DOKUZ EYLÜL UNIVERSITY

GRADUATE SCHOOL OF NATURAL AND APPLIED SCIENCES

OPTIMIZATION IN METALIC UNDERGROUND MINE DESIGN

by

Victor Abel MASSAWE

June, 2019

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OPTIMIZATION IN METALIC UNDERGROUND MINE DESIGN

A Thesis Submitted to the
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M.Sc THESIS EXAMINATION RESULT FORM

We have read the thesis entitled **"OPTIMIZATION IN METALIC UNDERGROUND MINE DESIGN"** completed by **VICTOR ABEL MASSAWE** under supervision of **PROF.DR. AHMET HAKAN ONUR** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Master of Science.

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Lastly, but not in the order of importance, I would like to specify the conference paper adapted from this thesis:

- Victor A. and Ahmet H. O, 2019. Delimitation of development costs towards stope boundary layout optimization, the paper has been presented and published following the 26th International Mining Congress and Exhibition of Turkey on 16-19th April 2019 held at Antalya.

Victor Abel MASSAWE

OPTIMIZATION IN METALIC UNDERGROUND MINE DESIGN

ABSTRACT

Optimization efforts to increase the value of metallic minerals extracted by underground mining methods have been mostly implemented for the past 20 years using both rigorous and heuristic approaches. Yet, the global optimal solution has not been achieved and consequently, this thesis approaches optimization from mine development perspective considering that the orebody reserve is accessed by a decline or ramp. In the algorithm which works for North-South and East-West direction, orebody is divided into small blocks, which contain geological and economic information. Blocks are then examined to determine the position of high mineralization, which depicts the position of a ramp. While applying ramp and drives development cost, drilling and material transportation costs, each block is accessed and aggregated to form stopes that maximize profit per tonne of ore constrained by cutoff grade. Moreover, stopes have a variable dimension with allowable length, width and height to cover the entire parts of the reserve at a given position constrained by permissible dilution. Finally, the algorithm is used in two real metallic ore posits with the help of visualization and quantitative analysis software such as Surpac™ and Excel. The approach produces practical results and with significant confidence, overcomes some of the challenges raised from past literature.

Keywords: Optimization, development cost, ramp/raise/decline, drive/level, sublevel, stope, stope boundary layout

YERALTI METALİK MADEN TASARIMI OPTİMİZASYONU

ÖZ

Yeraltı madencilik yöntemleri ile üretilen metalik madenlerin değerlerinin artırılmasına yönelik olarak sezgisel ve kesin yaklaşımlar kullanılarak geliştirilen optimizasyon yaklaşımları, son 20 yıl içerisinde uygulanmaktadır. Ancak, evrensel olarak kabul edilmiş optimum çözüm başarılamamış olması nedeni ile bu yüksek lisans tezi, desandre veya rampa ile ulaşılan cevher rezervlerine maden hazırlıkları perspektifinden optimizasyon yaklaşımı sunmaktadır. Kuzey-güney ve doğu-batı yönlerinde çalışan algoritmada cevher kütlesi jeolojik ve ekonomik bilgiler içeren küçük bloklara ayrılmıştır. Bloklar daha sonra rampanın lokasyonunu belirleyecek yüksek mineral içeriklerine göre incelenmiştir. Sınır tenörle sınırlandırılmış her bir bloğa ulaşılırken elde edilen galeri ve rampa açma, delme ve cevher nakliye maliyetleri birleştirilerek, karı maksimize eden pano oluşturulması amacıyla kullanılmıştır. Bunun yanında, panolar cevher yatağının tamamında izin verilen seyrelmeyi sağlayacak istenilen boy, en ve yüksekliğe bağlı değişken boyutlarda oluşturulabilir. Son olarak, algoritma, iki gerçek metalik maden yatağında, Surpac ve Excel yazılımlarının görsel ve kantitatif analizleri yardımları ile kullanılmıştır. Yaklaşım, geçmiş literatürde ortaya çıkan sorunlardan bazılarına çözüm bulmuş pratik güvenli sonuçlar üretmiştir.

Anahtar Kelimeler: Optimizasyon, hazırlık maliyeti, rampa/kuyu/desandre, arakatlı kazı, pano sınır tasarımı

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CHAPTER ONE

INTRODUCTION

Development cost shares a great part in the economic evaluation of a stope. The stope located at 400m from the surface, for example, will never have the same profit as a stope located at 800m given that other criteria of grade and size remain the same. Excluding development cost in early stope layout planning might result in inaccurate production scheduling due to stope overrating. In 2009, John Chadwick, the editor of Journal of International Mining quoted Tim Horsley; Mining Manager at Global Specialist Consultancy in Coffey Mining, that if you specifically focus on one part of the value chain there is a risk of transferring costs or reducing revenue elsewhere. There is a great emphasis on the insertion of every aspect during optimization including these development costs. Development costs can be categorized as primary and secondary. Primary development involves the establishment of ramp (decline), adit and/ or shaft to access the orebody while secondary development involves the activities to access a stope in an orebody by establishing levels, sublevels, raises, crosscuts, etc. Secondary development costs will be used in this thesis to establish the layout of stopes.

1.1 Problem Statement

Optimization efforts to increase the value of metallic minerals extracted by underground mining methods have been mostly implemented for the past 20 years using both rigorous and heuristic approaches. Yet, the global optimal solution has not been achieved and consequently, further studies are mandatory. The optimization process in the mining industry can be categorized into three groups, namely the optimization of development activities, stope boundary layout, and material scheduling and transportation. As stated in the Introduction part, if you specifically focus on one part of the value chain there is a risk of transferring costs or reducing revenue elsewhere. There is a great emphasis on the insertion of every aspect during optimization including development costs. As a contribution to the efforts that have been shown by other researchers, this thesis objects to approach the optimization of stope boundary layout by delimitating development costs from the top of an orebody and applying constraints to avoid stope overlapping and ensure appropriate geotechnical characteristics for both ore and waste materials.

Underground orebody reserve is accessed by a decline (ramp or adit), shaft or the combination of both. The selection of the best method depends on the depth of orebody from the surface, safety, economic consideration and time scheduled for production to kick up. During production planning, all of these options are considered and a choice that is significantly safe and maximizes NPV is regarded as the best option. According to Leandro et al. (2017) analysis on options of production and access ways in underground mines based on some case studies of Brazilian, South African, Australian and Turkish mines amongst others, concluded that the use of decline is usually considered favourable for mines with low production rate and shallow depth (up to 800 m). However, some mines, especially in Australia, use decline to depths greater than 1,000 m and with relatively high rates of production. They concluded that in mines near the surface, transportation by decline is recommended due to low investment requirement, quick return of investment, and it generally presents the largest NPV (net present value). However, Leandro agrees that it is difficult to establish a general rule for defining the main access because of each mine been formed with self-peculiarities.

Despite having the variety of access methods to an orebody from the surface, there should be secondary development activities such as establishing levels and sublevels, declines, raises, etc. These activities mainly depend on the type of mining method and orebody orientation. The secondary development starts from the main development and prioritizes the stope with high economic value. This thesis uses the secondary development costs supposing the development activities proceed from the top of orebody toward the stope with a high economic value per tonne of ore. This kind of development favours mostly the mines that use decline as access way from the surface. Unlike decline, when a shaft is used, secondary development can start from any point along the shaft to the most profitable stope. This scenario has not been covered in this thesis although the researcher believes that the primary access has an influence on the final stope layout. Therefore, to be precise, this thesis guarantees the solution of stope boundary layout as the orebody is accessed from the top to the preferable stope. When the word development or development cost appear in this report will precisely mean secondary development unless otherwise been used with another adjective such as tertiary or primary development. The same applies to stope layout where the name is hereby used to refer stope boundary layout.

1.2 Objectives

Objectives of this thesis are as follows

- Review specific development activities appealing to underground mining methods.
- Review the existing optimization approaches to stope layout and identify the major shortcomings directly either from their works or by using the analysis made by other researchers.
- Establish constraints that can be applied to a real metal deposit in the ground to develop a stope layout.
- Develop a heuristic based algorithm that incorporates development costs to achieve an optimal stope boundary.
- Suggest the better algorithmic approach that includes major development costs and uses profit/tonne of ore to create a stope layout.
- Implement the suggested algorithm to two real orebodies, observe and discuss the displayed characteristics of the results.
- Create an algorithm to extract the coordinates of created stope layout and visualize the results in an appropriate program in tree dimension.

1.3 Scope of the Thesis

The thesis is limited to an orebody which is accessed from any location at the top while satisfying the physical constraints of the mining environment. The main access that may allow the secondary development to start from any other location than the top of an orebody is out of this project context. Additionally, the production schedule; an important aspect of underground optimization is covered in neither the study nor the algorithm. The algorithm uses dimensional constraints and economic values such as metal price, mining and mineral processing costs to create a block model. Later, after studying the block model in different sections, the variable stope dimension and development costs are applied along with other constraints to create a stope boundary layout that ensures the geomechanical and geotechnical properties of the ore and waste rocks.

1.4 Methodology

The thesis approaches optimization from mine development perspective considering that the orebody reserve is accessed by a decline or ramp or another method that access the metal deposit from any position at the uppermost part. A mixed integer heusteric algorithm is created to solve an optimization challenge by creating the solution of a stope boundary layout. The process starts from a mining software (Surpac™) by developing a block model from an orebody by means of interpolating the composite borehole data using geostatistics methods. The blocks are thereafter imported to the algorithm written by Fortran (ft.95) in the form of centroid coordinates and grade values. In the algorithm which works for North-South and East-West direction, the regular size of blocks dimension is determined along with the contained grade information. This information is presented to Fortran in matrix form where the raws and columns represent the blocks along the x-axis (east-west) and y-axis (north-south) respectively. The propagation of blocks along the height (elevation) is presented in the form of a separate matrix. Since the elevation decreases as we go down the ground while the algorithm starts to mention the coordinates of blocks from minimum towards the maximum, the block model is presented as an inverted form in Fortran. The uppermost part of the orebody is found at the bottom of the matrix. Blocks are then examined to determine the position of high mineralization, which depicts the position of a ramp. While applying ramp and drives development cost, drilling and material transportation costs, each block is accessed and aggregated to form stopes that maximize profit per tonne of ore constrained by cutoff grade. Moreover, stopes have a variable dimension with allowable length, width and height to cover the entire parts of the reserve at a given position constrained by permissible dilution. Finally, the algorithm is used in two real metallic ore posits with the help of visualization and quantitative analysis software such as Surpac™ and Excel. The observations are discussed followed by conclusion and recommendations.

1.5 Significance of the Study

The research presents the following contributions:

i) Show the true definition of global optimization of the underground mining operation by incorporating the secondary development costs to a stope boundary layout.
ii) Provide a way forward to use development costs to further optimization studies in the underground mining industry.
iii) Develop an algorithm that can produce non-overlapping stopes within a short time and suggest the tips to minimize computation time in case of a big deposit.
iv) Expose the challenges facing optimization efforts in underground mining and suggestions toward an integral optimal solution.

1.6 The Thesis Outline

Chapter 1 of this thesis introduces the optimization concept and the research gap. The chapter explains the objective, the scope covered, methodology and lastly the significance of the study.

Chapter 2 introduces the mining concept and underground mining development methods. The chapter concludes by describing the collective properties to be considered in selecting a mining method.

Chapter 3 presents some efforts toward stope boundary layout optimization and technical hitches. It also explains the concept of the block model and stope access development.

Chapter 4 and 5 presents the stages the proposed algorithm has passed through to solve the optimization problem, the chapter provides a flow diagram indicating the inputs and optimization stages of the algorithm.

Chapter 6 discuss, interpret and visualizes the results from the algorithm. It shows the ability of the algorithm to produce result in a varying cutoff grade, dilution factor and wide range of stope dimension

Chapter 7 concludes the concept and observations made in this thesis. It also recommended further studies to improve performance and visualization of the proposed algorithm. The chapter proposed some research gaps in underground mining optimization.

CHAPTER TWO
MINING METHODS

Mining methods involve the collection of processes and characteristics of excavation that result in the extraction of ore from the ground to stockpile or to process plant. There are mainly underground and surface mining method. Selection and classification of any mining method depend on its proximity to maximum safety, positive cash flow and friendly to the environment. Technically, SME handbook (2011) stated that the key influences on selecting a mining method include the style of mineralization, stress, and characteristics of the rock mass. For some occasion, the ore is mined by surface method and letter been converted to the underground method. The answer to critical questions regarding the depth at which a transition should occur is dependent on many factors, including the relative scale of the surface mine and the underground mine, the lead-time required to develop the underground mine, and the optimum underground mining method. All of these scenarios are followed after detailed technical and economic study of the ore.

Under technical factors, the large orebody in size, which is in low depth and tabular shaped is probably amenable by open pit exploitation. However, there is a possibility to consider turn into underground methods in the future even if ore extends down. Contrarily, thin, inclined and deeply found orebody, with other economic consideration can be amenable by underground methods. Topography and accessibility also influence selection. Ore under the mountain provides a room to consider the underground method. Climatic conditions and environmental aspects, selectivity, dilution, and flexibility can also be a major effect to choose one mining method feasible than the other. The important part of the selection is to acknowledge that there is a large development process and requires time before mining by underground method than surface mining method. Enough attention is required to assure development costs in underground mining may not overweight less productive but possible and shorter development surface mining method (Komljenovic et al., 2015).

Economic reflections follow subsequently when knowledge of ore body is available. The reflection considers capital investment and operating costs, Cash flow, profitability and time value of money. Runge (1998) argued that failure to appreciate the purpose of the economic concept of cost might mean that efforts are misdirected. The capital investment decision is irrevocable hence links to the confidence level of geology and geotechnical parameters of the orebody. He also added that the mining method (or scheme) selected should have low operating cost to hedge against variation in market price. Money now is best than the same amount of money in the future hence investment dedicated to the mining method in terms of either capital intensive or delayed capital expenditure should focus on positive cash flow and early return of investment. This thesis is adapted to underground mining methods hence the techniques involved in underground mining and their economics (development costs) is described.

2.1 Underground Mining

Underground mining methods are selective mining procedures aiming to remove ore from the ground leaving waste materials on place. Atlas Copco MCT AB (2010) regards the method as always trying to take the meat out of the pie while leaving the crust. This method is eyed when there is an ore found deeper from the surface or there are concerns about the environment, which hinder the feasibility to mine by open pit (cast) method.

In order to access the ore, openings are created from the surface to the location of ore. Main development involves the construction of shaft, declines (ramp) and adit as an entrance from the surface to the underground. These excavation are permanent throughout the production period hence geotechnical considerations and wall support are evaluated and reinforced accordingly. Thereafter, levels (drives) parallel to the orebody are created to connect the entrance to different levels or zones of production. Finally, from each level; panels, crosscuts, or lateral developments are conducted across the orebody. After these three procedures, ore can be accessed from the surface although auxiliary development must be conducted to support access to sublevel, ventilation, materials transfer (ore passes) and dewatering purposes. There is also the excavation of places for workshops, magazines, and refuge chambers. Underground mining methods have common nomenclatures used to refer the different development and exploitation workings. Figure 2.1 presents most of the terminologies by which Hustrulid (2001) defines them as follows:

- *Dip*: The angle at which an ore deposit is inclined from the horizontal.
- *Drawpoint*: Excavation beneath the stoping area and specifically intends to aid the gravity loading of the ore to haulage equipment.
- *Drift:* Nearly horizontal underground opening.
- *Footwall:* The rock under the ore deposit, *Hanging wall* is located above the deposit
- *Level/Drives:* The horizontal underground workings connected to a shaft or decline, which forms the basis for excavation of the ore above or below.
- *Ore:* Any mineral deposit in the ground that can be economically mined at a profit.
- *Ore Pass:* Vertical to a nearly vertical underground opening used to transfer ore from one level to another.
- *Raise*: An underground opening; vertical or inclined, which is driven from one level to a higher level or to the surface
- *Ramp:* Inclined underground opening, which is driven downward to connects levels or production areas and allow the passage of motorized vehicles. Sometimes is regarded as *incline*
- *Shaft:* Vertical underground opening through which a mine is worked or accessed.
- *Stope:* An underground opening made by mining ore from surrounding rock.
- *Sublevel:* System of horizontal underground workings, which is used within stoping areas when needed for ore production.
- *Waste:* The rock materials of too low grade to be mined economically.

2.2 Underground Mining Methods

Underground mining methods for metallic ore can be categorized into three groups; natural supported, artificial supported and caving methods (unsupported). The effectiveness of applying these methods depends on both geometric, geomechanical and geologic properties of the deposit such as shape, size, grade, faults, and fractures. The following are the properties of the orebody that influence the selection of a mining method

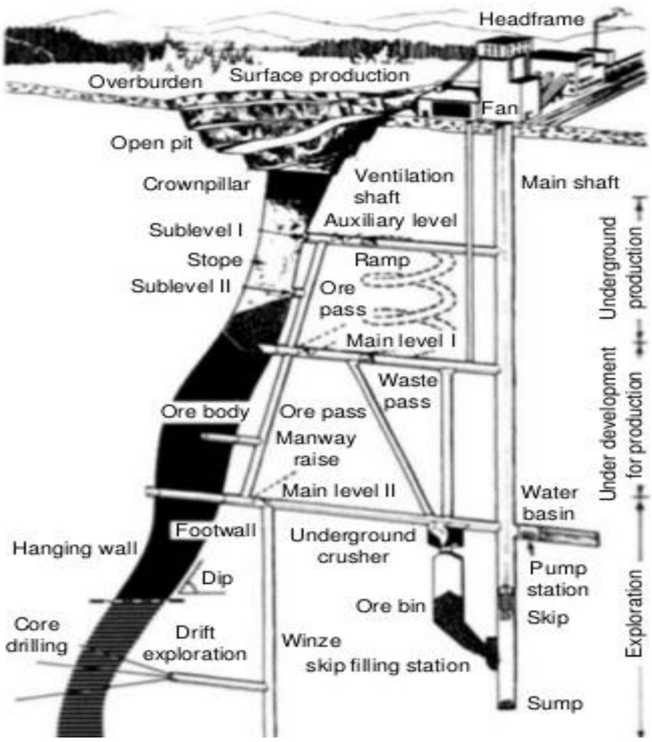


Figure 2.1 Common terminologies in metal mines (Tatiya, 2005)

2.2.1 Geometric Configuration of the Orebody

The property includes the size, orientation, and shape of the orebody. The shape of an ore body can be Isometric i.e. have an almost equal dimension in all direction or sheet which extend in two dimensions (constant thickness. Again Isometric shaped orebody can be stock and nests such as massive deposit or columnar (pipe deposit) which are vertical and thin e.g. vein and tabular deposits. Tatiya’s (2005) description of size or thickness of the deposit is as very thin deposit (< 0.7 m), thin deposit (0.7

m – 2.0 m) medium-thick deposit (2.0 m – 5.0 m), thick deposit (5.0 m – 20 m) and very thick deposit (> 20 m). Furthermore, he classifies the orientation of the orebody (dip) as flat dipping (0^0 to below 20^0), inclined dipping (20^0 to below 50^0) and steeply dipping (exceeding 50^0)

2.2.2 Physical and Mechanical Properties of the Ore and Surrounding Rock

These properties comprise rock mass strength and roof pressure. Rock mass strength is influenced by toughness, jointing, hardness, laminations, and presence of other types of rocks within the orebody (foreign inclusion). The strength of a rock material is determined by laboratory tests such as tensile, compression, shear and bending tests. In underground mining, rock strength of orebody and waste rock accounts for the stability of stope hence predetermine the necessary development method to ensure the safety of equipment and crew. Regarding exposure time and span, the stability of a rock can be classified from very unstable to very stable rock. For the case of roof pressure when excavation is made, factors such as the texture of roof and back rock, a dip of the deposit (which will form a stope), span, rate and duration of exposure are used as proof for suitability of a mining method.

2.2.3 Geological Properties of the Orebody and Surrounding Rock

Tatiya’s (2005) book argues one of the geological properties is the presence of geological disturbance and influence of the direction of cleats and partings including faults, joints, fissures, folds, dykes, etc. All of these factors affect the geomechanical stability of a rock during excavation by prompting water seepage and gas linkage to make mining tedious, lower mobility of the process, increasing cost and lower productivity. Another property is basing on a value of the product or ore grade and its distribution. This value depictures the selectivity and bulkiness of a mining method to be selected. The high-grade orebody is flexible to mine in any method depending or the other properties stated in 2.2.1 and 2.2.2. Ore value accounts to the economic return of mine and consequently, suggested mine method should avoid dilution and ore loss as possible.

Rock pressure increases with depth ($P = mgh$). Increase in pressure means the vulnerability of stope to collapse is high than in lower depth. Rock movement monitoring in mining methods that are artificial support is vital. This may trigger the use of costlier mining methods. In addition to these costs; ventilation, drainage, hosting and transportation costs also increases which affect the grade of ore and economic as well.

2.2.4 Degree of Mechanization and Output Required

To achieve high production, machines that have a large unit of production are suggested. The preferred mining method is the one having a suitable level of flexibility, excavation, and access size. One mine can have more than one mining method depending on geomechanical, geometric and geological consideration. Tatiya, (2005) asserted that any mining method should be planned and designed in a way that maximizes safety, mechanization and automation, gravity assistant, natural supports, flexibility, and adaptability while at the same time minimizing cost, dilution, time to achieve full production, retention time and pre-production development

2.3 Unsupported Underground Mining Methods

This category comprises methods that do not require systems of artificial supports to reinforce the rock rather; stopes are designed in such a way that the surrounding rock provides enough support. Both ore and waste rock is supposed to be strong to moderate strong to be able to support stope operations. To increase ore recovery and mine the ore that initially used as support, backfilling option is deployed. Mining methods in this category include room and pillar or stopes and pillar, shrinkage stoping and Sublevel stoping.

Room and pillar method is used to mine a thin, tabular and flat dipping orebody. The method is predominantly used in coal mines where the thickness of the coal seam is less than 5m. Series of a square to rectangular openings are driven into a coal deposit resulting in many box-like shaped structures. Additionally, Benching is required when the thickness of the coal seam is greater. The development method is called stopes and pillars when applied to metal mines. Depillaring, an attempt to recover pillars is possible once stopes (rooms in case of coal) have been backfilled or thoroughly supported. (Tatiya, 2005).

Shrinkage stoping is installed in tabular and thin deposit however, unlike room and pillar method, shrinkage stoping made possible to mine steeply dipping deposit. This method as described by Tatiya (2005), starts by developing an undercut throughout the width and length of the orebody. The height should be at least 10 meters from extraction or haulage level. Funnel-shaped like structures connects the undercut and extraction level to form the draw points. The undercut acts as a free face during blasting as well as a working platform during production drilling. In every successive blasting, only a fraction of material is recovered to provide a new working platform. The rest of ore is mucked when the whole stope height has been mined. Moreover, raises are developed at two ends of stope connecting the levels and provide access for both workforces, equipment, and ventilation.

Shrinkage stoping suffers some bottlenecks during stoping as just one third or 33% of ore can be recovered during production. Moreover, it possesses safety threats to the workers working under exposed unsupported roof. Open stoping method or sublevel stoping was developed to mine thicker orebody with same properties surrounded by competent rock. The orebody is vertically divided into several levels. Stopes of different dimension are found between the levels separated adjacently by rib pillar. Crown pillar above the stope protects the upper level from collapsing while the lower level forms series of operations ranging from production to ore handling. To provide more access, Stope is split into sublevels depending on ore competency and the reach of drilling equipment available. Due to the mode of production and the reach of drilling equipment available in the market, the method has been into classifications such as sublevel stoping with benching, blasthole stoping and big blasthole stoping (DTH). Furthermore, blasthole stoping can be undertaken longitudinally for thin orebodies and transversely for thick or wide orebodies. All of these classes share the common development stages hence regarded as open stoping mining method. The general method of development starts from establishing access to stopes from main haulage levels by driving crosscuts or drives. Service raise connects drives while haulage infrastructures are located at the lower level. A series of slot raise depending on the size of the orebody are also established to serve as a free face during production drilling. Backfilling is considered when there is an option to recover the crown or rib pillars.

2.4 Supported Underground Mining Methods

When an orebody is weak or not strong enough and surrounded by incompetent rock, the option left is to use artificial supports. Reinforcement materials include options such as backfilling, timbers, steels, mesh, bolts, and concrete. These materials are left in place throughout the mine life to avoid subsidence and caving of rocks hence jeopardize the production.

Stull stoping involves the use of timber to support a stope. Stopes of variable size and shape, which consist of steeply dipping thin orebody within a weak rock are favoured for this method. According to Tatiya, when the thickness of ore exceeds 4 m the method becomes impractical due to the scarcity of timber plus the complications involving its installation. It is recommended to use both stulls and fill if a deposit is thicker than four meters. Moreover, the ore should be strong to be able to hold stulls in place. The development processes begin with the establishment of access drives and/ or crosscuts followed by raises to join the levels. This follows the instituting of extraction infrastructures. Mining of stopes advances along strike, down dip or up dip under the favourable conditions.

Cut and fill involves cutting an orebody into slices and fill the void created by suitable materials. This means mining activities progress upwards from the level or sill pillar in a series of drilling, blasting, loading, and backfilling. The method is the best substitute for stull stoping as timber is getting expensive year after year. Furthermore, mining by cut and fill is more advantageous in terms of production rate, productivity, and safety. It can be applied to both thin and wide ore body in a variety of deposits.

Mining a stope which is susceptible to rock busts necessitate the use of cut and fill method. Filling materials include development waste rock, sand, mill tailings, high-density filling materials or crushed stone. Development activities start with driving a drive or crosscut to the stope from both main levels followed by raises at the proper interval for mine services such as ventilation and transportation of filling materials. After extraction level and infrastructures are set, further development proceeds in parallel with stoping operations depending on the equipment used.

Square set stoping consists of square set supports. The supports are made of the post, cap and grit as shown in Figure 2.2 whereby according to Tatiya, normally caps and grits are 1.8 m to 2.4 m in length whereas the posts are 2.4 m to 3 m high. The set made of timber or steel can be extended to the full length of a stope every after each blast. Square set stoping made possible to extract weak ore surrounded by a weak rock. A deposit is divided into levels in the interval of up to 30 m along with strike extension. Drives or crosscuts are used to access a stope from both levels connected by raises at both ends. Finally, extraction infrastructures such as extraction drive, chutes, undercutting and/ or draw points are developed depending on a feasible material handling system.

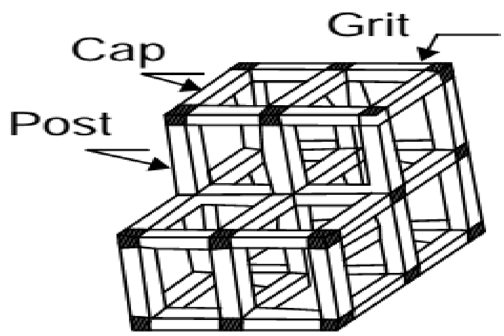


Figure 2.2 Assembly of square set modules (Tatiya, 2005)

2.5 Caving Methods

Longwall mining is the oldest method for extracting thin and flat regular deposit, particularly in coal mines. It is suitable for a weak ore with its face extends to more than a hundred meters and surrounded by weak cavable rock. The method has very limited application in metal mining however, according to Tatiya, it has been applied to some of the metal mines in a way quite similar to stope and pillar. The method starts by dividing a reserve into panels. They should be as long and wide as possible to guarantee high productivity and recovery. The main headings, which join the main access from the surface ensure the exposure of the panels. From the panel, longwall face of more than 100 m, is developed depending on the degree of mechanization. The process begins by driving gate roads at either side of the panel. The gate road consists of two drifts running along the length of the longwall and connected by crossroads in interval leaving giant pillars as supports. The gate roads are connected at the rear or end of the panel. The connecting drive serves as an initial stage or free face for mining operations to start. The same development procedures are applied to other panels. For the safety of access roads within the deposit during mining, some ore may be left as pillars (Drake, 2012).

Sublevel caving is applied in regular steeply inclined strong orebody within a weak rock. The development principles match with that of sublevel stoping but production starts from the upper stopes allowing the weak rock to cave into opened stope. The development can continue in lower stopes while mining is taking place in the uppermost stopes. Development starts by establishing a drive in waste rock, at least 10 m away from the hanging wall contact to the orebody (Tatiya, 2005). The drive advances from the top sublevel throughout the length of the stope and after each predetermined interval, crosscuts are established across the orebody to its hanging wall contact. Crosscuts are connected at the hanging wall contact by hanging wall drive. The hanging wall drive is expanded by drilling and blasting parallel holes to completely widen the drive into a slot. The slot is used as a free face to initiate stoping activities. Similar development procedures are performed in the rest of the sublevels. Finally, ore handling infrastructures are developed connecting each sublevel and levels by ore pass. For wide orebody, a series of crosscuts connected to a sublevel drive is developed to form a transverse sublevel caving system.

Block Caving mining technique is applied to the large, massive, weak and low-grade type of ore body. The surrounding waste rock is also weak and consequently, the shape of orebody should be preferably regular to minimize ore dilution during caving. The method consumes vastly plentiful development time before ore recovery begins, however, ensures continue ore production once development activities are completed. Unlike drilling and blasting, the weak and naturally fractured ore depends on rock pressure and gravity to cave into the extraction infrastructures. Before developing the deposit, the accurate study is required to confirm the ability of ore to break naturally and precise location of the loading system. The development activities can be subdivided into three groups; namely main access, extraction and undercut development. The main access is driven to the bottom of orebody where loading system is located. Since the rock is weak, the main access is reinforced thoroughly to avoid rock failure. Thereafter, a series of parallel extraction drives are developed from side to side. Between the drive system, funnel-shaped finger raises are drilled to link the extraction level and the undercut. Undercut drives can be developed at the same time as the extraction drive. Undercuts drives are constructed enough to provide enough drilling space for the ore above. Holes enough to initiate ore caving are drilled and blasted from the undercut drives. Finally, the finger rises are opened and ore starts to flow to extraction level. Before mucking the ore, walls around extraction drive are reinforced to avoid unforeseen accidents. Loading infrastructures are developed to favour the selected material handling system.

2.6 Collective Analysis of Mining Methods

Despite the irregularity of orebody shape, a mining engineer tries to split the orebody into blocks that fit within the ore boundary as possible during stope designing. The blocks are aggregated to form rectangular stopes that maximize profit in accordance with predetermined mining method. The strength of waste rock and ore combined with environment concern provide the early clue to use either support, unsupported, caving method or their combination depending on the nature of mineralization. Likewise, the range of stope size is constrained to match the technical parameters of the selected method. Development costs are related to the expenses incurred to expose a stope while the mining costs are those experienced during the development of a specific mining method within the orebody such us sublevel development, crosscuts, reinforcement, backfilling, raises and/ or slot. The other costs such as drilling, blasting, and transportation are related to the size, grade, tonnage, and location of a stope. Table 2.1 shows comparative features of an underground mining method based on operational parameters. Square set stoping is costly and less productive per man-shift compared to the rest of the mining methods. Sublevel and cut and fill stoping are popular mining methods in modern time regardless of the development process been slow and expensive. Caving methods are suitable for bulk production but compromised by its tendency of causing subsidence on the surface, dilution, and applicability (the ability of ore to cave).

2.7 Stope Design and Layout

The stope is a regularized ore block that forms a part of the deposit to be mined. Stope dimension is not constant. It can differ with respect to a mining method and even from mine to mine. While deciding stope dimensions; strength of orebody and its enclosing walls, rate of stoping, degree of mechanization, depth, size and shape of the orebody play an important role (Tatiya, 2005). The height of a stope is fixed within the level interval or equivalent to the level interval in case of a deposit with uniform mineralization. The width depends on the size of a deposit whereas in thin orebody, the width is equivalent to the width of the orebody and for a case of the thick orebody, rock properties and the accurate interval that will minimize ore loss and dilution will decide the width to be used. Likewise, as stated in the case of stope width, the ability of ore and waste rock to keep the stope upright without failure will determine the length of the stope. To acquire model information, a block model constrained by orebody is formed. The blocks are later aggregated to form stopes that conform with the prospected mining method. The model is analysed from various horizons in order to establish an accurate stope length and width as shown in Figure 2.3. The advantage of using the horizons is that it allows the user to decide the number of blocks to form a stope, which will reduce the ore loss or dilution.

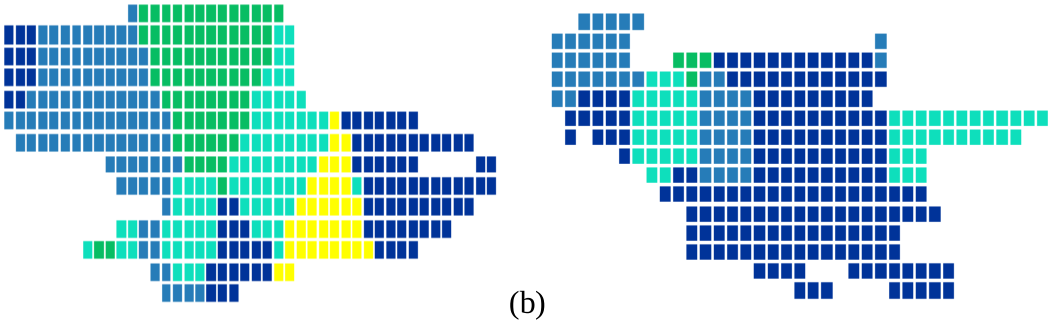


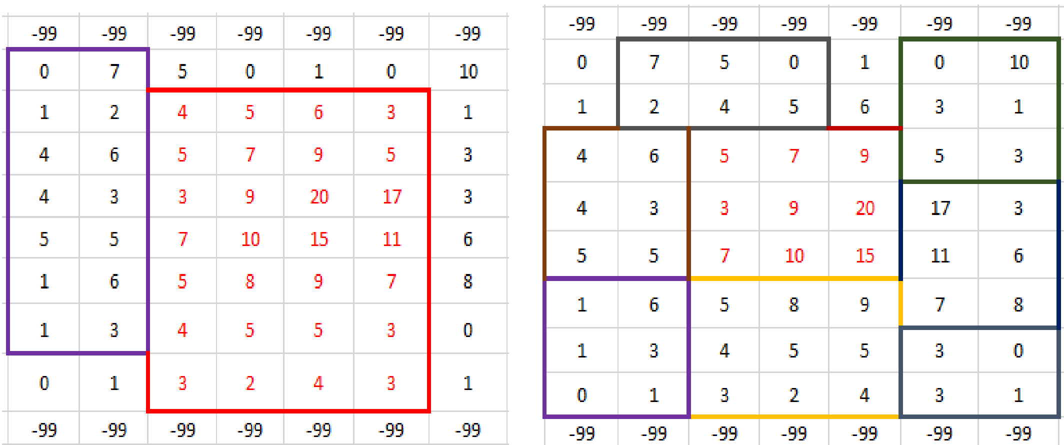
Figure 2.3 Sections developed at different elevation to ascertain the continuity of the ore deposit

Table 2.1 Comparison of stoping methods – based on operational parameters (Tatiya, 2005)

Stoping method	Stope and pillar	Shrinkage stoping	Sublevel Stoping	Cut and fill stoping	Square set stoping	Longwall mining	Sublevel caving	Block Caving
Relative cost %	30	50	40	60	100	20	50	20
Productivity in tons/man-shift	30- 50	5 - 10	15 - 30	10 - 20	1-3	75 - 180	20 - 40	15 - 40
Recovery %	75	75 - 80	75	90-100	≤ 100	70 -90	90 - 125	90-125
Dilution %	10- 20	< 10	≤ 20	5 -10	low	10 -20	10 -35	10 - 20
Development required	Little	Moderate	More, slow and expensive	little to moderate	low	Low	highest	Slow, extensive and costly
Applicability and special features	Limited metal mining method	Small scale, ore tie up to 60%	very popular modern method	Method of all times & varying conditions	Costliest method in worst ground conditions	Popular coal mining method continuous mining possible	Early production but high development	bulk mining method effective draw control critical for success

It is suggested that there should not be a wide range of stope dimension as it increases the optimization time and the chance of both ore loss and dilution. Alford (2000) cited that in a block model, there are two options for a block; either been selected or not selected. The number of possible alternatives for a combination of n blocks within the model is equal to 2^n and only one of which is the optimal solution. He added that, even for a small-sized model, can take millions of time unit. To contend his point, Alford provided an example of a situation where a 2D section of a stope is made of 10 x 10 blocks. In this particular case, the program should evaluate 2100 alternatives to generate one stope. For 3D, time increases even further. The current evolution of technology makes possible to find more powerful and efficient processors. However, still the wider the range of stope dimension, the more computation time an algorithm takes. Figure 2.4 demonstrates how stope range may contribute to ore loss or dilution during stope formation. On the figure, the numbers show block value with the red colour representing the most profitable stope.

Assume a block to be 10 m x 10 m in size, Figure 2.4 (a) represents a situation where the suggested stope dimension is between 20 m x 60 m to 40 m x 70 m. Supposing the numbers show the block value, the wide stope range will result in the formation of two stopes with other blocks at the edges being lost as we aim to maximize revenue per tonne. This huge loss would be recovered if the stope dimension were adjusted as shown in Figure 2.4 (b). In (b), the range of stope dimension is from 20 m x 20 m to 30 m x 30 m. with this narrow range, most part of the deposit is developed with only a small block loss of 8 unit value. Indeed, large stope sizes are preferred over small one in order to lower the cost of development but there should be a reasonable range of the stope dimension to avoid losses.



(a) (b)

Figure 2.4 Effect of (a) wide range and (b) narrow range of stope dimension

CHAPTER THREE
UNDERGROUND STOPE LAYOUT OPTIMIZATION EFFORTS

Surface planning and design methods such as Lerchs and Grossmann's (1965) algorithm have been improved while underground techniques been in the struggle for optimal solutions with little attention (Ataee-pour (2000) & Sandanayake (2015)). Since the year 1965, twelve algorithms have been developed; mostly intend to improve the Lerchs approach. Indeed, Carlson et. al. (1966), Korobov and Lerchs will be remembered for their efforts to generate historical methods such as moving cone and dynamic programming. The methods were the backbone of the evolution of modern and simpler methods such as network flow and graph theory solutions. Underground mine optimization has attracted more attention for the last two decades. The objective of stope optimization is to obtain the optimal ore geometry so that after the latter stages of scheduling and development, a reserve returns as maximum profit as possible. It may be considered as a starting point of the optimization process towards development and production scheduling.

Table 3.1 Some of the efforts of underground stope boundary layout optimization (Erdogan et al., 2017)

RIGOROUS ALGORITHMS	HEURISTIC ALGORITHMS
• Dynamic Programing Solution (1977)	• Octree Division (1995)
• Downstream Geotechnical Approach (1984)	• Floating Stope Algorithm (1995)
• Branch and Bound Algorithm (1995, 1999)	• Multiple Pass Floating Stope Process (2001)
	• Maximum Value Neighbourhood Algorithm (2000)
	• Generation of Stope Shapes for a Nominated Range of Cutoff Grade (2009)
	• Sens and Topal Heuristic Approach (2009)
	• Bai et al. Graph Theory and Network Flow Method (2013)
	• Sandanayake and Topal Heuristic Approach (2015)
	• Nelis et al (2016) Vertical Convexity Constraints

3.1 Rigorous Algorithms

3.1.1 Dynamic Programming Solution for Stope Optimization (1977)

This mathematical solution was proposed by Riddle (1977) specifically for defining stope boundaries in block caving mining method. Dynamic programming concept in mining was firstly introduced by Lerchs and Grossmann (1965) in their effort to find an optimal pit limit. Johnson & Sharp (1971) added more to Lerchs' efforts before Riddle came to find out that the algorithm is a good start for stope optimization in the field of underground mining. Therefore, this approach is one of the first attempts to optimize the boundary of underground mines in two dimensions (Erdogan et al., 2017). Riddle's algorithm requires a factor called "percentage draw control", which specifies the interval between drawpoint during mining by block caving. At first, the orebody is considered as a big block without footwall. The algorithm uses the draw control as a constraint to calculate the profit from cumulative net values of blocks. Thereafter, it defines a footwall by removing one column in operational regions and re-calculate the profit in the same way as without footwall. The process continues by eliminating one column until all columns have been eliminated (no more footwalls to be examined). Finally, the algorithm chooses the footwall with maximum profit as the optimal layout. This method attains optimal result for block caving in two dimensions only but does not guarantee optimality in 3D mining situation ((Ataee-pour, 2000; Sandanayake, 2014)

3.1.2 Downstream Geostatistical Approach (1984)

This method was developed by Deraisme et al. in 1984. It determines the outline of orebody mineable to cut and fill and sublevel stoping and based on 2D sections of a block model. Mathematical equations are used to define stope geometry constraints, ore-waste images, and for the transformation of the images (Ataee-pour, 2000). It accounts metal grades in the optimization process of stope geometry while excluding related economic profit, which in turn limits optimality in both 2D and 3D (Erdogan et al., 2017).

3.1.3 Branch and Bound Algorithms (1995, 1999)

Ovanic and Young (1995, 1999) developed a Branch and Bound Algorithm to determine the optimal mining layout. This mixed integer technique is also regarded as type two special ordered sets (SOS2) where two piecewise linear cumulative functions are used to optimize the start location and the finish location in each row of blocks. These functions calculate the cumulative economic values of blocks in each row of the model in order to create a monotonic series. Moreover, two special variables which sum up to one are used in cooperation with the stope geometry constraints to select an area in a row of blocks that will result in a stope of maximum value. The method is efficiently stated with an example by Ataee-pour (2000). This method, however, is developed specifically to optimize a panel or stope in one dimension use constraints such as stope dimension and economic values to achieve optimality, however, it optimizes the stope boundary in one direction. It is not clear if the method can be adjusted to fit in multiple dimension. Moreover, the algorithm supports the situation where only one stope can be formed along the direction of optimization.

3.2 Heuristic Algorithms

3.2.1 Stope Layout Optimization by Octree Division (1989)

Cheimanoff et al (1989) introduced this algorithm where geometrical stopes are evaluated in terms of economic values. Data of boreholes, geostatistics, and geological shapes are used to create a geometrical model that later is transformed into an economic model by introducing economic constraints. Thereafter, minable volumes are determined by establishing two models. Object manipulator and shape generator. Object manipulator collects mineralized veins into regular blocks while by using economic constraints, shape generator subdivides these blocks in octree until the smallest subdivision matches the smallest mining unit. As can be seen, the algorithm discards separately the blocks that do not match the minimum mining unit. It is possible that the accumulation of waste from allowable dimensions contain unnecessary waste in the final mine layout thus lower the possibility of the algorithm to guarantee the optimality (Sandanayake, Topal, Mohammad, 2015)

3.2.2 Floating Stope Algorithm (1995)

Alfrod (1995) has developed the algorithm whereby its approach is similar to the moving cone method when it comes to open pit optimization. The Mineable Reserves Optimizer (MRO) is a tool based on the Floating Stope algorithm in DATAMINE (2014) software. At first, two grades are specified, namely cutoff grade and head grade. The head grade is the grade to be targeted when a specific block above cutoff grade is floated to create a stope. Also at this stage, the minimum stope dimension is defined in 3D. This algorithm creates inner and outer envelopes. The inner envelope is constructed by collectively adding blocks that are above cutoff grade. The outer envelope is constructed on top of the inner envelope whereby each block above cutoff grade is floated to create a possible stope with required dimension, accumulated economic value and head grade. The final stope design should be closer to the inner envelope and fall within the outer envelope. (Alford, 1995). This algorithm generates overlapping stopes with shared mining blocks. Therefore, manual adjustments are required to exclude these blocks hence against optimization principles. (Sandanayake, 2015). In addition to this, in the study described Erdogan et al., (2017), the algorithm can only be used to guide a mine design; it cannot be directly used for final stope layout. It provides a poor approximation for narrow dipping deposits and in case of waste pillar consideration. Furthermore, it does not consider variable stope sizes, the inclusion of mining levels and barrier pillars. Another interesting finding from Ataee-pour (2000) which has also been observed in this thesis is the contradicting relation between Alford's floating stope algorithm and moving cone approach for optimizing the pit limit in surface mining. While in open pit, two cones may overlap in a waste block and add up to the feasibility of a resultant shape being mined by profit, in Alford's algorithm, the overlapping an unfeasible situation.

3.2.3 Multiple Pass Floating Stope Process (MPFSP) (2011)

Cawrse (2001) extends and improve the functionality of Datamine software that is based on Floating stope algorithm. MPFSP does not mitigate the shortcomings of Floating Stope algorithm such as overlapping stopes, absence of geotechnical constraints and the likes but in contrary focused on the informative evaluation of the block model and identifying possible areas for the development of stopes. (Sandanayake, 2015)

3.2.4 Maximum Value Neighbourhood Algorithm (2000)

Ataee-Pour (2000) has developed a Maximum Value Neighbourhood (MVN) algorithm to optimize stope boundaries. This algorithm requires pre-defined 3D stope size, a number of blocks required to form minable stope dimension, metal grades and all constraints necessary to create economic block model. The optimization process uses block economic values to form stopes with maximum net values without violating constraints of minimum stope size. The user specifies the starting location (block) for optimization to start before the algorithm starts to search for the feasible neighbourhood blocks that satisfy the minimum stope size. The neighbourhood with maximum economic value is preferred and flagged to include in the final stope layout. To avoid overlapping of stopes, every time when blocks that have been previously flagged are encountered, the algorithm stops the block for further proceedings. In addition, to minimize the inclusion of waste in the final stope layout, the algorithm prevents the origin block for stope optimization to be zero. The process continues until all blocks with positive economic values have been flagged (Ataee-pour, 2000). According to Erdogan et al., (2017), as MVN optimization process largely depends on the starting location, changing the starting location can change the set of stope layouts generated from the same ore body. Furthermore, a fixed economical block model is used in this algorithm and it does not consider stope-costing factors based on the size of the stope, or neighbourhood. It only takes into account the costs of individual blocks.

3.2.5 Generation of Stope Shapes for a Nominated Range of Cutoff grade (2009)

Alford and Hall (2009) developed a method for automated stope design that generates sets of stope shapes (nested stopes) for a nominated range of cut-off grades. This method uses a similar concept like that in pit optimization where nested pits are generated (Alford, 2009). Likewise, the nested stopes are used as input in the hill of values methodology for the evaluation of the optimum cut-off grade (Hall & Stewart, 2004). The method is able to define the best set of extraction levels and stope heights. However, block economic values and hence, stope economic values are not used in the optimization process. The guarantee for this method to provide an optimal solution is in question.

3.2.6 Sens and Topal Heuristic Approach (2009)

Sens and Topal (2009) established an algorithm based on a heuristic approach. At first, the algorithm converts irregular blocks to regular blocks in terms of length, width and height. Later, regularized blocks are aggregated to form stopes guided by the given economic values of blocks and range of stope dimension. The algorithm locates stope boundaries using different stope sizes and selection strategies in three dimensions. Furthermore, it selects the stopes in descending order of economic values while removing the overlapping stopes. Most of the profitable stopes might be ignored during the selection process simply because they have been deleted (overlapped) in the previous selections. Therefore, the final stope layout employed by this algorithm fails to generate the most profitable stope combination (Sandanayake, 2015)

3.2.7 Graph Theory and Network Flow Method (2013)

This method is based on graph theory and cylindrical coordinate systems around a vertical raise. Network flow algorithm is specifically used for stope layout optimization in sublevel stoping mining approach. Geotechnical constraints on the hanging wall and footwall slopes are translated as precedence relations between blocks in the cylindrical coordinate system. The program evaluates the maximum distance of a block from the raise and the horizontal width required to bring the farthest block to the raise. Then the algorithm optimizes the stope profit as a function of location and height of the raise. (Bai, et al. 2013)

3.2.8 Sandanayake and Topal Heuristic Approach (2014)

Sandanayake et al. (2014) developed a new 3D heuristic algorithm that incorporates stope size variation and challenges optimization problem. The algorithm uses blocks economic values as input for optimization. The blocks values are aggregated to form possible stopes (β_j). Stopes with positive economic values (Υ_j) are then extracted from β_j, which will obviously be overlapping each other. The algorithm identifies these overlapping stopes and adjusts to create family sets of non-overlapping stopes (F_k). Moreover, a family of unique sets of non-overlapping stopes (F_u) subset of F_k are created. From F_u, Stopes economic values of each set are summed where the set that provides Maximum value is considered as optimal stope layout. Results from this algorithm generate 10.7% more profitable solution than the commercially available Maximum Value Neighbourhood (MVN) algorithm. The downside of this approach relies on unique combinations to be analysed. In order to find the optimal solution, the algorithm needs to evaluate all the possible unique combinations which require significant computational power in large scale applications. This limits the algorithm to find the optimal solutions in large scale data sets on time (Erdogan et al., 2017).

3.3 Block Model of an Orebody

Block model has been introduced to the mining industry along with the concept of using computer technology in mine planning, designing, and production scheduling. The approach has been successful in both surface and underground mining operations. An orebody is divided into small blocks that depict the position and properties of contained features of ore and waste rocks. The size of blocks is set to reflect the expected stope dimension and mining method. There are different types of block model used in mining history as mentioned by Alford (1995). These are 3D fixed block model, 3D variable block model, 2D irregular block model, and 3D irregular block model. Among all, 3D fixed block model, which was first made available by Kennecott Copper Cooperation in the 1960s is widely used. This thesis will use this method to develop an algorithm to solve the optimization problem. The regular 3D fixed block model is capable of dividing a rock into ore and waste blocks based on specified cutoff grade. The only downfall of this model is incapability of modelling to a situation that requires the separation of lithological units and geological faces. In a 3D model, regular blocks are named using their respect coordinates or indices in both x, y and z-direction. The famous method of using integers made possible to avoid long and continuous coordinates. Considering there are indices $I \times J \times K$ along X, Y and Z directions respectively, a block in any position can be located as $B_{i,j,k}$ where i, j and k are proper subsets of I, J and K respectively.

To recognize the nature of the data presented, a block model is characterized as a geological or economic model. The geological model includes rock properties of waste and ore such as material grade, specific gravity, and general rock type or formation. The geometrical method, distance weighting or geostatistical analysis such as kriging is used to interpolate borehole composite data to all blocks in the deposit. Once geological information is available, the blocks can be converted easily to the economic model. Economic block reflects the geological properties into revenues by integrating the market price of contained material and cost of mine development, extraction, haulage, and processing. In this thesis, the block economic value is calculated as follows;

$$BEV = \begin{cases} B_m Pad - v\left(C_m + C_{mp}\right) & \forall\, d \geq cutoff\ grade \\ -(B_v C_m) & \forall\, d < cutoff\ grade \end{cases} \tag{3.1}$$

Where,

BEV = Block economic value in $\$$

P = Metal price as indicated by the market in $\$$

a = Recovery in %

d = average metal content in a block per tonne

B_m = Mass of a block in tonne

B_v = Block volume in m^3

C_m = Cost of mining a block of ore in $\$/m^3$

C_{mp} = Cost of processing a block of ore in $\$/m^3$

Consider a mining situation in 2D with the average percentage grade of metal indicated in Figure 3. 1 where recovery is 90% and cutoff grade is 5%. Cost of milling and mining 1 m^3 of ore from its respective stope is forecasted to be \$40 and \$50. The block dimension is 10 m x 10 m x1 m and an average specific gravity of 3 t/m^3. Assume the metal price in the market is equivalent to \$ 3300 per tonne of ore

(a)

2	4	-1	-3
4	10	5	2
0	7	4	3
-1	9	6	5
-3	8	9	2
-7	8	6	-1
-1	0	-2	-2

(b)

-5000	-5000	-5000	-5000
-5000	80100	35550	-5000
-5000	53370	-5000	-5000
-5000	71190	44460	85500
-5000	62280	71190	-5000
-5000	62280	44460	-5000
-5000	-5000	-5000	-5000

Figure 3.1 (a) Geological block model and (b) Economic block model, values calculated from (a)

There are basic rules for establishing block values for optimization purposes that have been established by Whittle (1989). Alford's (2000) PhD thesis has mentioned these rules as follows;

i) The values have to be calculated as if the blocks are uncovered. The cost related to access of stopes must not be included in the block costs.

ii) The value should be calculated based on the assumption that the block will be mined. A block that contains more waste than ore i.e. below cutoff grade is not going to be primarily chosen for the optimal layout. However, the block might be chosen to satisfy stope constraints and the ore content of positive blocks is going to pay for the waste blocks.

iii) Alford recited Whittle, that when considering the cost of mining or processing a block, only those costs that would stop when mining from the stope stops. This includes fuel and wage payment costs. Since these costs would stop if there were no further extraction of blocks, they should be included when there is corresponding mining or processing of a block. The reason is that the addition of each extra block in a mine layout extends the life of mine and therefore an extra block should pay for the costs incurred in the respective extended period.

Generally, mining costs relate to the predetermined mining method. Every mining method has its expenses as mentioned in Table 2.1. The total value of blocks in a stope should reflect the possibility of maximum and positive cash flow during mining operations in a stope. This is why any block that has a grade lower than cutoff is treated as waste. Volume has been preferred over the weight of a block in calculating the costs due to the truth that in underground operation, the void created should be minimum as possible in order to assure the rock stability. The material handling and mineral processing types of equipment are also more constrained in volume than mass.

3.4 Stope Access Development

Stopes are accessed by driving levels, crosscuts and/ or raises from the area that is already developed. Initially, primary development from the surface to the orebody is made. Costs related to primary development can be deducted from the revenue generated by the stope layout.

i.e.

$$R_T = \sum_{s=1}^{s=n} R_{i,j,k} - C_p \tag{3.2}$$

Where;

R_T = Total revenue after reducing primary development cost in *\$*

$R_{i,j,k}$ = Individual block revenue in *\$*

C_p = Primary development cost in $\$$

s = number of stopes to be mined

Primary development does not affect the final stope layout and therefore it will not be included in the optimization equation. However, secondary development, which varies with the stope layout will be considered in stope generation and optimization.

3.5 Summary

Development cost shares a boundless share in developing an optimal layout of stopes. The costs can be categorized as primary and secondary costs. Primary costs are those incurred during the establishment of the main entrance to the orebody from the surface such as the development of shaft, decline or adit. Primary costs assumed to have common less impact on the optimal layout and they can be deducted from the revenue generated from the final stope layout. However, primary development can be used to define the point where secondary development initiates. Shaft related development is mobile and its secondary development can start at any point along the shaft to the stope that maximizes the net value. In the other side, access methods such as a ramp, secondary development start from a location on top of orebody to the stope that maximizes the net value. This thesis will use this approach which favours orebody that is closer to the surface.

Underground mining methods are divided into three groups; namely supported, unsupported and caving methods. Every method has its procedures, which depends on both characteristics of ore and waste rock, safety, economic and environment. An orebody is translated into a geological block model and later to the economic model. Many researchers have applied a different set of constraints to the block model as efforts to generate optimal stope layout which will maximize value. This thesis work joins the effort of these researchers by using costs related to secondary development to achieve a layout which will maximize the net value. In this thesis, the term development will refer to secondary development.

CHAPTER FOUR
DEVELOPMENT OF THE PROPOSED ALGORITHM

The algorithm will pass through five stages to generate the stope layout and visualize the results. Fortran 95 (f.95) will be used to perform stage 1 to 4 and thereafter, the combination of Ms excel and Surpac™ will be used interchangeably for stage 5. The sequence of activities involved in this chapter is as shown in Figure 4.1.

The thesis will cover two types of the algorithm with a different approach to development cost assignment. One of them will account only the cost of establishing the main level while ignoring the need for accounting sublevel development cost for stopes distanced a bit far from the main level. Another algorithm will include sublevel development when necessary. The process begins by creating a geological block model that covers the whole dimension of the orebody in three dimensions. The model is presented by matrix form in the Fortran, one block layer after another from the bottom part of orebody to the top. The waste blocks are distinguished from ore by assigning them a big negative number, which would never agree with the combination of others during stope formation. The economic block model is generated by introducing the price of metal and costs associated with mining and mineral processing. The cutoff grade is used as constraint whereby materials below cutoff grade are treated as a waste rock with a negative value tag. Finally, the algorithm applies the stope constraints and by using development costs, a stope with the maximum value per tonne of ore will be selected. The algorithm removes the stope by assigning a large negative number into the void created and rerun again to find the next stope. These results are then transferred to other software for visualization and analysis.

4.1 Block Generation and Regularization

In this study, Geovia Surpac™ generated blocks with fixed dimension. The blocks are constrained by orebody and using inverse distance squared, composite borehole data (grades) are interpolated to each block. The centroid of each block along with grades are then extracted to a file ready to be incorporated in Fortran 95. Upon opening and reading the file with block centroids in Fortran, the algorithm is set to read the extreme coordinates and find the block dimension in three dimensions. Using the maximum and minimum coordinates and the block dimension, a total number of columns, rows, and layers are established in form of integers *I, J* and *K.* In this way, the location of a block is named as $B_{i,j,k}$ in three dimensional space where $i= 1, 2, 3 \ldots I$, $j = 1, 2, 3 \ldots J$ and $k = 1, 2, 3, \ldots K$.

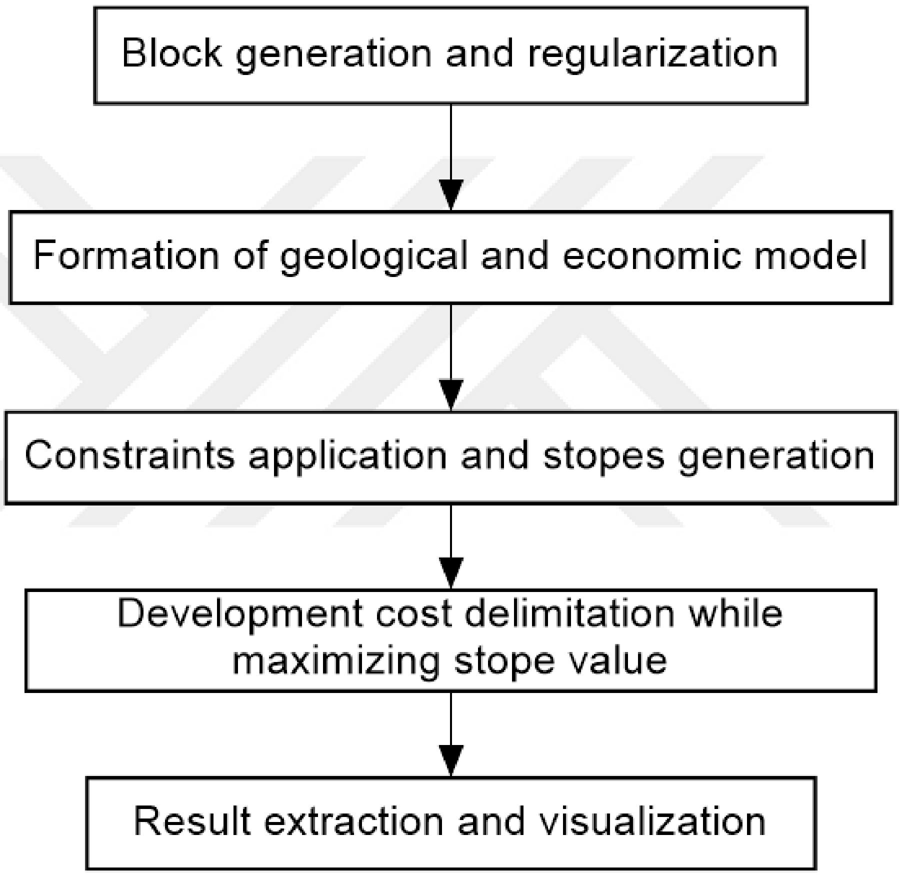


Figure 4.1 The stages performed by algorithm towards stope layout

Mathematically, Assume the maximum coordinates in *x*, *y* and *z* are *xmax*, *ymax* and *zmax* and minimum coordinates are *xmin, ymin* and *zmin* respectively. Also, suppose the block dimension in three dimensions is *xdim*, *ydim,* and *zdim* respectively. The total number of columns, rows, and layers will be as follows;

$$Number\ of\ rows\ (I) = \frac{xmax - xmin}{xdim} \tag{4.1}$$

$$Number\ of\ columns\ (J) = \frac{ymax - ymin}{ydim} \tag{4.1}$$

$$Number\ of\ layers\ (K) = \frac{zmax - zmin}{zdim} \tag{4.2}$$

The centroid of blocks can easily be relocated in x, y and z coordinates by using the following equations;

$$B_{i,j,k=(xmin+(xdim\ x\ i),\ \ ymin+(ydim\ x\ j),\ \ zmin+(zdim\ x\ k))} \tag{4.4}$$

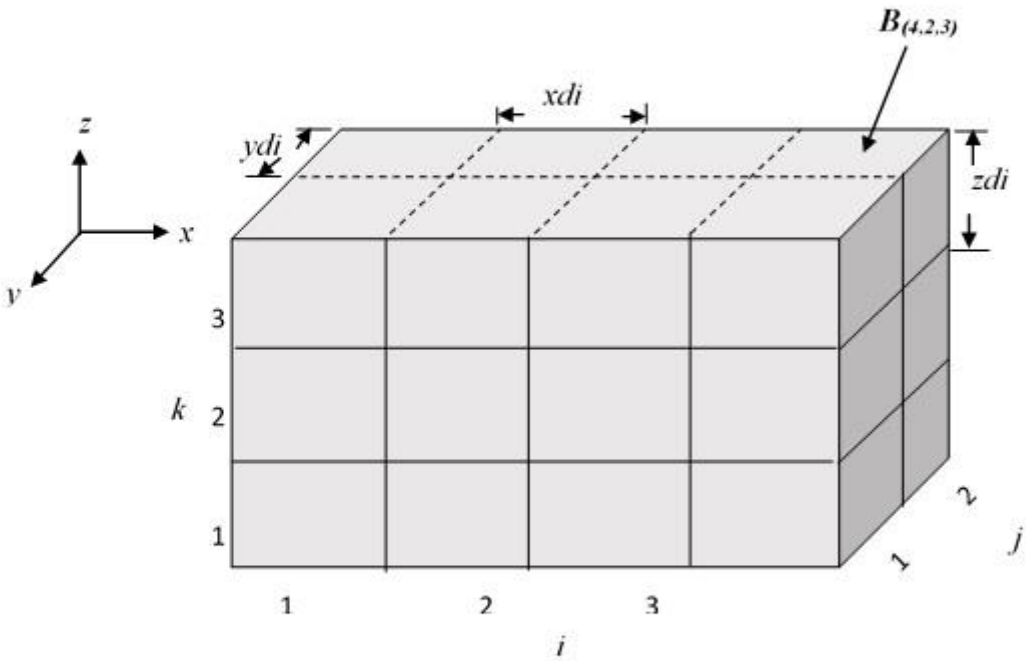


Figure 4.2 Naming a position of a block in the block model

4.2 Geological and Economic Block Model

Geological block model contains geologic information such as rock density and material grade. The geological data are interpolated from borehole composite data by using geostatistics estimation formulae such as kriging or inverse distance squared. Kriging is the optimal interpolation that uses the semivariogram model to create the best linear unbiased estimate at each location. Kriging assigns weights according to a data-driven weighting function rather than an arbitrary function, but it is still just an interpolation algorithm and will give very similar results to others in many cases (Isaaks & Srivastava, 1989). This thesis used two models as an example to discuss different aspects of the algorithm. Inverse distance squared is chosen arbitrarily as the interpolation method. The equation used is as follows;

$$Z_0 = \frac{\sum_{i=1}^{n}\left({}^{z_i}/_{d_i{}^m}\right)}{\sum_{i=1}^{n}\left({}^{1}/_{d_i{}^m}\right)} \tag{4.5}$$

Where,

Z_0 = Estimated grade

z_i = Borehole composite grade located at i

$d_i{}^m$ = The distance from i with a weight factor m

For inverse distance squared, the value of m is equal to 2.

The economic block model is created by using the block dimension, specific gravity, and cutoff grade from the geological model. The calculation for a block economic value is as shown by equation 3.3.1. Mass (B_m) and volume (B_v) of a block is calculated as follows;

$$B_m = xdim\ x\ ydim\ x\ zdim\ x\ g \tag{4.6}$$

$$B_v = xdim\ x\ ydim\ x\ zdim \tag{4.7}$$

Where,

g = metal density in t/m^3

Development costs are not assigned to the block model in this stage. Rather, they will be assigned during stope generation as a criterion of sorting the stopes and avoiding overlapping. However, the impact of these costs will be observed in a particular stope value per tonne.

4.3 Stope Generation

In this stage, stopes are generated randomly depending on the given constraints. The following are the list and description of symbols for parameters used to constrain the stope

- w_s x l_s x h_s = Stope dimension (*width x length x height*) in m^3
- (w_{smi} x l_{smi} x h_{smi}) and (w_{sma} x l_{sma} x h_{sma}) represents the minimum and maximum stope dimension respectively.
- $S_{iws\ ,jl\ ,khs}$ denotes the location of a block in a stope. The $S_{iwsmi\ ,jlsmi\ ,khsmi}$ and $S_{iwsma\ ,jlsma\ ,khsma}$, denote the position of a block at bottom right corner and upper left corner of the stope (see figure 8). The two blocks are used to name a stope
- R_i or $R_{iw,jl\ kh}$ in 3D denotes a particular block revenue in *$*
- R_s or $R_{iws\ ,jls\ ,khs}$ denotes a particular stope revenue in *$*
- T_r = Total revenue before development cost is encountered in *$*
- B_v= Volume of a block (m^3)
- B_m= Weight of a block *i* (*ton*)
- B_s represents the total weight of blocks that form a stope and b_s represents the total weight of blocks with positive revenues in a stope
- T_m = Total tonnage of all blocks with a positive value
- M_s = Total metal content in a stope (*ton*)
- ρ = binary number *(0 or 1)*
- *P* = Profit (*$*)
- P_{st} = Profit per tonne (*$/ton)*
- *rec* = Minimum percentage of blocks required for a stope to be established

A block in a stope is simply named according to its position in a block model. The name of a stope is stated according to a sequence of formation (maximum value per tonne). However, to be more specific, a stope is also named by using both bottom right and upper left blocks. Figure 4.3 elaborates this naming formula whereby the orange coloured blocks represent the lower and upper limit of a stope. Suppose the stope ranks

8^{th} compared to others in terms of metal value per tonnage. With this assumption, the stope can be named as 8^{th} stope or ($S_{17,1,8} - S_{19,3,10}$)

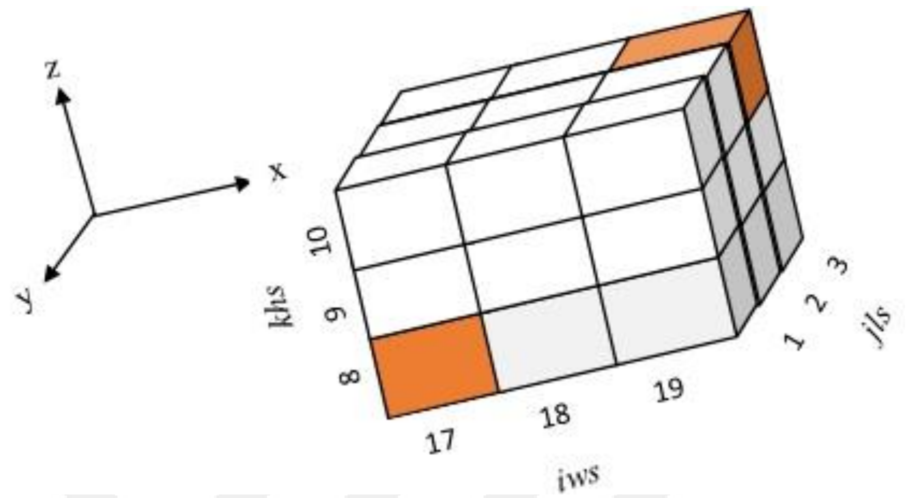


Figure 4.3 Naming a stope

The range of stope dimension is selected first and then the algorithm uses each block to combine with the others to form a stope that fulfils the predefined dimensional constraints. The weight of a stope B_s and respective revenue R_s is calculated. Again, the algorithm sums up the weight b_s of blocks with positive dollar value within the stope. Other parameters such as metal content in a stope M_s, Tonnage of all blocks with a positive value T_w and total revenue of all stopes T_r is calculated in this stage for the purpose of describing the ore loss and recovery after optimization. However, Stopes generated are unsorted, contains waste blocks and overlapping to each other. Every block in the block model played a role of stope formation. The only constrain that prevent a stope from formed is the size of a block model and dimensional constraints of a stope. For illustration, as shown in Figure 4.4, it requires 2 x 2 blocks for a stope to be formed, which makes impossible for the stope formation to start from any of blocks in the last column and raw.

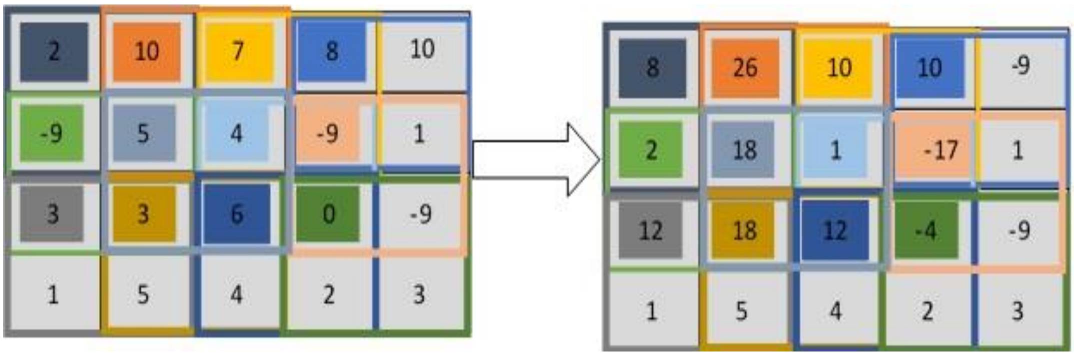


Figure 4.4 Stope formation by using block values and dimensional constraints

Some of the properties related to mining method, geotechnical and geomechanical properties are depicted by the definition of stope size and shape. For example, Block caving method requires a big and fairly regular stope with uniform ore grade distribution. For this algorithm to be able to generate stopes for such mining method, an orebody should be analyzed from different layers (sections) in order to define a logical range of stope dimension that will maximize the net value. Contrarily, Cut and fill or sublevel stoping provides the window for backfilling, flexibility and hence more recovery from comparatively small stopes. Generally, stope dimension should be logical as possible and the precise range should be considered during the optimization stage.

Number of the waste blocks (metal grade is less than cutoff) in a stope can be controlled in parallel with the efforts to minimize ore dilution. The algorithm allows a user to set the allowable percentage of waste blocks in a stope. The calculation is given as:

$$rec = \frac{b_s}{B_s}\, x\, 100 \tag{4.8}$$

This is an optional category although it has some advantages toward the orebody layout. One of the advantages is to assure the value per tonnage of a stope generated. The discussion about this optional factor is available on section 6.3

4.4 Delineating Development Cost and Optimization

The algorithm delineates secondary development cost from the top of orebody where primary development is supposedly ended. At the top of an orebody, the algorithm defines four alternative locations where the development is likely to start, mainly east, west, south, and east. Stopes are developed from the direction that minimizes waste development cost while maximizing profit/tonne. The algorithm works by defining logical constraints that determine the very east and west block with positive economic value in each layer. Every time a drive is required, two alternatives are checked of whether to direct the ore from stope towards east or west. The alternative that offers the maximum cumulative value per tonne of ore is chosen as the best location to start the development. The algorithm runs again for the south and north alternative. Finally, the best alternative from east-west calculations is compared with that of south-north to have the overall decision of the location to start the development activities. Figure 4.5 illustrates the different location of the ramp and possible stopes in 2D. In figure 4.5 (a) where a ramp is located at the east and drives driven towards west, visibly shows a low level of waste development compared to figure 4.5 (b) with a ramp located west and drives extending east. The same concept can be experienced in figure 4.5 (c) and (d) when a ramp is created in either north or south. All of these alternatives are considered when the algorithm is searching for the optimum stope layout. Moreover, when a stope is exposed, drilling and ore transportation cost is involved. All four costs are deducted from revenue of prospective stope

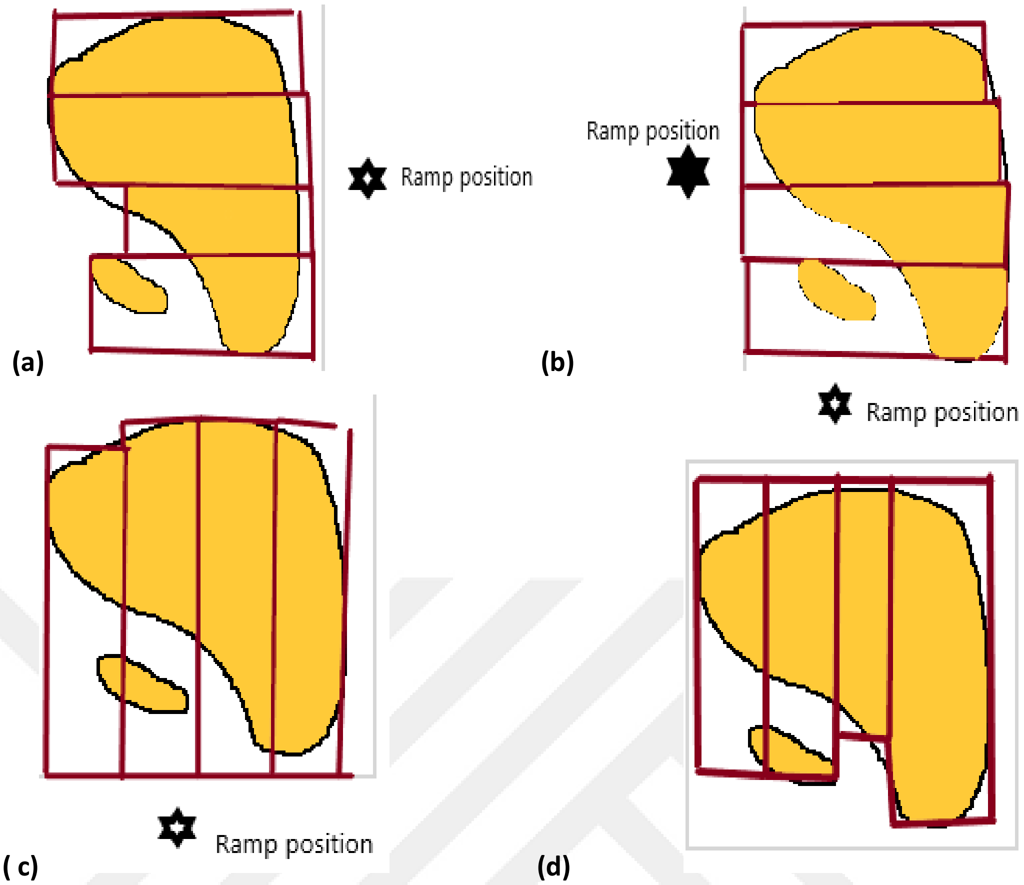


Figure 4.5 Sketch of stope layout with (a) ramp located east and drives extending west, (b) ramp located west and drives extending east, (c) ramp located south and drives extending north and (d) ramp located north and drives extending south

4.4.1 Development Cost Description

Development costs are secondarily expected expenses to access and remove ore blocks from a block model. Namely, ore drilling, transportation, ramp (raise) and drives development cost. The following symbols are used to define the costs and parameters involved in describing the secondary development.

- C_{dv} = Ramp development cost per metre (*$/m*)
- C_{dr} = Production drilling cost per tonne (*$/m*)
- C_{cr} = Stope access development cost (*$/m*)
- C_{tr} = Material transportation cost per tonne (*$/m*)
- *%g* = Haulage grade that is applicable to ramp development

d = Accounts the percentage of dilution per meter caused by blast-hole deviation

Level or sub-level access cost (R_c) is incurred when a decline (ramp) or raise is developed from the nearby-developed level. Ramp development cost is given as:

$$R_c = \frac{C_{dv}\ x\ kh_{j-s}\ x\ zdim}{\%g}\ \forall\ kh_{j-s} = \begin{cases} K - kh_s & (NP) \\ k - kh_s & (RNP) \\ 0.0 & (DPP) \end{cases} \qquad (4.9)$$

Where *NP* means no existing stope hence new ramp is required, *RNP*; develop ramp from the nearest stope, and *DDP*; Do not develop ramp

Drilling cost (D_c) relates to the number and length of holes to be drilled to access the block. The longer the hole, the higher the tendency of diversion and hence more ore dilution. This cost is calculated as follows:

$$Dc = kh_s\ x\ zdim\ x\ C_{dr}\ x\ B_s \qquad (4.10)$$

Stope access (Drive) development cost (Dr_c) involves the horizontal excavation that exposes and connects the stope to the main access. This cost is considered as follows:

$$Dr_c = jl_s\ x\ xdim\ x\ C_{cr} \qquad \forall\ E - W\ development \qquad (4.11)$$

$$Dr_c = iw_s\ x\ ydim\ x\ C_{cr} \qquad \forall\ N - S\ development \qquad (4.12)$$

Transportation Cost (T_c) includes the cost of hauling insitu blocks from a stope to the surface. It is calculated as:

$$T_c = kh_s\ x\ zdim\ x\ C_{tr}\ x\ B_s \qquad (4.13)$$

4.4.2 Optimization Process

$$Objective\ function:\ Max \sum_{i=1}^{n} \frac{P_s}{B_s} \quad \forall\ s_{min\leq} s \leq s_{max}$$

$$\boldsymbol{S}ubjected\ to: \quad R_s, Dr_c, D_c\ and\ T_c \tag{4.14}$$

4.4.3 Mathematical Equations of Physical Constraints

$$T_r = \sum_{i}^{n} \rho R_i \quad \forall\ \rho = \begin{cases} 1 & if \quad t_i \geq cog \\ 0 & elsewhere \end{cases} \tag{4.15}$$

$$T_m = \sum_{i}^{n} \rho\ B_m \qquad \forall\ \rho = \begin{cases} 1 & if \quad t_i \geq cog \\ 0 & elsewhere \end{cases} \tag{4.16}$$

$$\begin{cases} iw, jl, kh \subset i, j, k \\ iw_s, jl_s, kh_s \in iw, jl, kh \\ iw_{smi}, jl_{smi}, kh_{smi} \leq iw_s, jl_s, kh_s \leq iw_{sma}, jl_{sma}, kh_{sma} \end{cases} \tag{4.17}$$

$$R_s = \sum_{iw_s=iw}^{iw_{smi},iw_{sma}} \sum_{jl_s=jl}^{jl_{smi},jl_{sma}} \sum_{kh_s=kh}^{kh_{smi},kh_{sma}} R_i \tag{4.18}$$

$$B_s = \sum_{iw_s=iw}^{iw_{smi},iw_{sma}} \sum_{jl_s=jl}^{jl_{smi},jl_{sma}} \sum_{kh_s=kh}^{kh_{smi},kh_{sma}} B_m \tag{4.19}$$

$$b_s = \sum_{iw_s=iw}^{iw_{smi},iw_{sma}} \sum_{jl_s=jl}^{jl_{smi},jl_{sma}} \sum_{kh_s=kh}^{kh_{smi},kh_{sma}} B_m\ x\ (1-d)\ \forall\ R_i\ > 0.0 \tag{4.20}$$

$$M_s = \sum_{iw_s=iw}^{iw_{smi},iw_{sma}} \sum_{jl_s=jl}^{jl_{smi},jl_{sma}} \sum_{kh_s=kh}^{kh_{smi},kh_{sma}} B_{wei}\ x\ t_i \tag{4.21}$$

$$P_s = \sum_{iw_s=iw}^{iw_{smi},\ iw_{sma}} \sum_{jl_s=jl}^{jl_{smi},jl_{sma}} \sum_{kh_s=kh}^{kh_{smi},\ kh_{sma}} R_s - R_c - Dr_c - D_c - T_c \ \ \forall\ \frac{b_s}{B_s} \geq rec \tag{4.22}$$

The optimization process starts by selecting the stope that has a maximum value per tonne of ore without compromising other criteria of dimension and dilution control. The stope is removed by assigning a large negative number into the stope values, which will avoid overlapping during the selection of a next stope. Figure 4.7 shows an example of stope formation 2D. In the beginning, every stope is set to reflect the costs involved in development where obviously, the stopes close to the top of orebody or stopes near the decline (raise) would have least development costs. The example shows the value of +54 has been mined out of +60. Total of +6 blocks value is the ore loss caused by dimensional constraints of stope and orebody. The concept of using profit per tonne of ore during optimization is brought forward to account for the variable stope size.

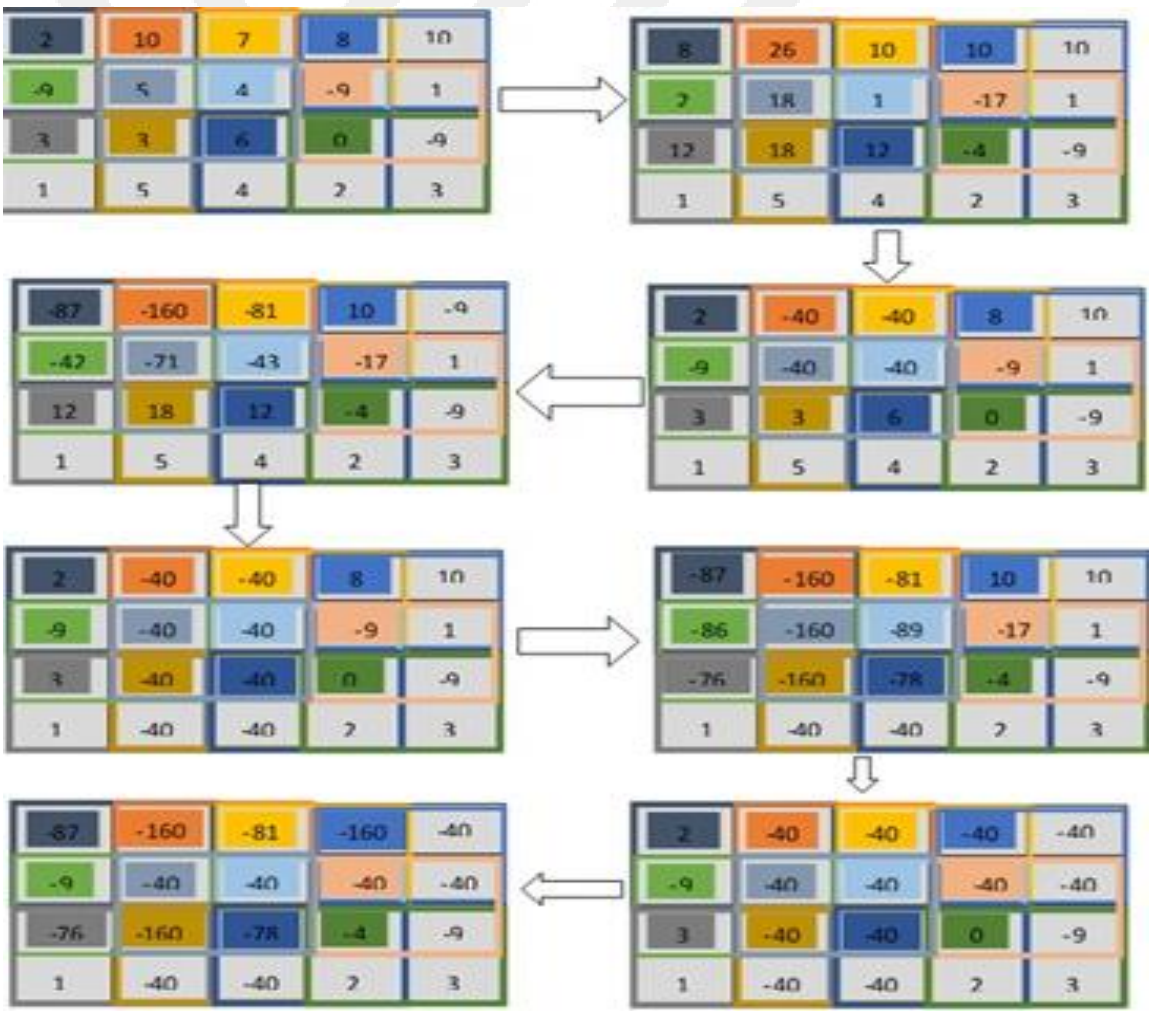


Figure 4.6 An example of stope formation and optimization by using block values in two dimensions

CHAPTER FIVE

EXTRACTION OF RESULTS AND FLOW DIAGRAM

Results obtained from the algorithm written in Fortran 95 (programing language) are imported to Microsoft Excel and Geovia Surpac™ for visualization and interpretation. These results include total value and tonnage of the ore within the stope layout, ore value per tonnage for each stope created, ore loss and recovery. Several examples of real mine data are used to interpret the capability of the algorithm to produce an accurate solution. The algorithm is developed (i) to limit the assignment of drive development cost whereby the cost is assigned to a stope within a specific distance from its neighbour. (ii) to allow a stope to be assigned a drive development cost regardless of the distance from its neighbour. Another small algorithm is set to create *x, y* and *z* coordinates of stopes from the indices *i, j* and *k*. The coordinates are then imported to Geovia Surpac™**.** Figure 5.1 below shows the exact coordinates extracted from the algorithm to plot a stope 1.

Stope	x	y	z
1	250221.8	4219771.0	937.8
1	250281.8	4219771.0	937.8
1	250281.8	4219771.0	1007.8
1	250221.8	4219771.0	1007.8
1	250221.8	4219771.0	937.8
1	250221.8	4219831.0	937.8
1	250281.8	4219831.0	937.8
1	250281.8	4219831.0	1007.8
1	250221.8	4219831.0	1007.8
1	250221.8	4219831.0	937.8

Figure 5.1 Sample coordinates extracted from the algorithm for the purpose of visualization

5.1 Stope Formation and optimization in Fortran

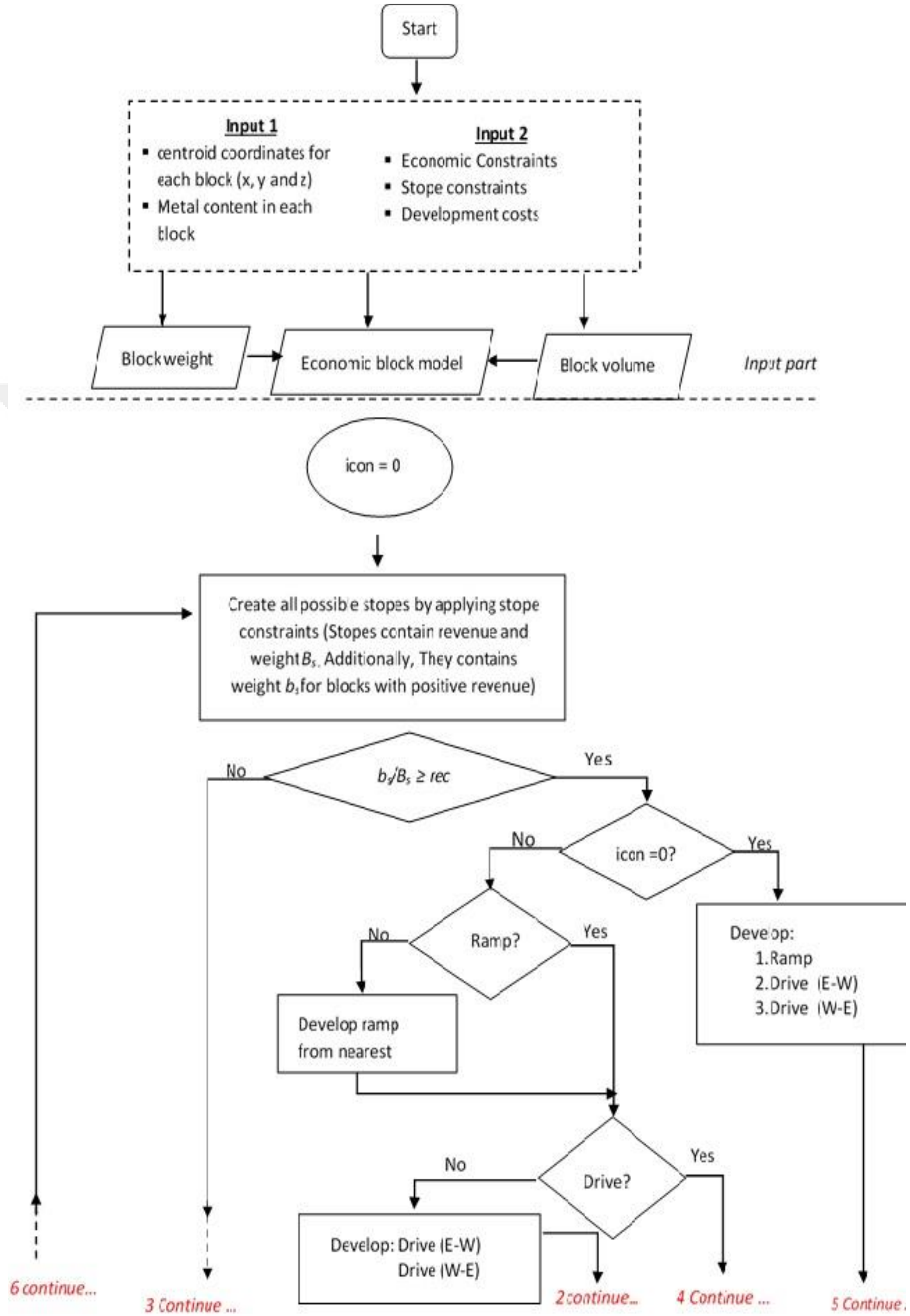


Figure 5.2 Stope formation and optimization in Fortran

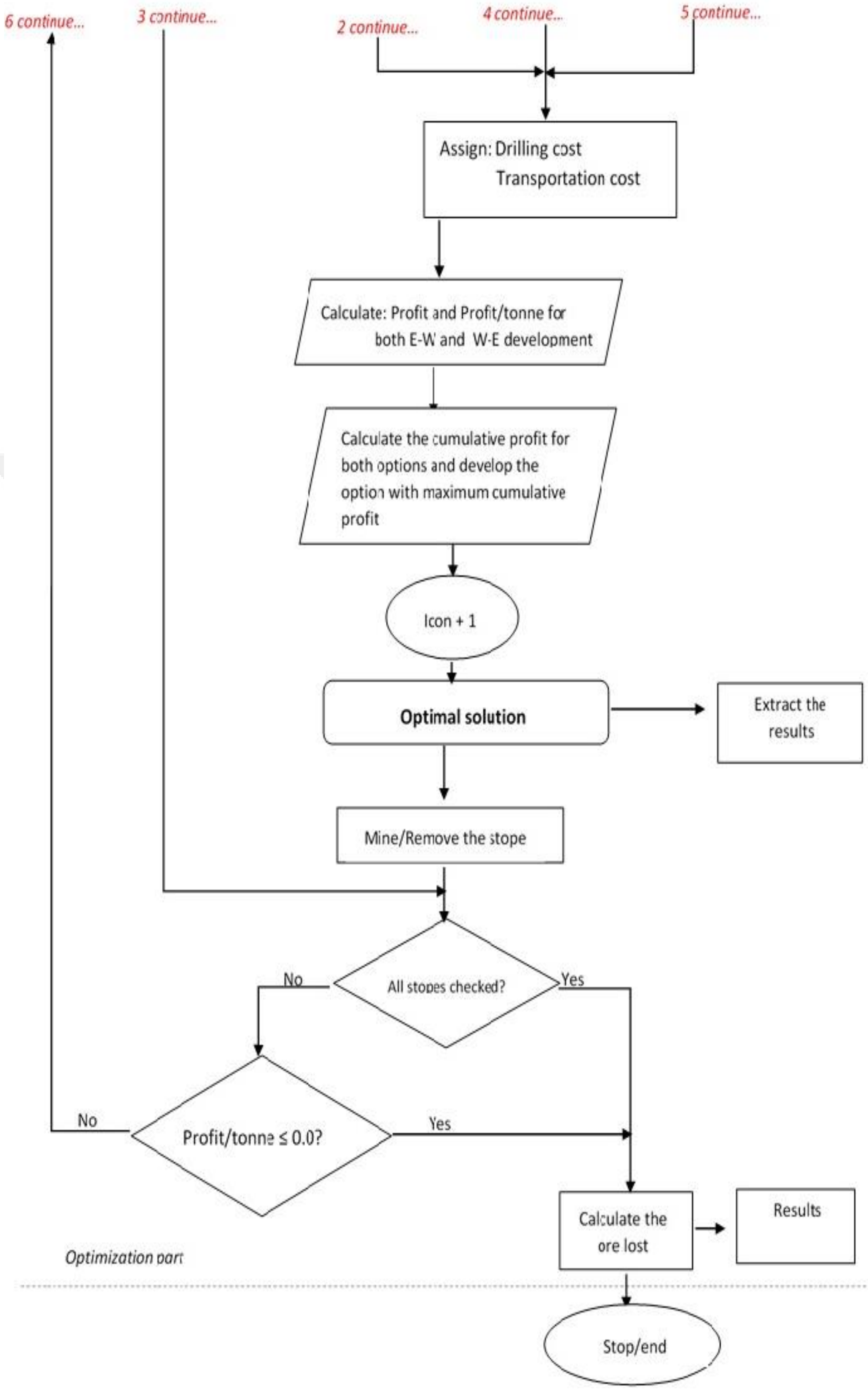


Figure 5.2 continues

CHAPTER SIX

DISCUSSION AND INTERPRETATION OF RESULTS

Two real orebodies of copper and zinc with blocks n = 22139 and 728 respectively were used to evaluate the algorithm. Block dimension for these orebodies were 10m x 10m x 5m and 5m x 5m x 2.5m respectively.

The copper deposit used in the interpretation of the Algorithm is generally dipping $48^0 – 51^0$ to the north. The analysis of a block model in different sections (50 m apart) toward north shows the total distance of mineralization ranges between 200 m to 650m in an east-west direction. However, the mineralization is discontinuous and the orebody is irregular. There are two to three blocks or merely four in raw before a series of waste blocks. In terms of height or elevation, two to four successive blocks are found to have a deposit of copper. In the other side, the block model was analysed in different sections (50 m apart) toward the east. The discontinuous mineralization runs in a total distance of 60 m to 280 m in a north-south direction. The continuity property is quite regular compared to the north whereby at least a series of six to seven blocks with copper mineralization can be found. The combination of these characteristics overrides the decision to choose the range of stope size as shown in table 6.1.

A standard computer system of 64-bit operating system, x64 based processor (Processor: Intel® Core™ i5-4210U CPU @ 1.70GHz – 2.40 GHz) was used to run both algorithms. Appendices 1.1 to 1.4 show the results obtained from the deposit of zinc. As mentioned earlier, two circumstances have been considered for the purpose of analysis and interpretation. One of the circumstances is when the need for sublevels are controlled. A stope is eligible to be assigned a drive development cost if and only if it is located within a specified distance from its neighbour. The stope is considered to be accessible by a sublevel whose cost is very small and hence been ignored in the calculation. In other circumstance, the cost of establishing main levels and raises are assigned freely to every stope. This means that each stope regardless of the distance from its neighbour is assigned a drive development cost. For referencing, *Algorithm 1* denotes the algorithm that evaluates the suitable distance to establish the drive development cost. Likewise, *Algorithm 2* denotes the algorithm that establishes drive development cost to each stope created. Table 6.1 shows the dimensional and economic constraints applied to the copper block model and hence produces the results for both *Algorithm 1* and *2*

Table 6.1 Input parameters to the copper block model

Input	Value	unit
Cutoff grade	*1.5*	*%*
Market price of copper	*6600*	*$/ton*
Ore recovery	*90*	*%*
Mining cost	*50*	$\$/m^3$
Milling cost	*25*	$\$/m^3$
Specific gravity	*3*	ton/m^3
Minimum stope dimension	*20 x 60 x 20*	m^3
Maximum stope dimension	*30 x 70 x 25*	m^3
Drive development cost	*2000*	*$/m*
Haulage grade	*10*	*%*
Level development cost	*1000*	*$/m*
Drilling cost	*0.1*	*$/m*
Material transportation cost	*0.1*	*$/m*
Block recovery	*0.1*	*%/m*
Alowable b_s/B_s ratio	*0.2*	*%*

6.1 Algorithm 1 Versus Algorithm 2

The algorithm required approximately 55 minutes to create possible stopes for all assumed ramp positions of the copper deposit. During the optimization, cutoff grade and range of stope dimension showed a pronounced impact on maximum profit, profit per tonne, number of stopes, profit per tonne and ramp position. The cutoff grade is determined by market price of metal while the stope dimension links with the available technology, geotechnical considerations and the allowable degree of ore dilution.

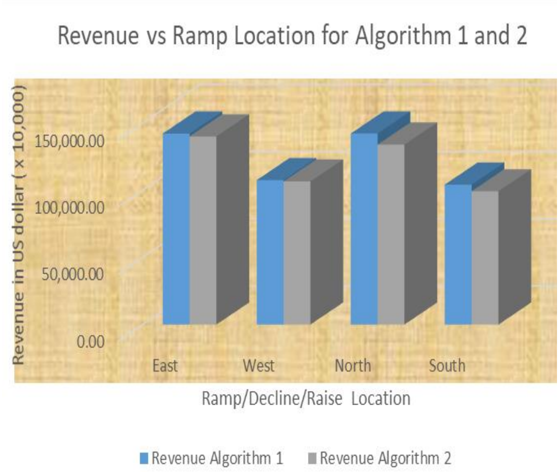
Table 6.2 and 6.3 summarize the key results observed from *Algorithm 1* and *2* when 13.43 million tonnes of the copper reserve was executed.

Table 6.2 The Calculated results from *Algorithm 1*

Results from *Algorithm 1*:				
Ramp position	Number of Stopes	Total profit $ (x 10,000)	Tonnage recovered (million tonnes)	Tonnage lost (Million tonnes)
East	353	142,251.00	10.47 (78%)	2.96 (22%)
West	347	107,591.00	10.35 (77%)	3.08 (23%)
North	298	142,293.00	10.38 (77.3%)	3.05 (22.7%)
South	295	104,240.00	10.31 (76.8%)	3.12 (23.2%)

Table 6.3 Strategic results from *Algorithm 2*

Results from *Algorithm2*				
Ramp position	Number of Stopes	Total profit $ (x 10,000)	Tonnage recovered (million tonnes)	Tonnage lost (Million tonnes)
East	352	140,145.00	10.37 (77.2%)	3.06 (22.8%)
West	346	106,608.00	10.17 (75.7%)	3.26 (24.3%)
North	292	134,025.00	9.94 (77.3%)	3.49 (22.7%)
South	285	99,260.00	9.75 (72.6%)	3.68 (27.4%)



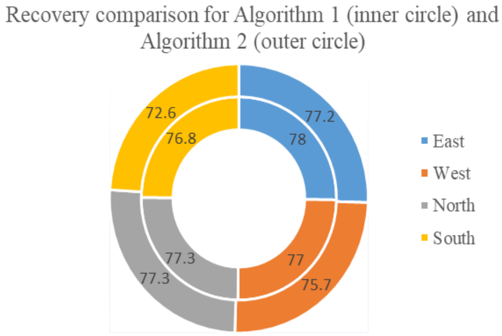


Figure 6.1 Revenue and ore recovery for *Algorithm 1 and 2*

Stope layout generated by *Algorithm 2* proves to be superior compared to that of *Algorithm 1*. The data of recovery and revenue in Table 6.2 and 6.3 are summarized in Figure 6.1. The figure clarifies that in any respective location of a ramp (the position where development activities are launched), the stope layout generated by *Algorithm 1* is profitable over *Algorithm 2*. For example, when the development starts from the north, the amount of copper ore recovered is the same for both algorithms. However, the costs of establishing the drives subordinate the value of the ore by almost 6% as seen in the bar chart. The stopes to which main drives have not been assigned are assumed to be developed from sublevels. Moreover, the distance from the developed neighbour stope(s) is very small compared to the distance from the decline/raise/ramp. The cost of establishing the sublevel is very small compared to that of the main drive and hence, the cost can be ignored. The data of revenue and recovery in Table 6.2 and 6.3 proves *Algorithm 1* to have an ability to maximize recovery and value compared to *Algorithm 2*.

(a) (b)

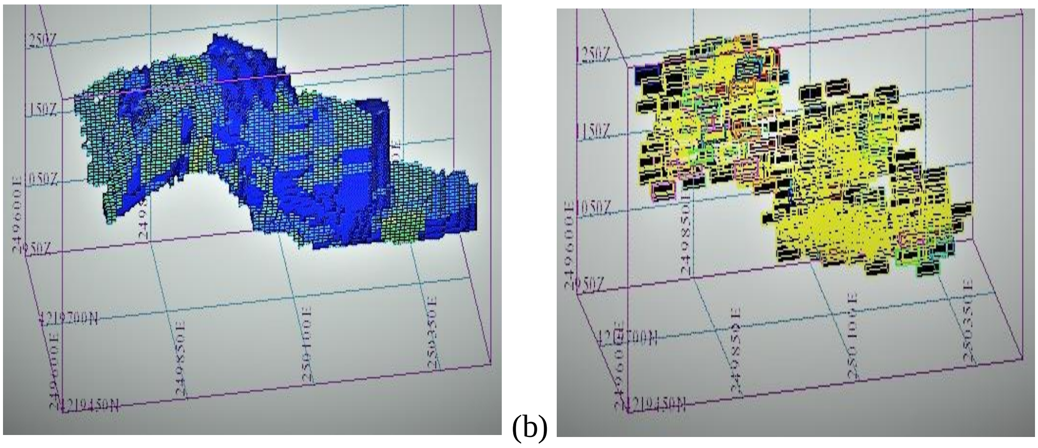


Figure 6.2 (a) Copper block model and (b) the stope layout – developed in the north - south direction

(a) (b)

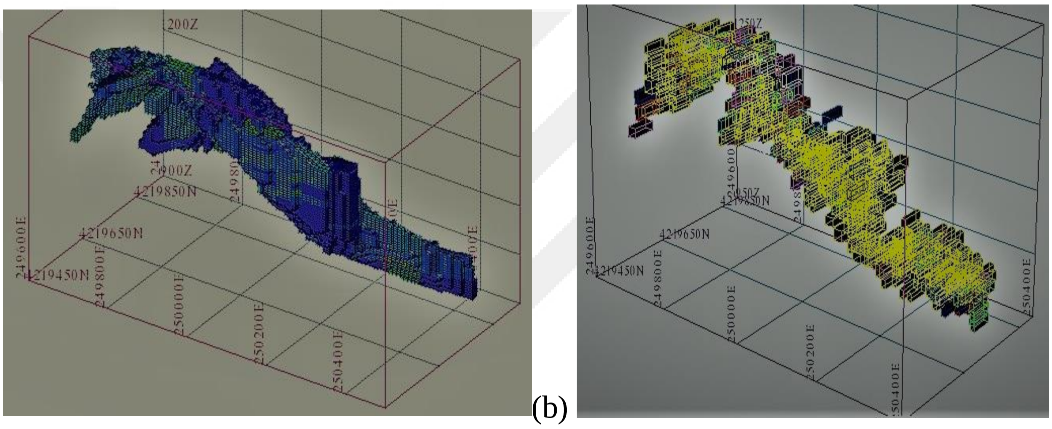


Figure 6.3 (a) Copper block model and (b) the stope layout – developed in the east-west direction

Both Figure 6.2 and 6.3 show the stope boundary layout resulted from *Algorithm 1*. The layout in Figure 6.2 (b) produces the highest revenue compared to other layouts but the recovery is lower than that of Figure 6.3 (b). The actual data of revenue and recovery are in Table 6.1. There are two reasons underlying this observation.

i) The nature of mineralization. The stopes formed toward the south are longer than those extending west. This is because of steadiness in mineralization in north-south. Since the algorithm works by maximizes revenue per tonne of every block included in a stope without breaking the specified dimensional limit, the direction which has continuity of rich mineralization is favoured to form longer stopes. Contrarily, in case of poor mineralization or continuity, there will be a high level of ore dilution influenced by waste blocks.

ii) The dimension of the deposit. The copper deposit has a wider span in the east than the north (see Figure 6.4 and 6.5). Due to poor mineralization towards the west, many short stopes are formed when development cost is applied from the east. Indeed, the stopes might include many blocks to form stopes but the cost of development and poor mineralization lower the value per tonne of the stopes formed. As shown in table 6.1, Development from north forms 298 stopes, more profitable than development from the east where 55 more stopes are formed.

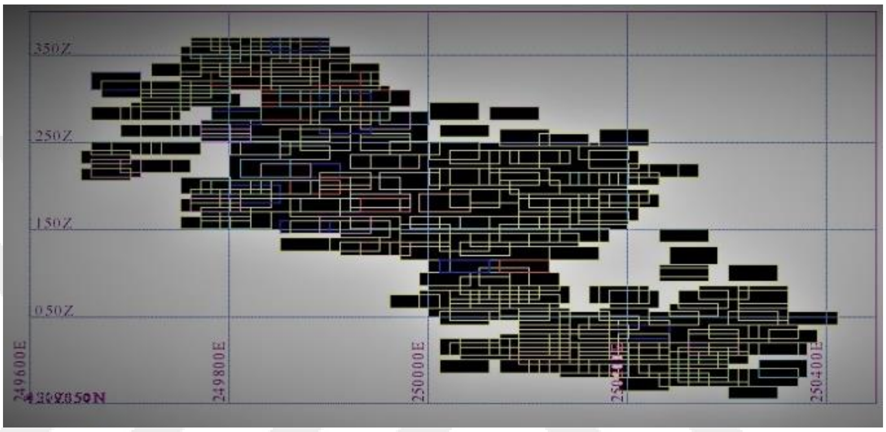


Figure 6.4 Stopes extending from north to south. The view has been taken from the east

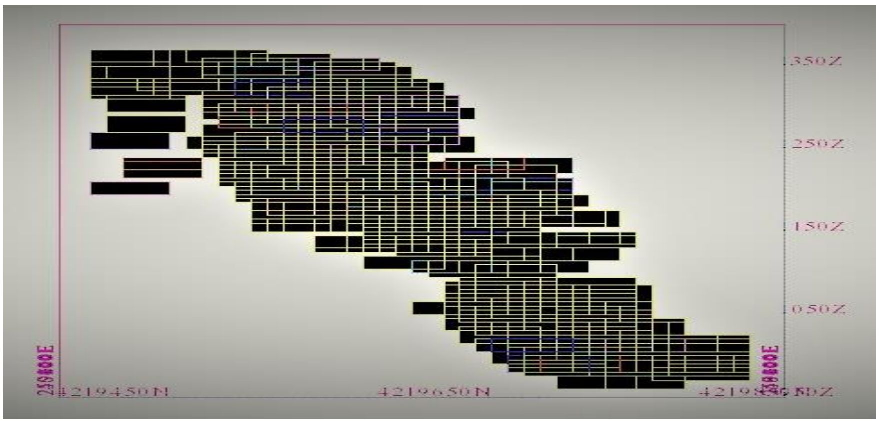


Figure 6.5 Stopes extending from east to west. The view has been taken from the north

To conclude about the observation from the copper deposit, *Algorithm 1* is capable of producing a stope layout that maximizes the value per tonne. The algorithm developed a copper reserve of 13.43 million tonnes and the option, which locates ramp at the north (in the side facing footwall) of the reserve produced 77.30% of copper ore by creating 298 stopes. This makes the highest profit of $ 1.34 billion. Furthermore, accessing the reserve from the south, which is facing the hanging wall increases drives development cost per tonne for the stopes located near the surface hence radically lower the revenue by nearly 27%. As shown in Figure 6.1, when a reserve is accessed in other locations more ore is left in the ground. The main reason behind this is negative profit per tonne of ore or the presented ore to be small pockets with less than 0.2% blocks (the minimum number of blocks above cutoff grade required to construct a stope). Ore loss is hereby defined as materials that cannot be mined when development cost is applied due to the generation of stopes with negative profit per tonne of ore or exist in a small amount to generate the minimum stope size. The ore lost can be recovered if there will be efforts to lower development cost per meter or the range of stope dimension.

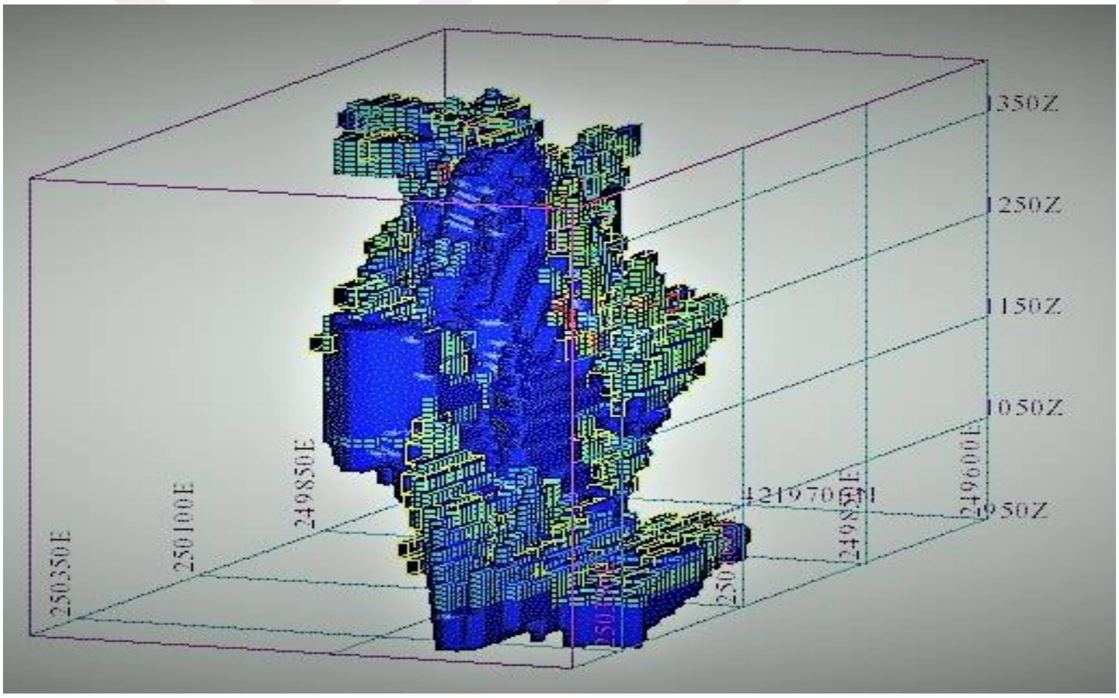


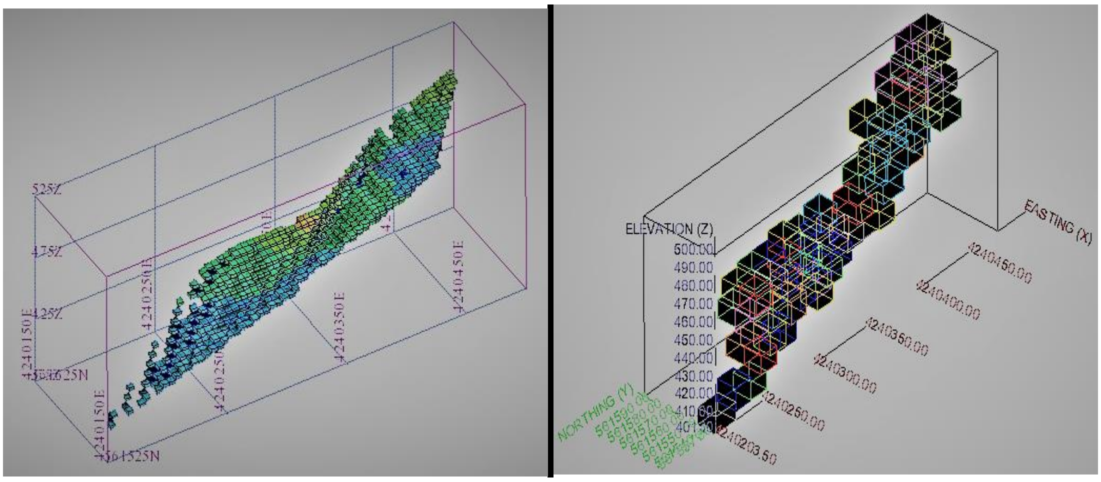
Figure 6.6 Ore loss at the ore-waste interface (black coloured blocks)

6.2 Varying the Cutoff Grade

To account for the effect of changing the cutoff grade, the *Algorithm 1* was executed five times with cutoff grade of zinc equals to 5%, 6%, 7% and 8% and 10% along with other inputs as shown in Table 6.4. This thin orebody is relatively uniform, which in turn a slight change in profit and number of stopes was observed with the change in ramp position. The rich ore is concentrated at the centre making the length quite similar to width. This shape and metal distribution results to formation of a relatively equal number of stopes in either choice of ramp position. With stopes dimension ranging from 15 m x 30 m x 15 m to 20 m x 40 m x 20 m and when the cutoff grade been 5%, 33 stopes are generated while the ramp is in any location (See Appendix 1.1 to 1.4). Furthermore, the number of stopes kept decreasing uniformly as cutoff grade increases. The algorithm took approximately 2 minutes to form result in each cutoff grade.

Table 6.4 Input parameters to the zinc block model

Inputs		
Input	Value	unit
Market price of copper	*3400*	*\$/ton*
Ore recovery	*90*	*%*
Mining cost	*50*	*\$/m^3*
Milling cost	*25*	*\$/m^3*
Specific gravity	*2.9*	*ton/m^3*
Minimum stope dimension	*15 x 30 x 15*	*m^3*
Maximum stope dimension	*20 x 40 x 20*	*m^3*
Drive development cost	*1800*	*\$/m*
Haulage grade	*10*	*%*
Level development cost	*950*	*\$/m*
Drilling cost	*0.1*	*\$/m*
Material transportation cost	*0.1*	*\$/m*
Block recovery	*0.1*	*%/m*
Alowable bs/Bs ratio	*10%*	



(a) (b)

Figure 6.7 (a) Zinc block model in z-x (towards east) and (b) represents its stope layout

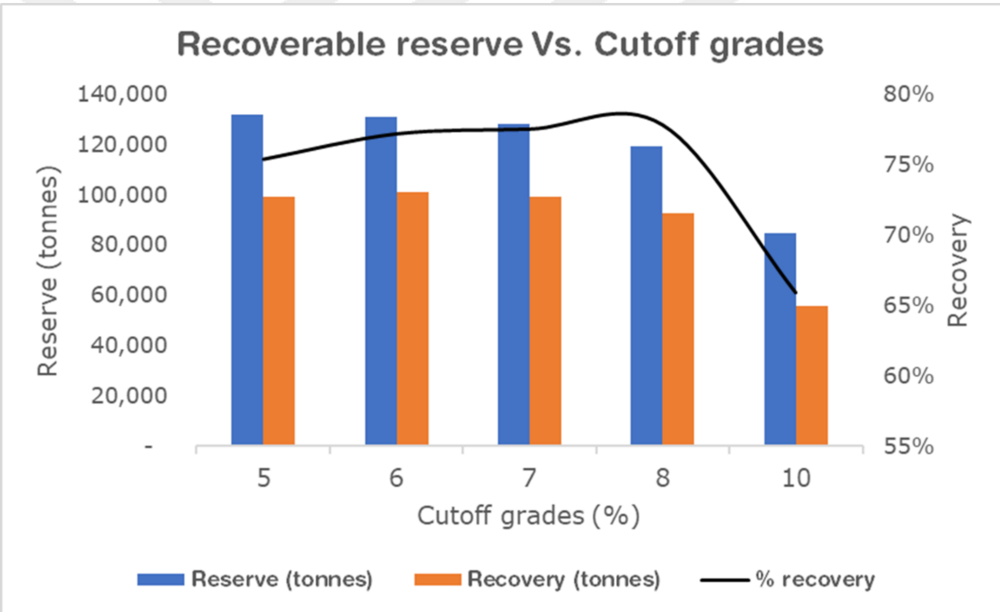


Figure 6.8 The effect of the varying cutoff grade of zinc to minable reserve and recovery

Cutoff grade influences the amount of reserve and hence the recovery. Figure 6.8 above manifests the exponential decrease in the amount of reserve and recoverable tonnage as the cutoff grade increases. However, considering the percentage recovery for an individual cutoff grade, large proportionate of ore can be recovered when the cutoff grade is somewhere between 7% and 8%. Indeed, the situation depends on grade distribution in the orebody. Additionally, a block with low grade can combine with blocks with high grade or value to form a stope with positive profit per tonne. Thus, including the secondary development cost in a formation of stope geometry results to uniform trend (exponential), which at somehow helps to predict the reserve or recoverable ore in case of varying cutoff grade.

6.3 The Significance of the Block Ratio

The ratio of blocks with a positive value over total blocks in a stope is introduced to the algorithm in order to control the ore dilution during stope layout formation. The ratio functions in parallel with the other constraint of revenue per tonne. When a stope with positive revenue per tonne is created, the algorithm checks for the number of waste blocks included. If the number of waste blocks is greater than a specified limit, the stope is skipped from being included in the layout at that particular period. For example, if a stope contains ten blocks and ${}^{b_s}/_{B_s}$ equals to 0.1, then a stope with more than one waste block will be avoided in the final layout. This cessation has about three consequences.

(i) Boost the recovery of high-grade ore

(ii) It may totally change the advancement of stope development from one direction to another.

(iii) The blocks will be lost, never participate in stope formation, and hence cause ore loss in the final stope layout.

Figure 6.9 shows the tonnes recovered from zinc stope layout when the ratio (${}^{b_s}/_{B_s}$) is changing from 0.01, 0.1, 0.15, 0.20 and 0.25. The result shows the ramp/raise/decline to be located in the south when the ratio equals 0.01 and later would shift to the east if the ratio changed to 0.25. The location is predominantly north for other percentage of ${}^{b_s}/_{B_s}$.

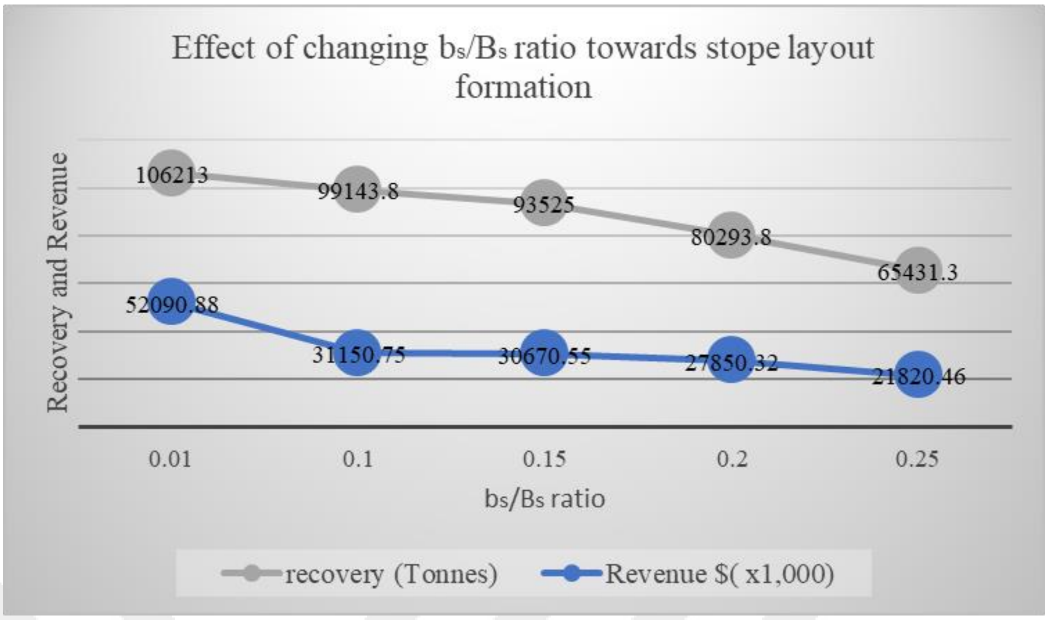


Figure 6.9 Effect of changing bs/Bs ratio towards the formation of zinc stope layout

In this particular deposit, a stope may consist of 12 to 16 blocks to fulfil the size criterion in three dimensions. Furthermore, a block consists of 181.25 tonnes and thus a formed stope may weigh around 2175 to 2900 tonnes. For example, when b_s/B_s ratio is 0.01, a stope with waste material (the grade of material is less than cutoff grade) less than 29 tonnes is allowed to be included in a final layout. The more bs/B_s ratio increase, the more waste materials are expelled from a stope thus when the ratio set to 0.25, almost 725 tonnes of waste can be denied from forming a stope even if the stope still generates a positive revenue per tonne of ore. The primary objective of the optimization is to increase profit and therefore the b_s/B_s ratio should be maintained as minimum as possible. Several iterations should be performed to generate the ratio that improves revenue and reduce tonnage. In Figure 6.9, changing b_s/B_s ratio from 0.01 to 0.001 evidenced the decrease of tonnage by almost 6.7% while revenues fall dramatically by nearly 40%. Definitely, the good ratio lies between zero and somewhere around 0.001.

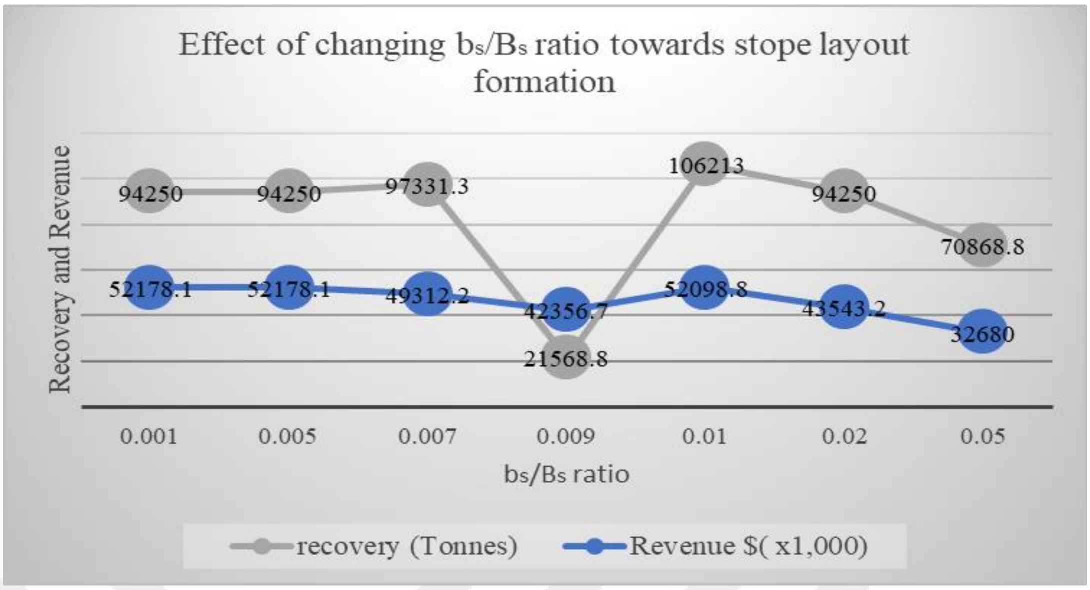


Figure 6.10 Effect of changing bs/Bs ratio towards the formation of zinc stope layout

From Figure 6.10, both recovery and revenue show a similar trend when b_s/B_s ratio is less than 0.007 and greater than 0.01. They appear to be nearly the same when the b_s/B_s ratio is less than 0.007 although starts to fall considerably with an increase in the ratio from 0.01. The ratio of 0.009 constructs a layout with high revenue per tonne of ore. Only 21,568.8 tonnes create $ 42.36 million. Compared to the other ratio, a long-term mining planner should critically answer the following questions with reference to the b_s/B_s ratio of 0.009.

i) Should I choose the b_s/B_s ratio less than 0.007 in order to mine more 77.1% of the material for only $ 9.82 million?

ii) Should I go with the b_s/B_s ratio of 0.009?

iii) Should I target the highest ore recovery at the b_s/B_s ratio of 0.01?

The answer to this question depends on how profit is defined. To this particular project, the aim is to maximize revenue per tonnage thus the b_s/B_s ratio of 0.009 will be fine. However, if the profit were to provide employment and target high recovery, the ratio of 0.01 would be favoured. Consider the zinc reserve at the cutoff grade of 5%. When the b_s/B_s ratio is constrained at 0.009, the revenue of $ 42.36 million will be generated by only 16% of the deposit. On the other side, if the target was maximum revenue regardless of tonnes recovered, the ratio less than or equal to 0.005 would almost guarantees 72% recovery of zinc reserve to make $ 52.18 million.

Generally, the definition of profit varies. It can be either to sustain employment, to achieve maximum revenue or to have a maximum value per tonne of ore mined. The b_s/B_s ratio helps to give the decision on how much material less than cutoff grade a stope should include for achieving the predefined sense of profit.

CHAPTER SEVEN
CONCLUSION AND RECOMMENDATION

This chapter is subdivided into section 7.1 and 7.2. Section 7.1 presents a summary of the basic aspects of the algorithm and their contribution to underground mining. Both copper and zinc reserve generated by the algorithm will be used to reflect the general position and challenges of including secondary development costs to create a stope layout. Section 7.2 provides the recommendation for improving the algorithm and further studies based on the observation, applicability and other demonstrated performance characteristics.

7.1 Summary of the Thesis

The algorithm presented in this thesis indorses the following

i) The use of secondary development costs to generate amenable stopes in a layout. The approach agrees to the suggestion posed by experts in the mining industry such as those mentioned by Chadwick, 2009 that global optimization should be preferred over individual aspects of the production cycle. The study has specifically focused the situations where the main access to an orebody is ramp/decline/adit. Furthermore, the thesis acknowledges the primary development costs required to establish the main entrance to access an orebody may vary depending chiefly on depth from the surface. However, the secondary development costs can affect the optimal layout with respect to changes in individual production and designing parameter. Secondary development costs are classified as drive/level development costs, ramp/raise/decline cost, drilling and transportation costs. These expenses are applicable soon after mining and mineral processing costs have been used to suggest the possible stopes. The stopes that generate positive revenue are included in a final layout.

ii) The starting location of secondary access to the reserve dictates the amount of revenue from the stope layout. A reserve can be accessed from one of the four compass direction, mainly north, south, east or west. The choice of either one of the approach affects the stope layout, recovery, and revenues. The location that maximizes profit per tonne of ore is selected.

iii) The selection of appropriate stope characteristics, including the range of stope dimension, number of waste blocks in a stope and cutoff grade. The range of stope dimension has a great correlation with the mining method and geotechnical characteristics of both ore and surrounding rock mass. Different sections of the orebody or block model help to analyse the continuity of mineralization and possible stope range to increase recovery. The b_s/B_s ratio limits the tonnes of materials below cutoff grade from forming stopes and a result of which, stopes with high grade per tonnes of ore are formed. In contrary, some blocks above cutoff grade may be outweighed by blocks with negative value objected from forming stopes. Indeed, the chance of ore loss is high and hence the ration should be maintained as low as possible. The cutoff grade should be chosen wisely while forecasting on the trend of the market price of a metal to be extracted.

iv) It is necessary to consider the option of using sublevels to access stopes that are near to each other. Developing drives is not only expensive but also have concerns with geotechnical properties of the surrounding rock. The use of sublevels minimizes unnecessary exposure of waste rock and development costs. Algorithm 1 generates a stope layout while assuming the cost of creating a sublevel being less as much as zero compared to the cost of establishing a drive.

The algorithm uses a heuristic approach to generate stopes. The approach is favoured due to the simplicity of choosing and combining appropriate blocks to include in a stope within multiple ranges of dimension while using a short time to produce results. The algorithm was implemented on a real copper and zinc deposit. The copper deposit was used to evaluate the recovery of ore and revenue generated with respect to a location of starting the secondary development. The deposit was also used to demonstrate the importance of using *Algorithm 1* over *Algorithm 2*. With a cutoff grade of 1.5% and the dimension of stope ranging from 20m x 60m x 30m to 30m x 70m x 25m, the blocks measured 10m x 10m x 5m were used to generate stope layout of copper deposit with 13.43 million tonnes. While employing the constraints of economic, development cost and b_s/B_s ratio, the algorithm iterates through the northern, southern and eastern location of secondary access to generate stopes with a positive value per tonne of ore. The optimum location was determined to be north and generates a stope layout that recovers 77.3% of ore; an equivalent to the reserve of 10.38 million tonnes with a revenue of $ 14.23 billion. It only took 55 minutes to work in such a big deposit and thereafter, the results were converted from integer back to Cartesian coordinates (x, y, and z) by using a small algorithm (sub-function). The stope layout as imported into Surpac™ for visualization showed to fit significantly to the copper block model with stopes covering the blocks with positive economic value.

The objective of using *Algorithm 1* to a zinc deposit was to analyse the effect of cutoff grade and b_s/B_s ratio towards the recovery and revenue. While cutoff grades of 5%, 6%, 7 %, 8% and 10% and the stope dimension ranging from 15m x 30m x 15m to 20m x 40m x 20m, the blocks measured 5m x 5m x 2.5m were used to generate stope layout from a zinc deposit. With other constraints remaining constant, it was discovered that reserve decreases with increase in cutoff grade however, in some peculiarity; recovery from a slightly small reserve may outweigh that of a bigger reserve. The reason for this situation was selectivity tendency whereby a block with high grade and value may combine with other blocks with a slightly lower grade than cutoff to form a stope with positive profit per tonne. The significance of the b_s/B_s ratio was analysed from recovery and revenue by using different ratios while maintaining constant other constraints. It was found that the definition of profit might suggest the b_s/B_s ratio whereby, some ratio increases recovery, revenue or revenue per tonne. A small ratio is recommended to avoid ore loss and dilution.

7.2 Recommendations

The research has a significant contribution toward global optimization. The algorithm uses development expenses to reflect the stope layout. It is a major step towards stope boundary layout optimization in underground mining compared to the other researches in the literature which ignore the secondary development costs by appealing to have a negligible effect on the layout. However, the applicability and performance proposed algorithm and mode of visualization could be further improved in the following areas:

The development costs are presented in the algorithm in the form of a constant value. An equation that may mimic the development costs in terms of involved operations and dimension of a drive, raise/decline/ramp may estimate the global expenses related to a mining condition. Additionally, a program that links the results to visualization may help to interpret the results more easily. Another suggestion, the scheduling part must be included in a stope optimization process. The plan of which stope should be mined before the other in order to maintain rock geotechnical quality may lower the value of a stope with respect to time.

The algorithm is set to produce results when the orebody is accessed from the top location. Other main access methods such as using the shaft to access the ore in other positions and hence, secondary development to start in any location other than the top are out of the scope of this thesis. Further studies are recommended to decide the starting point of secondary development. Furthermore, the algorithm is set to solve the optimization problem for only one metal type at a time. In case of a reserve of more than one material such as copper and zinc, the algorithm cannot provide a solution simultaneously. Improvement to suit the stated situation is vital.

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APPENDICES

Appendix - 1.1: Result extracted from the algorithm 1 when the cutoff grade of zinc is 5% and the drives extend along east-west

Minimum, maximum x-coordinates: 4240161.00, 4240483.50

Minimum, maximum y-coordinates: 561540.30, 561602.81

Minimum, maximum z-coordinates: 387.58, 503.83

Block width = 5.00 m   Block length = 5.00 m   Block height = 2.50 m

i Start	i end	j start	j end	k start	k end	stope number	profit ($) (x10000)	Weight (tonnes)	($/ton) (east)	($/ton) (west)
41	46	62	64	7	12	1	101.48	1993.75	0.050899	0.051915
15	20	41	43	7	12	2	141.68	2356.25	0.060129	0.058421
14	19	38	40	7	12	3	183.60	3443.75	0.053313	0.051899
19	25	35	37	7	12	4	160.40	3443.75	0.046578	0.046919
21	26	41	43	7	12	5	90.55	1993.75	0.045418	0.046536
20	25	38	40	7	12	6	98.03	2175.00	0.045070	0.046024
19	24	32	34	7	12	7	116.06	2718.75	0.042690	0.043299
13	18	35	37	5	10	8	116.06	2718.75	0.042690	0.041066
41	46	59	61	7	12	9	205.85	5437.50	0.037858	0.037808
25	31	44	46	5	12	10	113.81	3081.25	0.036935	0.036356
19	25	29	31	7	12	11	118.32	3262.50	0.036267	0.036352
13	18	32	34	7	12	12	118.32	3262.50	0.036267	0.035080
27	34	47	49	6	12	13	98.96	3081.25	0.032118	0.032288
19	25	26	28	7	12	14	139.62	4350.00	0.032096	0.032288
41	46	56	58	7	12	15	63.85	1993.75	0.032023	0.032088
24	31	40	43	1	6	16	63.85	1993.75	0.032023	0.031217
13	18	28	31	6	11	17	71.91	2356.25	0.030518	0.029205
35	40	56	59	7	12	18	69.03	2356.25	0.029295	0.029076
30	35	50	53	3	8	19	69.03	2356.25	0.029295	0.028498
17	24	23	25	7	12	20	69.03	2356.25	0.029295	0.028356
20	27	37	39	1	6	21	69.03	2356.25	0.029295	0.028052
38	45	53	55	7	12	22	62.21	2175.00	0.028600	0.027777
36	42	50	52	7	12	23	62.21	2175.00	0.028600	0.026795
17	24	32	34	1	6	24	62.21	2175.00	0.028600	0.025955
12	17	20	22	6	11	25	53.63	1993.75	0.026901	0.025562
40	46	57	59	1	6	26	91.66	3443.75	0.026617	0.025033
15	22	25	27	1	6	27	91.66	3443.75	0.026617	0.024776

12	17	17	19	4	9	28	45.71	1993.75	0.022925	0.021247
39	46	54	56	1	6	29	43.49	1993.75	0.021814	0.020896
36	42	50	52	1	6	30	43.49	1993.75	0.021814	0.020431
5	10	10	12	1	6	31	43.49	1993.75	0.021814	0.019672
10	15	14	16	3	8	32	42.47	1993.75	0.021300	0.019334
33	38	54	56	1	6	33	56.52	2900.00	0.019491	0.018586

Total profit (x10000) for east $ 3033.73; Total profit (x10000) for west $ 3059.24 Recoverable tonnage = 90262.5

Appendix - 1.2: Result extracted from the algorithm 1 when the cutoff grade of zinc is 5% and the drives extend along north-south

i Start	i end	j start	j end	k start	k end	stope number	profit ($) (x10000)	Weight (tonnes)	($/ton) (north)	($/ton) (south)
41	46	59	64	10	12	1	201.98	4350.00	0.046433	0.046796
19	25	38	43	10	12	2	256.85	4350.00	0.059046	0.057851
20	27	32	37	10	12	3	136.87	2718.75	0.050342	0.048110
13	18	35	41	9	12	4	96.39	1993.75	0.048347	0.048110
19	25	40	45	7	9	5	118.53	2537.50	0.046713	0.045663
19	25	34	39	6	9	6	125.47	3081.25	0.040719	0.041390
20	27	26	31	10	12	7	78.98	1993.75	0.039615	0.035759
40	45	53	58	10	12	8	121.91	3443.75	0.035400	0.035080
41	46	59	64	7	9	9	121.91	3443.75	0.035400	0.034039
13	18	29	34	8	10	10	121.91	3443.75	0.035400	0.033058
26	31	39	44	5	7	11	68.96	1993.75	0.034587	0.032657
16	21	19	24	8	10	12	65.32	1993.75	0.032762	0.031658
31	37	47	52	10	12	13	83.09	2537.50	0.032746	0.031658
28	34	39	44	2	4	14	81.79	2537.50	0.032232	0.030409
13	18	32	37	5	7	15	75.49	2356.25	0.032040	0.030409
32	38	44	49	5	7	16	72.61	2356.25	0.030816	0.029606
16	23	26	31	5	7	17	105.86	3443.75	0.030739	0.029143
30	37	45	50	2	4	18	200.17	6706.25	0.029848	0.029143
40	46	53	58	7	9	19	74.27	2537.50	0.029268	0.029143
22	27	38	43	2	4	20	63.28	2175.00	0.029095	0.027578
34	39	53	58	8	11	21	63.28	2175.00	0.029095	0.027371
19	26	32	37	3	5	22	63.28	2175.00	0.029095	0.025433
40	46	57	62	4	6	23	63.28	2175.00	0.029095	0.024883
15	22	26	31	2	4	24	63.28	2175.00	0.029095	0.024809
15	22	20	25	5	7	25	63.28	2175.00	0.029095	0.024766

31	36	50	55	5	7	26	69.27	2718.75	0.025479	0.024755
31	36	51	56	2	4	27	77.04	3081.25	0.025003	0.024320
14	20	20	25	1	4	28	63.16	2537.50	0.024889	0.021436
11	16	14	19	5	7	29	46.28	1993.75	0.023211	0.020416
7	14	10	15	1	3	30	64.62	2900.00	0.022283	0.020186
38	45	51	56	4	6	31	98.00	4712.50	0.020796	0.020186
8	13	18	23	1	4	32	74.53	3806.25	0.019581	0.018338
37	44	51	56	1	3	33	35.40	3262.50	0.010851	0.018118

Total profit (x10000) for north $ 3151.75 Total profit (x10000) for south $ 2900.11 Recoverable tonnage = 99143.8

Appendix - 1.3: Result extracted from the algorithm 2 when the cutoff grade of zinc is 5% and the drives extend along east-west

i Start	i end	j start	j end	k start	k end	stope number	profit ($) (x10000)	Weight (tonnes)	($/ton) (east)	($/ton) (west)
41	46	62	64	7	12	1	101.48	1993.75	0.050899	0.051915
15	20	41	43	7	12	2	141.68	2356.25	0.060129	0.058421
14	19	38	40	7	12	3	183.60	3443.75	0.053313	0.051899
19	25	35	37	7	12	4	160.40	3443.75	0.046578	0.046919
21	26	41	43	7	12	5	90.55	1993.75	0.045418	0.046536
20	25	38	40	7	12	6	98.03	2175.00	0.045070	0.046024
19	24	32	34	7	12	7	116.06	2718.75	0.042690	0.043299
13	18	35	37	5	10	8	116.06	2718.75	0.042690	0.041066
41	46	59	61	7	12	9	203.68	5437.50	0.037458	0.037371
25	31	44	46	5	12	10	113.81	3081.25	0.036935	0.036356
19	25	29	31	7	12	11	118.32	3262.50	0.036267	0.036352
13	18	32	34	7	12	12	118.32	3262.50	0.036267	0.035080
27	34	47	49	6	12	13	98.96	3081.25	0.032118	0.032288
19	25	26	28	7	12	14	139.62	4350.00	0.032096	0.032288
41	46	56	58	7	12	15	63.85	1993.75	0.032023	0.032088
24	31	40	43	1	6	16	63.85	1993.75	0.032023	0.031217
13	18	28	31	6	11	17	71.91	2356.25	0.030518	0.029205
35	40	56	59	7	12	18	69.03	2356.25	0.029295	0.029076
30	35	50	53	3	8	19	69.03	2356.25	0.029295	0.028498
17	24	23	25	7	12	20	69.03	2356.25	0.029295	0.028356
20	27	37	39	1	6	21	69.03	2356.25	0.029295	0.028052
38	45	53	55	7	12	22	62.21	2175.00	0.028600	0.027777
36	42	50	52	7	12	23	62.21	2175.00	0.028600	0.026795

17	24	32	34	1	6	24	62.21	2175.00	0.028600	0.025955
12	17	20	22	6	11	25	53.63	1993.75	0.026901	0.025562
40	46	57	59	1	6	26	91.66	3443.75	0.026617	0.025033
15	22	25	27	1	6	27	91.66	3443.75	0.026617	0.024776
12	17	17	19	4	9	28	45.71	1993.75	0.022925	0.021247
39	46	54	56	1	6	29	43.49	1993.75	0.021814	0.020896
36	42	50	52	1	6	30	43.49	1993.75	0.021814	0.020431
5	10	10	12	1	6	31	43.49	1993.75	0.021814	0.019672
10	15	14	16	3	8	32	42.47	1993.75	0.021300	0.019334
33	38	54	56	1	6	33	56.52	2900.00	0.019491	0.018586

Total profit (x10000) for east $ 3031.56 Total profit (x10000) for west $ 3056.87 Recoverable tonnage = 90262.5

Appendix - 1.4: Result extracted from the algorithm 2 when the cutoff grade of zinc is 5% and the drives extend along north-south

i Start	i end	j start	j end	k start	k end	stope number	profit ($) (x10000)	Weight (tonnes)	($/ton) (north)	($/ton) (south)
41	46	59	64	10	12	1	201.98	4350.00	0.046433	0.046796
19	25	38	43	10	12	2	256.85	4350.00	0.059046	0.058561
20	27	32	37	10	12	3	136.87	2718.75	0.050342	0.050857
13	18	37	42	9	11	4	96.39	1993.75	0.048347	0.050857
19	25	40	45	7	9	5	118.53	2537.50	0.046713	0.045663
14	19	31	36	10	12	6	90.76	2175.00	0.041727	0.042237
19	25	34	39	6	9	7	125.47	3081.25	0.040719	0.041390
20	25	26	31	9	11	8	78.98	1993.75	0.039615	0.036078
40	45	53	58	10	12	9	72.61	1993.75	0.036421	0.036078
13	18	31	36	7	9	10	184.43	5075.00	0.036342	0.036078
41	46	59	64	7	9	11	68.96	1993.75	0.034587	0.033268
26	31	39	44	5	7	12	68.96	1993.75	0.034587	0.032657
16	21	19	24	8	10	13	65.32	1993.75	0.032762	0.031658
31	37	47	52	10	12	14	83.09	2537.50	0.032746	0.031658
17	22	25	30	6	8	15	65.26	1993.75	0.032733	0.030661
28	34	39	44	2	4	16	81.79	2537.50	0.032232	0.030661
14	19	25	30	9	11	17	85.53	2718.75	0.031461	0.030661
32	38	44	49	5	7	18	72.61	2356.25	0.030816	0.029606
30	37	45	50	2	4	19	200.17	6706.25	0.029848	0.029097
40	46	53	58	7	9	20	63.28	2175.00	0.029095	0.028207
22	27	38	43	2	4	21	63.28	2175.00	0.029095	0.027578

34	39	53	58	8	11	22	63.28	2175.00	0.029095	0.027371
15	20	19	24	5	7	23	63.28	2175.00	0.029095	0.027371
13	18	31	36	4	6	24	57.64	1993.75	0.028911	0.027371
16	22	25	30	3	5	25	72.46	2537.50	0.028555	0.026973
11	16	25	30	6	8	26	59.14	2175.00	0.027193	0.026973
19	26	32	37	3	5	27	93.54	3443.75	0.027162	0.025433
40	46	57	62	4	6	28	69.27	2718.75	0.025479	0.024883
31	36	50	55	5	7	29	69.27	2718.75	0.025479	0.024755
31	36	51	56	2	4	30	77.04	3081.25	0.025003	0.024320
10	15	23	28	2	4	31	49.40	1993.75	0.024777	0.021513
13	19	17	22	2	4	32	56.63	2356.25	0.024032	0.020416
7	14	10	15	1	3	33	64.62	2900.00	0.022283	0.020416
9	14	15	20	5	7	34	44.28	1993.75	0.022208	0.020416
38	45	51	56	4	6	35	98.00	4712.50	0.020796	0.020186
37	42	54	59	1	3	36	34.81	1993.75	0.017459	0.018118
7	12	16	21	1	3	37	34.81	1993.75	0.017459	0.016008

Total profit (x10000) for north $ 3323.42 Total profit (x10000) for south $ 3181.68 Recoverable tonnage = 102406.