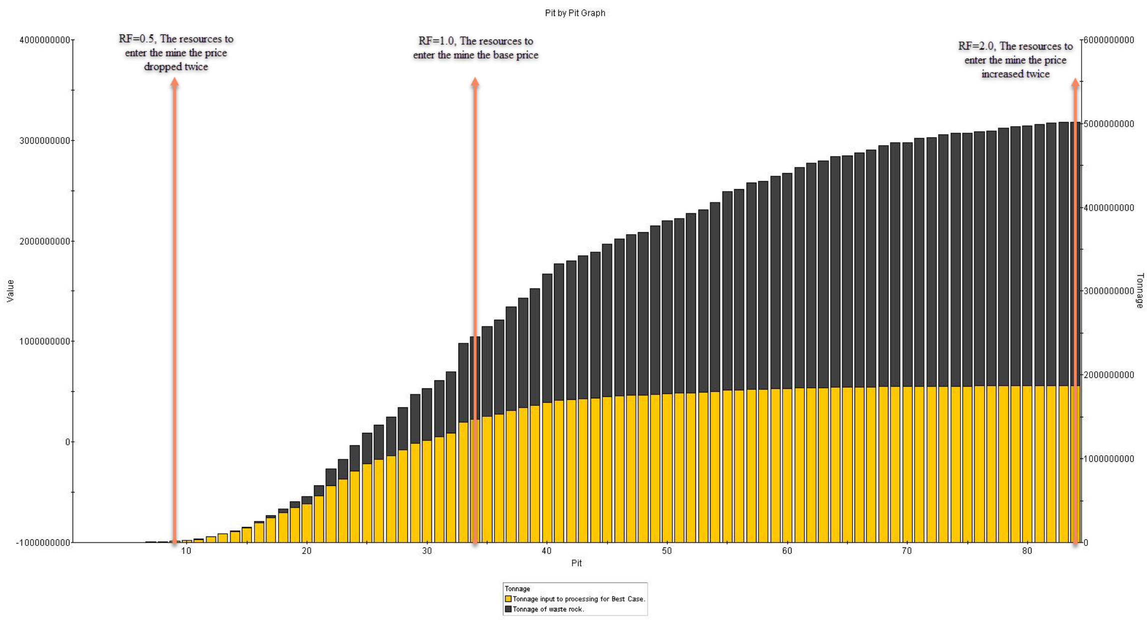

Figure 3-19 Nested pit shells by the open pit optimization

3.6.2. Cashflow analyses of specified case

The optimization of the open pit defines the optimal pit shells under mining, economic and other constraints, but the outcome is insufficient to reflect any changes in the life of mine. Therefore, it is important to determine the optimum value of the net present value NPV for the mine life period cash flow analysis. To calculate annual cash flows, a long-term product release plan is required to cover the life of mine. This planning is done under the "best case" and "worst case".

The "best case" planning is used for planning all the options within the specific case, and as a resulting the relatively low stripping ratio of the operation and the high grade of ore will have the highest net present value (NPV). This planning is difficult to implement in terms of mining practice (it is necessary to carry out mine operations in limited working conditions) and equipment utilization.

The "worst case" planning does not apply to any of the phases of the shells, and the mining operation scenario uses strip mining method by bench to bench within the final pit shell identified by the specific case. Mining operation will be used in the horizontal direction from the upper bench to bottom and the result will be relatively high in the early stages of waste rock and NPV will have the lowest value in this planning. This planning is simpler for use in practical aspects of mining, but it is difficult to implement the equipment usage.

Therefore, estimating both the planning and cash flows to the "Milwa Balanced", "Milawa NPV" and "Fixed Lead" algorithms between "best case" and "worst case" cash flows are more practical results. In that case, the discounted value of the specific case is taken as the mean value of a best and worst cases. Cash flow analysis reflects mining annual production capacity of 35 million tons of ore and discount rate of 10%. All other parameters are estimated to be the same as those used in the optimization of open pit.

Table 3.10. Cash flow analysis related to the specific case

Pit shells	Discounted open pit cash flow, mil.$ (dicount rate 10%)			Open pit cash flow	Ore	Waste	Mine life years		
	Best case	Specified case	Worst case	mil.$	mil.t.	mil.t.	Best case	Specified case	Worst case
1	2.02	2.02	2.02	2.02	0.05	0.00	0.00	0.00	0.00
2	6.49	6.49	6.49	6.49	0.15	0.00	0.00	0.00	0.00
3	16.07	16.07	16.07	16.09	0.42	0.00	0.01	0.01	0.01
4	28.62	28.62	28.62	28.68	0.80	0.00	0.02	0.02	0.02
5	54.65	54.65	54.65	54.91	1.69	0.00	0.05	0.05	0.05
6	103.50	103.50	103.50	104.53	3.62	0.00	0.10	0.10	0.10
7	178.41	178.41	178.41	181.87	7.05	0.17	0.20	0.20	0.20
8	249.23	249.23	249.23	256.61	10.71	0.20	0.31	0.31	0.31
9	347.20	347.20	347.20	362.94	16.28	0.68	0.47	0.47	0.47
10	502.34	502.34	502.34	540.40	26.82	0.87	0.77	0.77	0.77
11	658.43	658.30	658.30	724.95	38.88	0.99	1.11	1.11	1.11
12	987.57	981.68	981.68	1,128.45	69.64	2.05	1.99	1.99	1.99
13	1,306.21	1,287.78	1,287.78	1,551.93	104.31	4.37	2.98	2.98	2.98
14	1,535.42	1,502.38	1,502.38	1,881.68	133.03	6.88	3.80	3.80	3.80
15	1,777.42	1,720.36	1,720.36	2,268.26	171.00	8.88	4.89	4.89	4.89
16	2,099.99	1,987.68	1,987.68	2,859.44	234.89	15.28	6.71	6.71	6.71
17	2,321.22	2,145.26	2,143.65	3,343.46	293.12	25.55	8.37	8.37	8.37
18	2,498.80	2,248.63	2,248.63	3,804.72	353.04	46.85	10.09	10.09	10.09
19	2,643.92	2,315.52	2,296.74	4,248.08	417.08	70.37	11.92	11.92	11.99
20	2,731.13	2,337.33	2,284.37	4,554.93	465.60	85.86	13.30	13.30	13.52
21	2,851.76	2,339.68	2,239.79	5,087.90	561.56	114.44	16.04	16.04	16.46
22	2,954.38	2,352.34	2,144.83	5,738.88	681.69	194.28	20.07	19.48	20.23
23	2,996.94	2,334.32	2,053.75	6,084.46	754.35	237.93	22.26	21.55	22.58
24	3,037.16	2,299.15	1,950.00	6,518.49	856.18	305.24	25.35	24.46	25.69
25	3,058.97	2,260.40	1,828.38	6,829.44	940.51	365.24	27.92	26.87	28.36
26	3,069.15	2,252.70	1,743.26	6,997.59	990.88	407.73	29.36	28.31	30.04
27	3,076.24	2,245.63	1,668.07	7,140.33	1,039.67	457.32	30.95	29.71	31.62
28	3,083.56	2,238.73	1,595.75	7,306.99	1,106.42	506.46	32.86	31.61	33.72
29	3,089.17	2,256.87	1,498.28	7,470.26	1,186.44	581.66	35.16	33.90	36.31
30	3,090.93	2,246.16	1,436.64	7,528.15	1,220.81	620.97	36.23	34.88	37.46
31	3,092.39	2,226.77	1,372.99	7,585.12	1,265.49	668.73	37.69	36.16	38.89
32	3,093.19	2,291.07	1,288.22	7,624.63	1,306.30	727.24	38.99	37.33	40.33
33	3,093.23	2,370.69	1,046.21	7,698.29	1,438.69	940.33	44.52	41.11	45.01
34	**3,093.21**	**2,383.42**	**995.94**	**7,702.70**	**1,469.09**	**986.40**	**45.53**	**42.04**	**46.08**
35	3,093.03	2,369.97	908.55	7,694.98	1,503.62	1,071.07	47.07	43.19	47.44
36	3,092.87	2,359.43	860.02	7,680.23	1,530.37	1,122.59	48.23	44.17	48.40
37	3,092.36	2,371.30	759.90	7,634.79	1,573.36	1,237.33	50.52	45.84	50.10
38	3,091.93	2,357.60	719.79	7,583.61	1,612.58	1,307.68	51.96	47.15	51.38
39	3,091.57	2,350.71	660.24	7,529.12	1,641.09	1,387.38	53.47	48.58	52.51

40	3,091.05	2,345.08	573.42	7,444.93	1,672.39	1,530.03	55.88	50.32	53.88
41	3,090.70	2,311.25	523.92	7,367.19	1,697.05	1,630.39	57.64	51.62	54.88
42	3,090.57	2,331.33	510.18	7,337.63	1,705.36	1,657.91	58.15	52.06	55.20
43	3,090.41	2,329.61	480.30	7,291.58	1,716.59	1,707.30	58.99	52.72	55.72
44	3,090.27	2,318.22	459.44	7,250.59	1,726.01	1,741.80	59.61	53.22	56.13
45	3,090.02	2,301.27	417.72	7,170.56	1,740.81	1,820.24	60.93	54.02	56.82
46	3,089.87	2,301.34	391.21	7,117.26	1,748.78	1,871.31	61.76	54.60	57.24
47	3,089.73	2,298.77	371.51	7,060.86	1,757.26	1,913.42	62.47	55.12	57.64
48	3,089.65	2,296.67	360.38	7,031.06	1,761.16	1,935.87	62.84	55.43	57.85
49	3,089.48	2,298.32	331.33	6,953.11	1,770.63	2,004.76	63.95	56.18	58.39
50	3,089.33	2,295.24	307.17	6,883.47	1,778.30	2,058.32	64.81	56.80	58.88
51	3,089.26	2,295.67	294.02	6,845.28	1,782.41	2,087.38	65.27	57.15	59.16
52	3,089.13	2,295.92	273.10	6,774.86	1,789.44	2,135.58	66.06	57.70	59.63
53	3,089.04	2,294.32	256.56	6,721.14	1,794.10	2,176.11	66.69	58.13	59.99
54	3,088.87	2,286.81	222.14	6,615.14	1,803.09	2,254.55	67.92	59.01	60.83
55	3,088.66	2,286.65	175.47	6,460.15	1,818.90	2,368.33	69.75	60.33	62.21
56	3,088.61	2,286.24	164.14	6,422.71	1,822.02	2,396.01	70.18	60.63	62.54
57	3,088.52	2,284.08	141.62	6,345.07	1,828.10	2,460.90	71.18	61.32	63.24
58	3,088.50	2,287.94	135.90	6,322.40	1,829.57	2,476.82	71.42	61.45	63.40
59	3,088.42	2,287.03	115.35	6,250.68	1,834.68	2,532.54	72.28	62.12	64.02
60	3,088.38	2,287.20	103.66	6,205.19	1,837.50	2,566.85	72.80	62.60	64.42
61	3,088.30	2,287.80	83.49	6,116.58	1,842.62	2,632.18	73.80	63.50	65.12
62	3,088.25	2,300.50	67.20	6,052.61	1,846.90	2,680.68	74.54	64.18	65.73
63	3,088.23	2,284.31	61.15	6,022.34	1,848.31	2,702.49	74.87	64.48	65.98
64	3,088.18	2,284.00	46.81	5,954.27	1,851.82	2,752.04	75.61	65.17	66.56
65	3,088.17	2,283.31	44.30	5,940.36	1,852.58	2,761.90	75.76	65.29	66.68
66	3,088.14	2,282.89	36.56	5,900.26	1,854.53	2,791.57	76.21	65.75	67.05
67	3,088.11	2,283.14	26.85	5,849.78	1,856.85	2,827.74	76.75	66.28	67.48
68	3,088.06	2,307.54	15.28	5,779.16	1,860.12	2,881.18	77.55	67.00	68.02
69	3,088.04	2,302.20	9.30	5,741.62	1,861.31	2,906.55	77.92	67.38	68.31
70	3,088.04	2,302.17	8.04	5,736.60	1,861.51	2,910.27	77.98	67.43	68.35
71	3,088.01	2,304.91	-4.94	5,674.88	1,864.87	2,956.01	78.67	68.11	68.96
72	3,088.00	2,304.09	-5.88	5,663.36	1,865.33	2,964.29	78.79	68.23	69.05
73	3,087.97	2,305.42	-11.95	5,607.14	1,867.10	3,004.72	79.39	68.74	69.46
74	3,087.96	2,301.64	-14.01	5,593.58	1,867.68	3,014.36	79.53	68.86	69.56
75	3,087.96	2,304.51	-15.69	5,585.75	1,867.87	3,019.85	79.61	68.95	69.63
76	3,087.95	2,305.06	-20.49	5,559.93	1,868.82	3,037.45	79.87	69.27	69.85
77	3,087.94	2,305.48	-22.53	5,549.36	1,869.17	3,044.63	79.98	69.35	69.93
78	3,087.92	2,303.40	-31.16	5,502.59	1,871.07	3,077.88	80.47	69.81	70.39
79	3,087.91	2,302.97	-34.53	5,486.59	1,871.53	3,088.66	80.63	69.96	70.52
80	3,087.91	2,301.22	-36.97	5,471.62	1,871.90	3,099.18	80.79	70.16	70.65
81	3,087.90	2,295.52	-41.03	5,440.59	1,872.48	3,120.36	81.09	70.42	70.85
82	3,087.89	2,300.82	-44.34	5,417.46	1,873.10	3,135.96	81.32	70.61	71.09
83	3,087.88	2,299.20	-44.69	5,413.52	1,873.16	3,138.52	81.36	70.64	71.12
84	3,087.88	2,304.41	-45.43	5,408.46	1,873.30	3,142.26	81.41	70.68	71.19

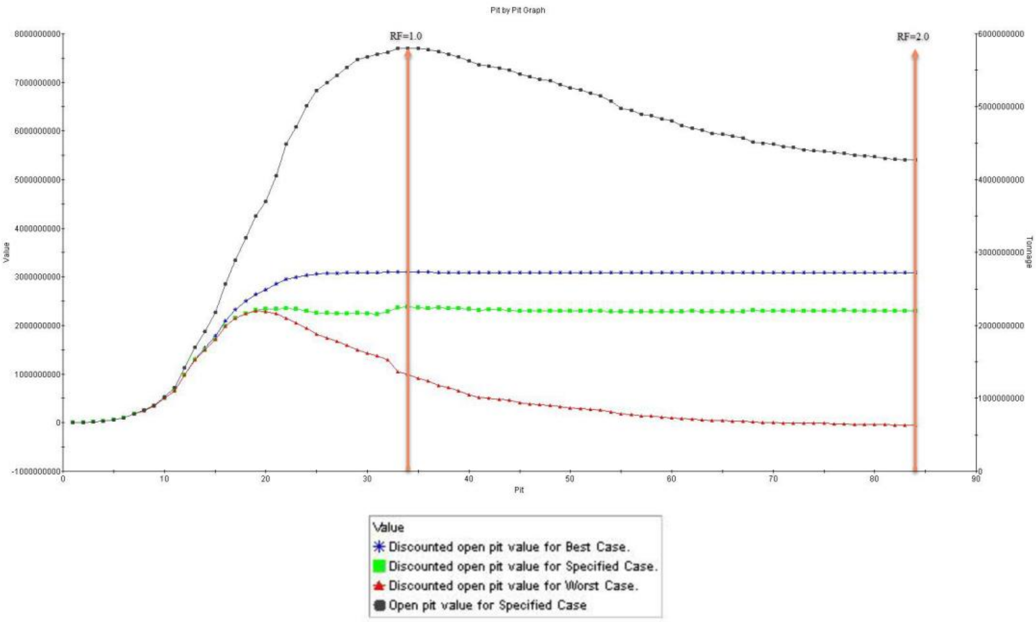

Figure 3-20 Results of cash flow analysis

The results show that the Pit shell-34 (RF=1.0) covers the maximum net present value (NPV). And the result differences between the Pit shell-34 (RF=1.0) and Pit shell-84 (RF=2.0) are 79 M$ of NPV (2294 M$ undiscounted cash flows) and 2155.8 Mt of waste rock.

3.6.3. Selection of the ultimate pit limit

Selecting the ultimate pit limit of mining based on the key indicators below:

- Maximum cash flow
- Ore reserves for maximum production
- Maximum lifetime
- Price risk for sale

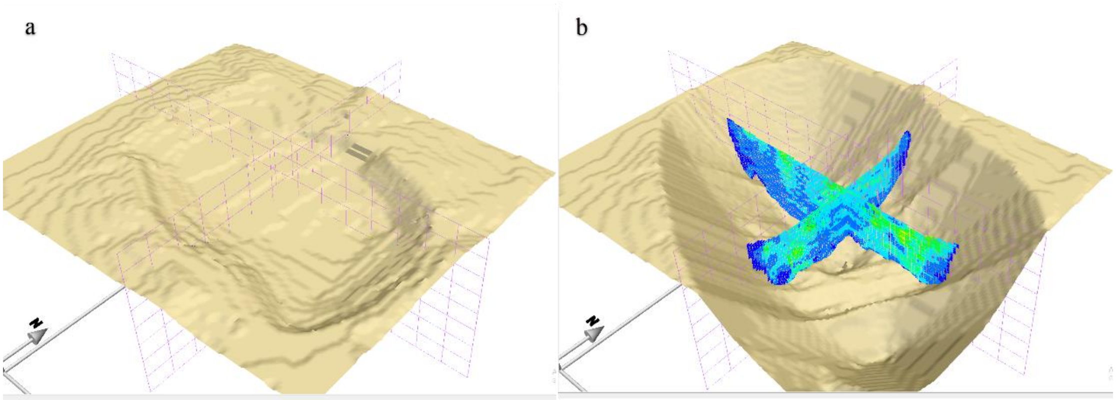

Figure 3-21 Nested pit shells a. Pit shell-1 (RF=0.34) b. Pit shell-84 (RF=2.0)

Based on the optimization calculations and cash flows analysis, the Pit shell-34 (RF=100%) was selected by ultimate pit limit. In this scenario, the NPV was relatively high and 1,469.09 million tons of sulphide ore with 0.42% average content of Cu and 0.02% average content of Mo. The stripping ratio 0.67 t/t and the life of mine is 42 years if the mining capacity is 35 Mt.

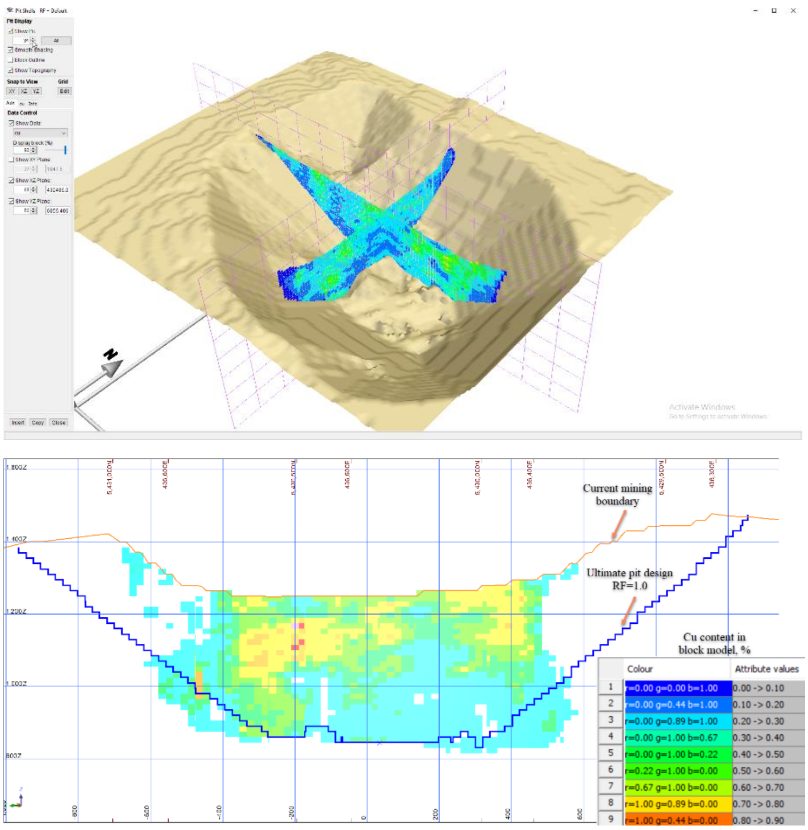

Figure 3-22 Ultimate pit design, Pit shell-34 (RF=1.0)

3.6.3. Sensitivity analysis

An additional sensitivity analysis studied on operational costs, metal prices and the overall slope angle for the Lerch-Grossman's analysis. The sensitivity analysis conducted on costs and prices was calculated from the specified case to + 30%, -30% by 5% intervals. Figure 3-23 – 3-25 shows results of the sensitive analysis.

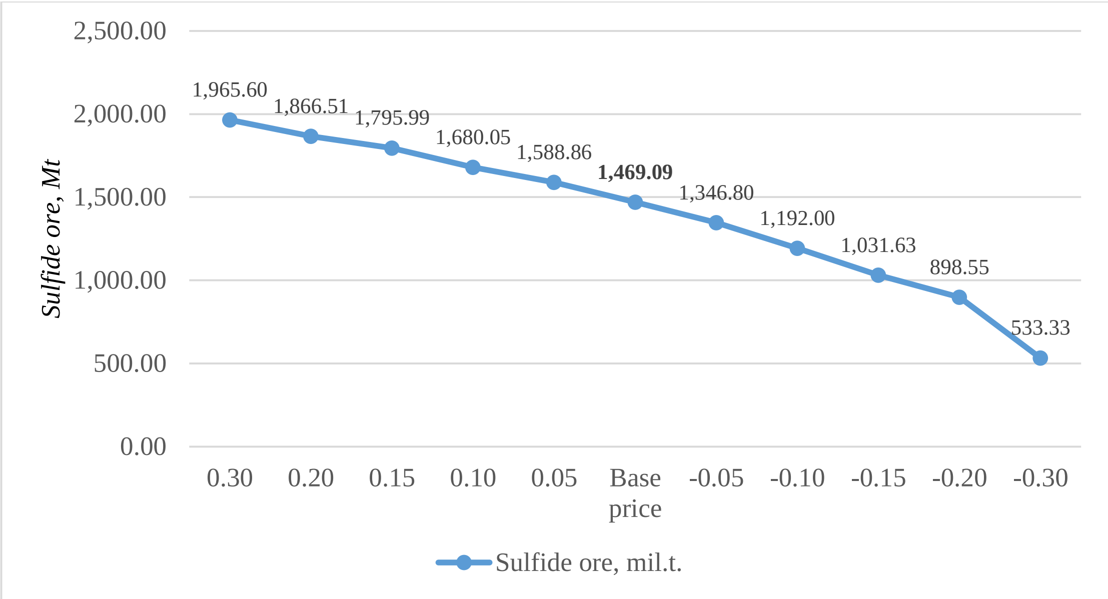

Figure 3-23 Sensitivity analysis of metal prices

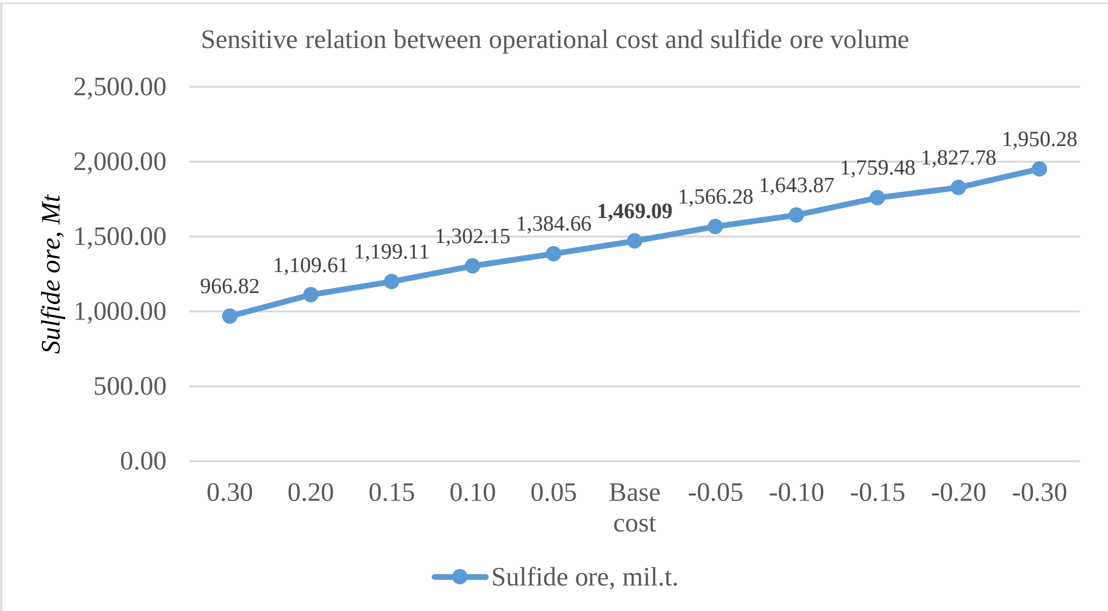

Figure 3-24 Sensitivity analysis of operational cost

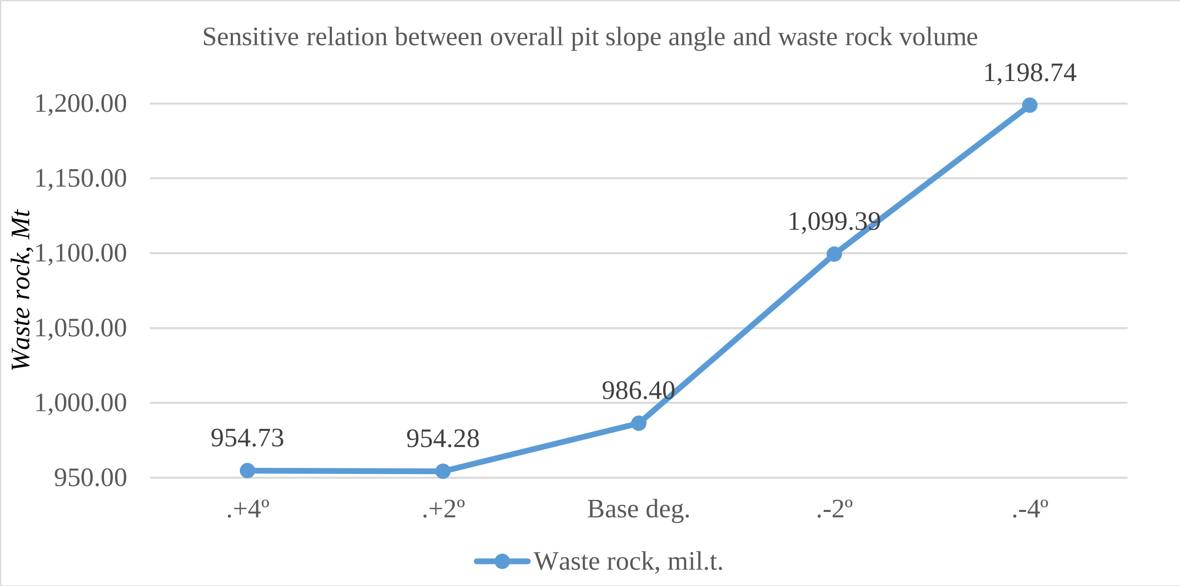

Figure 3-25 Sensitivity analysis of overall pit slope angle

In general, resource in open pit mine is the most sensitive to metal prices. When the metal price drops to 30%, while the sulfide ore decreases to 935 million tonnes and increases by 30% to 497 million tonnes. In addition, the sensitivity depends on operating costs also high as shown in the graph below. Increasing the overall slope angle of the open pit, amount of the waste rock decreased very low. However, decreasing the overall slope angle of the open pit, amount of the waste rock decreased quite high.

3.6.4. The prospective boundaries of open pit

In the future, there is a possibility to increase the market price of the minerals, reduce costs, and introduce new technologies that will allow the full range of resources to be expanded. Determining the location of the waste dumping and surface infrastructure and constructions based on the established open pit boundaries is quite risky. Because the open pit boundary is quite variable and is constantly changing from the beginning of the mine life to the end.

According to open pit optimization analysis, the Erdenet mine waste rock dumps have developed within the boundary of the economic benefits of the mine and there is a need for additional work to re-locate the waste dumping of 15 Mm^3.

Therefore, it is necessary to define the mine's future prospects and finalize the locations of waste dumps and surface infrastructure and constructions. The scope is based on long-term price forecasts for the sale of products and is predicted to be at an average of 8,994.98 $/t. and maximum of 12,500.00 $/t. for total mine-life expectancy. Figure 3-26 and Table 3.11 shows the historical Cu price and forecast.

Table 3.11. Historical Cu price in last 40 years (www.infomine.com/)

Year	Cu price, $/t.	Year	Cu price, $/t.	Year	Cu price, $/t.
1961	659.64	1981	1,846.22	2001	1,578.12
1962	674.61	1982	1,607.35	2002	1,559.38
1963	674.61	1983	1,716.52	2003	1,778.99
1964	704.59	1984	1,471.72	2004	2,865.62
1965	771.98	1985	1,445.47	2005	3,678.82
1966	797.40	1986	1,425.32	2006	6,721.94
1967	842.73	1987	1,787.86	2007	7,139.13
1968	922.56	1988	2,625.81	2008	6,955.01
1969	1,047.93	1989	2,855.71	2009	5,149.12
1970	1,272.05	1990	2,684.41	2010	7,166.25
1971	1,133.89	1991	2,341.93	2011	7,210.35
1972	1,115.90	1992	2,286.59	2012	7,960.05
1973	1,297.45	1993	1,789.36	2013	7,164.95
1974	1,689.80	1994	2,307.31	2014	7,054.72
1975	1,400.69	1995	2,935.30	2015	5,070.58
1976	1,517.29	1996	2,294.08	2016	4,497.38
1977	1,450.80	1997	2,276.44	….	
1978	1,444.23	1998	1,653.97	….	
1979	2,033.39	1999	1,572.83	….	
1980	2,235.82	2000	1,813.39	***2066***	***12,500.00***

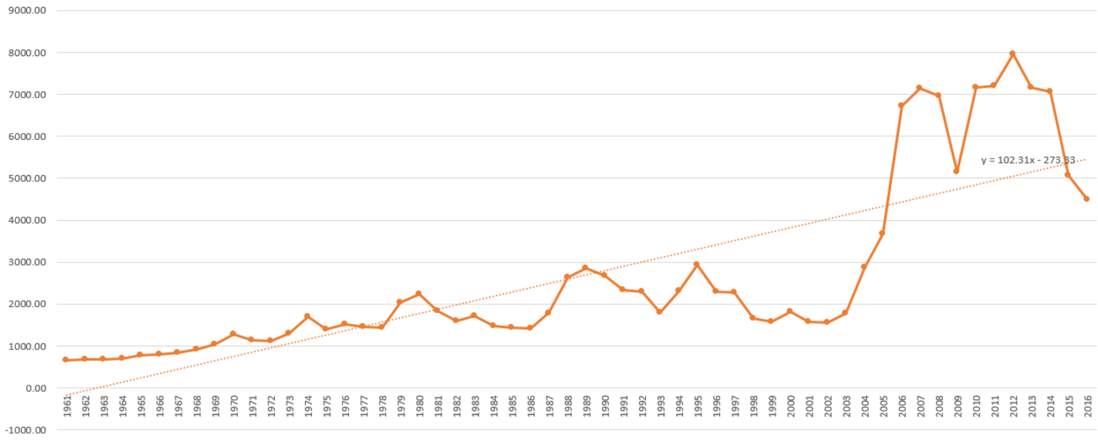

Figure 3-26 Historical Cu price in last 40 years (www.infomine.com/)

The results of the trend analysis of historical Cu price data and the nested pit optimization analysis shows the preliminary description of Pit shell-59 with RF=1.5 could be a prospective boundary design. Figure 3-27 and Table 3.12 shows the prospective pit shells.

Table 3.12. Prospective pit shells

Pit shells	RF	Cu price, $	Discounted open pit cash flow, mil.$ (dicount rate 10%)			Ore	Waste	Mine life years		
			Best case	Specified case	Worst case	Mt	Mt	Best case	Specified case	Worst case
34	1	6,000	3,093.21	2,383.42	995.94	1,469.09	986.4	45.53	42.04	46.08
59	**1.5**	**9,000**	**3,088.42**	**2,287.03**	**115.35**	**1,834.68**	**2,532.54**	**72.28**	**62.12**	**64.02**
84	2	12,000	3,087.88	2,304.41	-45.43	1,873.30	3,142.26	81.41	70.68	71.19

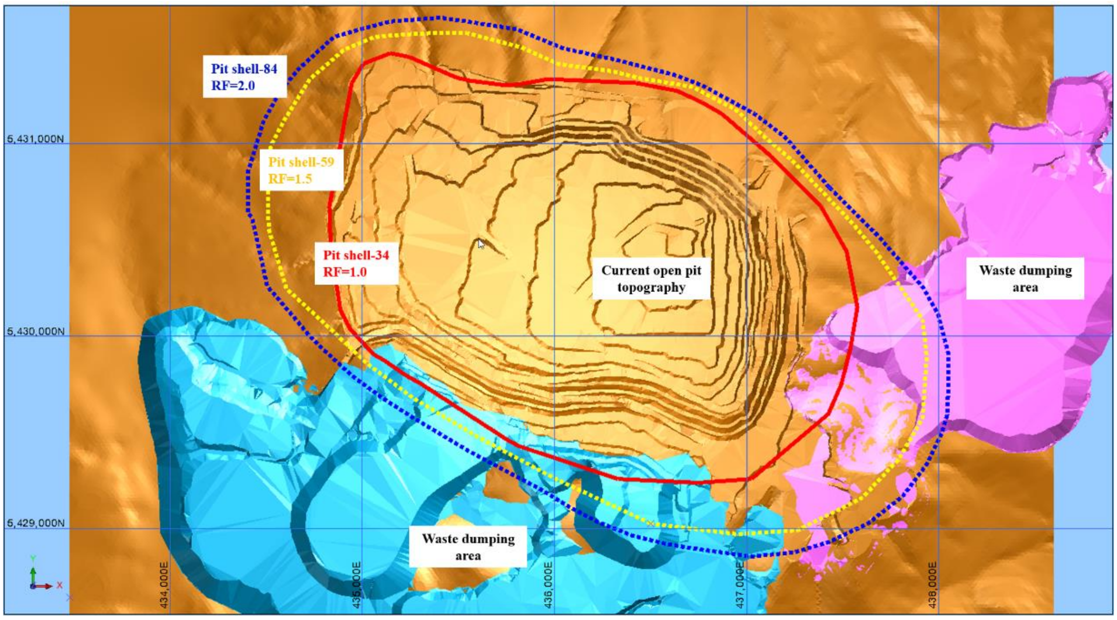

Figure 3-27 Pit shells RF=1.0, RF=1.5, RF=2.0 and current open pit mining

3.7. Summary

The large-scale open pit mining is a complex operation that may extend over several decades and require very large investments. Many factors govern the size and shape of an open pit. The pit slope, which may vary throughout the deposit, is one of the key factors governing the amount of waste to be removed in order to gain access to the mineral deposit. Small changes in slope angle can change the amount of waste to be removed and significantly affect the degree of selectivity in mining operations. For this reason, it is very important to change slope angles through the deposit in order to follow different structures and rock types and keep the total amount of waste as small as possible.

The dataset obtained to the block model for this study was exported from the geological database of the Erdenetyn-Ovoo Cu-Mo and basic economic indicators of several copper mining companies of Mongolia. The block model for this study estimated by basic size of the each block in the model is 20m x 20m x 15m by Geovia Surpac, total count of blocks are 430,117 and the 3D geological and deposit block model conducted by several attributes such as ore grade, volume, lithology, structures and rock types. Following this, the Rocscience Dips software was used for kinematic analyses. The purpose of these analyses was to identify the kinematically possible failure modes within each domain sector and to set appropriate overall slope angle for belonging different rock type domains for the Lerch-Grossman open pit optimization using the stereographic technique. From the results of kinematic analyses, probability of toppling and planar failures was relatively low in applied overall slope angles and wedge failures are kinematically possible in higher overall slope angles of 40° for most sections. The Geovia Whittle software was used in order to establish the final pit limit in terms of the maximum Cashflow, Net Present Value (NPV) and associated pushbacks to produce a best case mining scenario. To do this, all the required data such as grade, density and rock type and financial data were entered as numerical values into each of the deposit's block models.

From the results, Pit shell-34 with Revenue Factor=100% covers the maximum net present value (NPV). And the result differences between the Pit shell-34 (RF=1.0) and Pit shell-84 (RF=2.0) are 79 M$ of NPV (2,294.0 M$ undiscounted cash flows) and 2,155.8 Mt of waste rock. From the sensitive analysis, resource in open pit mine is the most sensitive to metal prices. When the metal price drops to 30%, while the sulfide ore decreases to 935 million tonnes and increases by 30% to 497 million tonnes. Increasing the overall slope angle of the open pit by 4°, amount of the waste rock decreased very low as 3.3%. However, decreasing the overall slope angle of the open pit by 4°, amount of the waste rock increased quite high as 21.5%.

The current concept of the Erdenet Mining Corporation has a total of 950 Mt of ore at the open pit mine depth of 905m. The results of pit optimization analysis show the possibility of open pit mine depth considering stability condition reach to the elevation of 780m which allows 125m more depth and to allow more than 550 Mt of ore reserve to be exploited by the current concept of Erdenet mining.

Determining the location of the waste dumping and surface infrastructure and constructions based on the established open pit boundaries is quite risky. Open pit mining boundary is relative variable and is constantly changing from the beginning of the mine life to the end. The size, location and final shape of open pit should be optimized based on prospective production prices and open pit revenue factors are important in planning the location of waste dumps, stock piles; processing plant, access roads and other surface constructions, facilities and infrastructures.

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CHAPTER IV. FORMATION MECHANISM OF BENCHES ON STABILITY OF LARGESCALE DUMPING

4.1. Introduction

The removal of waste rock is the first step in a mining operation, so as to expose underlying valuable mineral for excavation. The waste rock material being a waste, nonmarketable and low-grade products, it is removed and dumped safely and economically. The primary aim for construction of waste rock dump is to provide an effective stable working surface for the dump deposit.

Waste rock dumps can be external pit dumps created at a site away from the mining area or it can be internal created by in-pit dumping concurrent to the creation of voids by extracted area. Practice of waste rock dumping in the external dumps has some serious problems. Foremost among them are requirement of additional land which involves very high transport and re-handling cost. Therefore, it increases the cost of mining production substantially, stability and reclamation at the site.

Large scale, open pit mining in Mongolia involves the mobilization of large amounts of waste material from the pit to areas specifically prepared to store this material. Waste dumps, as opposed to dams, are generally built simply by overturning truckloads, which produce a low initial density. Additionally, the particles of material can reach sizes in the order of meters, which is not common in dumping construction.

The formation mechanism of benches on stability of dumping area to optimize waste dump design will be discussed in this Chapter. At the Erdenet open pit mine, the geological overview, dumping operation, waste particle distribution, and stable problems were investigated. Then, a series of the experiments was conducted in the laboratory to simulate the formation process of single bench, multiple benches, and the efficiency of dumping operation's design. Finally, the relationship between safety factor of dumping area and bench height, bench angle, bulk factor of waste rock, and truck transport were simulated by using numerical simulation.

Dumping operation at Erdenet open pit mine. The Erdenet Cu-Mo open pit mine utilizes shovel-truck-bulldozer dumping system for waste rock, low-grade ore and oxidized ore. Various rocks are found in the middle of the overburden. Figure 4-2 shows the dumping area of shovel-truck-bulldozer dumping operation. The particle size distribution varies widely. Generally, small blocks are centralized in the top bench and large blocks are located in the bottom bench. The waste rocks are dumped from the truck to the ground surface and slope. Afterwards, waste materials roll from the slope top to bottom. Bulldozers push the remaining materials from the ground surface to the slope.

Figure 4-1 Shovel-truck-bulldozer dumping operation

Table 4.1. Parameters for shovel-truck-bulldozer dumping operation

Parameters	Unit	Value	Parameters	Unit	Value
Bench height	m.	50-100	Final slope angle	deg.	20
Rock roll distance	m.	18	Minimum working width	m.	90~95
Repose angle	deg.	35-38	Bulk factor		1.2

Figure 4-2 Dumping area of Erdenet mine

4.2. Waste particle size distributions

The size and shape are two major parameters in waste particle. In the blasting operation of open pit mining, the size of waste particle is easy to be controlled, while the shape of particle size is not easy to be controlled. Therefore, in this Chapter, the shape of waste particle was not considered in this research. We selected 7 pictures from separate benches in the shovel-truck-bulldozer and dragline dumping areas. The waste particle size distribution is analyzed with Spilt-Desktop Version 2.0 software. Table 4.2 shows the analysis results of waste particle size distribution for the shovel-truck-bulldozer and dragline dumping areas. The table shows that the range of waste main particle sizes of is 126.5 to 391.3 mm.

Table 4.2. Analysis results of waste particle size distribution for the shovel-truck-bulldozer and dragline dumping areas

Material No.	P20 (mm)	P50 (mm)	P80 (mm)	Top size (mm)
1	88.7	177.1	362.1	642.4
2	109.9	199.2	300.0	524.4
3	151.4	264.7	393.9	577.9
4	91.4	146.7	293.2	565.5
5	143.1	274.0	539.8	845.4

6	186.0	302.9	482.1	800.6
7	114.7	215.0	368.0	664.7
Average	*126.5*	*225.7*	*391.3*	*660.1*

Note: P20 – Corresponds to the size in which 20% of the total mass is represented by smaller particles. P50 – Corresponds to the size in which 50% of the total mass is represented by smaller particles. P80 – Corresponds to the size in which 80% of the total mass is represented by smaller particles.

4.3 Experiment methods and results

4.3.1 Setting of tests

Experimental tests are useful to prove and verify theories (Bellaloui and Chtaini, 1999; Krahn et al., 1989; Pelkey et al., 2001). Experiments were conducted to clarify the mechanism of bench formation on stability of dumping area. Figure 4-3 provides an overview of the experimental device. A certain volume sample is dumped from a specified height with a constant initial velocity. Then, the pile shape is measured. To maintain a constant frictional force at the floor surface, a felt mat is placed on the plate surface. The particle size distribution, dumping volume, dumping height, floor angle, and interval distance can be set in the device.

The experimental procedures are as follows:

a) Select a group of samples that correspond to a particle size range with a sieve.
b) Measure a constant volume sample with a graduated cylinder.
c) Place the constant volume samples into the bucket.
d) Set the dumping height and floor angle.
e) Start the crane and dump them on the floor with a constant initial velocity.
f) Measure the pile height, length, width, and angle.
g) Repeat b) through f) 5 times. Record data and calculate averages.

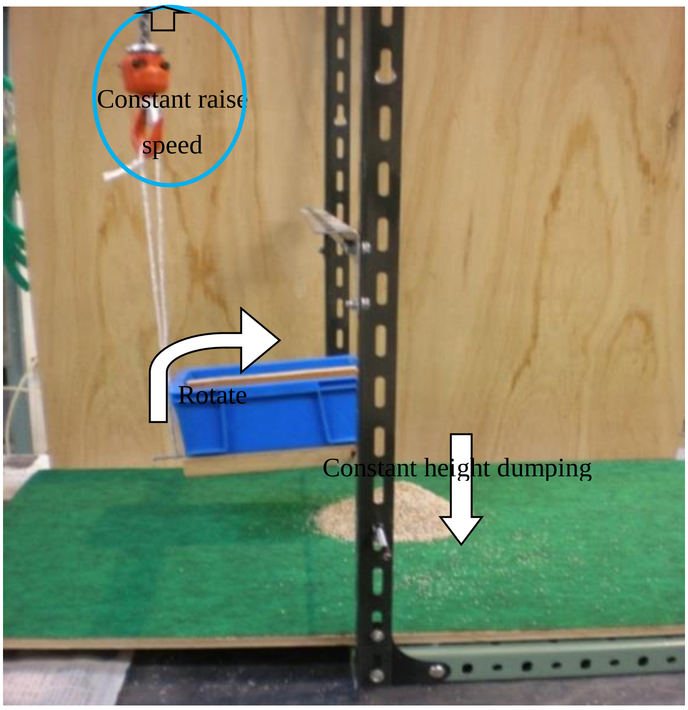

Figure 4-3 ***Overview of the experimental device***

Figures 4-4 and 4-5 show the measurement process in the experimental tests. At first, the effects of particle size, dumping height and drop volume on the formation of single bench were discussed. Secondly, multiple benches are formed and then the interaction between adjacent benches was discussed under different interval distances. In order to eliminate the effect of water contents of samples, all samples are dried before the test. Gravel and coarse sand were used in this test as the waste materials. Scale factor was set as 80 based on the results of field investigations and the condition of laboratory test. The average values of P20, P50, and P80 are converted to 1.58 mm, 2.82 mm, and 4.89 mm, respectively. According to the opening of standard sieve, the particle sizes of samples are set from 1.00-1.68 mm, 1.68-2.83 mm and 2.83-4.75 mm. A certain volume of a sample was dropped from a certain height to the floor and then the dimension of the pile was measured. A felt mat was laid on the floor in order to maintain the frictional force between bench/waste materials and floor constant. In this device, drop volume, drop height, drop rate and floor angle can be controlled.

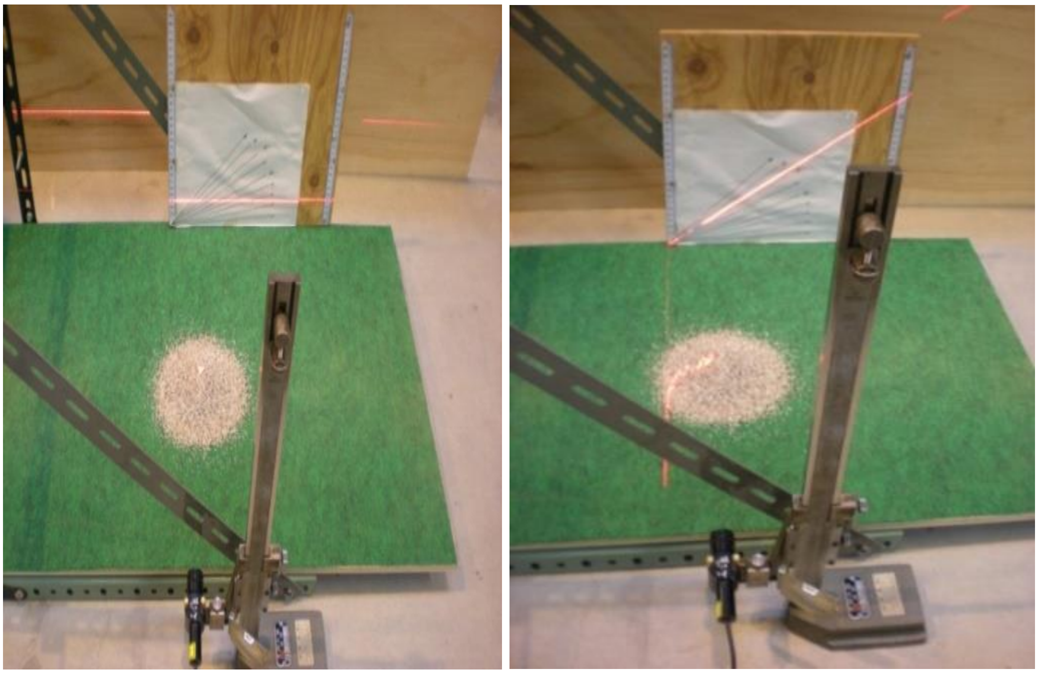

Figure 4-4 Process to measure the height and the repose angle

Figure 4-5** **Process to measure the length and width

4.3.2 Single bench

At first, effects of a particle size, a drop height, a drop volume, and a dip of floor on the formation of single bench are discussed.

Effect of a particle size and a drop height on the shape of bench

The relationship between the pile height / slope angle of bench and the drop height under different particle sizes of waste materials are shown in Figures 4-6 and 4-7, respectively. Slope angle is angle of repose of the bench.

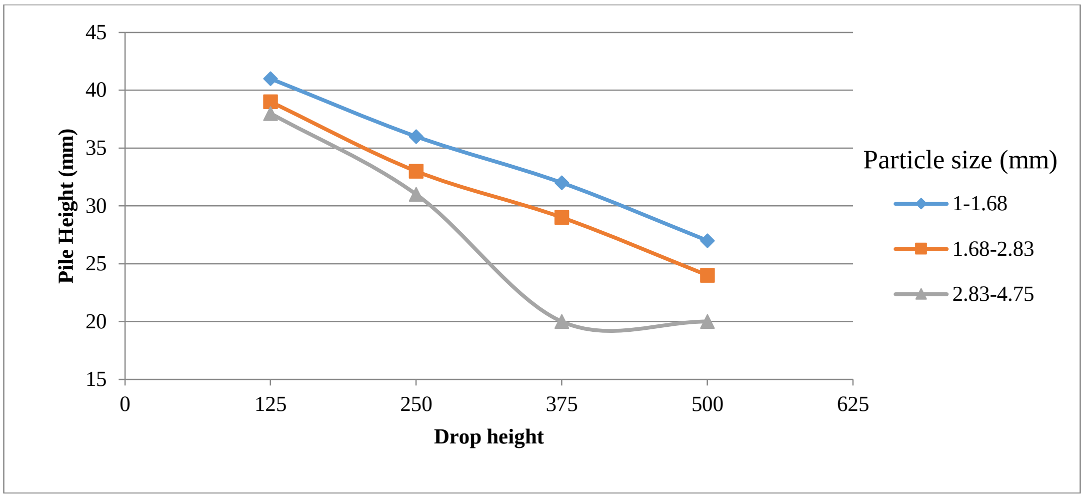

Figure 4-6 Relationship between pile height and drop height

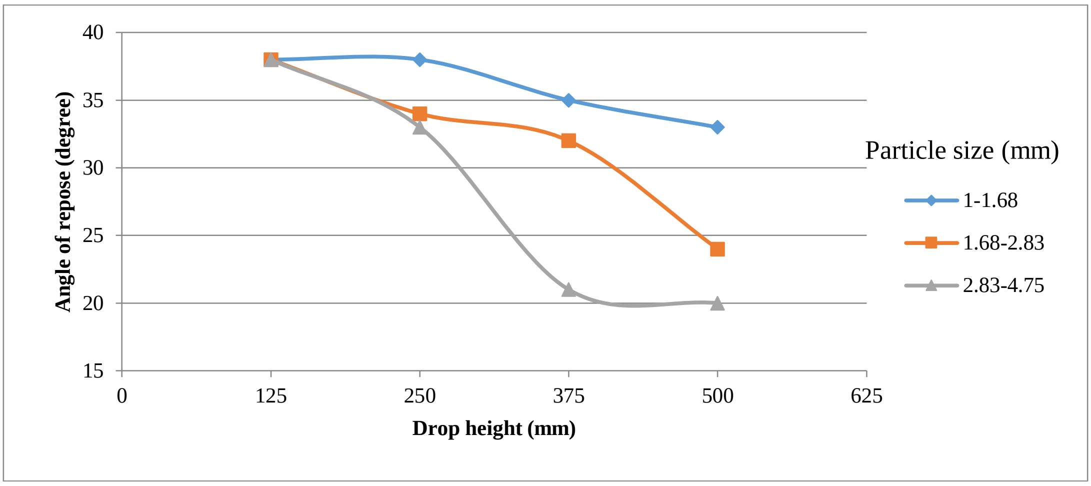

Figure 4-7 Relationship between slope angle and drop height

Figure 4-6 and 4-7 show the larger particle size is, the lower the bench height is and the smaller the slope angle is. It can be said that the larger the particle size, the more weight the dropping material and more inertia force they have, so the materials are well spread. The figures also show that the higher the drop height is, the lower the bench height is and the smaller the slope angle is. The velocity of dropping material when it reaches at the top of the pile increases with increasing the drop height. As a kinetic force of friction is fixed, dropping material rolled the slope faster and reached farther away. Moreover, as their impact force also increases with increasing of the drop height, waste material was compacted and the bench height was decreased. From the above results, it can be concluded that both particle size and dropping height have an obvious impact on the formation of bench.

Based on above laboratory tests and site investigation, the following conclusions were found:

a) The design of blasting needs to be changed to make the waste particle size of smaller. In the dumping area, the bench repose angle decreases and the roll distance of rock increase with increasing the particle size of the waste. The volume capacity of bench and stability of the dumping area decreases with decreasing the bench repose angle. Moreover, the safety distance by rock roll increases with increasing the roll distance of rock.

b) The volume capacity of bench is maximized and the safety distance by rock roll is minimized in the bucket wheel excavator-belt-stacker dumping operation. The value of shovel-truck dumping operation is in the mid-level. The volume capacity of bench is minimized and the safety distance by rock roll is maximized in the dragline dumping operation.

Effect of a drop volume on the shape of bench

The relationship between the pile height/slope angle and the drop volume of sample is given in Figure 4-8.

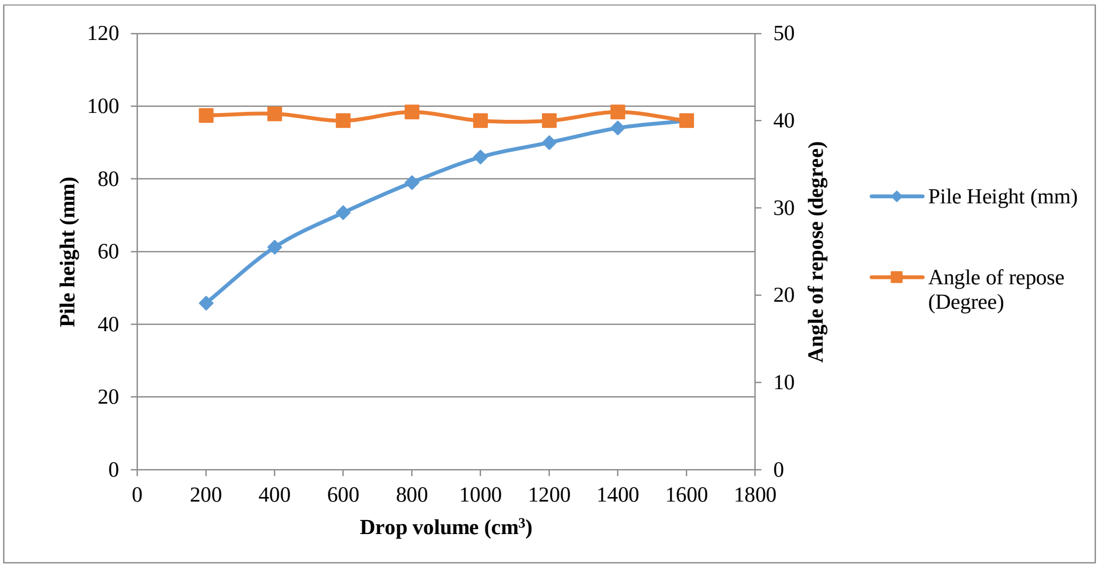

Figure 4-8 Relationship between pile height/slope angle and drop volume

It can be seen from this figure that the pile height increases with increasing the drop volume. However, there are no obvious changes on the slope angle and the bench shape due to the change of drop volume. Hence, it can be said that a drop volume do not have an obvious impact on the shape of bench.

Effect of floor dip on the shape of bench

Figure 4-9 shows length, height and repose of angle of bench. Table 4.3 shows the relationship between dips of floor and bench shape. Pile length is a length of tendency direction; pile width is a length of strike direction.

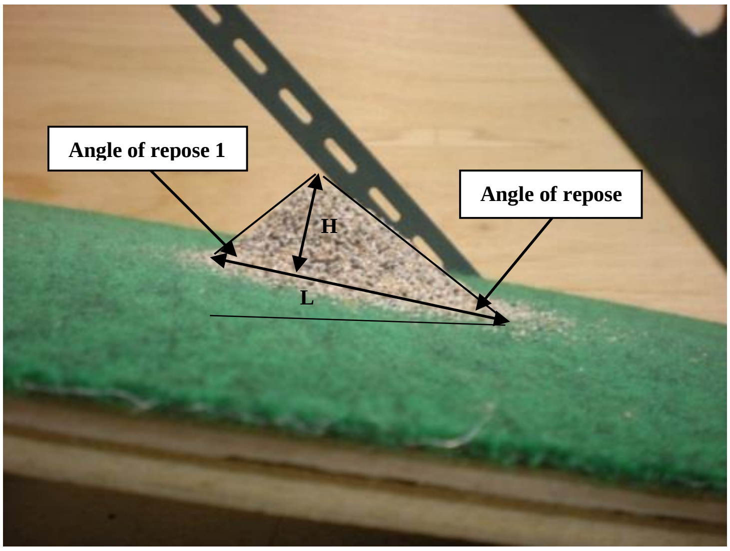

Figure 4-9 Diagram of length, height and repose of angle of bench

Table 4.3. Relationship between dips of floor and bench shape

Dip of foundation (deg.)	Pile height (mm)	Angle of repose 1 (deg.)	Angle of repose 2 (deg.)	Pile length (mm)	Pile width (mm)
20	32	37	21	155	145
15	37	39	23	145	145
10	40	40	30	140	140
5	41	40	32	140	145
0	40.5	38	36	143	153

Based on the above laboratory tests and site investigations, the following measures were found:

a) In the selection process of dumping area, the smaller the floor inclination is better than that of higher inclination. The bench height decreases and bench length increase with increasing the floor inclination. Moreover, the capacity of bench decreases and the roll distance of rock increases with increasing the floor inclination.

b) Some measures should be taken to increase the kinetic force of friction between waste material and floor surface. The kinetic force of friction between waste material and floor surface is no more than the kinetic force of friction of waste material. It is easy to a weak seam. Generally the inclined floor is flat and an entirety surface thus it is easy to be a slide surface. For example, in the treatment process of floor, we can remove the loose earth and all vegetation to make the floor strong seam. Some pits can be formed in the floor's surface with blasting method to make the floor not easy to be a slide surface.

4.3.3 Multiple benches

Next, the formations of multiple benches were investigated under different conditions. Figures 4-10 and 4-11 show the diagram of dumping order and drop interval distance and that of pile height and their difference, respectively.

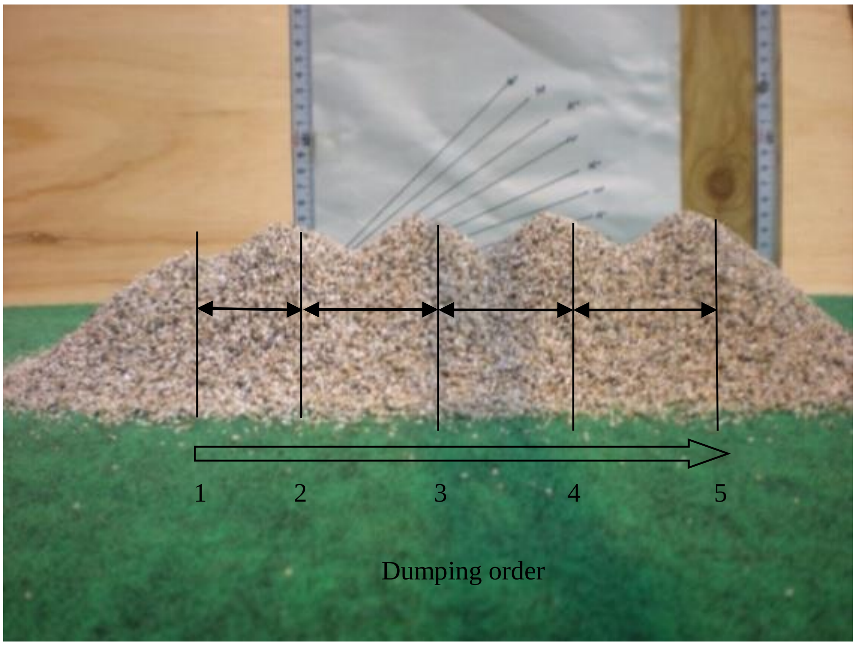

Figure 4-10 Diagram of dumping order and interval distance

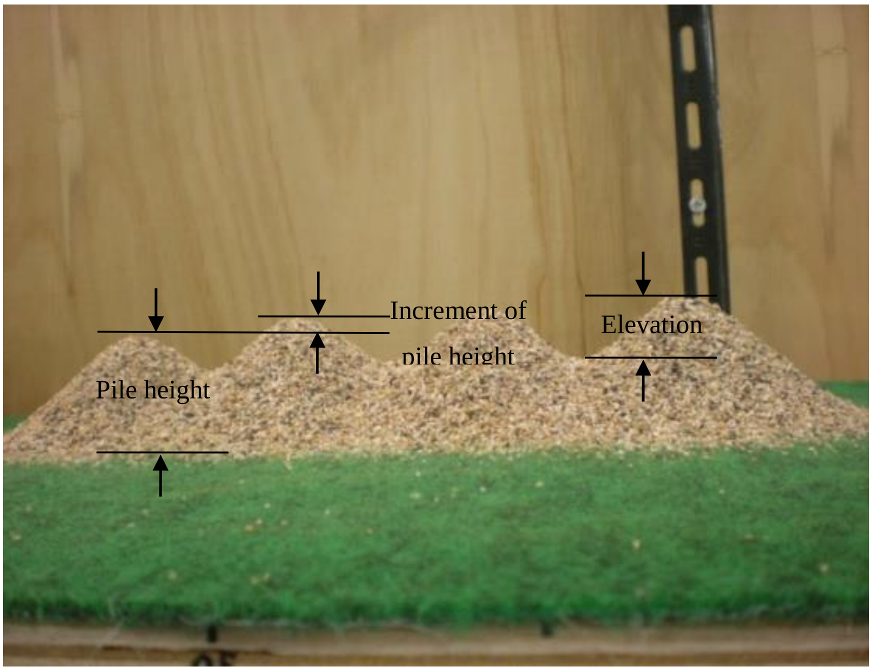

Figure 4-11 Diagram of elevation difference and pile height measurements

Figure 4-12 shows the relationship between the pile height and the pile number under different drop intervals. Pile numbers indicates the first, second, third, fourth and fifth bench. Here, the increasing of the pile height means the difference between the height of each bench and that of first one. It represents that the height of bench increases with the increasing pile number. The height of bench increases gradually and then becoming constant. Moreover, the shorter the drop interval is, the higher the bench is.

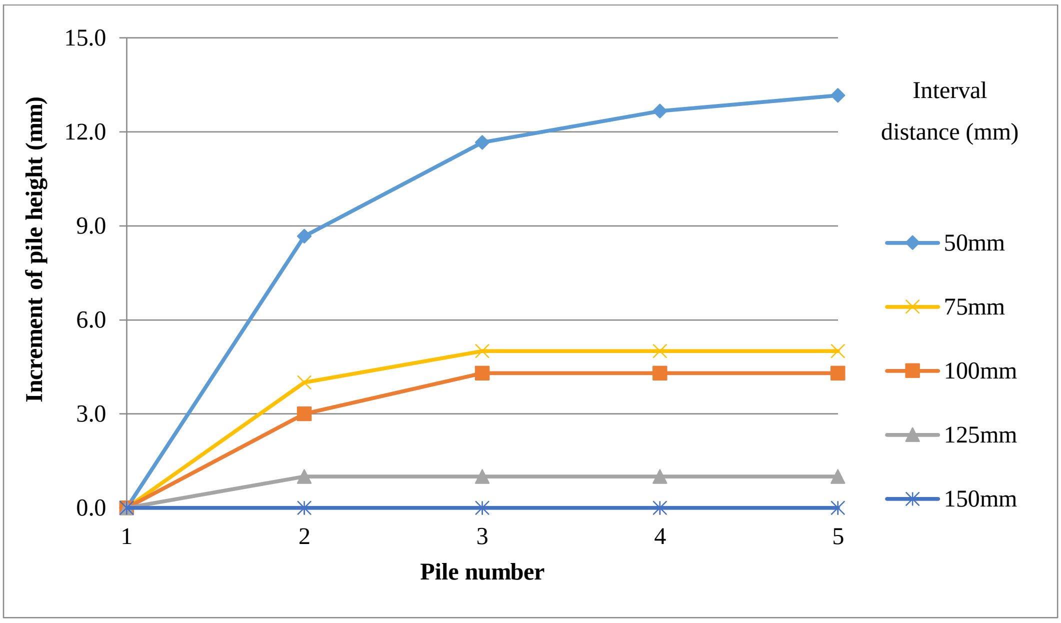

Figure 4-12 Relationship between the increase of pile height and the pile number

Based on the above laboratory tests and site investigations, the following conclusions were found. In bucket wheel excavator-belt-stacker dumping operation and dragline dumping operation, the dumping width should be decided through optimization efficiency of bulldozer and dumping device. The interaction decreases with increasing width of the dumping. In bucket wheel excavator-belt-stacker dumping operation and dragline dumping operation, the work amount of bulldozer increases with decreasing the interaction. Efficiency of dumping equipment decreases with decreasing of the width of dumping.

Figure 4-13 shows the relationship between the increase of fifth pile height and drop intervals under different drop volumes. It shows that the height of bench increases with decreasing intervals under all drop volumes. When interval distance reaches to a specific one at each drop volume, no interaction between neighboring piles are observed. This is because of the interference between two adjacent piles. As the drop interval decreases, the interference of previous bench on the next one appears and becoming large.

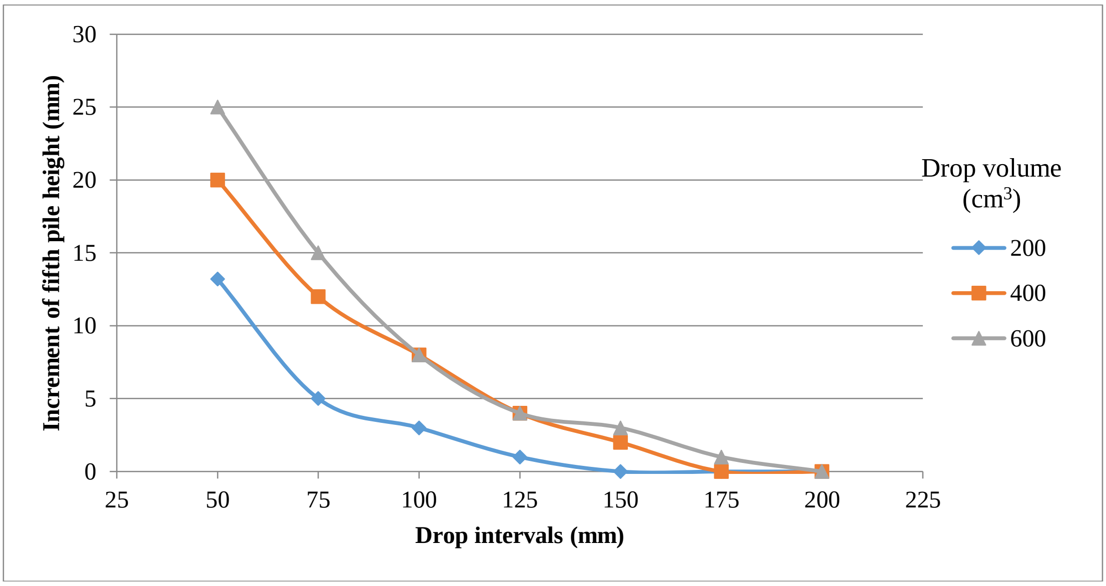

Figure 4-13 Relationship between the increase of the fifth pile height and drop interval

In the dumping operation, the volume of bucket does not have a pronounced interaction effect on dumping operation. The pile height and the minimum interval distance increases with increasing volume of the dumping. Meanwhile, the ratio of dumping volume to bench volume is too small.

Figure 4-14 represents the relationship between the ratio of the height increment and the ratio of the interval of drop point to volume. Here, the ratio of the height increment represents the increasing of fifth pile height divided by the height of the first pile. The ratio of the dropping interval is dropping interval (I) divided by the cubic root of volume ($V^{1/3}$). Figure 4-14 also shows the similar tendencies of height increment despite of drop volume. Thus, the pile height can be estimated by the following Equation (4-1).

$$\Delta H / H_o = -0.260 \ln (I / V^{1/3}) + 0.800 \tag{4-1}$$

ΔH–Increasing of fifth pile height, m.

H_o–Height of the first pile, m.

I–Drop interval, m.

V–Volume, m^3.

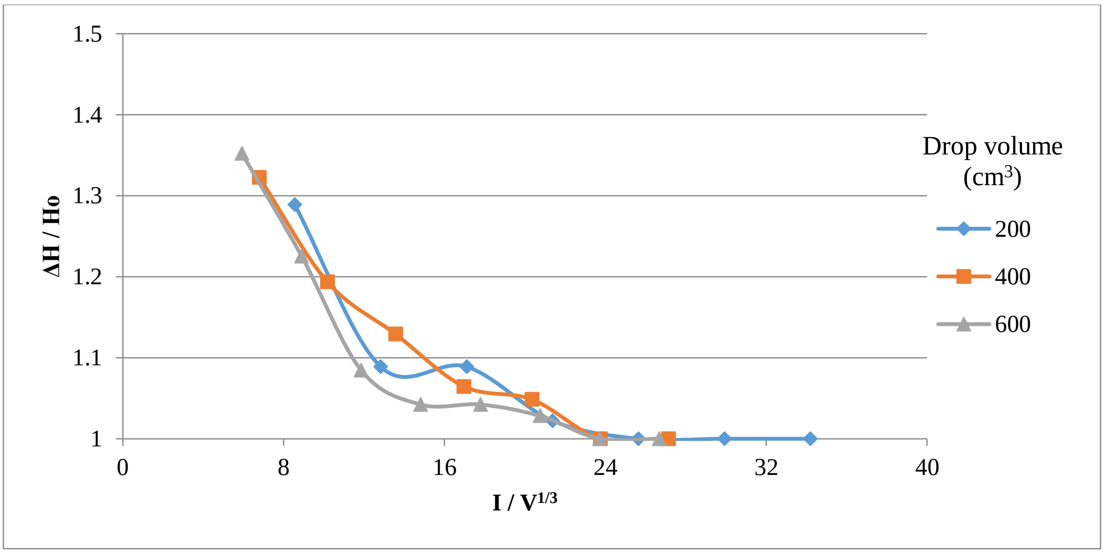

Figure 4-14 Ratio of increase height – ratio of interval of drop point to volume

From the above discussions, it can be concluded that in the dumping operation, the work amount of bulldozer decreases with increasing size of bench. The pile height decreases while interaction effect increases with increasing height of the dumping. Meanwhile, the work amount of bulldozer decreases with increasing the interaction effect.

4.3.4. Efficiency of dumping operation design

Land reclaiming must be carried out after dumping work. Figure 4-15 shows the process of ground leveling operation work. It is necessary to level the benches with bulldozer. Therefore, the elevation difference and pile numbers are measured to examine the amount of grading work needed under different dumping conditions.

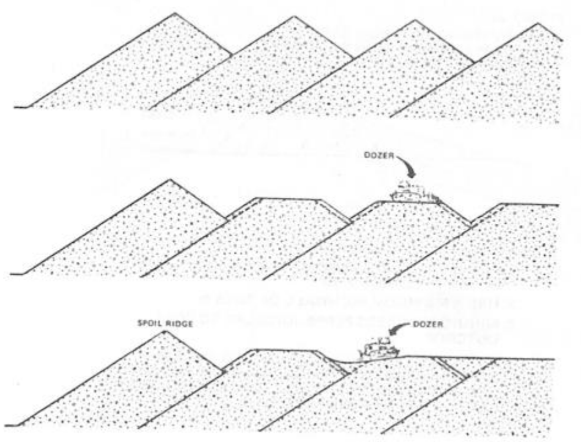

Figure 4-15 Process of ground leveling operation work

At first, we measure the elevation difference of the fifth bench. Figure 4-16 shows the relationship between elevation difference and interval distance. It shows that the elevation difference increases with increasing interval distance. Therefore, it is believed that the elevation difference of bench decreases with decreasing interval distance. The amount of grading work decreases with decreasing elevation difference of bench. Therefore, cost reduction of the ground leveling operation is possible.

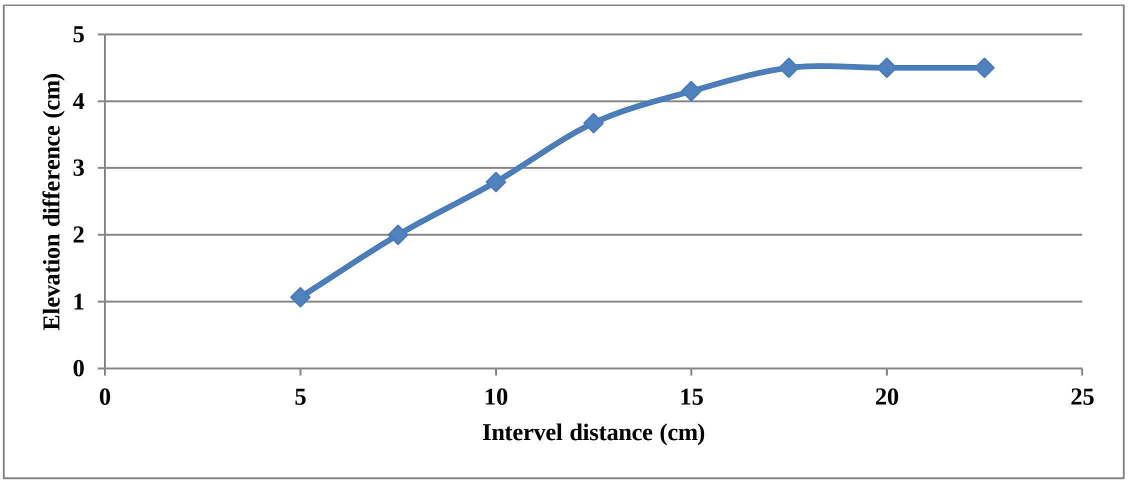

Figure 4-16 Relationship between the elevation difference and interval distance (200cm^3)

Second, the volume of each pile was changed to examine the relationship between pile numbers and amount of grading needed under the condition of constant total volume and constant floor area. An example of the results is shown in **Table 4.4**. It shows that the amount of grading increases with increasing the pile numbers. The result shows that the work amount of grading operation decreases with decreasing dumping volume of equipment.

Table 4.4. Relationship between the amount of grading work and pile numbers

Total volume V_0 (cm^3)			**1,200**	
Pile numbers	6	4	2	1
Floor area A (cm^2)			**570**	
Amount of grading ΔV (cm^3)	81.5	160.8	197.6	461.1

However, different equipment has different efficiency, different flexibility, and bucket volume. Equipment capability will decide the dumping volume, cost, the work time of building the bench and stripping work. Therefore, the design of the dumping operation must consider the total efficiency of ground leveling operation and formation of dumping area.

4.4. Simulation for the Effects of Dumping Formation and Operation on Stability of Dumping Area

4.4.1. Dumping formation effects

The annual ore production and waste rock removal of the Erdenet open pit mine is 26 million tons and 15 million cubic meters and it will be reached 35 million tons and 20 million cubic meters respectively in the near further. Mining operation direction is from east to west and depth of open pit currently reached around 300m depth, 2,500m length in end of the 2016. A substantial amount of materials from the open pit will be removed from the mine for a total of 43 years and will be dumped to the dumping area and the Figure 4-17 shows annual required volume of dumping area.

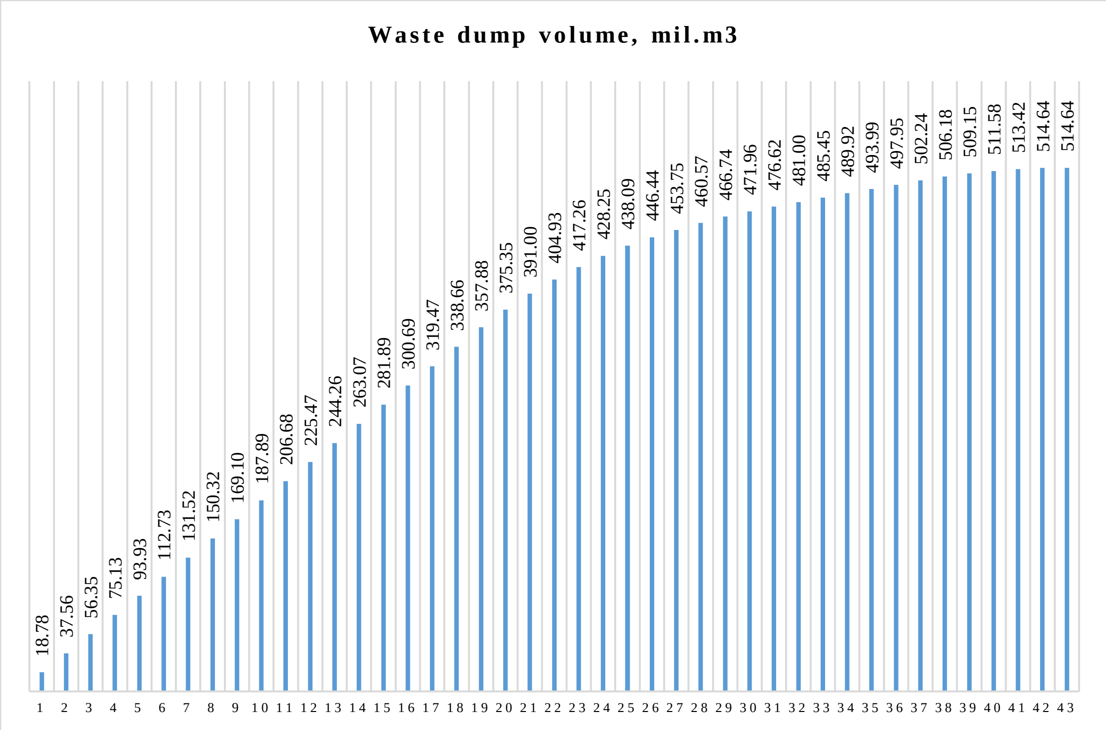

Figure 4-17 Annual growth of dumping area volume of Erdenet mine

In this section, based on the results of the site investigation and laboratory experiments, height of bench, the angle of bench were changed to analyze the effect of dumping on stability of dumping area.

The dump slope was numerically analyzed using a finite element method. The finite element method is a continuum model which can be used for analysis of complex geometries, stress modelling and material behaviour. The continuum structural system is modelled by a set of appropriate finite elements interconnected at points called nodes. Elements may have physical as well as elastic properties such as thickness, density, Young's modulus, shear modulus and Poisson's ratio.

Table 4.5. Input data in regards to material properties of the model

Material	**Unit weight, MN/m^3**	**Friction angle, °**	**Cohesion, MPa**	**Tensile strength, MPa**	**Young's modulus, GPa**	**Poisson's ratio**
Oxidation	0.023	26.5	0.5	0.6	1.5	0.254
Transition	0.026	32.4	0.6	0.8	2	0.254
Fresh	0.027	36.6	0.8	1	3	0.332
Waste dump	0.025	35	0.24	0.05	0.05	0.3

The elements are interconnected only at the exterior nodes, and altogether they cover the entire domain as accurately as possible. Nodes have nodal (vector) displacements or degrees of freedom which may include translations, rotations, and for special applications, higher order derivatives of displacements. A uniform mesh with 6 noded triangular elements was used for the analysis mine dump. A major advantage of the finite element shear strength reduction method is that it does not demand any earlier assumptions on the nature of failure mechanisms.

The current dump slope having an initial height of 50-150m and 35-38˚ slope angle. The simulation provides the height increase of 50 meters interval in total height of 50-200m waste dump designs to simulate the relationship between height of bench and safety factor of dumping area. Figure 4-18 shows input parameters regards to dump design model and Figure 4-19 shows the stability analysis and factor of safety (FOS) in each dumping height designs.

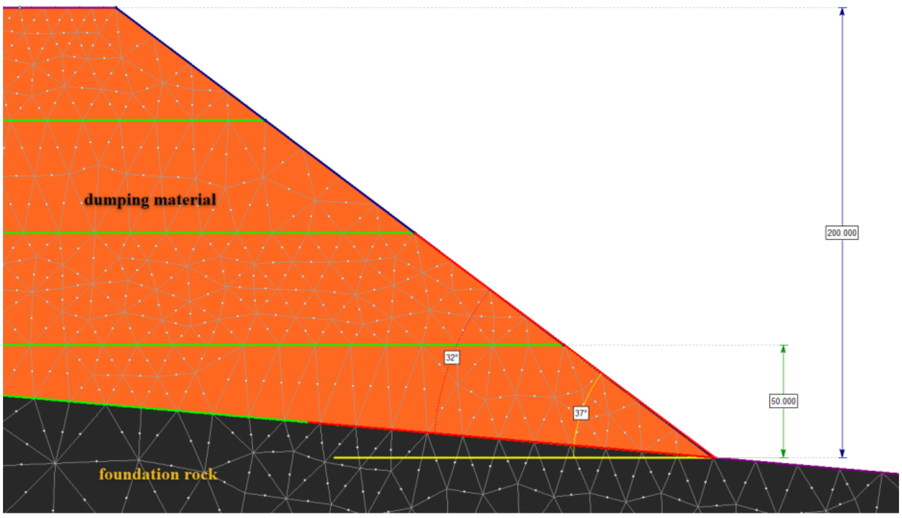

Figure 4-18 Input parameters regards to dump design model

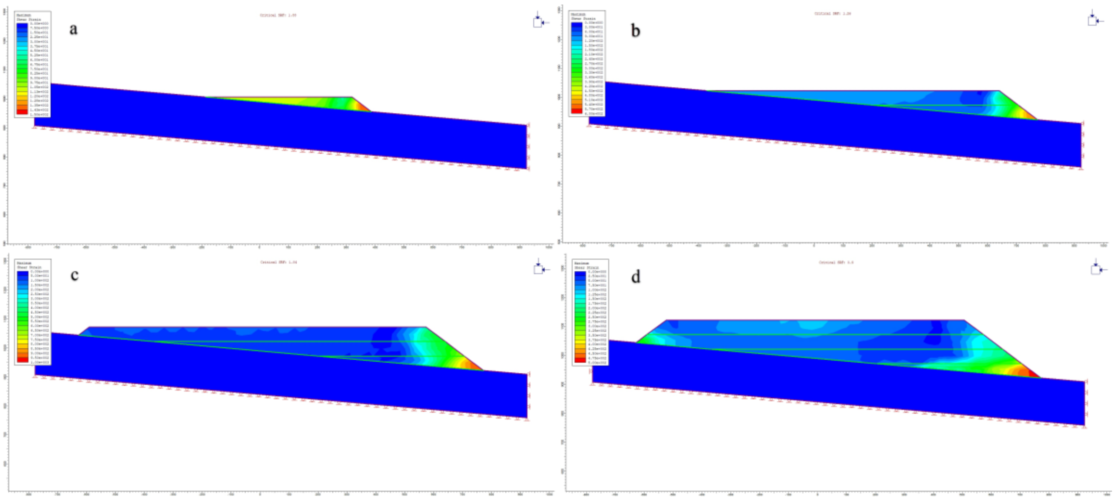

Figure 4-19 Maximum shear strain plots for the dump designs with a slope angle of 32˚, floor angle of 5°
a) H=50m FOS=1.88; b) H=100m FOS=1.26; c) H=150m FOS=1.04; d) H=200m FOS=0.9

In the dumping area, the efficiency of working and subsidence of dump increases with increasing the dump height. The subsidence of dump was a dangerous factor to workers and equipements. The height of dump can be as high as possible, up to the allowed safety values of workers and working equipment. A safety factor of 0.9 was yielded by the model at a slope angle of 32°, floor dip of 5° and 200 m height without benches. From the laboratory experiment results of dump shape and drop tests, dump angle ranging between 37° to 39°, but the FOS yielded for these dump angles are less than 1.0 and at very critical state.

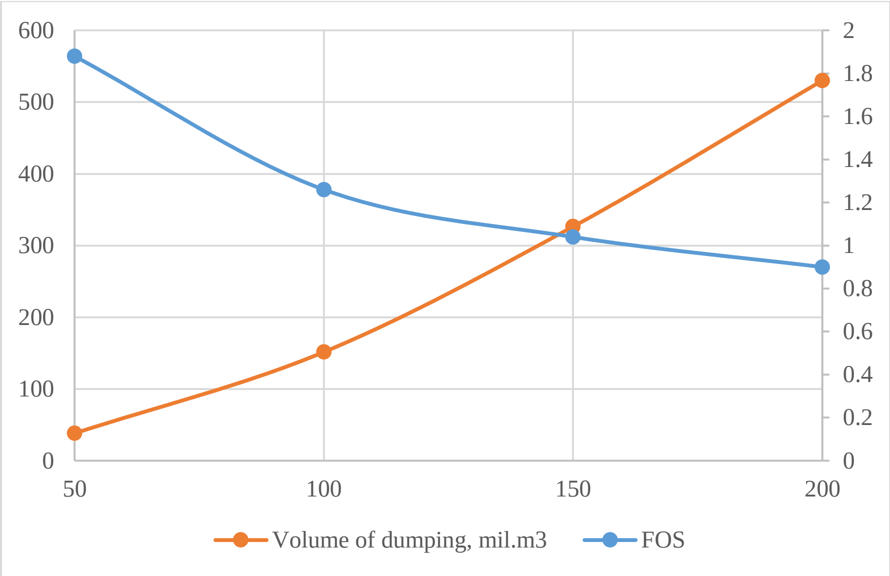

Figure 4-20 Relationship between dump height, safety factor and total capacity of dumping area

From the results of stability analysis regards to dump height increase, the safety factor of dumping area slightly decreases with increasing height of dump. Meanwhile, the volume capacity of dumping area has a sensitive increases as the height of dump increases. And the total capacity requirement of dumping can be reached at 200m of height. The height of bench can be as high as possible, up to the allowed safety values of workers and working equipment.

The another simulation was created for to reach safety factor of 200m of height dump design with the dump benches. Other parameters keep in constant value. Figure 4-21 shows the geological model under the angle of bench is 37° on 0° of floor dip and 32° in 5° of floor dip.

The simulation provides the bench height of 100m of 2 benches and safety berm increase by 20m interval in 30m to 110m berm width to simulate the relationship between width of safety berm and safety factor of dumping area. Figure 4-21 shows input parameters regards to dump design model and Figure 4-22 shows the stability analysis and factor of safety in each bench width designs.

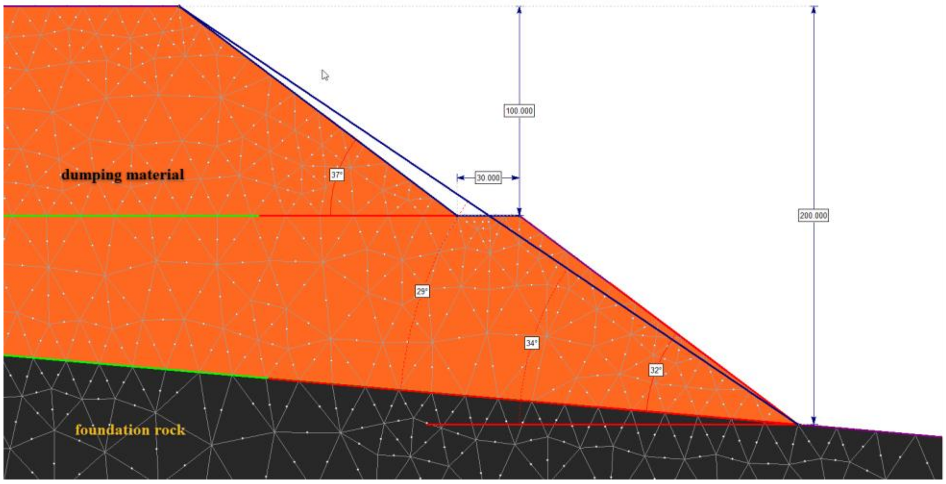

Figure 4-21 Input parameters regards to dump design model

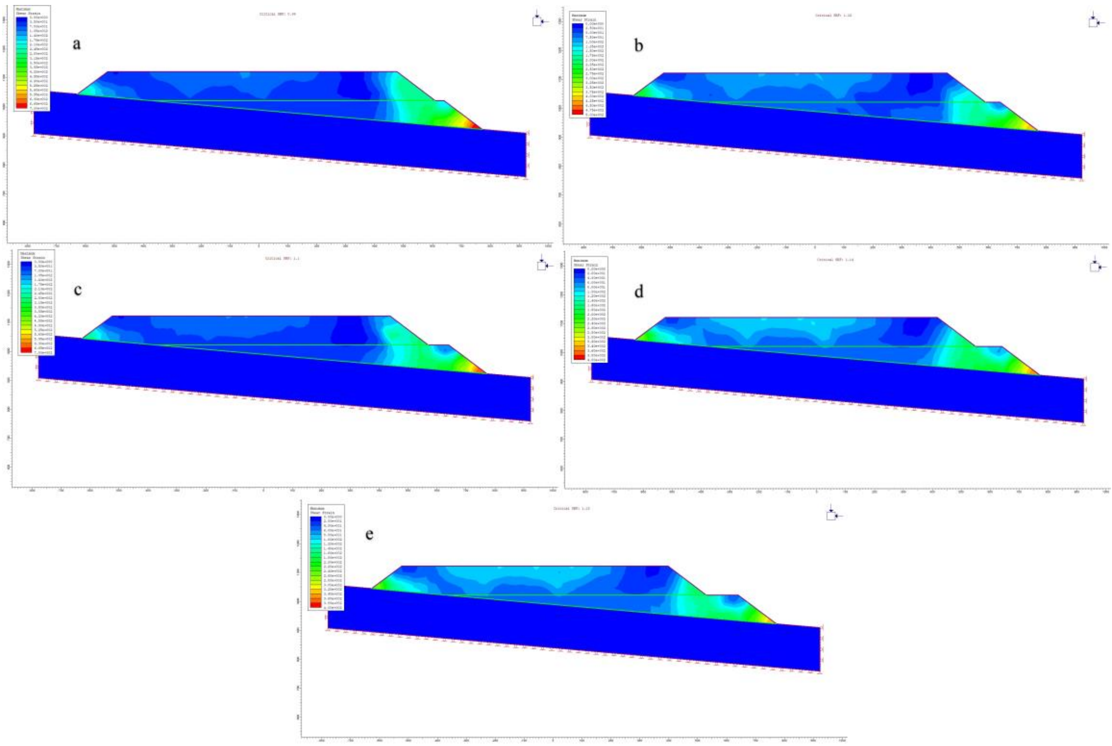

Figure 4-22 Maximum shear strain plots for the dump bench designs with slope angle of 32˚ on floor angle of 5°, slope angle of 37˚ on floor angle of 0°,
a) B=30m FOS=0.99 Overall slope angle 29°; b) B=50m FOS=1.02 Overall slope angle 27°;
c) B=70m FOS=1.1 Overall slope angle 26°; d) B=90m FOS=1.14 Overall slope angle 24°;
e) B=90m FOS=1.14 Overall slope angle 23°

Figure 4-22 shows the relationship between width of safety berm, overall slope angle and safety factor of dumping area. It shows that the safety factor of dumping area decreases with decreases the width of bench and increases the overall slope angle.

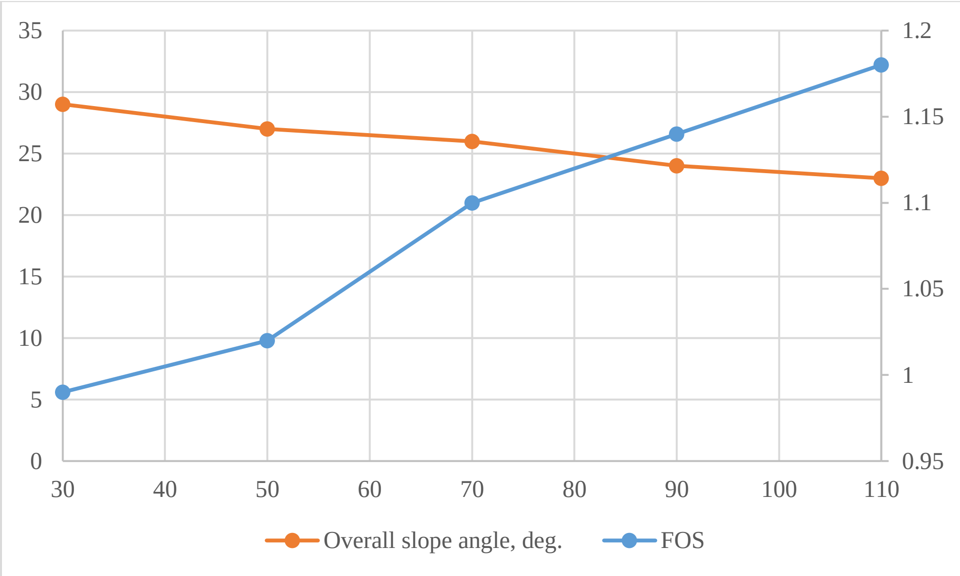

Figure 4-23 Relationship between dump height, safety factor and total capacity of dumping area

4.4.2. Dumping operation effects

In this section, based on the results of the site investigation and laboratory experiments, bulk factor, and the transporting truck were changed to analyze the effect of dumping operation on stability of dumping area.

The bulk factor of waste rock represents the ratio between volume after stripped to initial volume. The bulk factor of waste rock was changed to clarify the relationship between bulk factor of waste rock and stability of dumping area. The density of waste rock will be changed with changing the bulk factor. Other parameters keep in a constant value. Figure 4-24 shows the relationship between bulk factor of waste rock and volume capacity increase, safety factor of dumping area. It shows that the volume capacity increment increases with decreasing bulk factor of waste rock. Furthermore, the bulk factor of waste rock has a beneficial and small effect on safety factor of dumping area. The bulk factor of waste should be as small as possible to improve stability of dumping area.

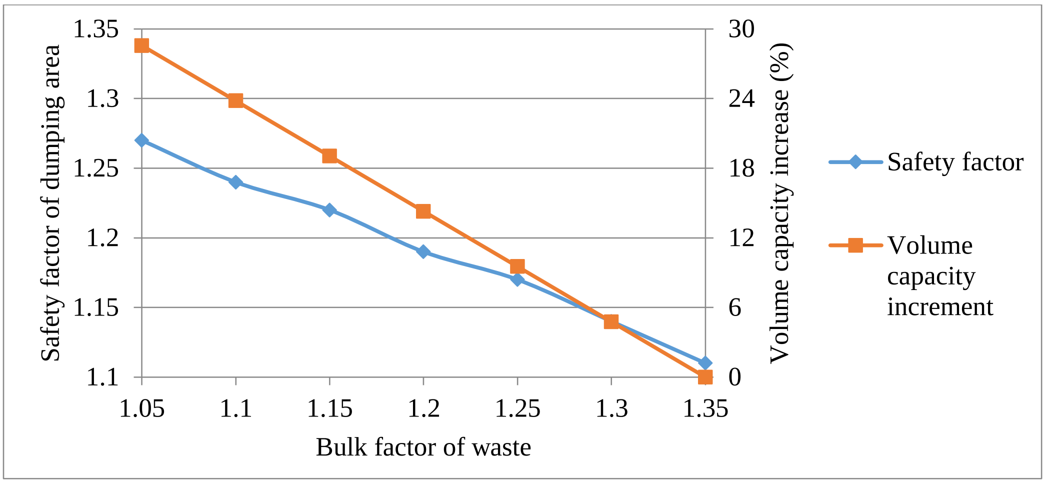

Figure 4-24 Relationship between bulk factor of waste rock and volume capacity increase, safety factor of dumping area

The stability effect of transport truck in roads on dumping area was analyzed. We assume that the weight of truck is 100 tonnes, 200 tonnes, 300 tonnes, 400 tonnes, the length of truck is 10 m. Therefore, the force is 0.1 MN/m, 0.2 MN/m, 0.3 MN/m, 0.4 MN/m, respectively. Figure 4-25 shows the geological model as the truck load force is 0.2 MN/m.

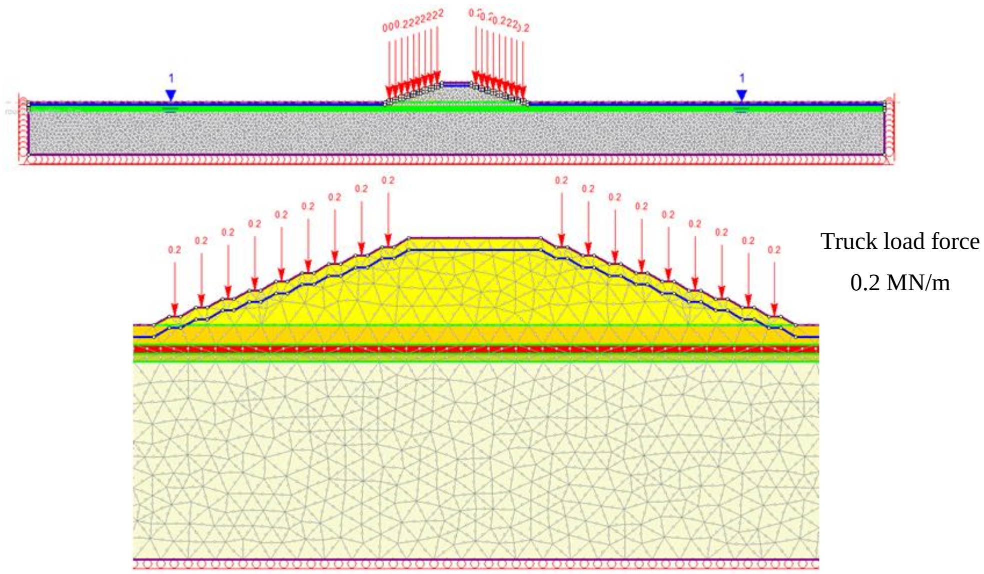

Figure 4-25 Geological model as the truck load force is 0.2 MN/m

Figure 4-26 shows the relationship between truck load and safety factor of dumping area. It shows that the truck load does not have effect on safety factor of dumping area. Meanwhile, the activity of truck transport can give compaction effect to dumping area. The activity of transport truck in dumping area has a beneficial effect on stability of dumping area.

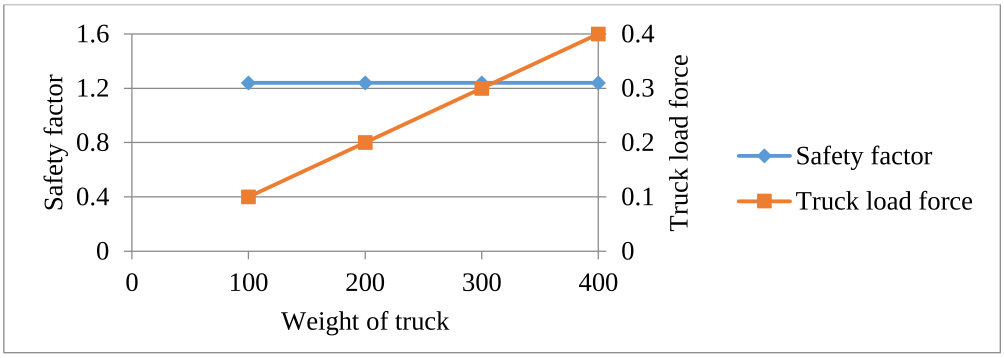

Figure 4-26 Relationship of truck load and safety factor of dumping area

4.5. Summary

In this study, a site investigation, a series of experiments, and a series of numerical simulations were performed to determine the formation mechanisms of benches on stability of dumping area. Some main conclusions can be summarized as follows.

The design of blasting is modified to make the particle size of waste smaller. In the dumping area, the capacity of bench decreases while safety distance by rock roll increase with increasing particle size of the waste. The volume of bucket does not have a pronounced effect on bench repose angle, volume capacity of bench, and stability of dumping area. In the selection process of dumping area, smaller floor inclination is better than that of higher inclination. Two methods are proposed to increase the stability of dumping areas. First, the loose earth and all vegetation need to be removed to make the floor strong seam. Second, floor surface of dumping area becomes rough by blasting, which can prevent the floor to be slide surface. Design of the dumping operation must consider the total efficiency of ground leveling operation work and forming dumping area work. Height of bench can be as high as possible, up to the allowed safety values of workers and equipment working. Angle of bench is not important to dumping operation. Bulk factor of waste rock should be as small as possible to improve dumping operation stability. The activity of transport truck in dumping area has a beneficial effect on stability of dumping area.

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CHAPTER V. Buffer Zone Optimization

5.1. Introduction

Creating waste dump near to the pit is one of the solutions when the waste rock contains low grade of valuable minerals that planned to be extracted in future. Waste dump alongside the pit also gives advantage in regards to waste hauling cost. However, from geotechnical point of view, constructing a waste dump alongside the pit should be planned well particularly distance between final pit's boundaries to waste dump's boundary which is referred as buffer zone. This chapter discusses a buffer zone's design for a particular pit wall as well as waste dump design. The buffer zone geometries must be chosen to fit the bench configuration of pit as well as the waste dump.

The study has been done at the Erdenet porphyry copper and molybdenum (Mo) deposit that has been adopting the dump alongside pit wall. The Erdenet mine is one of the largest mine in Mongolia which has annual production of 25-30 million metric tons of Cu ore. The waste, which contain low grade ore, is dumped in 3 locations of dumping area where located alongside the pit. The distance from the pit to waste dump is approximately 0.05-0.6 km. The total capacity of the waste dump is 300-350 million cubic meters. The illustration of pit and waste dump's current situation is given in Figure 5-1.

Figure 5-1. Aerial photos of location of waste dump (source: Google maps).

A problem arises in the Erdenet mine when a minor collapse is occurred in the wall that located near to the waste dump; accordingly, influencing stability of bench of the waste dump. It has been proven that over capacity or weight of deposited waste may cause slope loss and slope deformation (Scott et al., 2007; Cho and Song, 2017). Stability of pit wall as well as dump's bench is essential for smooth mining operations to avoid any slope failure and hazardous accordingly for mining activities. Therefore, a proper design of buffer zone between wasted dump and pit should be done in order to prevent slope failure and preserve the safety of workers and machineries accordingly. In order to prevent more serious happen in future in regards with pit wall failure, the Erdenet mine plans to relocate waste dump material that located close to the pit to area that does not give any influence on pit wall stability as illustrated in Figure 5-2 and create a buffer zone an shown in Figure 5-3.

Figure 5-2. Re-located dump area.

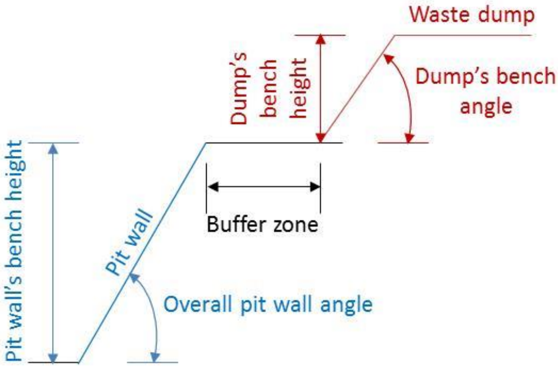

Figure 5-3. Sectional illustration of pit wall and waste dump configuration.

One of most concerns in relocation project is the amount of waste material that should be moved. This material may need to be removed from area adjacent the pit wall in order to reduce gravity loading; accordingly reduce shear stress along pit wall, putting personal safety and equipment at risk of slide hazard. The material that relocated should be as few as possible to cut the cost. It has been reported that, in period of 2012 to 2015, the waste material that has been relocated is around 7.5 million cubic meters. Considering it, the relocation is done by cutting the area of waste dump that located adjacent the pit wall. The volume that cut is done based on the study on buffer zone configuration that satisfies stability criteria of pit wall as well as waste dump.

There are several factors that together contribute to stability of pit wall as well as waste dump. The large scale geometry, that is the pit wall's overall height and slope angle as well as waste dump's bench height, is governed largely by the quality of the rockmass and overburden properties respectively, and its stability within the context of the stresses imposed. The wide of buffer zone and waste dump's bench configuration must be chosen to fit the stability of pit wall. It can therefore be seen that buffer zone and waste dump's bench configuration is governed in combination by the stability of pit wall.

This chapter studies the effects of different wide of buffer zone for different pit wall as well as waste dump's bench configuration that fit a given stability of the slope. Optimization of the wide of buffer zone and waste dump's bench configuration is the key aim. This is carried out based on the need to minimize the volume of dump material that removed to relocated area. Moreover, this chapter also studies about the influence of cohesion and friction angle on pit wall stability. It is well known that in reliability analysis of rock slopes, parameters of shear strength of the rock slope such as cohesion (c) and friction angle (φ) are required (Li et al., 2011; Jimenez-Rodriguez and Sitar, 2007; Low, 2007; Jimenez-Rodriguez et al., 2006).

5.2. Method of Study

This chapter describes current condition of pit wall and waste dump and then discusses the optimization of buffer zone for different pit wall as well as dump's configuration as well as shear strength properties on the pit wall as well as waste dump's slope performance. Optimization of the geometry is carried out by investigating and comparing the performance of geometrical combinations of buffer zone, and dump's bench height and angles by means of numerical modeling.

In regards to slope stability analysis, finite element methods and limiting equilibrium are the most common method used. Both methods are applicable to be used to analyze homogeneous and inhomogeneous slopes. However, in a particular case, finite element methods are better than that of limiting equilibrium method in regards to provide more appropriate analysis. The limiting equilibrium methods often face computational difficulties in locating the critical slip surface, and moreover numerical inconsistencies may occur in this case.

Owing to these inherent limitations of limit equilibrium methods, the method has been increasingly used in slope stability analysis (Verma et al., 2013; Hughes, 1987; Strang and Fix, 1973; Clough and Woodward, 1967). Despite the problems that associated with limit equilibrium methods, the finite element methods do not consider assumption about the shape or location of the critical failure surface. The finite element methods also can be easily used with others to calculate stresses, movements, pore pressure in embankments and seepage induced failure as well as for monitoring failure (Zienkiewicz and Taylor, 1989). One more of the advantages of finite element method are that the program can analyze problems in man-made slope, in this case is waste dump slope. In this method, the safety factor is determined using the φ/c reduction approach where the strength parameters ($\tan \varphi$) and (c) of the soil are successively reduced until failure of the structure occurs (Hammouri et al, 2008). A slope can be regarded as being stable if the strength reduction factor (SRF) is greater than 1. Considering to the advantages, finite element method is adopted to analyze waste dump slope stability in this study.

In order to study of different combinations of buffer zone, dump's bench height, wall slope configuration and shear strength parameters on pit wall slope performance, a fixed waste dump's bench angle was chosen. The dump's bench angle is considered in accordance with current dump's bench angle design i.e. 45^{o}. The maximum length of bottom base of waste dump is 500 m, whereas the minimum length of top base is 200 m. The selection of maximum length of bottom base of waste dump is taken in accordance with the topography of mining area, whereas the selection of minimum length of top base of waste dump, however, is most often a function of the equipment size. The 200 m length of top base of waste dump is large enough in regards to safety and equipment maneuver. Considering to bench angle, maximum length of bottom base as well as minimum length of top base of dump, the maximum height of the waste dump should be not more than 150 m. The configuration of the pit and waste dump for calculation is given in Figure 5-4.

In general, the most general factors affecting stability of any slope are: (1) slope's geometry; (2) material properties; and (3) forces acting on the slope. In this study, all of those parameters were considered. The objective of this study is to compare the performance of different pit wall as well as waste dump geometries such as,

1. Pit wall's bench height: 240 m, 340 m and 440 m,
2. Pit wall's bench angle: 45^{o} and 60^{o},
3. Length of buffer zone: 0, 100 m and 200 m,
4. Dump's bench height: 50 m, 100 m and 150 m, and
5. Shear strength's properties of the pit: $0.75c$, c, $1.25c$, 0.75φ, φ, 1.25φ.

The analysis were conducted by means of numerical simulation for a predetermined bench stack angle i.e. 45^{o} and thickness of oxidation and transition zone i.e. 60 and 120 m, respectively. In this study, discussion is divided into two categorize i.e. numerical simulation without buffer zone consideration and numerical simulation with buffer zone consideration.

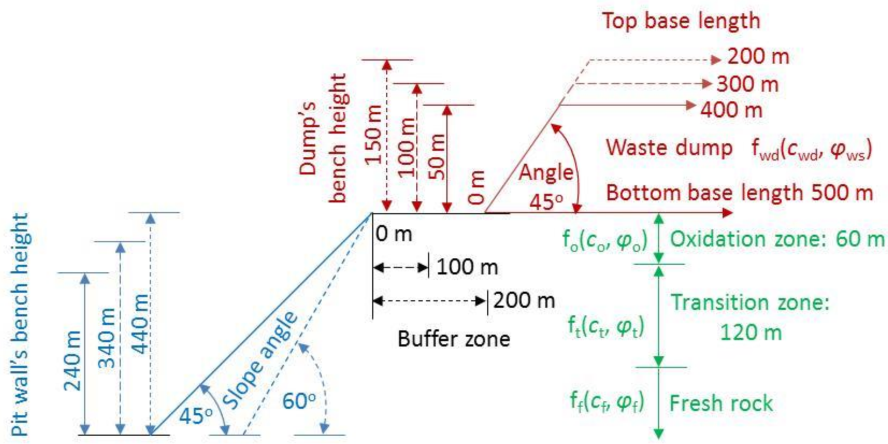

Figure 5-4. Configuration of numerical calculation.

According to Figure 4, the simulation considers three layers of rock in the pit such as oxidation zone, transition zone and fresh rock. The input data in regards to rock properties for oxidation zone, transition zone, fresh rock and waste dump material of the model is given in Table 5.1. It is shown in Table 5.1 that the properties of waste rock dump are much weaker than that, even though, of oxidation material's properties. It may due to waste rock dump is made up of a mixed rock, unconsolidated rock, and sometimes also contains soil; accordingly the physic-mechanical properties are changed compared to the original materials.

Table 5.1. Input data in regards to material properties of the model.

Material	Unit weight, MN/m^3	Friction angle, °	Cohesion, MPa	Tensile strength, MPa	Young's modulus, GPa	Poisson's ratio
Oxidation	0.023	26.5	0.5	0.6	1.5	0.254
Transition	0.026	32.4	0.6	0.8	2	0.254
Fresh	0.027	36.6	0.8	1	3	0.332
Waste dump	0.025	35	0.24	0.05	0.05	0.3

5.3. Numerical simulation

5.3.1. Without buffer zone

The analysis was begun by developing a model in accordance with the current design; the waste dump is located adjacent the pit. The model is given in Figure 5-5. The height of pit is 240 m, and possible to extent up to 440 m high. The pit wall slope is 45°. The waste dump's height and angle is 50 m and 45°. The result of bench stability simulation is given in Figure 5-6. The SRF of the model is estimated 1.33 which means the slope is stable and satisfies the stability criteria. However, although the slope is safe, the figure suggests that there is a high stress concentrated in toe region of the pit wall. It should be compressive in both radial and circumferential directions (Stacey et al., 2003). It indicates that the toe of pit wall receives high burden pressure. It should be a warning when the waste dump's bench height is increased. The shape potential sliding surface was defined as the circular type based on the simulation analysis. The location of shear stress below the crest of the waste dump was considered as the starting point of sliding surface. The weight of the circular area gives severe pressure on the toe region of the pit wall, accordingly the tensile stress on the area is increased. It has been proven by simulation result of bench stability for 240 m height pit wall with 100 m and 150 wasted dump height without buffer zone that the SRF is decreased to 1.14 and 1.06, respectively.

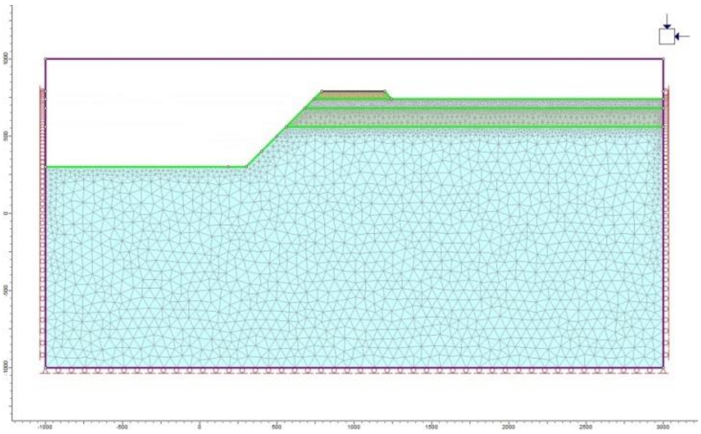

Figure 5-5. Simulation model by means finite element method phase2.

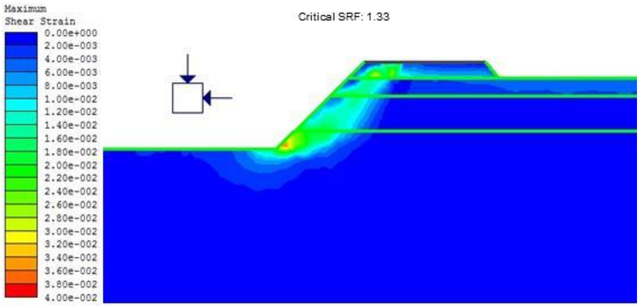

Figure 5-6. Result of bench stability simulation for 240 m pit wall height with 50 m wasted dump height without buffer zone.

Moreover, the probability of a large-scale failure is high as increasing the pit height when the exaction is continued up to pit limit. Figures 5-7 and 5-8 shows result of bench stability simulation for 340 m and 440 m pit wall height, respectively, with 50 m wasted dump height without buffer zone. Compared with Figure 5-6 that shows the simulation result for 240 m pit wall height under same condition of waste dump bench height as well as buffer zone, the SRF of bench stability for 340 m and 440 m pit wall height is lower i.e. 1.22 and 1.15, respectively. It is shown in Figures 5-7 and 5-8 that the shear stress region under toe region of the pit wall is higher for deeper pit wall. The potential volume slide slope is also higher for deeper pit wall.

Furthermore, when the waste dump bench height is increased to 100 m and 150 m, the stability is in critical for 100 m waste dump bench height and failure for 150 m waste dump bench height. It indicates that the stability of pit wall reduces with increasing the pit wall height under without buffer zone condition. The resume of result of bench stability simulation under without buffer zone condition for 240 m, 340 m and 440 pit wall height and 0, 50 m, 100 m and 150 m waste dump bench height is given in Table 5.2. When the SRF is plotted as function of waste dump height for different pit wall height, it is likely to form a unique correlation. The correlation shows exponential correlation with a great correlation coefficient, the R is more than 0.99. The correlation between pit wall height, waste dump height and SRF is described in Figure 5-9.

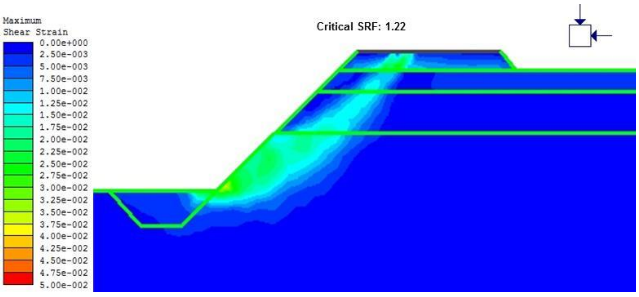

Figure 5-7. Result of bench stability simulation for 340 m pit wall height with 50 m wasted dump height without buffer zone.

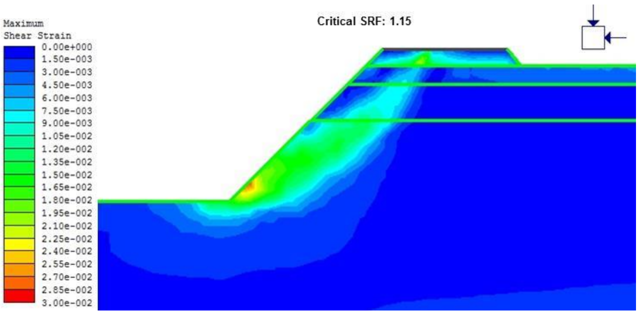

Figure 5-8. Result of bench stability simulation for 440 m pit wall height with 50 m wasted dump height without buffer zone.

5.3.2. With buffer zone

Considering to instability of pit wall that has found in some cases in the Erdenet mine, the buffer zone, which is defined as distance between toe of the pit wall and crest of the waste dump, is proposed. In case of the Erdenet mine, construction of buffer zone is done by excavating and relocating the waste material where located adjacent the pit wall to the other side of the waste dump or to other locations (Figure 5-2). In engineering point of view, the re-excavating of waste dump material where located adjacent to pit wall is very risk. Thus, the excavation should be done carefully. In order to ensure the slope is not so disturbed by the activity, the excavation is done from the safe distance, and continued toward toe of the pit wall. In this case, 100 m and 200 m in length of buffer zone are considered for different pit wall height as well as waste dump height. The pit wall and waste dump's configuration model is constructed based on the current mining design such as 240 m pit wall height, 45^{o} pit wall angle, 50 waste dump height and 45^{o} waste dump angle. During simulation, the height of pit wall is increased from 240 m to 340 m and 440 m, whereas the height of waste dump is increased from 50 m to 100 m and 150 m.

Table 5.2. Result of bench stability simulation for pit wall and wasted dump without buffer zone.

No.	Pit wall height, m	Waste dump height, m	SRF
1	240	0	1.43
2		50	1.33
3		100	1.14
4		150	1.06
5	340	0	1.39
6		50	1.22
7		100	1.12
8		150	0.94
9	440	0	1.27
10		50	1.15
11		100	1.07
12		150	0.87

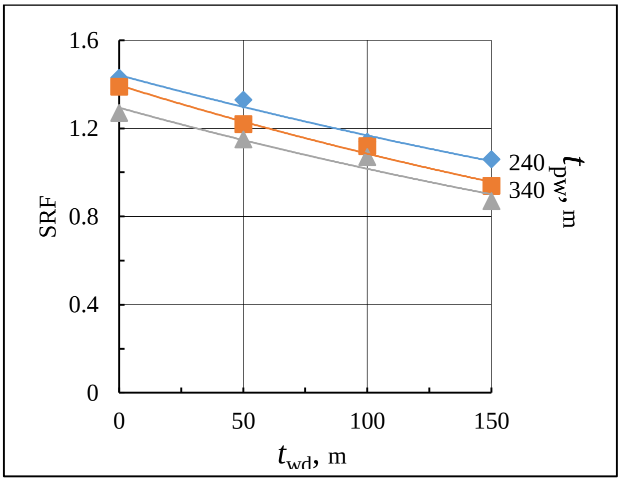

Figure 5-9. SRF for different pit wall height and waste dump height under without buffer zone condition.

According to the simulation result, in general, having a buffer zone is able to improve the stability. The example of improving SRF by increasing buffer zone can be seen at Figure 5-10 for case of 240 m pit wall height with 50 m waste dump height. The figure shows that The SRF increases from 1.33 for without buffer zone to 1.56 and 1.62 for 100 and 200 m buffer length, respectively. The figure also shows that when the buffer length reaches 200 m, the weight of waste dump does not influence the gravity load of the pit wall. This phenomenon is also seen for cases of 340 m pit wall height with 50 m waste dump height (Figure 5-11) as well as 440 m pit wall height with 50 m waste dump height.

Furthermore, when the waste dump height is increased, a significant tensile stress is induced around toe of the waste dump. When the height is increased continuously, a shear stress is generated along the face of waste dump. In case of 340 m pit wall height with 100 m buffer length, the SRF reduces from 1.26 for 50 m waste dump height to 1.12 for 150 m waste dump height. This phenomenon can be seen in Figure 5-12. The resume of SRF for case of with buffer zone is given in Table 5.3. When the SRF is plotted as function of waste dump height for different pit wall height and buffer length, it is seen a great exponential correlation with correlation coefficient, the *R* is more than 0.99. The correlation between pit wall height, buffer length, waste dump height and SRF can be seen in Figure 5-13.

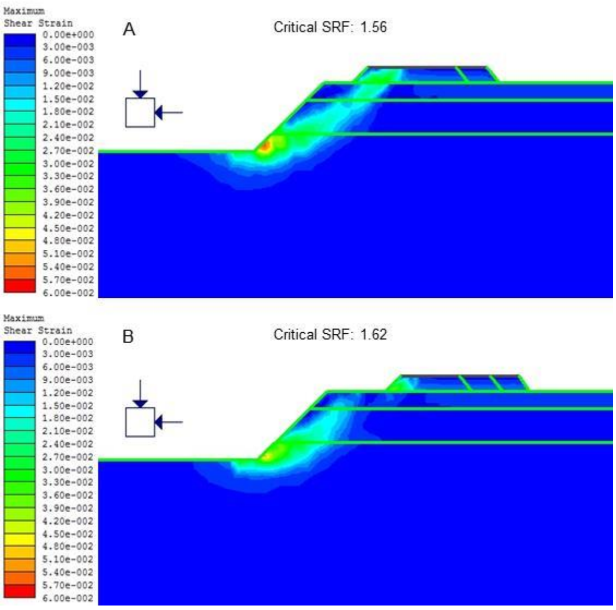

Figure 5-10. Buffer zone's influence on bench stability for case of 240 m pit wall height with 50 m wasted dump height: (A) 100 m buffer length; (B) 200 m buffer length.

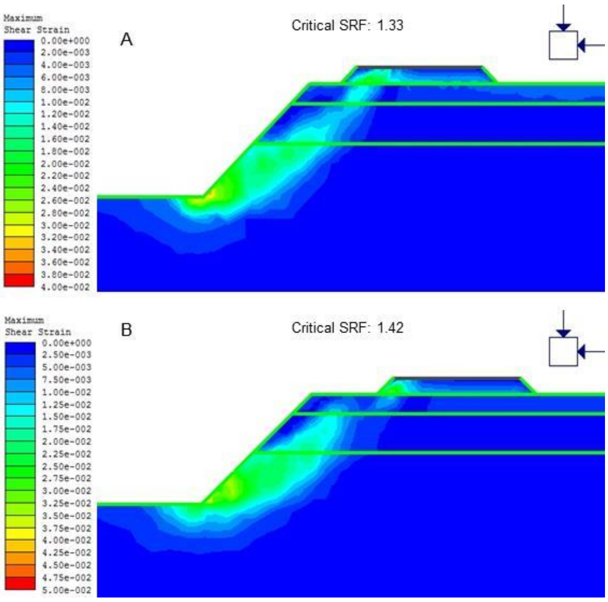

Figure 5-11. Buffer zone's influence on bench stability for case of 340 m pit wall height with 50 m wasted dump height: (A) 100 m buffer length; (B) 200 m buffer length.

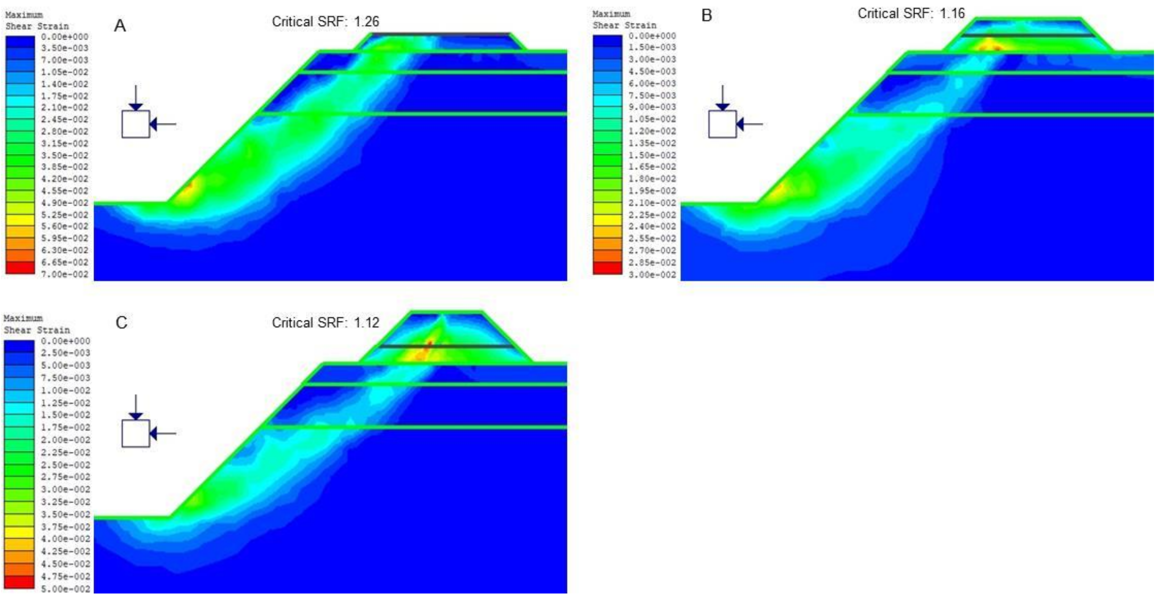

Figure 5-12. Response on a changing of waste dump height to buffer zone's role on bench stability for case of 340 m pit wall height with 100 m buffer length: (A) 50 m waste dump height; (B) 100 m waste dump height; (C) 150 m waste dump height.

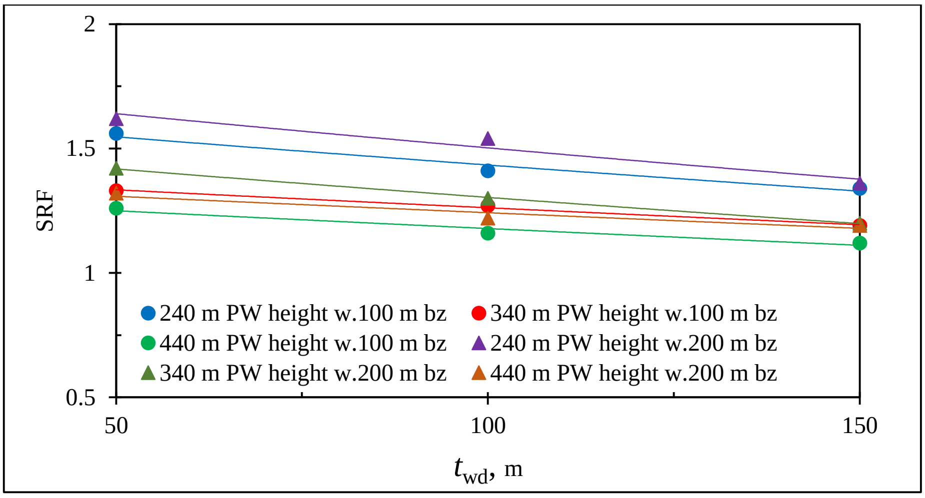

Figure 5-13. SRF for different pit wall height, buffer length and waste dump height.

Table 5.3. Result of bench stability simulation for pit wall and wasted dump under different length of buffer zone condition.

No.	**Pit wall height, m**	**Waste dump height, m**	**SRF for simulation under different length of buffer zone**		
			0	**100 m**	**200 m**
1	240	50	1.33	1.56	1.62
2		100	1.14	1.41	1.54
3		150	1.06	1.34	1.36
4	340	50	1.22	1.33	1.42
5		100	1.12	1.27	1.3
6		150	0.94	1.19	1.2
7	440	50	1.15	1.26	1.32
8		100	1.07	1.16	1.22
9		150	0.87	1.12	1.19

5.4. Discussion

5.4.1. Effect of height of pit wall and waste dump on slope stability under without buffer zone condition

Figure 5-9 shows great correlation between pit wall height and waste dump height under without buffer zone condition on determination of slope stability (SRF). The correlation shows exponential correlation that can be written in an equation as follows:

$$\mathrm{SRF} = ke^{-bt_{\mathrm{wd}}} \tag{5.1}$$

where t_{wd} is the waste dump height (m). The constants of the equation for each pit wall height are given in Table 5.4.

Table 5.4. Constant k and b of SRF equation under without buffer zone condition.

No.	PW height	Exponential correlation (Equation: SRF $= ke^{-bt_{\mathrm{wd}}}$)			
		K	***b***	R^2	***R***
1	240 m	1.4418	0.002	0.9766	0.9988
2	340 m	1.3962	0.003	0.9826	0.9991
3	440 m	1.2941	0.002	0.9454	0.9972

Considering to constant *k* and *b* that given in Table 5.3, constant *k* is likely related to pit wall height. Constant *k* decreases with increasing the pit wall height. It is clearly shown in Figure 5-14. The correlation coefficient, *R*, between pit wall height and constant *k* is 0.9765. The correlation is described with an equation as follows:

$$k = -0.0007t_{pw} + 1.6285 \text{ (linear)}$$
$$k = 1.65e^{-0.0005t_{pw}} \text{ (exponential)} \tag{5.2}$$

where t_{pw} is the pit wall height (m).

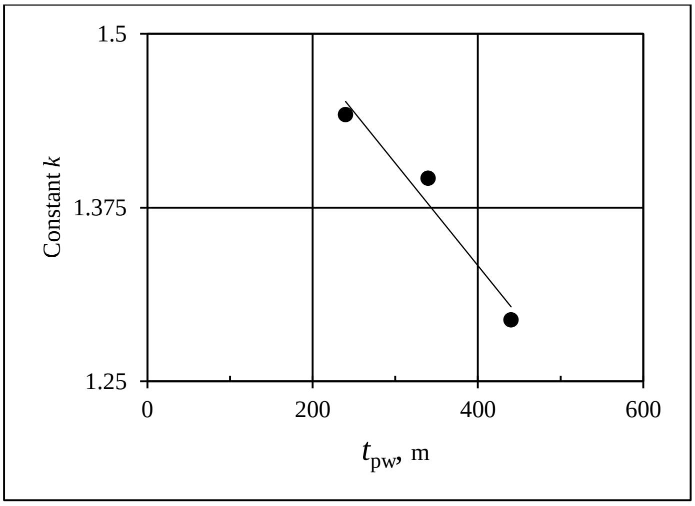

Figure 5-14. Correlation between constant k of SRF equation with pit wall height under without buffer zone condition.

Meanwhile, concerning to Equation (1), constant *b* has likely interdependency with waste dump height. Table 5.4 shows that constant *b* slightly increases with changing the pit wall height from 240 m to 340. However, it is back when the pit wall height increases from 340 m to 440 m. The increasing and decreasing of constant *b* toward pit wall height indicates that there is no direct relationship between constant *b* and pit wall height. Regardless increasing and decreasing of constant *b* toward pit wall height, considering to safety and engineering point of view the constant *b* can be assumed 0.002 (Table 5.4).

By considering constant *b* is 0.002, the SRF can be calculated for a particular pit wall height and waste dump height substituting Equation (5.2) to Equation (5.1). For example, when the excavation has reached 400 m deep, the constant *k* should be given 1.3485, which then is substituted to Equation (5.1) to calculate SRF. In case of the waste dump height is 50 m, 100 m and 150 m, the SRF should be given 1.22, 1.1 and 1, respectively. Compared to the SRF of 340 m pit wall height and 400 m pit wall height for 50 m, 100 m and 150 m waste dump height, the value is almost same with 340 m pit wall height and slightly higher than that of 400 m pit wall height.

5.4.2. Effect of height of pit wall and waste dump on slope stability under different buffer zone condition

Tables 5.5 and 5.6 give constants *k* and *b* of exponential correlation for a case of 100 m and 200 m buffer length, respectively, that derived from Figure 5-13. The table show that the correlation is follows Equation (1). It is shown in the tables that the constant *b* is ranged from 0.001 to 0.002, which is slightly different with constant *b* under without buffer zone that ranged from 0.002 to 0.003. Considering to it, it is technically acceptable, if 0.002 is chosen as value of constant *b*.

Table 5.5. Constant k and b SRF equation for case of 100 m buffer length.

No.	PW height	Exponential correlation (Equation: SRF $= ke^{-bt_{\mathrm{wd}}}$)			
		k	***b***	R^2	***R***
1	240 m	1.6692	0.002	0.965	0.9982
2	340 m	1.4105	0.001	0.9905	0.9995
3	440 m	1.3259	0.001	0.9484	0.9974

Table 5.6. Constant k and b SRF equation for case of 200 m buffer length.

No.	PW height	Exponential correlation (Equation: SRF $= ke^{-bt_{\mathrm{wd}}}$)			
		k	***b***	R^2	***R***
1	240 m	1.7899	0.002	0.9442	0.9971
2	340 m	1.5426	0.002	0.9992	1.0000
3	440 m	1.3778	0.001	0.9174	0.9957

Considering to factors that influencing constant *k*, it is likely pit wall height and buffer length give a lot influence to constant *k*. The correlation between constant *k* and pit wall height is descripted at Figure 5-15. The correlation gives the equation as follows:

$$k_{100} = 2.1617e^{-0.001t_{\mathrm{pw}}} \quad \textbf{(5.3)}$$

$$k_{200} = 2.4357e^{-0.001t_{\mathrm{pw}}} \quad \textbf{(5.4)}$$

The correlation coefficient of Equations (5.3) and (5.4) is given 0.9661 and 0.9969, respectively.

It is shown in the Equations (5.3) and (5.4) that the constant b for case of 100 m and 200 m buffer length is same i.e. 0.001. It is indicates that there is strong correlation between 100 m and 200 m buffer length.

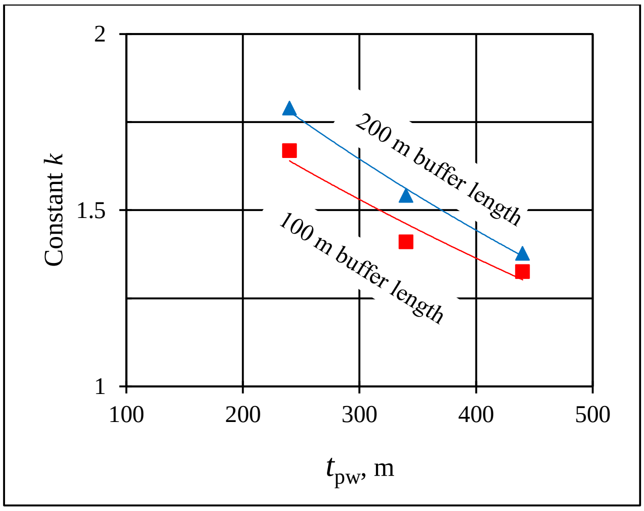

Figure 5-15. Constant k as function of t_{pw} for 100 m buffer length and 200 m buffer length.

When the constant *k* of Equations (5.3) and (5.4) and constant *k* of Equation (5.2) after changed into exponential correlation are plotted toward buffer length is plotted toward buffer length, there is an excellent relationship that can be used to determine a constant that be found on Equations (5.3) and (5.4). Let we say this constant with constant *c*. The relationship is illustrated in Figure 5-16 and Equation (5.5).

$$c = 1.9e^{0.001l_{\text{bz}}} \qquad \textbf{(5.5)}$$

where l_{bz} is buffer length. The correlation coefficient of Equation (5.5) is given 0.9763.

By substituting Equation (5.4) to Equations (5.3) and (5.4), we can generate new equation to determine constant *k* as follows:

$$k = 1.9e^{0.001l_{bz}}e^{-0.001t_{\text{pw}}} \qquad \textbf{(5.6)}$$

Equation (5.6) can be simply being:

$$k = 1.9e^{0.001(l_{\mathrm{bz}}-t_{\mathrm{pw}})} \qquad \textbf{(5.7)}$$

Considering to Equation (5.1) can be written as,

$$\mathrm{SRF} = ke^{-0.002t_{\mathrm{wd}}} \qquad \textbf{(5.8)}$$

The SRF for case of buffer zone is adopted, can be calculated by substituting Equation (5.7) to Equation (5.8). The equation is as follows:

$$\mathrm{SRF} = 1.9e^{0.001(l_{\mathrm{bz}}-t_{\mathrm{pw}})}e^{-0.002t_{\mathrm{wd}}} \qquad \textbf{(5.9)}$$

Equation (5.9) can be simply being:

$$\mathrm{SRF} = 1.9e^{0.001(l_{\mathrm{bz}}-t_{\mathrm{pw}}-2t_{\mathrm{wd}})} \qquad \textbf{(5.10)}$$

Equation (5.10) can be used to calculate SRF when the model adopts buffer zone. The example of calculation is given in Table 5.7 for a case of 100 m buffer length. Compared to SRF generated by numerical simulation that given at Table 5.3, the SRF that calculated by Equation (5.10) gives lower SRF. It is acceptable considering to the Equation (5.10) is generated from pessimistic point of view for safety reason.

Table 5.7. Predicted SRF for case of 100 m buffer length.

No.	Pit wall height, m	Waste dump height, m	SRF for 100 m buffer length
1	240	50	1.48
2		100	1.34
3		150	1.21
4	340	50	1.34
5		100	1.21
6		150	1.09
7	440	50	1.21
8		100	1.09
9		150	0.99

5.4.3. Effect of changing pit wall angle

In many cases, a deep excavation of mineral deposit in an open pit mine method is followed by the changing of pit wall angle, whether to satisfy the stability criteria or to reduce cost. In case of reducing operational cost, a changing of pit wall angle can be interpreted as increasing the overall angle. In order to clarify effect of increasing pit wall bench angle on the stability of pit wall which have waste dump without buffer zone, the study by means of numerical simulation is carried out. In this study, 60^o pit wall angle is considered. The result is then compared with the result of 45^o pit wall angle.

In general, the SRF resulted by 60^o pit wall angle is worse than that of 45^o pit wall angle. The SRF decreases 0.25 point in average. The SRF resulted from simulation for 60^o pit wall angle is given in Table 5.8. The illustration of shear stress distribution for case of 60^o pit wall angle is given in Figure 5-16. It is shown in the figure that the tensile stress on toe region is higher for 60^o pit wall angle compared to 45^o pit wall angle. It is because increase in pit wall angle may induce increase in gravity loading. By having buffer zone, the SRF can be improved thus may satisfy stability criteria.

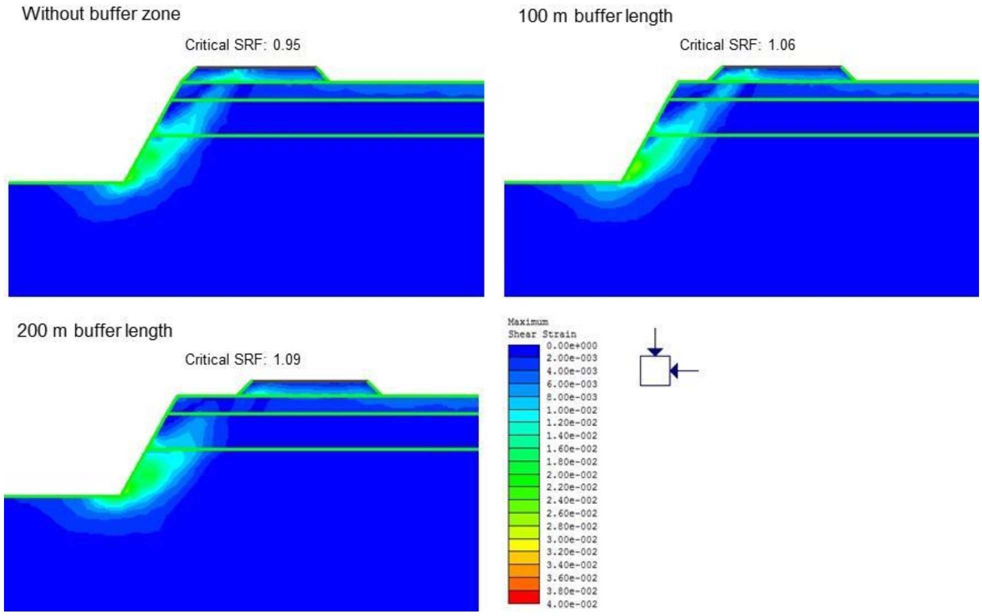

Figure 5-16. Buffer zone's influence on bench stability for case of 60^o pit wall angle of 340 m pit wall height with 50 m wasted dump height.

5.4.3. Effect of changing friction angle and cohesion

Friction angle and cohesion is one of the main parameter in slope stability analysis. The quality of friction angle and cohesion may being a key in constructing a stable slope. Considering to this matter, the effect of changing friction angle and cohesion is investigated in this study. In this study, the friction angle and cohesion quality has been weakened to 0.75 times and strengthened to 1.25 times of the original friction angle and cohesion. The simulation results are given in Table 5.8 to 5.12.

Table 5.8 to 5.12 shows that the SRF is improved for better friction angle and cohesion. When the quality of friction angle and cohesion is lower up to 0.75 times of the original quality, the SRF can reduce up to 0.179 and 0.099 in average, respectively. In other hand, the SRF can improve to 0.185 and 0.12 in average when the quality of friction angle and cohesion, respectively, is higher up to 1.25 times of the original quality. According to Table 5.8 to 5.12, changing friction angle generates more significant SRF shifting than that of cohesion.

Table 5.8. Result of simulation for case of changing pit wall angle.

<table>
<tr><th rowspan="2">No</th><th colspan="2">Pit wall</th><th rowspan="2">Buffer zone</th><th colspan="2">Waste dump</th><th rowspan="2">SRF</th><th rowspan="2">SRF45 - SRF60</th></tr>
<tr><th>Angle, º</th><th>Height, m</th><th>Angle, º</th><th>Height, m</th></tr>
<tr><td>1</td><td rowspan="23">60</td><td rowspan="10">240</td><td rowspan="4">0</td><td>No dump</td><td>0</td><td>1.21</td><td>0.22</td></tr>
<tr><td>2</td><td rowspan="9">45</td><td>50</td><td>1.09</td><td>0.24</td></tr>
<tr><td>3</td><td>100</td><td>0.98</td><td>0.16</td></tr>
<tr><td>4</td><td>150</td><td>0.85</td><td>0.21</td></tr>
<tr><td>5</td><td rowspan="3">100</td><td>50</td><td>1.19</td><td>0.37</td></tr>
<tr><td>6</td><td>100</td><td>1.14</td><td>0.27</td></tr>
<tr><td>7</td><td>150</td><td>1.09</td><td>0.25</td></tr>
<tr><td>8</td><td rowspan="3">200</td><td>50</td><td>1.25</td><td>0.37</td></tr>
<tr><td>9</td><td>100</td><td>1.24</td><td>0.3</td></tr>
<tr><td>10</td><td>150</td><td>1.19</td><td>0.17</td></tr>
<tr><td>11</td><td rowspan="10">340</td><td rowspan="4">0</td><td>No dump</td><td>0</td><td>1.09</td><td>0.3</td></tr>
<tr><td>12</td><td rowspan="9">45</td><td>50</td><td>0.95</td><td>0.27</td></tr>
<tr><td>13</td><td>100</td><td>0.86</td><td>0.26</td></tr>
<tr><td>14</td><td>150</td><td>0.8</td><td>0.14</td></tr>
<tr><td>15</td><td rowspan="3">100</td><td>50</td><td>1.06</td><td>0.27</td></tr>
<tr><td>16</td><td>100</td><td>1.02</td><td>0.25</td></tr>
<tr><td>17</td><td>150</td><td>0.97</td><td>0.22</td></tr>
<tr><td>18</td><td rowspan="3">200</td><td>50</td><td>1.09</td><td>0.33</td></tr>
<tr><td>19</td><td>100</td><td>1.07</td><td>0.23</td></tr>
<tr><td>20</td><td>150</td><td>1.02</td><td>0.18</td></tr>
<tr><td>21</td><td rowspan="3">440</td><td rowspan="3">0</td><td>No dump</td><td>0</td><td>0.995</td><td>0.275</td></tr>
<tr><td>22</td><td rowspan="2">45</td><td>50</td><td>0.9</td><td>0.25</td></tr>
<tr><td>23</td><td>100</td><td>0.83</td><td>0.24</td></tr>
</table>

24					150	0.77	0.1
25			100		50	0.92	0.34
26					100	0.9	0.26
27					150	0.87	0.25
28			200		50	0.99	0.33
29					100	0.97	0.25
30					150	0.96	0.23

Table 5.9. Result of simulation for case of weakening friction angle to 0.75x

No	**Pit wall**		**Buffer zone**	**Waste dump**		**SRF**	**$SRF_{fric.}$ – $SRF_{0.75fric.}$**
	Angle, °	**Height, m**		**Angle, °**	**Height, m**		
1	45	240	0	No dump	0	1.4	0.03
2				45	50	1.14	0.19
3					100	1.02	0.12
4					150	0.92	0.14
5			100		50	1.31	0.25
6					100	1.18	0.23
7					150	1.13	0.21
8			200		50	1.37	0.25
9					100	1.33	0.21
10					150	1.27	0.09
11		340	0	No dump	0	1.16	0.23
12				45	50	1.03	0.19
13					100	0.94	0.18
14					150	0.89	0.05
15			100		50	1.11	0.22
16					100	1.06	0.21
17					150	0.99	0.2
18			200		50	1.18	0.24
19					100	1.12	0.18
20					150	1.09	0.11
21		440	0	No dump	0	1.07	0.2
22				45	50	0.97	0.18
23					100	0.9	0.17
24					150	0.86	0.01
25			100		50	1.02	0.24
26					100	0.96	0.2
27					150	0.93	0.19
28			200		50	1.05	0.27
29					100	1.03	0.19

30					150	1	0.19

Table 5.10. Result of simulation for case of strengthening friction angle to 1.25x

No	Pit wall		Buffer zone	Waste dump		SRF	$SRF_{fric.} - SRF_{1.25fric.}$
	Angle, º	Height, m		Angle, º	Height, m		
1	45	240	0	No dump	0	1.91	-0.48
2				45	50	1.51	-0.18
3					100	1.16	-0.02
4					150	1.1	-0.04
5			100		50	1.81	-0.25
6					100	1.63	-0.22
7					150	1.42	-0.08
8			200		50	1.87	-0.25
9					100	1.66	-0.12
10					150	1.63	-0.27
11		340	0	No dump	0	1.64	-0.25
12				45	50	1.43	-0.21
13					100	1.16	-0.04
14					150	1.07	-0.13
15			100		50	1.57	-0.24
16					100	1.49	-0.22
17					150	1.4	-0.21
18			200		50	1.67	-0.25
19					100	1.54	-0.24
20					150	1.48	-0.28
21		440	0	No dump	0	1.31	-0.04
22				45	50	1.18	-0.03
23					100	1.11	-0.04
24					150	1.06	-0.19
25			100		50	1.49	-0.23
26					100	1.39	-0.23
27					150	1.32	-0.2
28			200		50	1.5	-0.18
29					100	1.43	-0.21
30					150	1.41	-0.22

Table 5.11. Result of simulation for case of weakening cohesion to 0.75x

No	Pit wall		Buffer zone	Waste dump		SRF	$SRF_{coh.} - SRF_{0.75coh.}$
	Angle, º	Height, m		Angle, º	Height, m		

1	45	240	0	No dump	0	1.46	-0.03
2				45	50	1.17	0.16
3					100	1.03	0.11
4					150	0.97	0.09
5			100		50	1.38	0.18
6					100	1.27	0.14
7					150	1.25	0.09
8			200		50	1.43	0.19
9					100	1.4	0.14
10					150	1.37	-0.01
11		340	0	No dump	0	1.26	0.13
12				45	50	1.1	0.12
13					100	1	0.12
14					150	0.92	0.02
15			100		50	1.21	0.12
16					100	1.14	0.13
17					150	1.09	0.1
18			200		50	1.28	0.14
19					100	1.21	0.09
20					150	1.18	0.02
21		440	0	No dump	0	1.19	0.08
22				45	50	1.07	0.08
23					100	0.97	0.1
24					150	0.82	0.05
25			100		50	1.14	0.12
26					100	1.07	0.09
27					150	1.02	0.1
28			200		50	1.2	0.12
29					100	1.13	0.09
30					150	1.1	0.09

Table 5.12. Result of simulation for case of strengthening cohesion to 1.25x.

No	**Pit wall**		**Buffer zone**	**Waste dump**		**SRF**	**$SRF_{coh.}$ – $SRF_{1.25coh.}$**
	Angle, º	**Height, m**		**Angle, º**	**Height, m**		
1	45	240	0	No dump	0	1.82	-0.39
2				45	50	1.48	-0.15
3					100	1.17	-0.03
4					150	1.12	-0.06
5			100		50	1.72	-0.16
6					100	1.54	-0.13

7					150	1.4	-0.06
8					50	1.8	-0.18
9			200		100	1.63	-0.09
10					150	1.5	-0.14
11				No dump	0	1.55	-0.16
12					50	1.35	-0.13
13			0		100	1.18	-0.06
14					150	0.98	-0.04
15		340			50	1.46	-0.13
16			100	45	100	1.4	-0.13
17					150	1.29	-0.1
18					50	1.56	-0.14
19			200		100	1.43	-0.13
20					150	1.39	-0.19
21				No dump	0	1.41	-0.14
22					50	1.28	-0.13
23			0		100	1.16	-0.09
24					150	0.95	-0.08
25		440			50	1.35	-0.09
26			100	45	100	1.26	-0.1
27					150	1.21	-0.09
28					50	1.42	-0.1
29			200		100	1.32	-0.1
30					150	1.26	-0.07

5.5. Summary

Creating waste dump near to the pit is one of the solutions when the waste rock contain low grade of valuable minerals that planned to be extracted in future. However, from geotechnical point of view, constructing a waste dump alongside the pit should be planned well particularly distance between final pit’s boundaries to waste dump’s boundary which is referred as buffer zone.

According to simulation result, this chapter suggests formulae to calculate optimum buffer length that satisfy stability criteria for any conditions in regards to pit wall height and waste dump height. When the configuration is without buffer zone, the SRF can be calculated by using equation:

$$k = 1.65e^{-0.0005t_{pw}}$$

While, when the configuration considers the buffer zone, the SRF can be determined by using equation:

$$\mathrm{SRF} = 1.9e^{0.001(l_{\mathrm{bz}}-t_{\mathrm{pw}}-2t_{\mathrm{wd}})}$$

The ratio $t_{\mathrm{pw}}/t_{\mathrm{wd}}$ is 4 to satisfy stability criteria (in case of SRF = 1.2). This study also shows that the increase in pit wall angle, the SRF reduces due to increasing gravity loading. The tensile stress around toe region increases significantly. Moreover, shear stress along the slope is also increased that may induce slope slide. The study shows that increase in pit wall angle from 45^{o} to 60^{o}, the quatity of SRF is reduced up to 0.25 point in average.

In the slope stability, friction angle and cohesion is proven plays important role. In this chapter, we also conduct a study to investigate the influence of friction angle and cohesion. The study shows that the SRF decreases with decreasing quality of friction angle and cohesion, and increases with increasing quality of friction angle and cohesion. It has been found that when the quality of friction angle and cohesion is lower up to 0.75 times of the original quality, the SRF can reduce up to 0.179 and 0.099 in average, respectively. In other hand, the SRF can improve to 0.185 and 0.12 in average when the quality of friction angle and cohesion, respectively, is higher up to 1.25 times of the original quality.

This conclusion is important to be considered when optimization of dumping area is discussed

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CHAPTER VI. Optimization of Dumping Area

6.1. Introduction

The waste rock dump does not have any intrinsic value; however, it is inevitable for most open pit mines. Due to the large volume of material handling and the associated haulage cost, a waste rock dump could be the most expensive structure to be built in an open pit mine. The implementation of truck-shovel operations within open pit mines in Mongolia is an extremely popular material extraction method. The haulage and dumping aspect of open pit mining operations is one of the largest cost components of the mining cost constituting approximately 50-60% of mining operational costs.

The overall aim of correct waste dump design is to plan a series of waste disposal stages that will effectively minimize the vertical and horizontal distances between the pit and potential waste dump site. Therefore the designing and planning of waste dumps can significantly affect the total operational expenses. The two major parameters that dictate the overall design and staging of waste dumps are pit mining sequence and the production schedule, as these parameters influence the waste dump starting location, advancing rate and the ultimate dump volume (Kennedy, 1990).

The dumping can be defined as a massive structure formed by placing large amounts of material in lifts of a restricted vertical expansion that laid one on top of each other and form a stable slope at the angle of repose. The dumping is so formed, however, needs a horizontal base at first, which is built by push dumping material from a certain elevation and levelling off the required footprint area.

Generally, this first phase of the dump construction takes the irregular shape of the topography where is placed. Subsequent lift height is constant, though is restricted to prevent shear stresses on the foundation and is a factor to control consolidations and permeability variations (Zanbak, 2012).

The total height of the dump is also restricted by formation mechanism (Zhang et al., 2014) and carrying capacity limitations (Peng, Ji, Zhao, & Ren, 2013). As in most of the large open pit operations, haulage is performed by heavy trucks, the access to the successive dump lifts is achieved by establishing ramps of a suitable width, super elevation and gradient in order to minimize travel distance and therefore to reduce haulage costs. The dominated factors in dumping designing procedures as follows (Ortiz, 2017):

Geometry. Usually designed to handle a total capacity throughout the life-of-mine. Over dimensioning can cause underutilization of valuable areas. Under dimensioning can result in the increase of the total haulage distances.

Operating costs. Costs resulting from fuel, energy, maintenance and labour of the haul trucks.

Haulage distances. Minimizing the total haulage distance while meeting the required capacity by strategic placing of the ramps, exits, entrances and dumping sequence.

Stability control. It will define the angle of repose and the nature of the underlying material. Maintaining the stability of the dump may require relocation of weathered rock or material blending, especially if water is present.

Although every dump is unique (Zástěrová et al., 2015) and some of its cost maybe be given by its own factors, the above description includes all the general concerns one would have to elaborate the most economical dump design

In the previous Chapters, the possiblity expansions of open pit boundary, formation mechanism and stability of dumping area and buffer zone optimization between open pit and dumping area have been discussed. Based on the previous parameter optimization results, this chapter studies a methodology to optimize the ultimate dump design in a mining operation by minimizing the unit haulage cost based on haul length and cost comparisons of dumping planning direction options using Geovia Minesched software.

6.2. Model implementation

In current practice, many of the mine scheduling software treats a rock dump as a single point or multiple points based on the number of lifts. The simplification improves the solution time, yet such a schedule often contains little information about the waste rock placement, i.e., the actual spatial dumping location is not available. The lack of detailed waste rock dumping information could potentially cause misalignment between the engineering design and the site operation, ultimately failing to achieve the long-term design objectives, such as the rock dump height, capacity, footprint, and slope angle.

The purpose of the dump block model is to derive more spatial information about a rock dump, i.e., the detailed locations within a rock dump and the capacity at each individual location, in such way that a practical dump schedule can be realistically generated and utilized by mining operations. Furthermore, it allows direct estimation of haulage distance and haulage cost with much higher accuracy (Li, 2014).

6.2.1. Dump design

The dump designs with same volumes and two different height scenarios which named d100 and d200 were generated using the Geovia Surpac software based on dumping space availability and total dumping volume requirement of ultimate pit design of Erdenet open pit mine as mentioned in Chapter 3 and 4.

Table 6.1. Summary of the dump design scenarios

	Unit	**d100**	**d200**
Base area	km2	7.90	4.87
Average height	m2	100.00	200.00
Volume capacity	mil.m3	549.21	552.15
Bench height	m	50	100
Bench numbers	-	2	2

According to the open pit design, the total required waste rock storage capacity is approximate amount of at least 550 million m^3 during its 43 years life-of-mine operation. Land properties extend its limits on the open pit area about 8 km^2 of surface available. Figure 6-1 shows the three design options.

The boundary of d200 scenario is reduced in Y axis by 800m from respectively while retaining the same north side and X axis For the three limit areas, the west side and the horizontal axis are the same to keep the shortest distance from the open pit exit. For operational convenience, property limits have been made squared, although the dump design maintains smoothed boundaries.

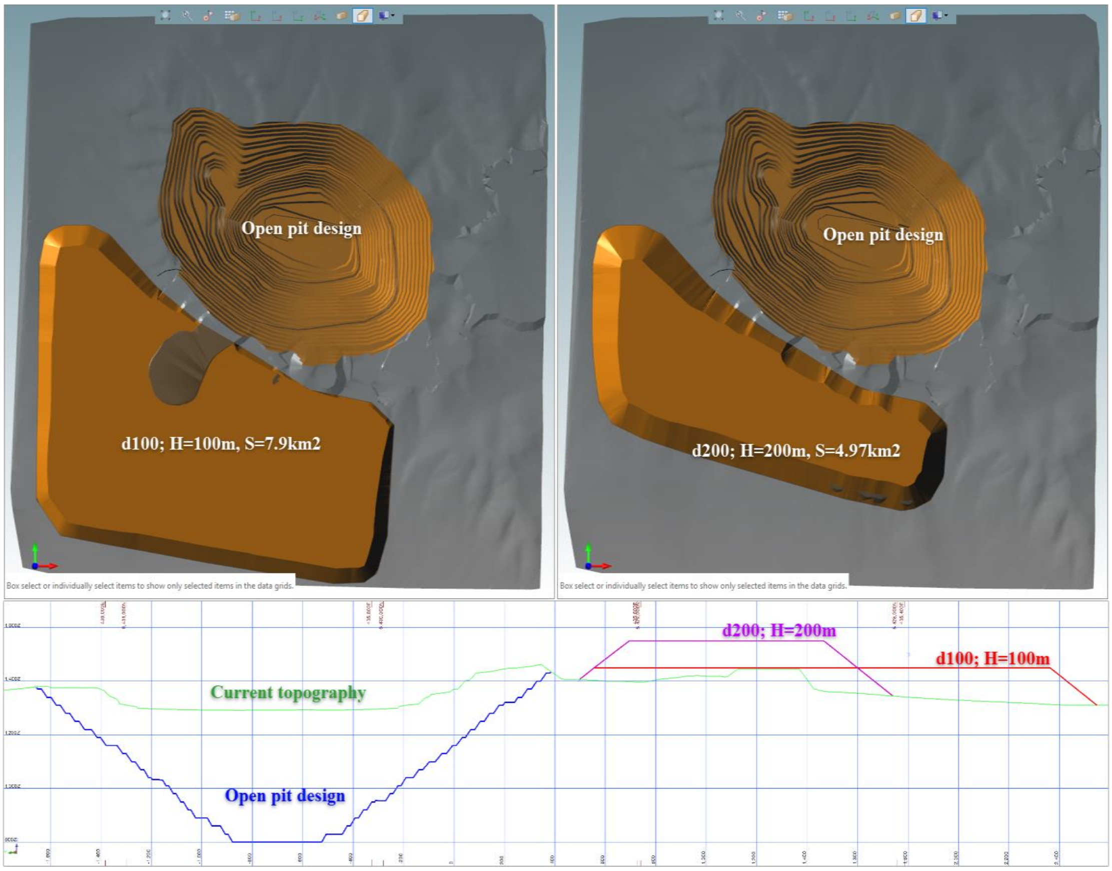

Figure 6-1 Dumping design scenarios

The 3D block model of the waste dumping area were generated using the Geovia Surpac software using design scenarios of d100 and d200. The block model for this study estimated basic size of the each block in the model is Y10 x X10 x Z10, The total number of possible dump blocks in d100 is 126,615, and in d200 is 95,105 and the block model conducted by material attributes and summarized in Figure 6-2.

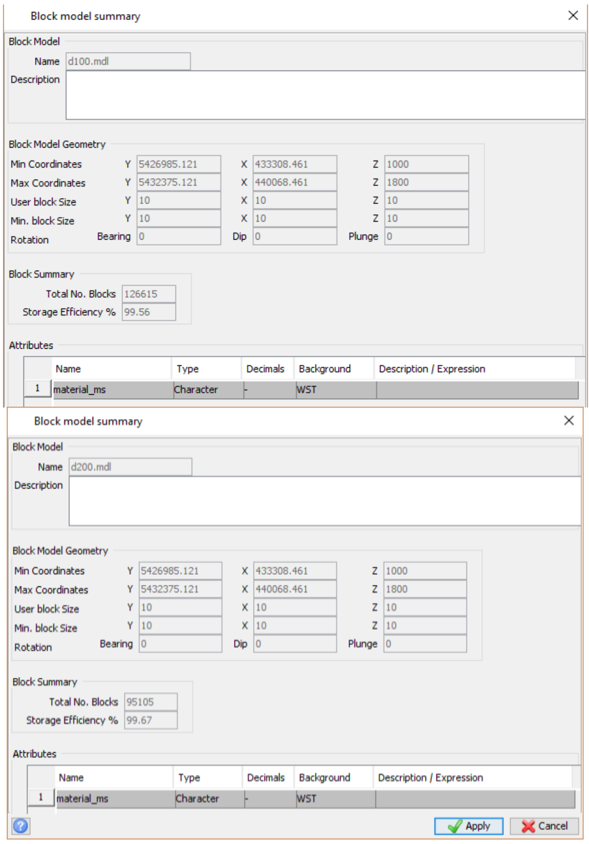

Figure 6-2 Dumping area block model summary; a.d100, b.d200

6.2.2. Haul road design

The implement of main waste rock hauling road design from bottom level of open pit to exit point was carried out considering the shortest length and without sharp curve for this study. Open pit mine design also generated as simple option such as without ore hauling design due to much computer memory and time usage for many numbers of production scheduling simulations.

The parameters of haul road in ultimate pit design are; total length is 8.84 km, minimum circle radius is 70m, total curve points are 267 and total elevation change is +-565.13m. The waste rock haul road design shows in Figure 6-3.

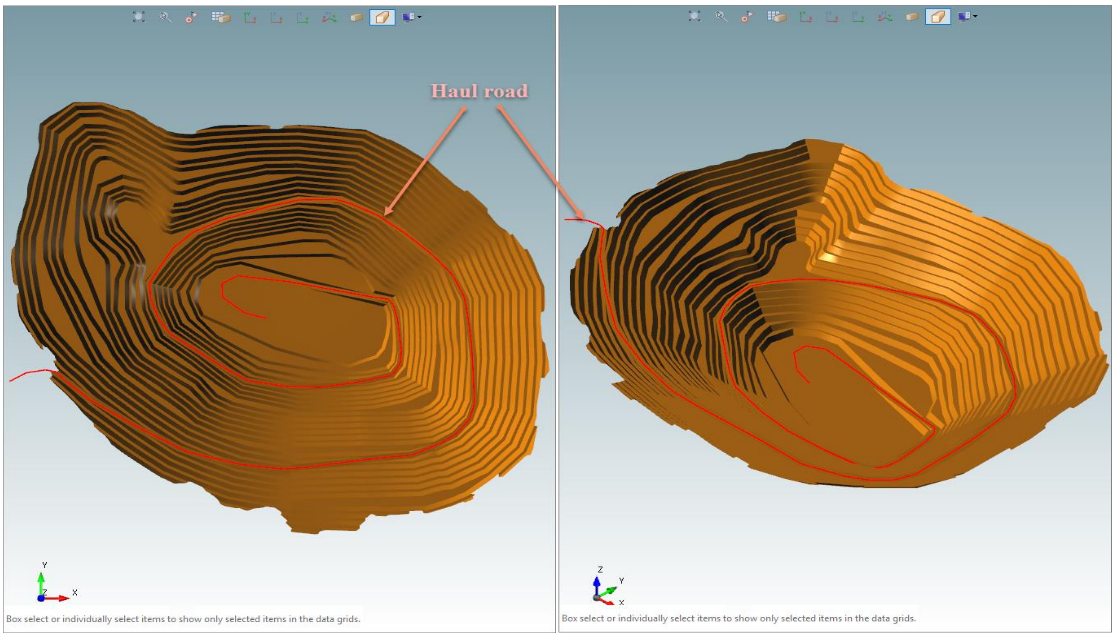

Figure 6-3 Haul design

6.2.3. Haulage routes and truck types

Round-travel speeds are given by the Erdenet mine's current equipment's technical specifications of the equivalent fleet truck in route; and operating costs include maintenance, fuel consumption, and labor. The ramp grade and lift height comply with the internal mine haul road design manual of the mine operation. Haulage profiles were designed for dumping locations. They contain horizontal, lift and drop segments.

Table 6.2. Summary of the route and truck

		Unit	Description
Truck type		-	BELAZ-7513
Tonnes per load		t.	130
Availability		%	85
Utilization		%	85
Efficiency		%	90
Velocities	Full path	kmph	20
	Empty path	kmph	25
Haul length		m	8,844.91

6.2.4. Material flows

The open pit design of Erdenet mine as mentioned in Chapter-3 (RF=1.0) covers totally 1,469.09 mil. tonnes of ore and 986.40 mil. tonnes of waste rock planned as totally 43 years of mine-life. In this study, production rates of waste rock operation from open pit was 90,000 t. per day at max rate. Figure 6-4 shows material movement from the open pit. Apart from waste rock, ore is to be transported and stockpiled to processing plant nearby with an assumed infinite capacity.

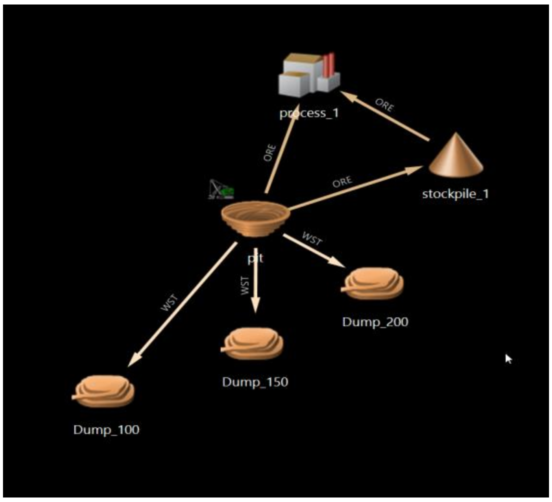

Figure 6-4 Scheme of material movement

6.3. Numerical simulation for the production scheduling and the comparisons between the dump design scenarios

The analysis of the haulage and production scheduling considering the widening and lifting influence on total operational costs at the Erdenet mine truck-shovel operation contains several design factors that could influence the end results. Therefore to ensure sufficient realistic designs were modelled and simulated to closely resemble the operation, a detailed haulage analysis was conducted in the software packages of Geovia Whittle, Surpac and Minesched.

The implementation of these software packages enabled realistic modelling of the truck-shovel operation by incorporating collected site data including surveyed topography files, pit and dump shells, current void profiles, planned haul routes, equipment specifications and accurate volume estimation. The production schedule is provided, and the annual movement is presented in Figure 6-5.

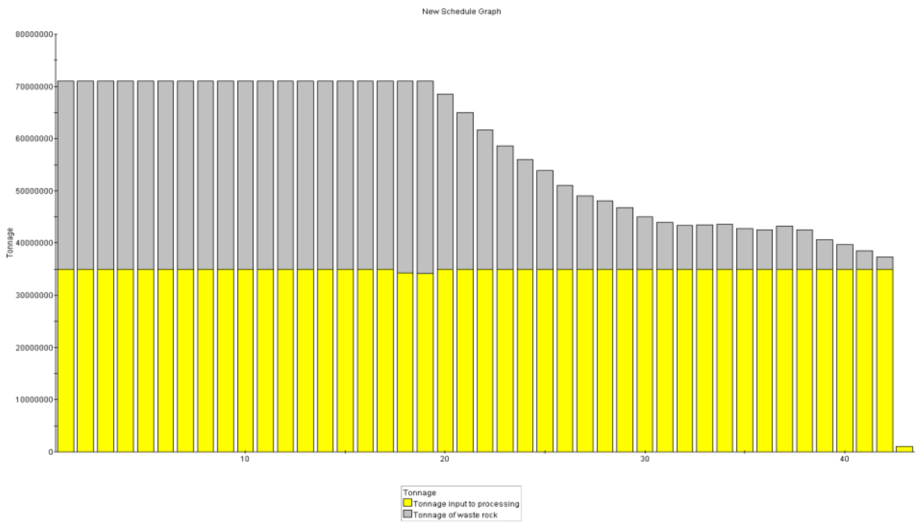

Figure 6-5 Material movement schedule from open pit mine

6.3.1. Directions of dump construction

The primary differing factor separating each scenario within the detailed haulage analysis is the pre-determined direction of the dump construction. The direction of dump construction is separated as 'North to All and South to All' in whole dumping area. Figure 6-6 shows the directions of dump construction.

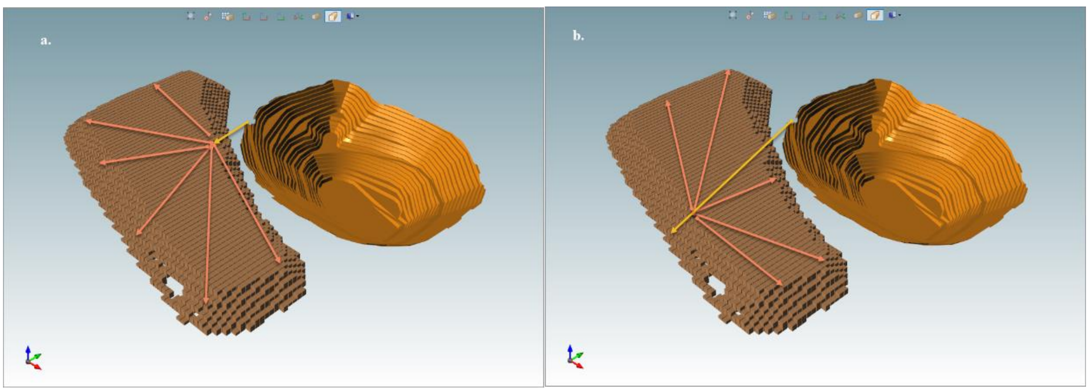

Figure 6-6 Directions of dump construction

6.3.5. Graphical results of dumping progressions

The dump block filling schedule enables predicting the final landform footprint and landform yearly progression, thus providing guidance for mine planning engineers in staged landform design. The dumping progression in d100 and d200 scenarios generated by the waste dump model, are illustrated in Figure 6-7 and Figure 6-8, respectively

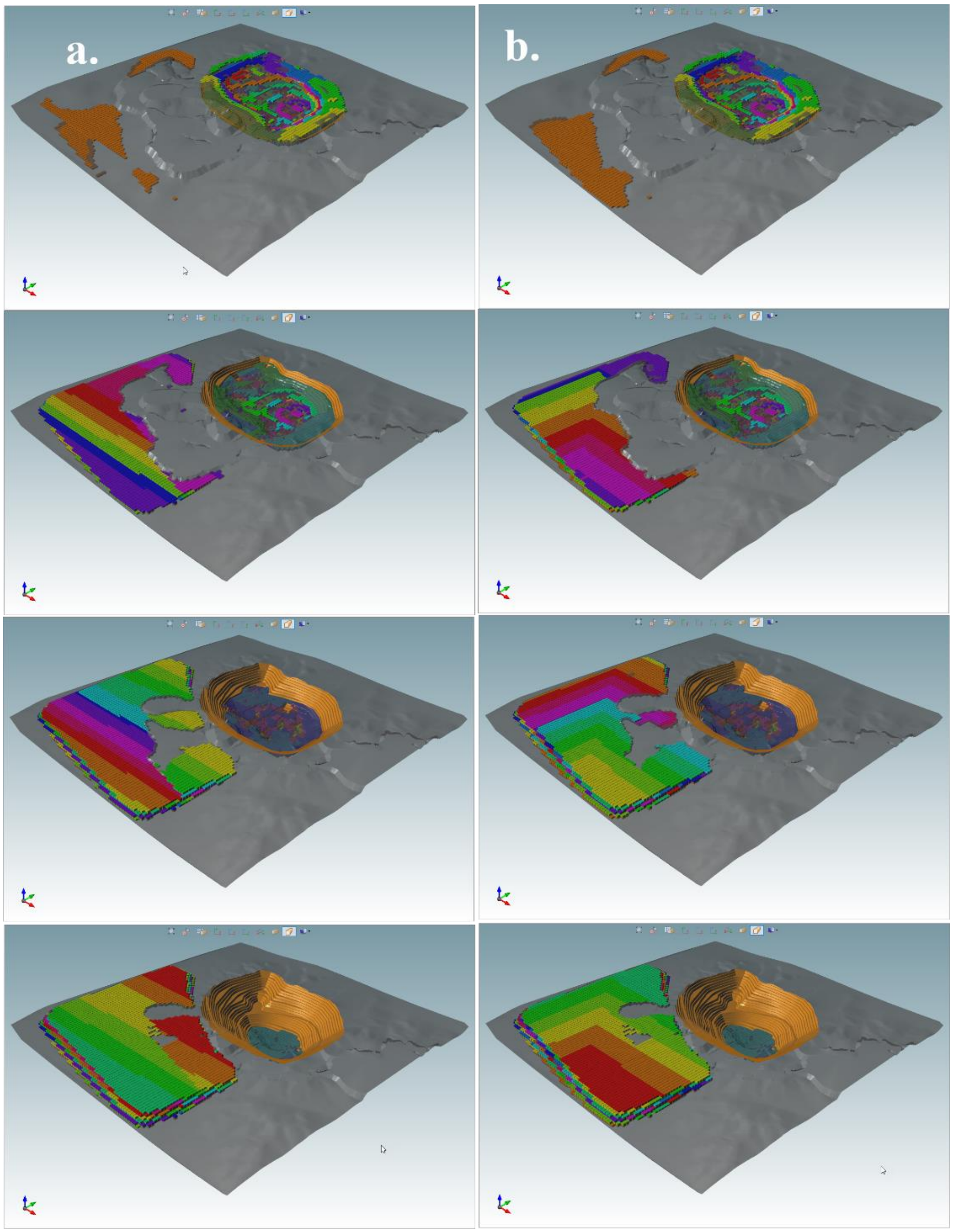

Figure 6-7 Dumping progressions in d100 scenario according to the: a. N to All, b. S to All (end of the Period 1, Period 15, Period 30 and Period 41)

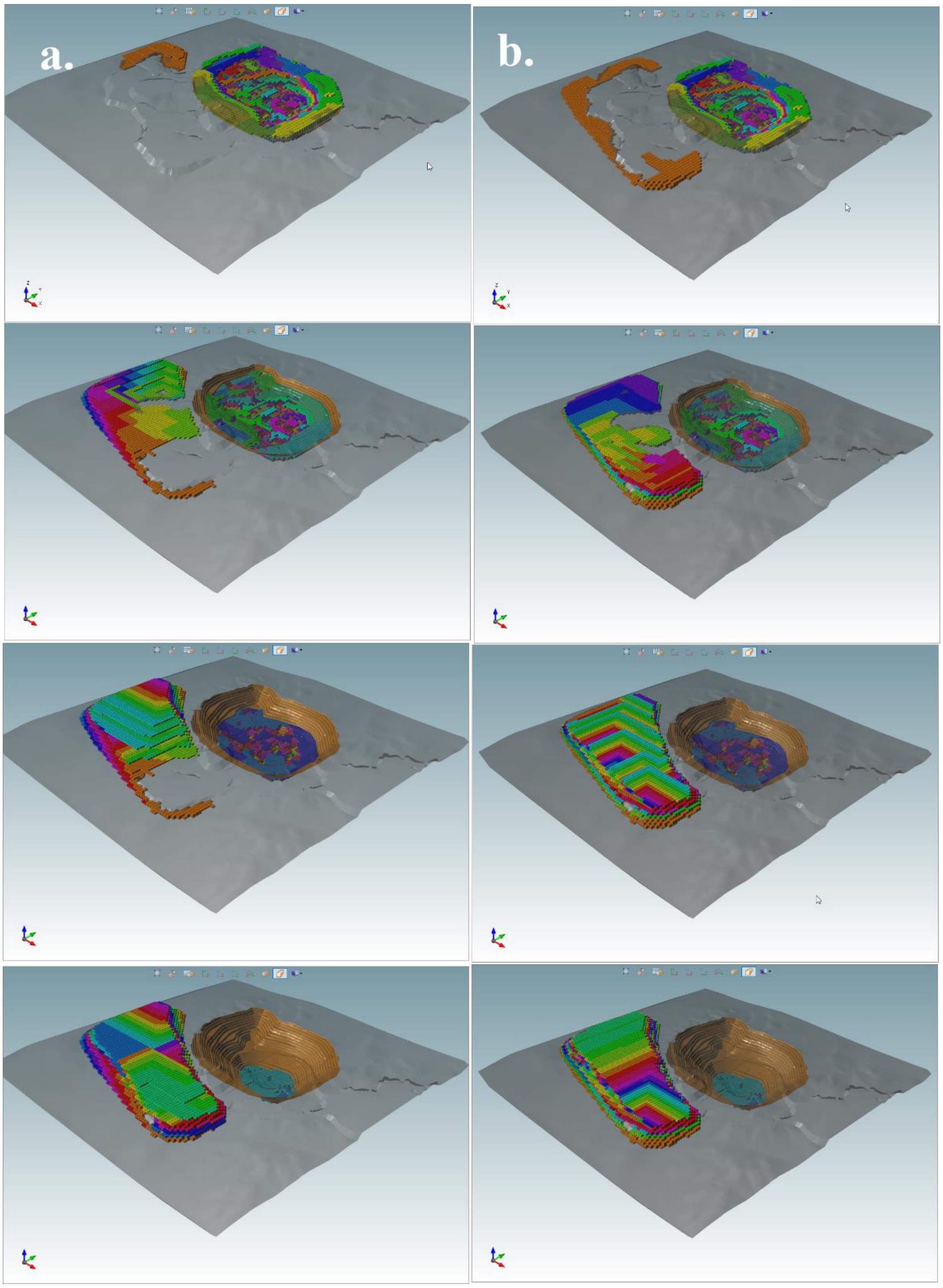

Figure 6-8 Dumping progressions in d200 scenario according to the: a. N to All, b. S to All (end of the Period 1, Period 15, Period 30 and Period 41)

6.3.2. Haulage distance

The waste rock movement from the open pit to dumping area schedules are generated by the design scenario block models. For each time period, the total equivalent flat haulage distance, the truck hours required and truck productivity are analyzed. The table 6.3 summarized the overall haulage distance for each dump design scenarios and directions of dump constructions. The estimated distances are presented in Figure 6-9.

Table 6.3. Summary of haulage distance

Scenarios	d100		d200	
	Haulage distance, km			
Period Number	N to ALL	S to ALL	N to ALL	S to ALL
1	4.77	4.63	3.05	5.39
2	4.87	5.00	4.43	7.50
3	4.84	4.54	5.56	9.20
4	5.18	4.46	7.07	9.65
5	4.05	3.75	7.89	11.76
6	4.81	4.91	6.48	11.27
7	4.47	4.63	7.70	10.27
8	2.89	3.25	7.51	10.83
9	3.08	3.66	7.52	8.98
10	4.95	3.53	6.39	9.37
11	4.62	3.77	7.54	10.55
12	3.55	3.27	8.06	10.56
13	4.17	4.24	7.95	10.17
14	4.46	4.69	7.80	10.53
15	4.04	4.51	7.27	10.33
16	3.84	4.82	8.11	10.09
17	7.78	8.28	8.12	9.93
18	11.25	10.54	8.46	10.35
19	10.55	10.36	7.90	9.61
20	10.77	10.84	7.87	9.48
21	10.66	10.63	8.40	9.63
22	10.34	10.46	8.49	8.77
23	10.53	10.73	8.11	8.78
24	10.30	10.48	7.79	9.70
25	9.85	10.73	7.77	9.13
26	10.45	9.48	8.11	8.59
27	9.53	8.86	7.71	9.41
28	9.62	9.24	7.58	8.60
29	9.27	9.19	7.62	8.41
30	8.85	8.93	7.52	8.85
31	8.90	9.01	7.43	7.98
32	8.41	8.73	7.51	8.53
33	8.51	8.80	7.41	7.71
34	7.94	8.34	7.14	7.74
35	7.36	8.25	6.88	7.32
36	7.72	6.92	7.12	7.01
37	7.06	6.64	6.96	6.50
38	6.14	6.26	6.58	5.85
39	5.53	5.83	6.83	5.24
40	5.37	5.61	6.70	4.76
41	3.52	4.17	6.25	4.48
42	3.80	3.90	6.09	1.65
43	3.10	3.20	5.00	2.5
Average	**6.78**	**6.79**	**7.20**	**8.44**

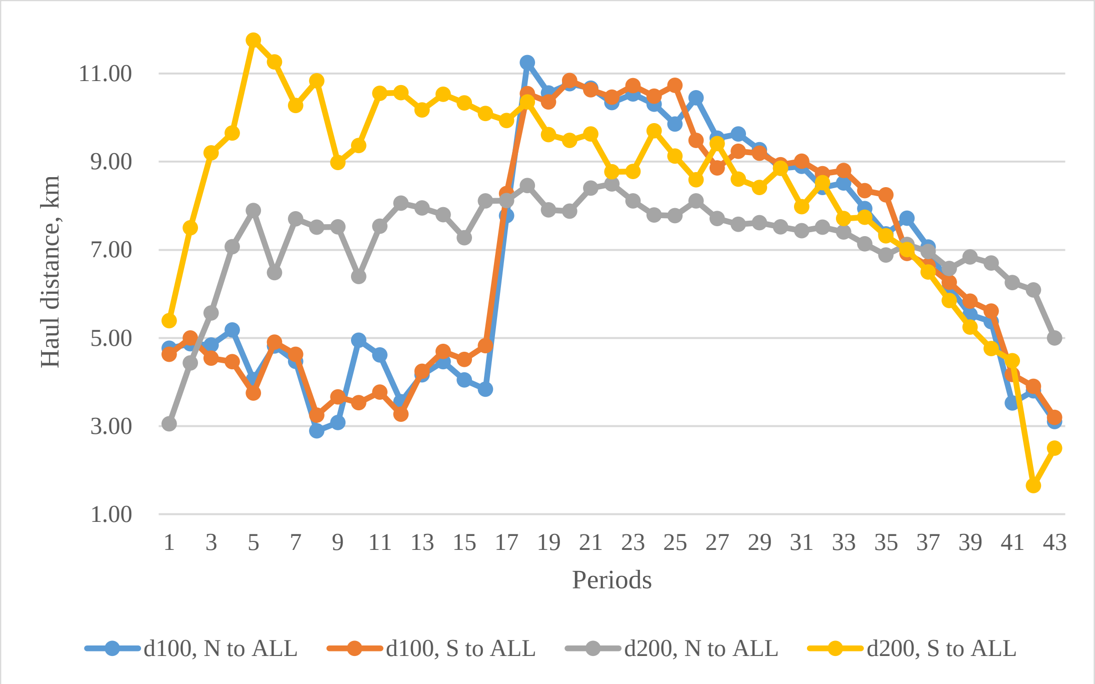

Figure 6-9 Overall haulage distance in each scenarios and dumping directions

From the results of haulage distance analysis, the average haulage distance in both directions of d100 scenario are relativity low but the graph shows rough results from the beginning of operation. The avarage haulage distance in both directions of d200 scenario are slightly high but the relativity smooth in whole mine-life.

6.3.3. Estimated truck hours

The haulage distance is divided by the average truck travel speed of 20km/h in full path and 20km/h in empty path to facilitate the estimation of the total truck hours and required numbers of trucks. This estimation results shows the quite same trend as the haulage distance. These results are summarized in Table 6.4.

Table 6.4. Estimated truck hours and total number of trucks

Scenarios	**d100**				**d200**			
Directions	**N to ALL**		**S to ALL**		**N to ALL**		**S to ALL**	
Period Number	**Truck hours, tous.hrs.**	**Total numbers of trucks**	**Truck hours, tous.hrs.**	**Total numbers of trucks**	**Truck hours, tous.hrs.**	**Total numbers of trucks**	**Truck hours, tous.hrs.**	**Total numbers of trucks**
1	140.62	24.69	144.04	25.29	95.51	16.77	159.82	28.06
2	149.98	26.33	146.60	25.74	130.37	22.89	213.04	37.40
3	138.64	24.27	146.33	25.62	159.36	27.90	256.66	44.94
4	136.26	23.92	154.58	27.14	197.00	34.58	267.49	46.96
5	118.26	20.76	126.03	22.12	217.72	38.22	320.70	56.30
6	147.53	25.90	145.23	25.50	182.04	31.96	308.25	54.11
7	140.92	24.67	136.79	23.95	213.48	37.38	283.93	49.71
8	105.61	18.54	96.68	16.97	208.13	36.54	297.35	52.20
9	116.10	20.38	101.38	17.80	208.33	36.57	250.59	43.99
10	112.78	19.80	148.66	26.10	179.93	31.59	260.24	45.69
11	119.26	20.88	140.62	24.62	209.40	36.66	290.97	50.94
12	106.29	18.66	113.41	19.91	222.02	38.98	290.50	51.00
13	130.79	22.96	128.86	22.62	219.16	38.47	280.66	49.27
14	142.17	24.96	136.35	23.94	215.36	37.81	289.65	50.85
15	137.94	24.15	126.12	22.08	202.73	35.49	285.40	49.97
16	145.47	25.54	120.59	21.17	223.20	39.18	278.64	48.92
17	232.76	40.86	220.09	38.64	223.48	39.23	274.51	48.19
18	289.90	50.89	307.88	54.05	232.08	40.74	285.25	50.08
19	286.03	50.08	291.07	50.96	218.69	38.29	267.23	46.79
20	297.54	52.24	295.73	51.92	217.29	38.15	263.24	46.21
21	292.12	51.28	293.07	51.45	230.57	40.48	266.81	46.84
22	287.97	50.55	284.80	50.00	232.91	40.89	245.27	43.06
23	295.46	51.73	290.58	50.87	223.90	39.20	246.05	43.08
24	288.44	50.64	283.97	49.85	215.10	37.76	268.69	47.17
25	294.72	51.74	272.50	47.84	214.79	37.71	254.22	44.63
26	263.09	46.19	287.55	50.48	223.28	39.20	240.63	42.24
27	248.05	43.43	265.21	46.43	213.79	37.43	262.11	45.89
28	257.02	45.12	266.79	46.84	209.91	36.85	240.88	42.29
29	255.91	44.93	257.88	45.27	210.82	37.01	236.14	41.46
30	249.18	43.75	247.15	43.39	208.28	36.57	247.21	43.40
31	251.97	44.11	249.04	43.60	206.77	36.20	225.86	39.54
32	244.06	42.85	236.12	41.45	208.11	36.54	239.01	41.96
33	246.02	43.19	238.72	41.91	205.49	36.08	218.40	38.34
34	234.27	41.13	224.16	39.35	198.68	34.88	219.09	38.46
35	232.64	40.73	210.10	36.78	192.78	33.75	209.02	36.59
36	198.44	34.84	218.65	38.39	198.22	34.80	200.66	35.23
37	191.42	33.60	202.11	35.48	194.28	34.11	187.81	32.97
38	181.76	31.91	178.82	31.39	184.59	32.41	171.39	30.09
39	171.47	30.02	163.69	28.66	191.55	33.54	156.53	27.40
40	165.28	29.02	159.30	27.97	187.55	32.93	143.87	25.26
41	123.47	21.68	107.05	18.79	176.35	30.96	136.80	24.02
Tot.Average	**199,21**		**199.13**		**202.51**		**244.89**	
		34.95		**34.93**		**35.53**		**42.96**

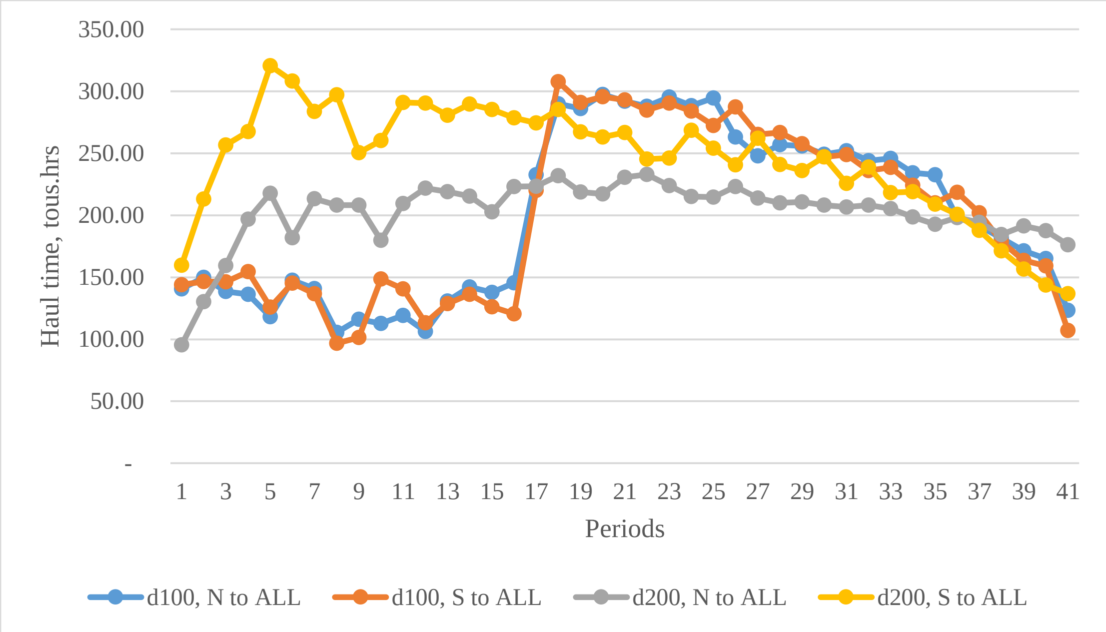

Figure 6-10 Truck hours

The truck hours and the required numbers of trucks of the d100 scenario design was relativity low in this simulation. However, the total number of truck in open pit mine demands relativity smooth numbers in long-term planning and optimizations.

6.3.4. Truck productivity per flat haulage distance (t/km)

The total waste rock mass handled in each year is divided by the total return trip distance to calculate the truck productivity, measured in t/km. Generally, the higher the t/km value, the better efficiency the haulage system. The yearly truck productivity yielded by the three schedules is summarized in Table 6.5, and shows in Figure 6-11.

Table 6.5. Truck productivity

Scenarios	d100		d200	
	Haulage distance, km			
Period Number	N to ALL	S to ALL	N to ALL	S to ALL
1	152.15	156.59	100.32	177.10
2	164.32	159.92	145.63	246.29
3	149.48	159.48	183.26	302.92
4	146.47	170.30	232.25	317.08
5	123.08	133.17	259.19	386.25
6	161.13	158.13	212.80	370.06
7	152.45	147.09	253.61	338.37
8	106.63	95.02	246.73	355.89
9	120.27	101.13	246.98	295.11
10	115.96	162.60	210.06	307.65
11	124.30	152.06	248.30	347.52
12	107.52	116.77	264.78	346.99
13	139.37	136.85	261.06	334.20
14	154.16	146.60	256.12	345.89

15	148.57	133.21	239.63	340.27
16	158.45	126.11	266.31	331.57
17	271.93	255.46	266.67	326.21
18	346.21	369.59	277.86	340.16
19	341.10	347.64	260.38	316.65
20	356.15	353.79	258.63	311.55
21	349.09	350.33	275.90	316.20
22	343.70	339.58	278.93	288.19
23	353.36	347.01	267.16	289.12
24	344.31	338.50	255.79	318.63
25	352.47	323.59	255.38	299.82
26	311.36	343.16	266.42	282.16
27	291.72	314.02	254.02	310.00
28	303.47	316.17	249.04	282.48
29	302.02	304.58	250.22	276.33
30	293.28	290.63	246.92	290.72
31	296.82	293.01	244.89	262.88
32	286.62	276.30	246.70	280.05
33	289.16	279.67	243.29	253.26
34	273.89	260.75	234.44	254.15
35	271.69	242.38	226.70	240.98
36	227.31	253.58	233.84	230.20
37	218.19	232.08	228.71	213.49
38	205.63	201.81	216.12	192.15
39	192.17	182.05	225.11	172.74
40	184.20	176.43	219.97	156.37
41	136.98	115.64	205.41	147.18
42	192.17	137.97	200.13	54.12
Total	**9,559.32**	**9,500.75**	**10,015.67**	**11,848.95**

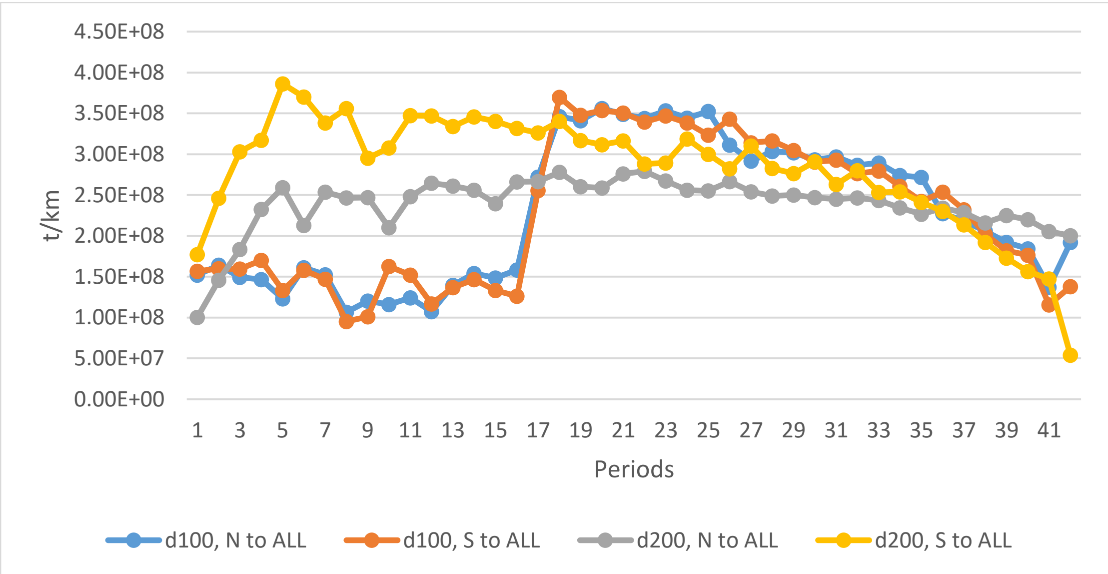

Figure 6-11 Truck productivity per haulage distance, t/km

6.4. Estimation of the dumping cost

The process of cost estimation modelling is a common aspect associated within the mining industry, allowing easy estimation of project costs through mathematical algorithms and equations.

The importance of cost modelling to the mining industry is to provide a realistic detailed result for budgets and financial planning in order to obtain approval for new designs or operational plans. Therefore the cost modelling of this research will estimation and comparisons of the optimal waste dumping in scenarios and dumping directions of the open pit mine. The cost modelling process involves inputting mining cost rates.

The discounted cash flow analysis is more sophisticated than other methods because it reflects the time value of money. Judgment is required in setting the appropriate discount rate to properly reflect risk, inflation and the cost of capital to the company making the investment. Operational costing can be easily applied to support discounted cash flow investment analysis.

NPC = Net Present Cost; a stream of cash flows over a specified period discounted to its present value at an appropriate discount rate:

$$NPC = \sum_{t=0}^{n} \frac{C_t}{(1+r)^t} \tag{6.1}$$

where:

t = time of the cash flow

n = total time of project

r = discount rate

C_t = Cash flow for a specific period

The NPC calculation of waste dumping in Erdenet open pit mine is shown on Table 6.6 using a 10% discount rate. NPC per ton, or discounted cost per ton (DCPT), is shown at the bottom of the table.

Discounted cash flow methods are most useful when evaluating investment alternatives, for example, when assessing competitive bids to supply equipment or when evaluating different fleet combinations or machine sizes to do the same job.

Table 6.6. Summary of discounted cash flow analysis in waste dumping

Scenarios		d100		d200	
Directions	-	**N to All**	**S to All**	**N to All**	**S to All**
Avarage truck numbers		34.33	34.18	35.25	41.38
Truck price, mil.$		1.50	1.50	1.50	1.50
Capital Cost, mil.$		90.00	90.00	90.00	90.00
Inflation Rate	0.03				
Discount Rate	0.10				
Gross Hours		5,699.82	5,699.82	5,700.18	5,699.82
Gross Hours (Total)		195,687.83	194,845.90	200,940.37	235,853.29
Maintenance					
Running Repairs, hrs		100.00	100.00	100.00	100.00
Planned Components Gross, hrs		200.00	200.00	200.00	200.00
Unplanned Maintenance, hrs		50.00	50.00	50.00	50.00
Planned PM's, hrs		150.00	150.00	150.00	150.00
Maintenance Subtotal, hrs		500.00	500.00	500.00	500.00
Scheduled Operating Delays					
Shift Change, hrs		365.00	365.00	365.00	365.00
Annually Breakup, hrs		336.00	336.00	336.00	336.00
Other, hrs		50.00	50.00	50.00	50.00
Operating Delays subtotal, hrs		751.00	751.00	751.00	751.00
Uptime, hrs		194,436.83	193,594.90	199,689.37	234,602.29
Use of Uptime (utilization)		0.90	0.90	0.90	0.90
Operating Hours, ave.hrs		174,993.15	174,235.41	179,720.43	211,142.06
Production/ Hour, ave.hrs		187.58	189.85	167.84	192.40
Total Production, mil.t.		1,252.62	1,252.51	1,263.98	1,223.53
Operating costs					
Operator, mil.$	20$ per o.hrs	150.49	149.84	154.56	181.58
Fuel, mil.$	100$ per o.hrs	752.47	749.21	772.80	907.91
Tires, mil.$	50$ per o.hrs	376.24	374.61	386.40	453.96
Operating (total), mil.$		***1,279.20***	***1,273.66***	***1,313.76***	***1,543.45***
Maintenance costs		-	-	-	-
Total SMU, tous.hrs		7,634.71	7,587.34	7,875.48	9,290.25
Running, PM, mil.$	9$ per Cum	1,392.53	1,401.70	1,520.70	1,974.76
Maintenance (total), mil.$		***1,392.53***	***1,401.70***	***1,520.70***	***1,974.76***
Total O&M Costs, mil.$		***2,671.73***	***2,675.36***	***2,834.45***	***3,518.20***
Cash Flow, mil.$		2,763.23	2,766.86	2,925.95	3,609.70
NPC, mil.$		**386.74**	**392.24**	**455.34**	**620.42**
Total Production, mil.t.		**1,252.62**	**1,252.51**	**1,263.98**	**1,223.53**
DCPT, $		**0.31**	**0.31**	**0.36**	**0.51**

The results show that the 'S to All' direction of d200 scenario is covers the maximum Net present cost (NPC) and the both directions of d100 scenario are getting minimum NPC and DCPT values. And the result differences between the d200, 'S to All' and d100, 'N to All' is totally 233.7 mil.$ of NPC (846.4 mil.$ of undiscounted cash flows) and 0.2$ of DCPT.

Summary

The research project aimed to optimize the fleet efficiency and utilization through a detailed haulage analysis, and to identify any potential cost savings available within the dumping operation of Erdenet open pit mine.

The haulage and dumping aspect of peint pit mining operations is one of the largest cost components of the mining cost constituing approximaterly 50-60% of mining operational costs. The overall aim of correct dump design is to plan a series of wste disposal stages that will effectively mininmize the vertical and horizontal distances (buffer zone) between the pit and potential waste dump site that has been discussed at Chapter 5. Two haulage distance i.e. 100 m and 200 m buffer length are considered.

The detailed material movement, haulage analysis and cost modelling was generated for 2 different dumping design and 2 different dumping directions. The haulage distance analysis, the average haulage distance in both directions of d100 scenario are relativity low but the graph shows rough results from the beginning of operation. The avarage haulage distance in both directions of d200 scenario are slightly high but the relativity smooth in whole mine-life. The study shows that the estimation of dumping cost analyses ‘S to All’ direction of d200 scenario is covers the maximum Net present cost (NPC) and the both directions of d100 scenario are getting minimum NPC and DCPT values. And the result differences between the d200, ‘S to All’ and d100, ‘N to All’ is totally 233.7 mil.$ of NPC (846.4 mil.$ of undiscounted cash flows) and 0.2$ of DCPT.

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CHAPTER VII. CONCLUSION

From the results discussed in previous chapters, the following conclusions were obtained:

7.1. Introduction

A relative deeper and large scale open pit mines developed in last 10 years and which are mainly operated in copper ore and hard coal mines in Mongolia. The depth of open pit mines was mainly less than 150m before 2000's. However, the depth and scale of open pit mines has been increasing with the mining capacity, mineral products and high economic growth in major projects. The most of deposits were explored and the feasibility studies were developed before 1990s. The main problems occurred in large-scale open pit mines in Mongolia which established before the 1990s, are optimization of open pit mine design, locations of waste dump and other surface structures due to feasibility studies and other studies developed at different politics and economic situations, the estimations of price increases of mineral products and future growth of resources are not proper to current economic and technical conditions. Furthermore, geotechnical and other base investigations and data are insufficient in many deposits it is the main problems for mining planning such as open pit optimizing, slope stability studies and appropriate large waste dump design etc. at most large scale open pit mines. In order to develop an appropriate open pit mine and waste dump design, these investigations have been conducted. In addition, the geotechnical and economic data has to be collected and optimizing to stable and economical design of open pit and waste dumping area are urgent issues for further large-scale open pit mine development were discussed. From this background, constructing a guideline for optimization of waste dumping considering the stability and economic situations is paramount.

7.2. Site description: Northwest section of Erdenetyn-Ovoo Cu-Mo deposit

Mining industry in Mongolia plays an important role in national economic growth. Especially, the Erdenet Cu-Mo deposit, which is the biggest porphyry copper-molybdenum deposit in eastern Asia exists and the majority of the state budget was created as the main pillar of the country economy. The total area of the Erdenetyn-Ovoo Cu-Mo deposit mine project is 5,500 km^2. On the basis of exploration surrounding the mine and feasibility study, ore reserves in the Erdenet Central deposit and the Erdenet Southeast (Oyut) deposit were calculated to be 1,250,000 tons (0.43 % Cu, 0.018 % Mo) and 41,890,000 tons (0.40 % Cu, 0.007 % Mo) respectively. The open pit mine currently covers an area of 2.5x1.5 km^2. The geological explorations in the deposit have been studied for the purpose of estimating and increasing the reserves and there are very insufficient studies for geotechnical research. Due to the insufficient of geotechnical investigation and researches, the stability angle, the dimensions of the design of the open pit mine and the underlying dimensions of the open pit mine are justified. Based on these real problems, the study has been subsequently conducted focus on development of an innovative method to optimize appropriate design of open pit and dumping area from geotechnical, geochemical and economic’s point of view.

Based on the current world price of mineral product and conditions of economic situations, it is necessary to optimize the Erdenet open pit mine boundaries, to explore the possibility of mining operation below 905 m and optimize the location and design of dumping area and surface structures in relation to the large open pit mine boundaries. Having taken insufficient geotechnical and operational investigation and economic data analyses, the planning and stability problems of open pit mine and waste dump area at Erdenet mine are considered from mining activity. Reviewing current planning, management and developing some corrective actions are required in order to design and optimize the appropriate open pit and waste dumping area.

7.3. Determination of ultimate pit design

Determination of the nested pit shells, pushbacks and ultimate pit limit is very important for a mining industry. The optimization should be conducted by considering geological model, and rock mass characterization and space factors to optimize the design of mining and dumping. The optimization pit design has been conducted by means of the Geovia Surpac software which has ability to generate a 3D geological and deposit block model, the Rocscience Dips software is utilized for kinematic analyses, and the Geovia Whittle software is used to establish the final pit limit in terms of the maximum Net Present Value (NPV) and associated pushbacks to produce a best case mining scenario were used. From the results, Pit shell-34 with Revenue Factor=100% covers the maximum net present value (NPV). And the result differences between the Pit shell-34 (RF=1.0) and Pit shell-84 (RF=2.0) are 79 mil. $ of NPV (2294 mil.$ undiscounted cash flows) and 2155.8 mil.t. of waste rock. From the sensitive analysis, resource in open pit mine is the most sensitive to metal prices. When the metal price drops to 30%, while the sulfide ore decreases to 935 million tonnes and increases by 30% to 497 million tonnes. Increasing the overall slope angle of the open pit by 4°, amount of the waste rock decreased very low as 3.3%. However, decreasing the overall slope angle of the open pit by 4°, amount of the waste rock increased quite high as 21.5%. The current concept of the Erdenet Mining Corporation has a total of 950 mil.t of ore at the open pit mine depth of 905 m. The results of pit optimization analysis show the possibility of open pit mine depth considering stability condition reach to the elevation of 780m which allows 125m more depth and to allow more than 550 mil.t of ore reserve to be exploited by the current Concept of Erdenet mining. Determining the location of the waste dumping and surface infrastructure and constructions based on the established open pit boundaries is quite risky. Open pit mining boundary is quite dynamic and is constantly changing from the beginning of the mine life to the end. The size, location and final shape of open pit should be optimized based on prospective production prices and open pit revenue factors are important in planning the location of waste dumps, stock piles; processing plant, access roads and other surface constructions, facilities and infrastructures.

7.4. Formation mechanism of benches on stability of large scale dumping

In open pit mine, providing a proper bench configuration thus satisfy stability criteria is crucial to mine's successful operation. The improper bench configuration result in stability issues which may affect safety and production of the mine. Considering to it, geological overview, dumping operation, waste particle distribution, and stable problems were investigated at the Erdenet open pit mine in order to design a stable bench configuration. In attempt to achieve the goal, a series of the experiments was conducted in the laboratory to simulate the formation process of single bench, multiple benches, and the efficiency of dumping operation's design. Moreover, the relationship between safety factor of dumping area and bench height, bench angle, bulk factor of waste rock, and truck transport were simulated by using numerical simulation. From the results, two methods are proposed to increase the stability of dumping areas. Firstly, the loose earth and all vegetation need to be removed to make the floor strong seam. Secondly, floor surface of dumping area becomes rough by blasting, which can prevent the floor to be slide surface. Design of the dumping operation must consider the total efficiency of ground leveling operation work and forming dumping area work. Height of bench can be as high as possible, up to the allowed safety values of workers and equipment working. Angle of bench is not important to dumping operation. Bulk factor of waste rock should be as small as possible to improve dumping operation stability. The activity of transport truck in dumping area has a beneficial effect on stability of dumping area.

7.5. Buffer zone optimization

Creating waste dump near to the pit is one of the solusions when the waste rock contain low grade of valuable minerals that planned to be extracted in future, as adopted by Erdenet open pit mine. Waste dump alongside the pit gives advantage in regards to waste hauling cost. However, from geotechnical point of view, constructing a waste dump alongside the pit should be planned well thus satisfy the stability criteria by adopting buffer zone, the distance between crest of pit wall and toe of waste dump slope. This chapter also discussed about the influence of cohesion and friction angle on pit wall stability. According to simulation result, it is found that buffer zone can reduce gravity loading on the pit wall thus able to reduce the shear stress along the wall; SRF increases with increasing buffer length. Furthermore when the buffer length is more than 200 m, it has also found that influence of waste dump on stability of pit wall is very small. The equation that suggested to be used to predict SRF for case of without buffer zone condition and with buffer zone condition is given $SRF = 1.65e^{-0.0005t_{pw}}$and $\mathrm{SRF} = 1.9e^{0.001(l_{\mathrm{bz}}-t_{\mathrm{pw}}-2t_{\mathrm{wd}})}$respectively. It has also been found that when the pit wall height is increased, the tensile stress around toe of pit wall is increased due to increase in self-load of the wall. The ratio $t_{\mathrm{pw}}/t_{\mathrm{wd}}$ is 4 to satisfy stability criteria (in case of SRF = 1.2). A same phenomenon has also been found in case of increase in pit wall angle. In case of increase in pit wall angle from 45^{o} to 60^{o}, the qualitity of SRF is reduced up to 0.25 point in average. The study also shows that the stability is changed when the quality of friction angle and cohesion is changed. However, a changing of friction angle will give more impact than that of cohesion. Based on the study result, a proper configuration can be designed for any conditions of rock quality particularly friction angle and cohesion. It has been found that when the quality of friction angle and cohesion is lower up to 0.75 times of the original quality, the SRF can reduce up to 0.179 and 0.099 in average, respectively. In other hand, the SRF can improve to 0.185 and 0.12 in average when the quality of friction angle and cohesion, respectively, is higher up to 1.25 times of the original quality.

7.6. Optimization of dumping area

The haulage and dumping aspect of peint pit mining operations is one of the largest cost components of the mining cost constituing approximaterly 50-60% of mining operational costs. The overall aim of correct dump design is to plan a series of wste disposal stages that will effectively mininmize the vertical and horizontal distances (buffer zone) between the pit and potential waste dump site that has been discussed at Chapter 5. Two haulage distance i.e. 100 m and 200 m buffer length are considered.

The detailed material movement, haulage analysis and cost modelling was generated for 2 different dumping design and 2 different dumping directions. The haulage distance analysis, the average haulage distance in both directions of d100 scenario are relativity low but the graph shows rough results from the beginning of operation. The avarage haulage distance in both directions of d200 scenario are slightly high but the relativity smooth in whole mine-life. The study shows that the estimation of dumping cost analyses ‘S to All’ direction of d200 scenario is covers the maximum Net present cost (NPC) and the both directions of d100 scenario are getting minimum NPC and DCPT values. And the result differences between the d200, ‘S to All’ and d100, ‘N to All’ is totally 233.7 M$ of NPC (846.4 M$ of undiscounted cash flows) and 0.2$ of DCPT.