UNCERTAINTY ASSESSMENT FOR THE EVALUATION OF NET PRESENT VALUE OF A MINERAL DEPOSIT

A THESIS SUBMITTED TO
THE GRADUATE SCHOOL OF APPLIED AND NATURAL SCIENCES
OF
MIDDLE EAST TECHNICAL UNIVERSITY

BY

ÖMER ERDEM

IN PARTIAL FULFILLMENT OF THE REQUIREMENTS
FOR
THE DEGREE OF MASTER OF SCIENCE
IN
MINING ENGINEERING

DECEMBER 2008

Approval of the thesis:

UNCERTAINTY ASSESSMENT FOR THE EVALUATION OF NET PRESENT VALUE OF A MINERAL DEPOSIT

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ABSTRACT

UNCERTAINTY ASSESSMENT FOR THE EVALUATION OF NET PRESENT VALUE OF A MINERAL DEPOSIT

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December 2008, 123 pages

The profitability of a mineral deposit can be concluded by the comparison of net present values (NPV) of all revenues and expenditures. In the estimation of NPV of a mineral deposit, many parameters are used. The parameters are uncertain. More accurate and reliable NPV estimation can be done with considering the related uncertainties.

This study investigates the probability distributions of uncertain variables in estimation of NPV and evaluation of NPV using Monte Carlo simulation. @Risk 4.5.7 software package is used to apply Monte Carlo simulation method. At the end of the study, all possible net present values and their probabilities are given as a probability distribution.

Dereköy copper ore reserve is selected to apply uncertainty assessment in NPV of ore reserves. The reserve is evaluated using both conventional polygonal method and a mining software which is Micromine. The southeastern part of the reserve was selected as a study area because average grade of the reserve is relatively low and the reserve extends to a larger area.

At the end of the assessment, NPV of the southeastern part of Dereköy ore reserve was found to be between $\$77.97\times10^6$ and $\$318.78\times10^6$ with 68.27% ($\bar{x}\pm\sigma$) probability and between $-\$45.37\times10^6$ and $\$443.54\times10^6$ with 95.45% probability ($\bar{x}\pm2\sigma$).

Keywords: Uncertainty, Monte Carlo Simulation, Net Present Value, Risk, Probability.

ÖZ

BİR MADEN YATAĞININ NET BUGÜNKÜ DEĞERİNİN TAHMİNİNDE BELİRSİZLİK DEĞERLENDİRMESİ

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Aralık 2008, 123 sayfa

Bir maden yatağının işletilebilirliği tüm harcamaların ve kazançların bugünkü değerlerinin mukayesi sonunda anlaşılabilmektedir. Maden rezevlerinin net bugünkü değerinin hesaplanması birçok parametre kullanılarak yapılmaktadır. Bu parametrelerin hemen hemen hepsi belirsizlik içermektedir. Daha doğru ve güvenilir bir net bugünkü değeri (NBD) tahmin etmek için bu parametrelerin en olası değerleri ve olası veri aralıkları göz önünde tutularak hesaplama yapılmalıdır.

Bu çalışma, maden rezervlerinin net bugünkü değerlerinin tahmininde kullanılan belirsiz parametrelerin olasılık dağılımlarını tanımlamak ve net bugünkü değeri Monte Carlo benzetişimi metoduyla değerlendirme üzerinedir. Monte Carlo benzetişimi metodunun uygulanması için @Risk 4.5.7 programı kullanılmıştır. Çalışmanın sonucunda projenin olası net bugünkü değerleri olasılık dağılım grafiği olarak verilmiştir.

Bir maden rezervinin net bugünkü değerinin tahmininde belirsizlik değerlendirmesinin uygulanması için, çalışma sahası olarak Dereköy bakır rezervi

seçilmiştir. Dereköy bakır yatağı bütün olarak bir bilgisayar yazılımı olan Micromine ve poligon metodu ile değerlendirilmiştir. Ortalama tenörün düşük çıkması ve rezervin çok geniş bir alana yayılmış olmasından dolayı, yatağın sadece güneydoğu bölümü çalışma için kullanılmıştır.

Çalışmanın sonucunda, Dereköy bakır yatağının net bugünkü değerinin %68,27 ($\bar{x}$±σ) olasılıkla da $77,97×10^6 ile $318,78×10^6 arasında ve %95,45 ($\bar{x}$±2σ) olasılıkla -$45,37×10^6 ile $443,54×10^6 arasında olduğu bulunmuştur.

Anahtar Kelimeler: Belirsizlik, Monte Carlo Benzetişimi, Net Bugünkü Değer, Risk, Olasılık.

To My Parents and Wife

ACKNOWLEDGEMENTS

I highly express my sincere appreciation to Prof. Dr. Tevfik Güyagüler for his kind supervision, support and guidance in preparation of this thesis.

I would also like to thank Dr. Nuray Demirel for her suggestions and comments.

I offer my thanks to the examining committee members for serving on the M.S. thesis committee.

Barış Çakmak, Recep Kaya, Ebru Demir, Halil Sözeri and Çiğdem Alkılıçgil are gratefully acknowledged for their help during the various stages of the thesis.

I finally want to thank to my family and my wonderful wife Tuğba for their support.

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LIST OF ABBREVIATIONS

NPV	**:**	Net Present Value
NBD	**:**	Net Bugünkü Değer
IRR	**:**	Internal Rate of Return
MARR	**:**	Minimum Attractive Rate of Return
ERR	**:**	External Rate of Return
B	**:**	Benefit
C	**:**	Cost
DTM	**:**	Digital Terrain Model
VBM	**:**	Variable Block Model
VZM	**:**	Variable Zone Model
i	**:**	Interest Rate
n	**:**	Investment Period
F	**:**	Future Value
P	**:**	Principal
PW	**:**	Present Worth
AW	**:**	Annual Worth
PV	**:**	Present Value
D	**:**	Density
GAM	**:**	Grade after Metallurgy
ML	**:**	Mining Life
MR	**:**	Mining Recovery
Met.R	**:**	Metallurgical Recovery
PR	**:**	Processing Recovery
OG	**:**	Ore Grade
SP	**:**	Selling Price
TO	**:**	Total Overburden
PC	**:**	Processing Cost
MC	**:**	Mining Cost

CHAPTER 1

INTRODUCTION

1.1 General Remarks

A mining investment is a risky business because of uncertainties that exist in the reserve evaluation and economical analysis. The main uncertainties are involved in the amount of overburden and reserve, ore grade, capital and operating costs, recovery, dilution, commodity prices, gross revenue and profit/loss, also in economical analysis, determining if the feasibility of orebody utilizes the cut-off grade technique. This technique assumes that operation cost and commodity price remain constant although they may vary. The modern approach in the evaluation of the mineral resources is based on determining the frequency distribution of each variable involved in the calculations and find the result within some confidence interval. At the end, the net present worth that will be achieved will be obtained as a probability distribution, so a decision maker can decide the probability of feasibility of the project. Lower limit and upper limit of the NPV can also be indicated by the distribution. In other words, the distribution can indicate the probability for which project become feasible.

An important example of the uncertainties for mining is commodity price, which has increased unexpectedly since 2003. In July 2008, copper price was more than $8,400 per ton (London Metal Exchange [LME], 2008). Therefore, some low grade copper deposits may become feasible in today's market conditions. In Turkey, there are some low grade copper deposits which were considered uneconomic and infeasible to operate. One of the low grade copper deposits is in Dereköy, Kırklareli. Exploration studies were conducted with drillings between 1981 and 1986 at Dereköy location which is 25 km far from Kırklareli. After the exploration studies, a feasibility report was prepared by MTA for Dereköy copper deposit in 1987. When the feasibility report was prepared for Dereköy copper deposit by MTA, copper price was around $2,000 per ton. Therefore, this deposit was evaluated as not feasible. However, copper price has increased 355.6% between January 1990 and July 2008 so Dereköy copper deposit should be evaluated again to check whether the deposit is feasible or not.

In this study, Dereköy copper deposit is evaluated as a case study. All uncertainties related with the evaluation of mining resource are considered. After the evaluation, a probability distribution for NPV of the deposit is estimated. The distribution indicates the probability of the values that can be achieved by the exploitation of the deposit.

1.2 Statement of the Problem

When net present value of an ore reserve is estimated, some uncertainties are not considered conventionally. Fix values are considered as inputs of estimation of NPV. However, the inputs are uncertain and their risks should be defined before the estimation of NPV of ore reserves. When fix values are used as inputs, a fix value is estimated as value of NPV. This type of estimation does not give any probability of occurrence of NPV. A simulation method should be used to estimate the value of NPV.

1.3 Objective and Scope of the Study

The objective of the study is to develop a method that will contribute to the evaluation of the value of ore deposits with considering uncertainties related to mining and NPV estimation. The developed model aims to estimate the possible NPV values of the ore deposit and to produce a probability distribution for NPV. The elements of the main objective are:

i. To develop a model to find out if the deposit is feasible or not by considering the desired probability levels.
ii. To conduct a case study to validate the model.
iii. To investigate the condition for different uncertainty to have a feasible poor grade Dereköy copper mine.

The scope of this thesis is evaluating the southeastern part of the Dereköy copper reserve with considering the related mining and economical uncertainties.

1.4 Research Methodology

This study was completed in two stages. In the first stage, the related uncertainties were determined. In the second stage, a financial model was developed to simulate the NPV to estimate the profitability of the deposit.

Dereköy copper deposit exploration was carried out by MTA through a series of drillings. Related data and topographic map were obtained from MTA to estimate the reserve amount. A database is prepared in Micromine, mine design program, with the data set of exploration drillings and the copper reserve was estimated. Conventional polygonal method is applied to check the results obtained from Micromine. The created database includes information about drillholes and topography such as grade, depth, elevation and coordinates of drillholes. Using the created database, the deposit is modeled in 3-D in Micromine environment. Reserve and overburden amounts were estimated with solid model. Grade distribution and average grade of the deposit are estimated using block modeling. The two important uncertainties which are grade and reserve tonnage are defined precisely.

In feasibility studies, Monte Carlo Simulation technique is used in @Risk 4.5.7 environment. Monte Carlo simulation defines the probability distribution for the NPV of Dereköy copper deposit with considering related uncertain variables.

1.5 Outline of the Thesis

This thesis is composed of six chapters. In the first chapter, general information is supplied about the thesis and problem. In the second chapter, a literature survey is given. The literature survey is related with copper, ore reserve evaluation, uncertainty in reserve estimation, risk assessment and risk management and investment analysis. In the third chapter, Dereköy copper deposit and the southeastern part of the deposit are evaluated. In the fourth chapter, a model is developed to evaluate the risk of investment of the deposit using @Risk 4.5.7. In fifth chapter, the results of the simulation and related discussions are explained. In the last chapter, related conclusions and recommendations are denoted.

CHAPTER 2

LITERATURE SURVEY

2.1 General Information about Copper

Copper has been known since 8000 B.C. It has contributed to development of human civilization from Stone Age to now. It can be found as native copper in nature but it generally exists as composite. Primitive human beings managed to create tools with native copper. They hardened native copper to make pots and pans, weapons, tools, etc. It is estimated that melting processes of copper was initially done at Mesopotamia at 3500 B.C. It is determined that some copper mines were operated at Sina peninsula and Egypt at 3800 – 2600 B.C. and Cyprus at 2500 B.C. Cooper mines have been operated in Europe since 1600 B.C., in North America since 1709 A.D., in Chile and Peru since 16th century and Middle Africa since 18th century. Copper mine operation in Anatolia started many centuries ago. It was documented that the copper mine at Ergani was operated by Assyrians and Küre copper mine was operated by Romans. The Ottoman Empire was started to operate Ergani and Murgul copper mines during 1890's (Turkish Republic Prime Ministry State Planning Organization [SPO], 2007).

Copper can be classified in two groups; copper products (concentrate copper, cathode copper, wire-rod and others) and alloys (copper-zinc alloy and others). Alloy of copper-zinc is named as brass. Brass is widely used copper alloy in the form of bar and sheet. There are about 20 types of brass used in different industries. Brass is classified considering the copper content. Alloys of copper with metals, except for zinc, are named as bronze. They are classified according to the type of metal they contain.

2.1.1 Production and Consumption of Copper

SPO (1996) stated that 83% of world copper production is supplied from primary resource, mining, whereas 17% of it is supplied from secondary resources, such as copper junks, waste copper materials. Porphyry type copper reserves supplies 60% of world primary cooper production whereas 25% of it is supplied from sedimentary deposits, and 15% of it is obtained from volcanic deposits.

Copper industry is very important at each county's economy because copper is an essential commodity for many important industry. World copper production and consumption areas of copper are given in Table 2.1 and Table 2.2 respectively. Copper industry is one of the most important industries in Turkish economy with more than one billion U.S. dollar. Turkey's copper consumption is presented in Table 2.3.

Table 2.1 World Copper Production (LME, 2008)

Region	**Percentage, %**
America	41
Asia	31
Europe	21
Oceania	4
Africa	3
TOTAL	100

Table 2.2 Industrial Consumption of Copper (LME, 2008)

Industry	**Percentage, %**
Building	48
Electrical	17
Engineering	16
Light Engineering	8
Transport	7
Other	4
TOTAL	100

Table 2.3 Copper Consumption per Capita in Turkey (SPO, 2006)

Year	**Consumption per Capita, kg**	**Change, %**
1999	3.84	-
2000	4.32	12.50
2001	4.00	-7.41
2002	4.52	13.00
2003	4.62	2.21
2004	5.11	10.61
2005	5.18	1.37

2.1.2 Copper Import and Export of Turkey

It is indicated in SPO report (2007) that Turkey exports the concentrate and blister copper while imports cathode copper which is more than 200,000 tons as semi product. The only smelting plant of Turkey is at Samsun and it produces almost 30,000 tons of copper concentrate annually. This amount meets only 10% of needs of Turkey. The imported concentrate copper amounts in 2004 and 2005 are 29,200 tons and 45,700 tons respectively. Over 1.2 billion U.S. dollar was paid for copper imports at 2005. After the mid of 2003, supply deficit ascended all over the world. Therefore, relatively copper price has increased unforeseen.

2.1.3 Trend of Copper Price in Recent Years

The base metal prices have risen abruptly from low prices in 2000 to high prices in the last couple years (Wright, 2007). Although the decline in house construction reduced the demand for building wire and copper semi products, copper price increased in the U.S. during 2007. Low demand for copper in the U.S. and in the North East Asia has been outweighed by continuing development elsewhere in 2007. Strong demand from China is the main factor, but other developing countries are also reasonably strong about copper demand (International Cablemakers Federation [ICF], 2008).

The copper price went over $8,000 per ton more than one time during 2007, before declining back to roughly $7,000 per ton (LME, 2008). Copper price reached $8,000 per ton in May 2006 (LME, 2008). The lowest level of last two year's copper price as $5669.66 per ton was reached in early 2007 (LME, 2008). Even this low price is much higher than the price level when compared with the previous years. Trend of copper price between January 1990 and September 2008 is shown in Figure 2.1. The monthly copper prices between January 1990 and September 2008 are given in Table B.1 (see Appendix B). Price changes of three important metals are given in Table 2.4.

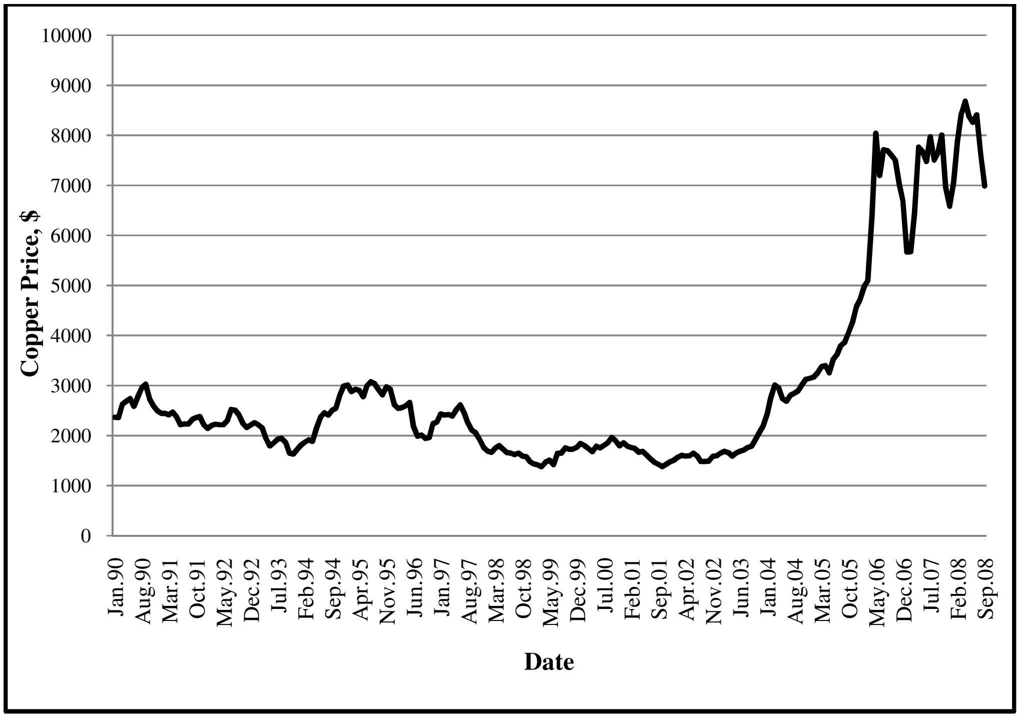


Figure 2.1 Trend of Copper Price between January 1990 and September 2008 (LME, 2008)

Table 2.4 World Metal Prices as Percentage Change between 2000 and 2005 (United Nations, Economic Commission for Africa [UNECA], 2007)

METAL TYPE	2000	2001	2002	2003	2004	2005	2002-2005
Aluminum	13.8	-6.8	-6.5	6.0	19.8	10.6	40.6
Copper	15.3	-13.0	-1.2	14.1	61.0	28.4	135.9
Gold	0.1	-2.9	14.4	17.3	12.6	8.7	43.5

2.2 Ore Reserve Evaluation

2.2.1 Classification of Mineral Resources and Mineral Reserves

Mineral resources are classified considering available geological, technical and economical information about the resource. Mineral resources can be divided into three sub-categories namely: (i) inferred, (ii) indicated and (iii) measured categories. The geological confidence increases in the same order. Indicated mineral resource has higher confidence than inferred category but has lower confidence than measured resources (Canadian Institute of Mining, Metallurgy and Petroleum [CIM], 2000). The details of the sub-categories of mineral resources are given based on Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards (2000) are given in this section. Inferred Mineral Resource's quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. Indicated mineral resource's quantity, grade, density, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. Measured mineral resource's quantity, grade, density, shape and physical characteristics are well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit.

A mineral reserve is the economically mineable part of a measured or indicated mineral resource. Classification of the mineral reserves is done with the same parameters like mineral resource. Mineral reserves can be divided to three sub-categories namely, possible, probable and proven mineral reserves. Proven mineral reserve has higher confidence than probable mineral reserve category. Explanations of the sub-categories of mineral reserve are mainly based on CIM standards (2000).

Probable mineral reserve means the economically mineable part of an indicated, and in some conditions a measured mineral resource confirmed by at least a preliminary feasibility study. Proven mineral reserve is the economically mineable part of a measured mineral resource confirmed by at least a preliminary feasibility study. The feasibility study must include enough information on mining, processing, metallurgical, economic, and other relevant factors that show, at the time of reporting, that economic extraction is reasonable.

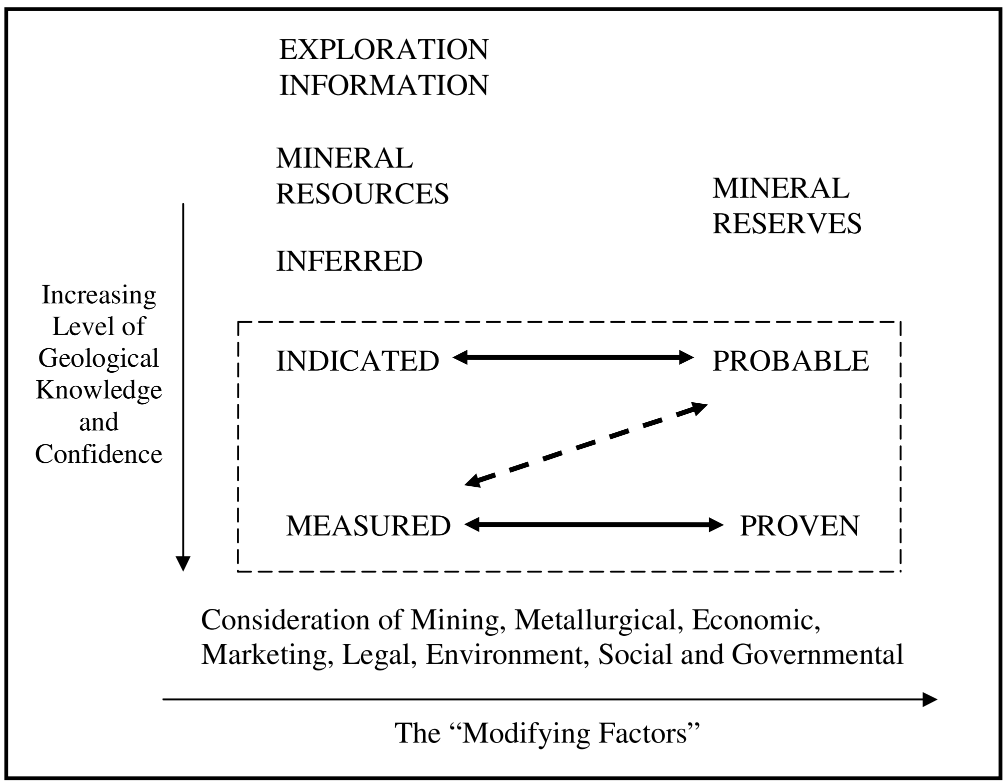


Figure 2.2 Relationship between Mineral Resources and Mineral Reserve (CIM, 2000)

2.2.2 Influence of the Type of Reserve in Investment

The main purpose of mineral deposit evaluation is to determine grade and tonnage of reserve with a certain degree of accuracy so as to minimize the economical risk. Only visible ore can be used as a reliable basis for medium and short-term planning, production scheduling and operation. Possible ore can affect only exploration and company's long-term strategic policy. Long-term decision making in the mining process is based on probable ore reserves.

2.2.3 Mineral Reserve Estimation

Reliable and accurate estimation of ore reserves is very important for determining the feasibility of mineral deposits. Reserve estimation should be started with exploration and continued throughout the life of the mine. Some technical and economical parameters should be applied at reserve estimation (Dominy *et al.*, 2002). An engineer can make reserve estimation with applying some special calculation techniques. These techniques include the estimations of average grade, extent and the amount of ore. Using these estimations, it is possible to determine whether mining is feasible or not.

Two general types of estimation methods are used in the ore exploration stage, namely conventional methods and weighting methods. The main principal of conventional methods is using the area of influence principle included area or excluded principle in estimations of the tonnage and grade. These methods utilize a limited number of samples in assigning a grade to an area or a volume. The weighting method consists of assigning mathematical weights to numerous surrounding samples to provide a statistically more accurate estimation of the grade in a given area or block.

The main consideration for the weighting method is that the influence of any given point in determining the grade of an area or block is a function of hole distant of sample from the area or block to be estimated. The assigned weight is reduced as the samples used for estimation get away from the block to be estimated. There is no defined limit about how many samples can be used for single block estimation in these methods. The weighting method uses more data for estimation. Therefore, weighting methods are statistically better than the conventional methods and used in most reserve estimation problems (Hartman and Mutmansky, 2002).

One of the members of second category is inverse distance weighting method. Inverse distance grade estimation method is mostly used because it is a simple estimation method with better accuracy. The main principle behind the inverse distance weighting method is Tobler's first rule of geography (Tobler, 1970). Tobler (1970) denoted that ''*everything is related to everything else, but near things are more related than distant things.*" Inverse distance weighting method gives a higher weight to samples in the immediate closeness to determine the assigned grade of block on a statistical base. Inverse distance squared, cubed, fourth and fifth can also be used. Inverse distance weighted method is used mostly because of better reconciliation with production (Jewbali and Mausset-Jones, 2002). The main advantages of inverse distance method are its simplicity. It produces a trend map which contains bull-eye effect, and giving a first indication immediately.

2.2.3.1 Conventional Methods

2.2.3.1.1 Cross-Sectional Method

Cross-sectional method is a collection of parallel or nonparallel sections. They dissect the orebody. These sections, together with the grade and physical properties contain the geological information of the deposit along the section lines. A model is built essentially by connecting sections to one another by linear interpolation and assuming a gradual change from one section to the next or by extending each section halfway to the next, reflecting a sudden change in the deposit (Badiozamani, 1992).

2.2.3.1.2 Polygonal Method

Polygonal reserve estimation method is based on geometrical method. The sample value of the hole is extended halfway to any adjacent hole. In this way an area of influence is defined within which the drillhole value is assumed to be valid for this part of the deposit. In the polygonal method, an area of influence of drillhole at the center of the polygon is assigned to whole polygon area. The polygon construction procedure based on the theory that the influence of a hole extends halfway to the next adjacent hole is illustrated in Figure 2.3. Areas then measured by planimeter or calculated analytically and subsequent calculations tabulated in finding tonnage and grade (Hartman, 1987).

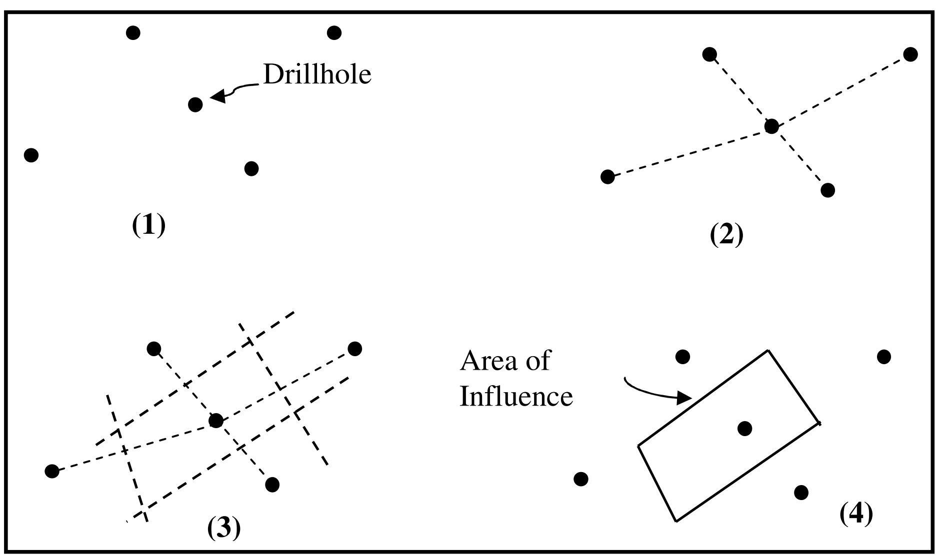


Figure 2.3 Steps of a Polygon Construction (Hartman, 1987)

2.2.3.1.3 Triangular Method

The triangular method is obtained by the modification of the polygonal method. The difference between triangular method and polygonal method is that three drillholes are used for constructing each triangle. Rule of gradual change is basis of triangular reserve estimation method. The key point of this method is that there is a gradual, linear and continuous change between the sample points. The advantage of the triangular method is that the data from three vertices are used to estimate the grade and thickness for the triangular prism. Calculations go on in the same manner as for the polygonal method (Hartman, 1987).

2.2.3.2 Ore Reserve Estimation with Software Packages

Nowadays, some software packages are used in exploration, drilling, mine design, mine operation, reclamation, accounting and marketing stages of mining. Badiozamani (1992) claimed that the main objective of computer application is to simplify the process of data collection, retrieval, analysis, and modeling. In addition, computer usage for modeling detects and removes undetected errors in calculation within shorter interval than the manual methods. As a result, computer modeling assists in achieving the ultimate aim of mine planning.

The optimum mining processes can be possible with maximum profit. In order to maximize the profit, the cost must be minimized and ore recovery must be maximized and revenue must be maximized. To minimize the cost or maximize the revenue, evaluation of multiple alternatives must be done in a short period of time and have the ability to modify the assumptions used, based on new sets of information. Computerized mine planning is an appropriate way to accomplish both objectives, that is, to allow evaluation of multiple alternatives in a short period of time and to be able to change the assumptions quickly (Badiozamani, 1992).

Computer applications in mining can be categorized under two headings. The first one includes software packages that provide basic analysis of the data and allow a design generation analogous to the manual method. The second category consists of software packages that provide simulation, modeling and optimization capabilities beyond the reach of the classical methods. Early stage of computer utilization in mining industry is concentrated on automating classical methods to decrease the unobserved errors and increase the efficiency of estimation. During the time, the automated methods have increased, along with the complexity of ore bodies, so did the need for more sophisticated techniques. Geostatistics, estimation methods and complex mathematical modeling provide ore reserve approximation techniques and simulation approaches that are basically impractical to perform manually (Badiozamani, 1992).

2.2.4 Working Algorithm of the Mine Design Software Packages

Software packages are used for many important stages of mining such as storage, retrieval, analysis of geologic data, design and simulation. Micromine, Surpac, Vulcan and Datamine are some of the important mine design software packages. They can be used for mining and geology projects. The capabilities of the packages range from data import to mine design. The most important feature of them is their ability to convert raw data into usable information. If user inputs some irrelevant and not valuable data, this problem will be solved by the software packages with using their data validation and processing tools.

There are seven main steps in the application of mine design software packages. These steps are entering geological data input to software, data verification, visualization, selecting estimation model, grade and reserve estimation and constructing pit or underground mining components.

2.2.4.1 Geological Data Input, Data Verification and Visualization

The initial task for geological data input is to organize the geological data in categories, due to the fact that each set of data is handled differently. The basic categories are grade, lithology and coordinates of the drillholes.

Geologic data input requirements vary depending on the commodity of interest, the complexity of the geologic conditions and the size of the property. As a general rule, the more complex the geology, the more information is required. The geological data input needs a great deal of attention because the input determines the usability of the model (Badiozamani, 1992).

After loading the data into the database, various validation reports should be generated and reviewed for accuracy of input data and for detection and elimination of invalid data. Accuracy is very important at mining industry because a small error can cause a significant economic loss.

The software packages indicate the validation of results as a report. User can detect each error with using this report. Data validation can also be done by drillhole validation and 3-D viewer. The other data verification is generating contour maps. If some bull-eyes are detected in these contour maps, it is highly probable that there is some invalid data in the database.

The visuals make results of the software estimation more understandable. The visuals can be created with the raw data. The visualization applications are points, strings, outlines, wireframes, and digital terrain model (DTM). Usages of these symbols are practical because they are perceptible on the screen or hard copy. Strings are used for create a line so string is suitable for digitize topographic maps, pit design, *etc*. Outline is used to determine the boundary of reserve. Drillholes indicate the length and lithology or geologic formation of drills. Some data can be converted to 3-D visuals by using wireframe. Wireframe is used to design topography, pit and underground design. The basic point of creating a wireframe is triangulations between points. 3-D modeling can also be generated with DTM format.

2.2.4.2 Selection of Estimation Model

The computer models and the estimations allow both evaluation and the economic analysis of these resources. Therefore, selection of estimation model is very important. Badiozamani (1992) stated that there are basically three main modeling techniques and a few other approaches that do not meet the strict definition of modeling. For example, polygonal estimation should not be regarded as a modeling. The three major modeling processes are the grid model, block model, and cross-sectional model (solid model). Each of these models is used for specific conditions and specific mining operations. Selection of unsuitable model for an orebody may result in imprecise results.

i. The grid model is normally used for bedded deposits such coal, phosphate, limestone, oil shale and sands.

ii. The cross-sectional model (solid model) is commonly used for complex folded and faulted or steeply dipping deposits.

iii. The block model is usually used for disseminated deposits such as porphyry copper, uranium, gold, and other nonstratabound deposits. Block model is good for estimating grade and other properties of the deposit. Many geostatistical estimation techniques can be applied with block model easily. In block model, each block is identified by the X, Y, and Z coordinate at the center of the block and contains the percentage values for each value of interest.

There are two other variations to the conventional block modeling technique namely variable block model (VBM) and variable zone model (VZM). In a variable block model, X and Y dimensions are fixed and Z dimension is variable to better approximate the geologic variability of the deposit. However, even though the Z dimension may change from block to block, they form rectangular blocks of different size. In the variable zone model, Z dimension may vary for each side of the block, resulting in blocks of trapezohedron shapes. Selection of suitable block size is vital in generation of representative models. This process is a balancing act between generating blocks that are not too far apart so that they mask variations in the data, or too small a spacing that requires extensive time and resources, or too many blocks or grids that exceed the program limits. The block size selection is more an art than a science (Badiozamani, 1992).

2.2.4.3 Grade and Tonnage Estimation with Software Package

Users can apply different type of geostatistical methods to estimate the grade of an ore body or calorific value of a seam with the aid of mine design software. Grade estimation can be done with software packages more precisely than manual methods because some complex geostatistical applications can be applied without calculation errors. The software packages can conclude the grade estimation within a small time. This is also an advantage of utilization of computer for grade estimation.

Badiozamani (1992) indicated that the reserve calculation is the simplest and yet the most elusive part of any related computer program. It is the simplest because it is a multiplication and summation processes. There are many intricate points that need to be considered to estimate an ore reserve accurately. Therefore, reserve estimation is a complex process. Reserves can be calculated by various techniques, that is, by a simple polygonal method where the area of influence around each drillhole is multiplied by the thickness of the unit to find volume, then multiplying by average density to find tonnage or the percentage grade. This method of volume calculation is not very accurate, and it is only used when an order-of-magnitude calculation is intended or when very dense drilling is available. Therefore, polygonal method generally is not used in software. Another method of reserve calculation is double-end area calculation from cross sections. This is the same process as the manual method; only a digitizer is used to expedite the calculation. An automatic cross-sectional calculation approach is used for steeply dipping deposits, which is basically the most accurate volumetric method for such deposits. Cross-sectional method can also be applied the other types of deposits to find volume of the deposit.

2.2.4.4 Surface/Underground Mine Design

Mine design software can also create surface and underground mine design. In surface mining, benches and berms can be created easily with giving angle of bench, height of bench, width of berm, *etc*. Shafts, drifts, faces, *etc*. can also be designed.

2.2.5 Dilution and Recovery

Dilution is a term which refers to any waste material within the mining block, involving barren and sub-grade rock and backfill. Increase in ore dilution results in a decrease of grade in comparison with the mining reserves. The global competitive market forces mining companies to maximize production and increase revenue. Unplanned ore dilution has a direct and large influence on the profitability of a mining company. The economic impact of dilution is due to costs related with mining operations and mineral processing such as mucking, haulage, crushing, hoisting, milling, and treatment of waste or low-grade rock having little or no value, displacing profitable ore and processing capacity. Dilution is a qualitative parameter that enables the mine operator to evaluate quality of design. Traditionally, the mining industry has used the dilution idea to describe the negative differences between forecasts and production results. Dilution and sacrifice of ore tend to be inseparable factors, with a trade-off between optimum recovery and impairment of grade (Henning and Mitri, 2007).

There are four basic dilution types that the mining companies encounter. (i) The mine call dilution, which is simply a correction factor applied to the mine-estimated production grade to arrive at the mill-reported feed grade. (ii) The contact dilution or external dilution, which is low-grade material taken accidentally during mining. (iii) The internal dilution, which is waste material so ingrained within the ore that physical separation is not practical. (iv) The Murphy dilution, which is caused by mining errors (Pincock Allen and Holt [PAH], 2004).

Recovery is the opposite side of the dilution. The meaning of the recovery is that the expected material from the model is not recovered in production. Recovery is as important parameter as dilution. Assumption of the recovery needs a good geological knowledge. There is a strong relationship between dilution and recovery. More ore can be recovered with higher dilution. Conversely, selective mining can be applied to reduce the dilution but selective mining increases the value of recovery. Because of these reasons, optimum point should be estimated to increase the portability of the orebody.

2.2.6 Grade/Tonnage Relationship

There are two main types of grade and tonnage models, the models deal with grades and tonnages of samples or blocks within deposits, and the models which use tonnages and average grades of whole deposits as samples. The first one primarily designed for ore reserve and economical analysis within deposits, while the second type of model is designed for evaluation and comparison of groups of undiscovered ore deposits (Donald, 1996).

The classification process of ore and waste material during mining operation is sensitive to whether the cut-off grade is based on sample grades or on recoverable block grades. This is due to the regression effect and results in some real waste blocks being accidentally classified as ore and some real ore blocks being classified as waste. Therefore, so as to improve this misclassification error ore and waste should be classified in accordance with recoverable block estimates. This will make possible a decrease in the variance of estimation and hence to less potential misclassification (Thomas and Snowden, 1990).

When attempting to reconcile exploration estimates with grade control estimates and finally with true head grades and production tonnages, it is important to recognize the relationship between grade and tonnage. For example, considering a hypothetical orebody represented by a bulk kriged block model, if the tonnage estimations are plotted against the estimated grade for a variety of cut-off grades, the resultant curve is called a grade/tonnage diagram. In other words the relationship between tonnage and grade at different cut-offs is illustrated by grade/tonnage diagram. If a more selective estimation for the same deposit is plotted on the grade/tonnage diagram, the resulting curve will be above the bulk block curve and this plotting shows the change in tones and grade estimated for each cut-off grade. If the selectivity is increased, tones may either decrease, or remains constant or increase but grade of the deposit does not decrease. It remains constant or increases. Therefore, a line joining a specific cut-off grade on the bulk curve to the same cut-off grade on the selective curve will show the direction of change in tones and grade with increasing selectivity for this cut-off grade. For example at higher cut-off grades, both tones and grade may increase with selectivity while at lower cut-off the grades may increase while the tonnage decreases. It can be then possible to plot on the same curve the estimates derived from comparable grade control data, either by manual allocation of blasthole assays or by kriging. Thereby, the difference between blasthole and selective mining unit size estimations may be compared in order to determine the effective mining cut-off grade and these estimations may be reconciled with bulk and selective exploration models. Finally, the real production data should be plotted on the diagram due to investigate how robust the models are and what level of selectivity is actually being achieved (Thomas and Snowden, 1990).

2.3 Modeling the Uncertainty Reserve Estimation, Risk Assessment and Risk Management

2.3.1 Risk and Uncertainty in Mining Industry

The usage of the term "risk" as a synonym for uncertainty is not right because their definitions are not the same. Ross (2004) stated that risk (or chance) can be described as the probability that a discrete event will or will not occur. Risk is denoted by single probability estimation. For example, the chance of discovery of copper resource with the exploration drillings is 30%. In contrast, uncertainty denotes the inability to estimate a value exactly, for example, the future price of copper, for the remaining reserve amount. Uncertainty can be denoted by a continuous distribution that defines a range of estimates and the likelihood of occurrence of event. Risk and uncertainty can be used as a combination, e.g. there is a 70% probability that the reserve amount lie between 80 and 150 million tons.

Snowden *et al.* (2002) described the risk in two categories namely objective risk and subjective risk. When the risk is modeled by some mathematical model, the risk is categorized as objective risk. The risk is categorized as subjective when personal judgment alters the perceived risk. Snowden *et al.* (2002) indicated that decision makers and engineers must have much information about the potential risks and opportunities that exist in mining projects. At the feasibility stage of the project, the most likely scenarios should be considered. The scenarios' upside and downside cases should also be tested to determine their effect on economic decision of mining projects. The authors also expressed that communication and compilation of all related mining uncertainties and the uncertainties' likelihood and distribution of occurrences is very important to obtain reliable results for better decision making.

There is a risk at the estimation of grade and reserve amount. Therefore, the outputs should be given as most likelihood and the occurrence interval as a distribution due to the significance of the uncertainties of reserve amount and deposit grade in mineral deposit valuation. Each uncertainty has a risk and the risks should be evaluated using frequency distributions. The distributions can be generated using historical data and engineering judgments.

The uncertainties and the risks should be managed properly to prevent the money loss and the opportunities loss. Risk management is optimizing the level of received risks. This optimization can be done using simulation which estimates the mining risks and their occurrence probabilities. Managing of the risks related to mining can be managed with the aid of the output of the simulation. Snowden *et al.* (2002) indicated an example that if the upside at Sunrise Dam in Western Australia had not been considered, it may never have been mined. The company produced 60 percent more gold than estimated value. Managing risk does not mean minimizing risk because this may result in loss of opportunities. However, there are so many mines where planning has been applied on the basis of the most optimistic estimates but at the end the companies encountered financial disaster. For example, Morley *et al.* (1999) indicated that the 70% of small mining companies in South Africa was mainly failed during the 1980's just because of over estimation of the reserve tonnage and grade.

2.3.2 Sources of Uncertainty in Mining Industry

Mining industry is a very risky industry when compared with the other industries because the decision maker must consider so many uncertain inputs in mining industry. The uncertainties begin with exploration and continue up to end of mine life. The decline and increase of value of each uncertainty affects to the profitability of the mining projects. The uncertainties have an important impact on project investment decision. Identifying the potential sources of uncertainties is very important to get accurate results. Therefore, each uncertainty and their impacts on the project should be analyzed carefully. The main sources of uncertainties are reserve amount, grade distribution of orebody, selling price of the product and density of in-situ reserve.

2.3.2.1 Uncertainty in Geology and Reserve Estimation

Reserve estimation is the process that defines which part of the resource can be economically extracted (Morley *et al*. 1999). The importance of the reserve estimation on the value of the mining project is emphasized by several researchers (Dimitrakopoulos (1998), Yamamoto (1999), Morley *et al* (1999), Snowden *et al.* (2002), Dominy *et al.* (2002), Rendu (2002), Ross (2004) and Emery *et al.* (2006)).

Dominy *et al.* (2002) and Morley *et al.* (1999) indicated that mineral resources and ore reserve reports generally contain a single tonnage and grade values. The tonnage and grade values do not contain any reference to the potential uncertainties in the estimations.

"Any resource and reserve estimation is guaranteed to be wrong. Some however, are less wrong than others" (Morley *et al*. 1999). Ore amount is the only input as money for mining companies. Therefore, if the estimation risk of the reserve amount is low, the variance of NPV reduces.

The main points of reserve estimation are estimation of volume of the orebody and then multiplying the estimated volume with the estimated in-situ density. Estimation of the reserve boundary is also important to find the volume of the orebody precisely. Geological boundaries of mineral reserves may not be well decided and boundary of the reserve is an uncertainty of reserve estimation caused by lack of information. For example, porphyry copper or disseminated gold orebodies' boundaries are poorly known and the boundaries are decided by mineral grade rather than by any particular geological property (Dominy *et al*., 2002). Density of the whole orebody is determined using the results of the drillholes but the drillhole samples are not indicative of the physical characteristics of the large mass. Volume of the orebody can be defined well but the tonnage value is computed using average density. Therefore, density of the orebody is one of the important uncertainties.

Snowden *et al.* (2002) stated that in mining, the main risk source is the orebody because knowledge of the orebody is based largely on estimates. Different authors express the importance of the variability of the reserve. Variability of an ore reserve can significantly affect and alter the critical and important decisions.

Geologic conditions can be described as "the state of nature". These conditions can not be changed and there is limited information about them because "the state of nature" is poorly known. Geologic uncertainty can be reduced with getting more and better information (Rendu, 2002).

2.3.2.2 Uncertainty in Grade Estimation

One of the important uncertainties in the estimation of mining deposit is the grade of the orebody. Dominy *et al.* (2002) stated that engineers recognize grade estimation as challenging. Selecting an appropriate model is not enough to estimate the exact value of reserve grade because there are two uncertainties remain, which are sampling error and estimation error.

Risk assessment of orebody grade is important in determining the metal content of the reserve. The potential errors can be evaluated by the kriging variance or a simulation method. Ordinary kriging is one of the popular estimation methods. Ordinary kriging can estimate the grade of a block with using the neighbor samples of the block. Therefore, an estimated grade value and the error involved can be estimated as kriging variance for each block (Yamamoto, 1999). However, Yamamoto (1999) and Dominy *et al.* (2002) discussed that simulation technique is better than kriging because of two reasons. The first reason is that the kriging variance can not recognize the local grade changes. Local grade variability is an important issue to estimate the grade of the heterogeneous reserves with richer and poorer ore being estimated. The second reason is that kriging application can produce a single set of estimated block grades while conditionally simulation can simulate set of possible sample grades of orebody. Size of the blocks can also be re-blocked easily into meaningful block sizes and shape.

2.3.2.3 Uncertainty in Cost Estimation

Specific assumptions are needed to estimate the cost. The closeness of estimated cost and actual cost indicates the quality of the assumptions and the estimator's expertise. Contingency cost is added to minimize the cost risk. The uncertainty of project determines the magnitude of the contingency. The risk of capital and operating risks can be reduced by administrative controls. The controls are judicious choice of the engineering, technical and financial supervision, appropriate project management and contract structures. When the amount of capital cost is determined, the relation between capital cost and operating cost should be considered. There is often an inverse relation between them. The optimum point should be taken because higher amount of capital cost means higher financial exposure during the early stages of the project (Rendu, 2002).

2.3.2.4 Uncertainty in Price Forecasting and Revenue Estimation

There is a strong relationship between price forecasting and revenue estimation. If the price is estimated precisely, revenue estimation can also be done precisely. Rendu (2002) indicated that the most important risk factor is the lack of knowledge of the future price of product of mining. Millions of dollars are spent to forecast future prices but most of them results with a poor success. There are two approaches to estimate the future price. They are technical and fundamental approaches. The technical approach consists of analyzing historical prices, studying long-term trends and short-term variability and developing a statistical model. The fundamental approach consists of forecasting supply and demand. The most accurate price forecasting can be done using combination of technical and fundamental approaches.

2.3.3 Estimation of NPV of Ore Reserves Considering Mining Risks

Estimation of the value of an orebody is not an easy process. There are many uncertainties to estimate the value of an orebody. There are many critical nodes of uncertainty along the mine value chain, starting from the orebody estimation to the mineral production. The uncertainties affect the estimated value and they compose the value chain. Therefore, the inputs should be analyzed to optimize the overall mine process. Optimization of the value chain must be done starting from the beginning to the end process to identify the high-risk areas and remove their impact on the maximization of the profit. Evaluation of the value chain is inter-disciplinary process. The inter-disciplinary components of value chain are geology, geomechanical, mining and metallurgical engineering. They are related to each stage from exploration through feasibility study, to grade control, mining processing and marketing (Snowden *et al*. 2002). A simplified example of mine value chain process nodes of uncertainty is illustrated in Figure 2.4. Two examples of mine value chain for estimation of NPV are presented in Figure 2.5 and Figure 2.6.

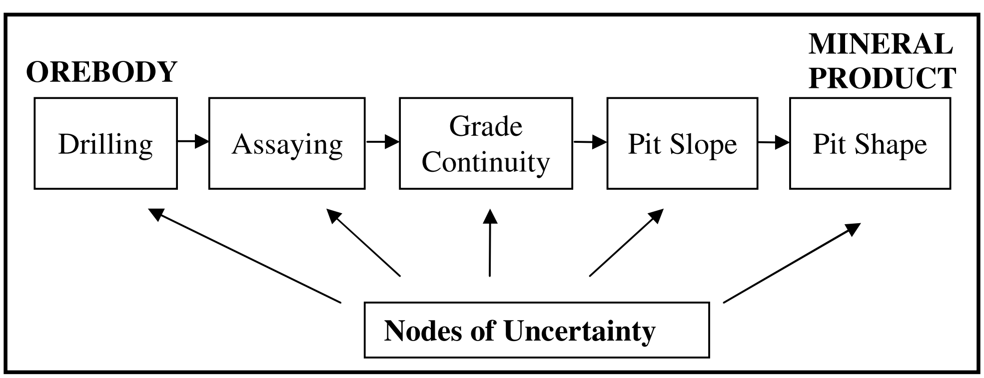


Figure 2.4 Simplified Mine Value Chain Illustrating Nodes of Uncertainty (Snowden *et al.*, 2002)

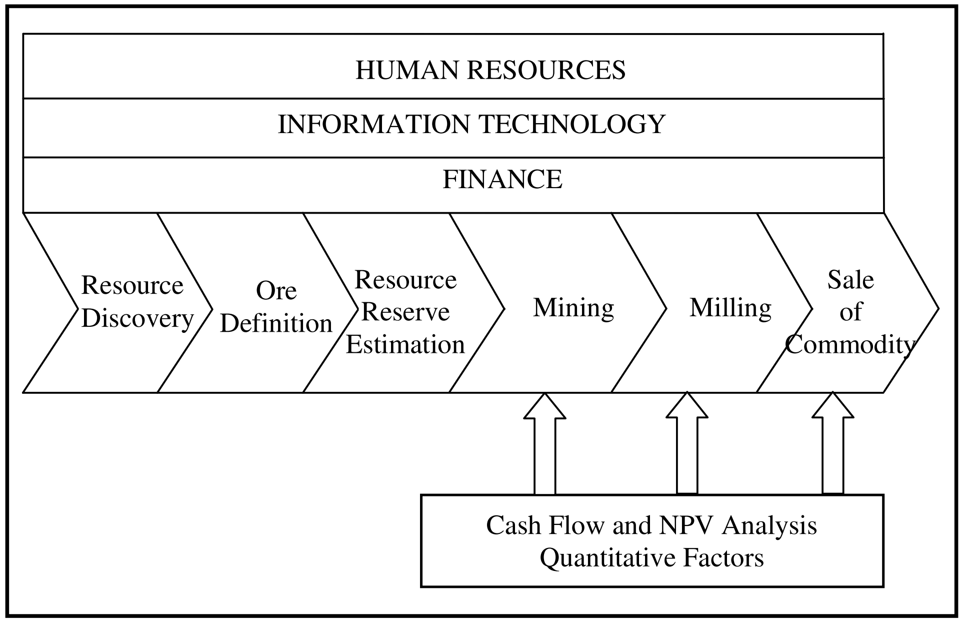


Figure 2.5 Mine Value Chain (Sample 1) (Morley *et al.*, 1999)

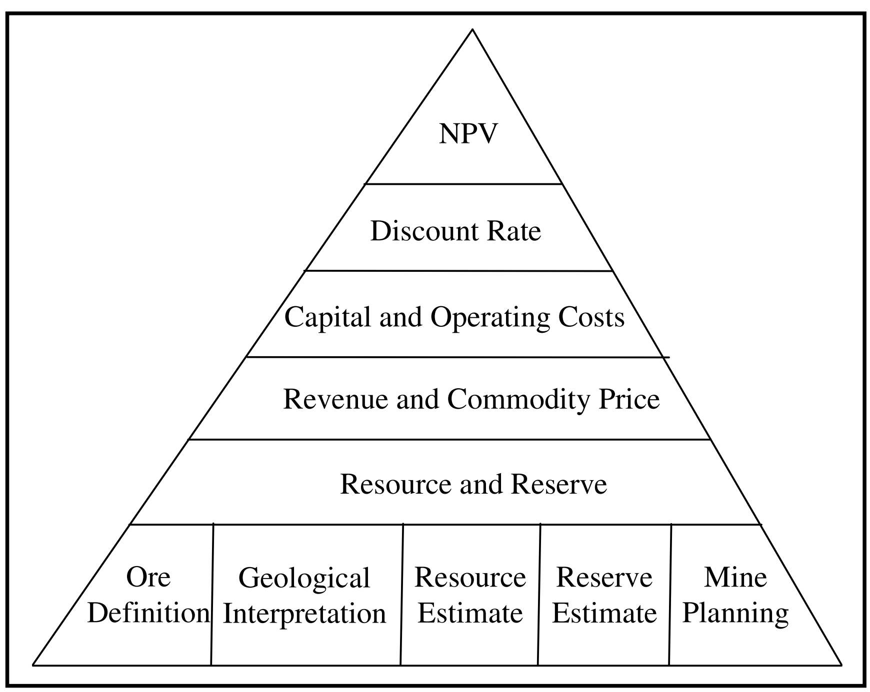


Figure 2.6 Mine Value Chain (Sample 2) (Morley *et al*, 1999)

2.3.4 Uncertainty Assessment using Monte Carlo Simulation

The simulation procedure consists of generating random numbers according to assumed probabilities associated with a source of uncertainty such as selling price, interest rate and grade of the orebody. The estimated outcomes related to the random drawings are then analyzed to determine the possible results and associated risks. Monte Carlo simulation technique is widely used for dealing with uncertainty in many aspects of operations (Chance, 2008).

Investigation of stochastic permutations of uncertainties is done and unbiased and consistent estimation can be obtained using Monte Carlo simulation. The principle behind Monte Carlo simulation technique is summarized as a sequence of the following steps:

i. Establish distributions for each input or independent variable.

ii. Set up equations which will allow the calculation of the independent variables. This can be done by determining expressions for the integrals of probability distributions. It should be recognized that the integral of a probability distribution is the cumulative frequency of the distribution.

iii. Generate random numbers for each independent variable.

iv. Use the random numbers to calculate values for the independent variables.

v. Calculate dependent variable and store the result in a class interval.

vi. Return to step 3 and repeat 3 through 5 many times.

vii. Construct relative and cumulative frequency diagrams (Eriçok, 2004 and Rezaie *et al.*, 2007).

The number of iteration is determined regarding the project size and the importance of risks. The number of iteration can be set as 1000, 2000, 5000, and so on. It could be said that while the number of runs increases, much more stochastic scenario are searched in the solution space. In other words, the higher number of runs gives the more accurate results. In each run, a stochastic value is allocated to each uncertainty in the range of its lower bound and upper bound. The frequency of each value is followed by determined distribution function (Rezaie *et al.*, 2007).

2.4 Investment Analysis

The purpose of an investment analysis is to determine whether the investment is profitable or not. An investor must know the profitability and estimated profit from the investment. The base point of the investment analysis is time value of money.

2.4.1 Time Value of Money

It is important to recognize the concept of time value of money, when items of an alternative which can be quantified in terms of cash. It is generally said that money makes money. This statement is indeed true, for if an investor selects to invest money today (for example, in a bank or savings and loan association) by tomorrow the investor will have accumulated more money than the investor had originally invested. The change in the amount of money over a given time period is called the time value of money. This concept is most important in engineering economy (Blank and Tarquin, 1989).

Time value of money is also explained with an example here. It is assumed that an investor invests his $1,000 to a bank with 5% annual interest rate can earn an 8% annual interest. Then, the investment will be worth $1,276 in five years as indicated in Table 2.5.

Table 2.5 Time Value of Money

Year	Value Begining of Year	Interest Rate		Annual Interest	Amount at End of Year
1	$1,000	5%	=	$50	$1,050
2	$1,050	5%	=	$53	$1,103
3	$1,103	5%	=	$55	$1,158
4	$1,158	5%	=	$58	$1,216
5	$1,216	5%	=	$61	$1,276

2.4.2 Interest Rates

Interest and interest rate are different terms. Interest is the difference between originally invested money and the final accrued money. Interest is calculated as equation (2.1). Blank and Tarquin (1989) explained that when interest is expressed as a percentage of the original amount per unit time, the result is an interest rate. This rate is calculated as equation (2.2). Time value of money and interest rate are evaluated together to generate the concept of equivalence. The mean of the equivalence is that different sums of money at different times can be equal in economic value. For instance, $1,000 at present would be equivalent to $1,050 one year from now, if the interest rate is 5% per year.

$$Interest = Total\ Amount\ Accumulated - Original\ Investment \tag{2.1}$$

$$Percent\ Interest\ Rate = \frac{Interest\ Accured\ per\ Unit\ Time}{Original\ Amount} \times 100\% \tag{2.2}$$

There are two type of interest which are simple and compound interest rate. They should be considered when more than interest period is involved. Simple interest is calculated using the principal (present value) only. It ignores any interest that was accrued in preceding interest period. Simple interest rate is calculated by equation (2.3). In contrast simple interest, compound interest is the interest for an interest period is calculated on the principal plus the total amount of interest accumulated in previous periods. Thus, compound interest means "interest on top of interest". Compound interest reflects the effect of the time value of money on the interest too. Future worth (F) and principal (P) are the basic terms in compound interest calculation. The calculation of F is presented in equation (2.3) and equation (2.4) (Blank and Tarquin, 1989).

$$Simple\ Interest = Principal\ \times Number\ of\ Periods\ \times Interest\ Rate \tag{2.3}$$

$$\begin{cases} \text{Year 1:} & F = P + (P + i) = P\ \times (1 + i) \\ \text{Year 2:} & F = P \times (1 + i) + \ \{[P \times (1 + i)] \times i\} = P \times (1 + i)^2 \\ \text{Year n:} & F = P \times (1 +\ i)^n \end{cases} \tag{2.4}$$

2.4.3 Cash Flow Diagram

A cash flow is the difference between total cash received and total cash disbursements for a given period of time for example one year. Cash flows are very important in engineering economics because they form the basis for evaluation projects, equipment, and investment alternatives. The easiest way to visualize a cash flow is through a cash flow diagram in with the individual cash flows are represented a vertical arrows along a horizontal time scale. Positive cash flows, which are net inflows, are represented by upward-pointing arrows and negative cash flows which are net outflows, by downward-pointing arrows. The length of an arrow is proportional to the magnitude of the corresponding cash flow. Each cash flow is assumed to occur at the end of the respective time period. Net cash flow can be calculated with equation (2.5). (Güyagüler and Demirel, 2008). A simple cash flow is represented in Figure 2.7. The cash flow is established for eight years. The value at present (0^{th} year) is capital cost as \$5,000,000. The investor get profits at first, second, forth, sixth and eighth years. In engineering economy there are some different cash flow types, namely uniform series of cash flows, gradient series of cash flows and geometric series of cash flows.

$$Net\ Cash\ Flow = Annual\ Revenue - Annual\ Expenses \tag{2.5}$$

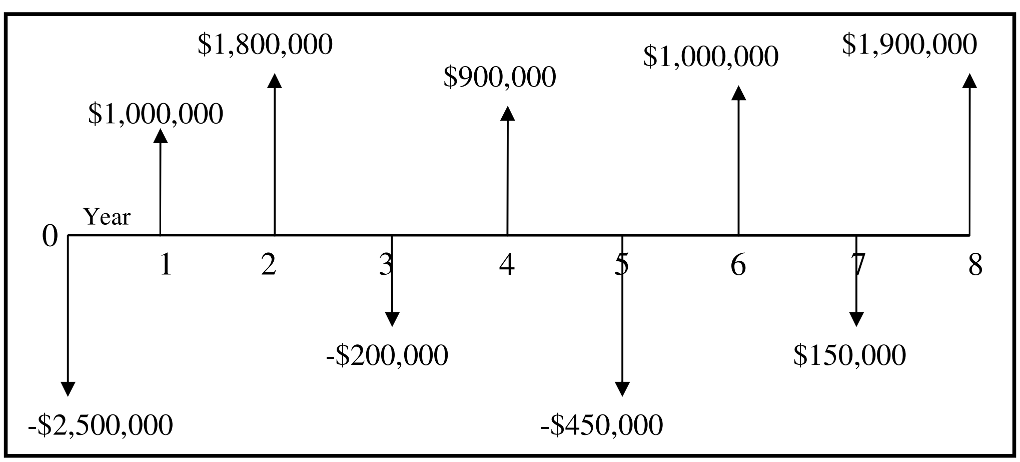


Figure 2.7 Cash Flow Diagram

2.4.4 Investments Analysis Techniques

Some techniques are applied to economic analysis of the projects. These techniques involve deciding whether or not to realize an investment project or choosing between or among alternative projects. The techniques are related to time value of money and the equivalence calculations. The methods are explained in this section.

2.4.4.1 Net Present Value Method (Equivalent Present Worth Method)

When equivalence is computed relative to time zero or the present, the analysis method is known as the equivalent present worth analysis**.** When the future amount of money is converted into its equivalent present value, the magnitude of the present amount is always less than the magnitude of the cash flow from which it was estimated. This is because for any interest rate greater than zero, all Present/Future factors are less than one. Because of this reason, present worth calculations are often referred to as discounted cash flow method. Similarly, the interest rate used in the calculations of present worth method is referred as discount rate. The important terms in the net present value method are present value (PV), and net present value (NPV) (Blank and Tarquin, 1989). The formulation of the net present value method is presented in equation (2.6) (White, Agee and Case, 1989).

$$PW_j(i) = \sum_{i=0}^{n} A_{jt}(1+i)^{-t} \tag{2.6}$$

In equation (2.6):

$PW_j(i)$ = Present Worth

n = Investment Period

A_{jt} = Net Cash Flow at the End of Period t

i = Interest Rate

Six some steps should be followed in the estimation of net present value (NPV) of an investment. The steps are listed here.

Step 1: Interest rate, or discount rate is chosen. Time value of money is determined by the value of interest rate. When NPV method is used to evaluate different projects, minimum attractive rate of return (MARR) is chosen for interest rate.

Step 2: Calculation of the yearly net cash flow should be done at step 2.

Step 3: Present value (PV) of the annual cash flow is conducted

Step 4: NPV is computed with the summation of PV of the annual net cash flow.

Step 5: The investment is accepted or rejected with considering the amount of NPV of the investment (Boehlje and Ehmke, 2008).

2.4.4.2 Equivalent Uniform Annual Worth Method

When the equivalent annual worth is determined, the method is called as the annual equivalent worth method. In other words, all cash flow values are converted to an equivalent uniform series of cash flows over the life of an investment. To calculate the uniform series of cash flow, present worth (PW) of the investment is calculated and then annual worth (AW) is calculated using PW. The mathematical formulation of AW is given in equation (2.7) (Güyagüler and Demirel, 2008).

$$AW_j = PW_j \times \left[\frac{i \times (1+i)^n}{(1+i)^n - 1}\right] \tag{2.7}$$

In equation (2.7):

PW_j = Present Worth

n = Investment Period

AW_j = Annual Worth

i = Interest Rate

2.4.4.3 Equivalent Future Worth Method

When equivalence is computed relative to the future, or time n, the analysis method is known as the equivalent future worth analysis**.** All cash flows are converted to an equivalent at the end of cash flow or end of the project. To calculate the future worth of an investment, summation of each cash flow's future value is found. The formulation of equivalent future worth method is presented in equation (2.8) (Güyagüler and Demirel, 2008).

$$FW_j = \sum_{i=0}^{n} A_{jt}(1+i)^{n-1} \tag{2.8}$$

In equation (2.8):

FW_j = Future Worth

n = Investment Period

A_{jt} = Net Cash Flow at the End of Period t

i = Interest Rate

2.4.4.4 Internal Rate of Return Method

Internal rate of return (IRR) method is one of the most used investment analysis methods. In IRR method, the objective is to find the interest rate at which the present sum and future sum are equivalent. In other words, the present or future sum of the all cash flows is equal to the zero if IRR value is used as interest rate as seen in equation (2.9) (Blank and Tarquin, 1989). Therefore, it can be understood that IRR value is the interest rate at which the investor recovers the investment. If summation of all positive cash flows are equal to the summation of all negative cash flows, IRR value can be found by equation (2.10) (Güyagüler and Demirel, 2008).

$$FW_j = \sum_{i=0}^{n} A_{jt}(1+i)^{n-1} = 0 \tag{2.9}$$

In equation (2.9):

i = IRR

A_{jt} = Net Cash Flow at the End of Period t

n = Number of Period

FW_j = Future Worth

$$\sum_{i=0}^{n} A_{jt}^{(+)}(1+i)^{n-1} = \sum_{i=0}^{n} A_{jt}^{(-)}(1+i)^{n-1} \tag{2.10}$$

In equation (2.10):

i = IRR

$A_{jt}^{(+)}$ = Positive Net Cash Flow at the End of Period t

$A_{jt}^{(-)}$ = Negative Net Cash Flow at the End of Period t

As mentioned above, IRR value is the recovered interest rate by an investment by investor. Therefore, IRR value must be greater than minimum attractive rate of return (MARR) if an investment is feasible. The minimum attractive rate of return, also known as the minimum acceptable rate of return, is a lower limit for investment acceptability set by companies, investors and organizations. MARR is designed to make the best possible use of limited resources. MARR values vary widely according to the type of organization. Government agencies and regulated public utilities have utilized lower MARR than competitive industrial enterprises. Within a given enterprise, the required MARR may be different for various divisions or activities. These variations often reflect the risk involved (Riggs, Bedworth and Randhawa, 1996). Güyagüler and Demirel (2008) state that every company, or investor, defines a certain MARR value for its investments and evaluates its investment alternatives based on this MARR value. Usually this value is somewhat related to the prevailing financial conditions in the country. A MARR of 15% would be acceptable for stable economy and currency.

2.4.4.5 External Rate of Return Method

External rate of return (ERR) is an investment analysis method where the main appeal is its pragmatic assumption that receipts (positive cash flow) are actually reinvested at MARR. The flaw is that this method does not base the reinvested on project cash flow balances. It is based solely on project receipts. An unknown rate of return (é) is found by equating the future worth of positive cash flow (receipts) compounded at an interest rate (i) to the future worth of negative cash flow (disbursements) compounded at é as illustrated in equation (2.11). When i is the MARR and é exceeds i, it is assumed that the investment is profitable because it promises a yield greater than the lower limit of acceptability (Riggs *et al.*, 1996).

$$FW\ (receipts\ compounded\ at\ i) = FW\ (disbursements\ compounded\ at\ \text{é}) \tag{2.11}$$

2.4.4.6 Benefit/Cost Ratio Method

Benefit/Cost Ratio (B/C) method is used for the evaluation of the governmental projects. Therefore, government agencies are used this method to analyze the desirability of public works. As indicated in name of the method, B/C method is based on the ratio of the benefits to costs related with a particular project. A project is considered to be acceptable when the benefits derived from its implementation exceed its associated costs. In other words, B/C is bigger than 1. Therefore, the first step in this method is to decide which of the elements are benefits and which are costs. In this method, benefits (B) are advantages, express in terms of dollar. When the project contains disadvantages, these are known as disbenefits (D). Costs (C) are the anticipated expenditures for construction, operation, and maintenance. The usually used B/C methods types are conventional B/C and modified B/C methods. The formulas of them are presented in equation (2.12) and equation (2.13) respectively (Blank and Tarquin, 1989).

$$Conventional\ B/C = \frac{Benefits - Disbenefits}{Costs} = \frac{B - D}{C} \tag{2.12}$$

$$Modified\ B/C = \frac{Benefits - Disbenefits - O\&M\ Costs}{Initial\ Investment} \tag{2.13}$$

2.4.4.7 Payback Period

The payback method, sometimes called pay out method, is a simple method used to obtain a rough estimate of the time that an investment will take to pay for itself.

This method is generally applied to relatively small investment proposals that originate from operating departments. The formula of payback period method is given in equation (2.14) (Riggs *et al.*, 1996).

$$Payback\ Period = \frac{Required\ Investment}{Annual\ Receipts - Annual\ Disbursements} = \frac{First\ Cost}{Net\ Annual\ Cash\ Flow} \tag{2.14}$$

2.4.5 Sensitivity Analysis

Riggs *et al* (1996) explained that sensitivity analysis contains repeated computations with different cash flow elements analysis factors to compare results obtained from these substitutions with results from the original data. In other words, Güyagüler and Demirel (2008) and White *et al*, (1989) stated that sensitivity analysis shows the effect of making error in the estimation of some parameters on the worth of an investment.

The purpose of sensitivity analysis is that decision makers should accept some factors (inputs) as certain and then the decision makers should focus on one or more critical factors and investigate what would happen to a proposal as a result of variations in those inputs (factors). If a small change in an input causes a proportionately greater change in the results, the situation is called to be sensitive to that variable (input). Consideration of sensitivity analysis begins preproposal stage and continues to the final acceptance or rejection decision (Riggs *et al.*, 1996). The software allows more than one basis of comparison. For example, PW and rate of return analysis (Blank and Tarquin, 1989).

2.4.6 Break-Even Analysis

It is often necessary to determine the quantity at which revenue and cost will be equal, to analyze or estimate profit or loss. The equivalence of cost and profit is called the break-even point. At beak-even point there will not be profit and loss. Break-even point is calculated using the best estimates of relations for revenue and cost for different quantities (Q). Quantity (Q) may be expressed in units per year, percentage of capacity, hours per month and many other dimensions. Units per year are usually used quantity (Blank and Tarquin, 1989).

Steiner (1996) and Riggs *et al.* (1996) indicated that break-even analysis is a limited form of sensitivity analysis. An investor is interested in determining a set of values for which an investment alternative is justified economically.

2.4.7 Risk Analysis

Risk analysis is defined as the process of developing probability distributions. Typically, probability distributions are developed for either present worth (PW), annual worth (AW) or the rate of return for a single project. Therefore, probability distributions are required for random variables for example cash flows, planning horizon and interest rate. The probability distributions are then aggregated analytically or through simulation to obtain the probability distribution the measure of merit (White *et al.*, 1989).

The cash flow value in a given year is often a function of a number of other variables for example selling price, size of the market, market growth rate, required investment, inflation rate, interest rate, tax rates operating costs, and salvage value of all assets. The values of a number of these random variables can be correlated with each other. Therefore, an analytical development of the probability distribution for the measure of merit is not easily produced in real world situations. Therefore, simulation is widely used in performing risk analyses (White *et al.,* 1989).

Risk analysis is suitable when significant outcome variations are likely for different future states and meaningful probabilities can be assigned to those states. Time is also important factor in the risk assessment because inputs and outputs for short-term investments have less variability then long-term investments. Mathematical models are developed at the end of the assessment for riskiness of the inputs and outputs. These models can assist the investor with suggesting the impact of chance event on economic outcomes (Riggs *et al.*, 1996).

Riggs *et al.* (1996) stated that risk analysis simulation can be improved with better assumptions and developing accurate distributions. The assumptions can be improved by getting more data (better estimates of future conditions and their effects), more reliable information (reduction of biases and inaccuracies) and more analytical experience (familiarity with similar decisions that avoid misapplication of models and misinterpretation of results).

White *et al.* (1989) listed the advantages and disadvantages for using simulation in risk analysis.

The major advantages are:

- Analytic solutions are impossible to obtain without great difficulty.
- Simulation can be used as a verification of analytical solutions.
- Simulation is very versatile.
- Less background in mathematical analysis and probability theory is generally required.

The major disadvantages are:

- Simulation can be quite expensive.
- Simulations introduce a source of randomness not present in analytic solutions.

- Simulation does not reproduce the input distributions exactly (especially the tails of the distributions).
- Validation is easily overlooked in using simulation.
- Simulation is so easily applied it is often used when analytic solutions can be easily obtained at considerably less cost.

CHAPTER 3

A CASE STUDY: UNCERTAINTY ASSESSMENT OF DEREKÖY COPPER DEPOSIT

3.1 General Information about Dereköy Copper Deposit

3.1.1 Location

Dereköy is a town in Kırklareli province and it is close to the boundary of Bulgaria. The distance between Dereköy and Kırklareli is 30 km. Location of Dereköy is shown in Figure 3.1. Accessibility of the location is easy because there are many access roads to the location. An international road pass near Dereköy and there is a custom station at Dereköy location. Dereköy copper deposit is located between Dereköy and Karadere, 25 km far from Kırklareli.

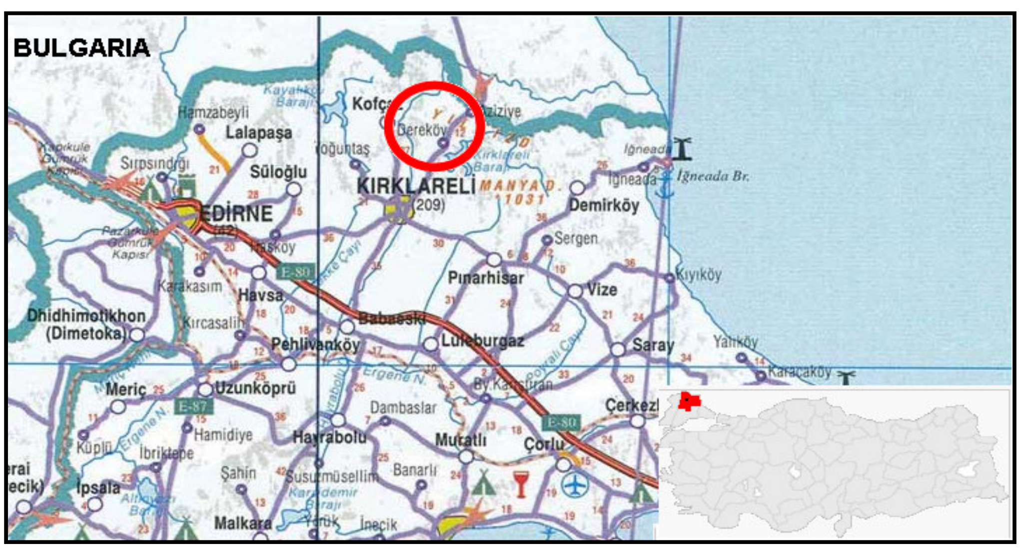


Figure 3.1 Location of Dereköy

3.1.2 Geology of the Site

Dereköy copper deposit is a porphyry type and ore formation is established in Istranca massive. The Istranca massive is underlain by a basement of gneissic rocks unconformably overlain by a cover of paleozoic and mesozoic sedimentary rocks. These types of rocks are composed of shale, limestone and sandstone which are regionally metamorphosed into the greenschist facies. The sedimentary rocks are, in turn, largely intruded by upper cretaceous granodioritic rocks and in part covered by volcano–sedimentary rocks of upper cretaceous age. Many ore deposits and mineralization known from the Istranca massive generally form in the contact zone between the intrusive and sedimentary cover rocks. They are the results of hydrothermal contact mineralization. Disseminated and stockwork types of mineralization occurred by primary ore minerals such as pyrite, some chalcopyrite, traces of molybdenite and scheelite are found close to the contact zone within the granodiorite porphyry; locally, hydrothermal quartz veins containing trace of molybdenite cut the granodiorite porphyry. The more important deposits and mineralization form in some contact zones; they consist of primary ore minerals, the most important of which are magnetite, chalcopyrite, bornite, the fahlerz group, pyrite, pyrrhotite, the Bi-minerals (e.g., bismuth, bismuthinite, emplectite, wittichenite, gladite, tatradymtte), sphalerite, cubanite, valleriite and scheelite in association with such secondary ore minerals as chalcocite, coveilite, malachite, azurite and limonite. The paragenesis, the texture and structure of the ore minerals have been studied in detail (Taner and Çağatay, 1983). The mineral cubanite and valleriite indicate a temperature of formation of 250° - 300°C, elsewhere mineralization generally occurs in some skarn zones where chalcopyrite contains exsolution stars of sphalerite. These phenomena show that these ore deposits and mineralizations have been formed in meso – katathermal conditions (Taner and Çağatay, 1983).

3.1.3 Mineralization of Dereköy Copper Deposit

Dereköy copper deposit is a typical porphyry type copper deposit. Porphyry type copper reserve has two meaning as economically and geologically. Economically, porphyry type copper reserves has average copper grade of 0,8% and geologically reserve amount of more than 500 million tons containing small amount molybdenum, gold, and silver. They can be mined with open and underground mining methods.

Geologically, formation in porphyry copper reserves has to be acidic, porphyry texture, connected originally with intrusive rocks, ore is scattered with small veins and stockwork type ore formation and alteration is observed with ore formation. Porphyry copper reserves are studied in two different categories with respect to their tectonic positions. They are porphyry copper reserves formed in island arcs and porphyry copper reserves formed near continents.

The Istranca massive, which is a part of Sredno-Gora zone, starts from Romania and passes over Yugoslavia and Bulgaria. This zone is important for mineralization especially for porphyry type mineralization. Therefore, there are many porphyry, skarn, and vein type mineralization. Some ancient mines are observed through Istranca massive. There are some slags, whose amount is higher than a few hundred tons, in Istranca massive and also the old Fatih foundry was located at this region. The clues indicate that Istranca Massive is very important for mineralization.

3.2 Exploration Drilling at Dereköy Copper Deposit

The exploration was started at 1981 between Dereköy and Karadere with one drilling to investigate an ore reserve. The exploration was continued with one drilling at 1982, two drillings at 1983, 13 drillings at 1984, and eight drillings at 1985. Up to 1986, totally 25 exploration drillings was conducted with the total drilling length of 8,776 m. Some topographical information about drillhole locations is given in Table 3.1 and locations of drillholes are indicated in Figure 3.2. Wire-line method was applied during the drillings and all drillings were conducted perpendicularly. Drilling cores were divided into two equal parts longitudinally. One of them was sent to General Directorate of Mineral Research and Exploration (MTA) laboratories in Ankara for chemical analysis. Sampling was done at every two meter from drilling cores. Each core was analyzed for the content of copper, molybdenum, gold, silver and wolfram. In MTA Laboratory, atomic absorption spectrometer and optical spectrographic analyzing methods were applied. In the conclusion of macroscopic observations and norm analyzing, content of pyrite changes between 1.5% and 3.5%. Average density of copper deposit and overburden is estimated as 2.7 ton/m^3 (Doğan, 1987).

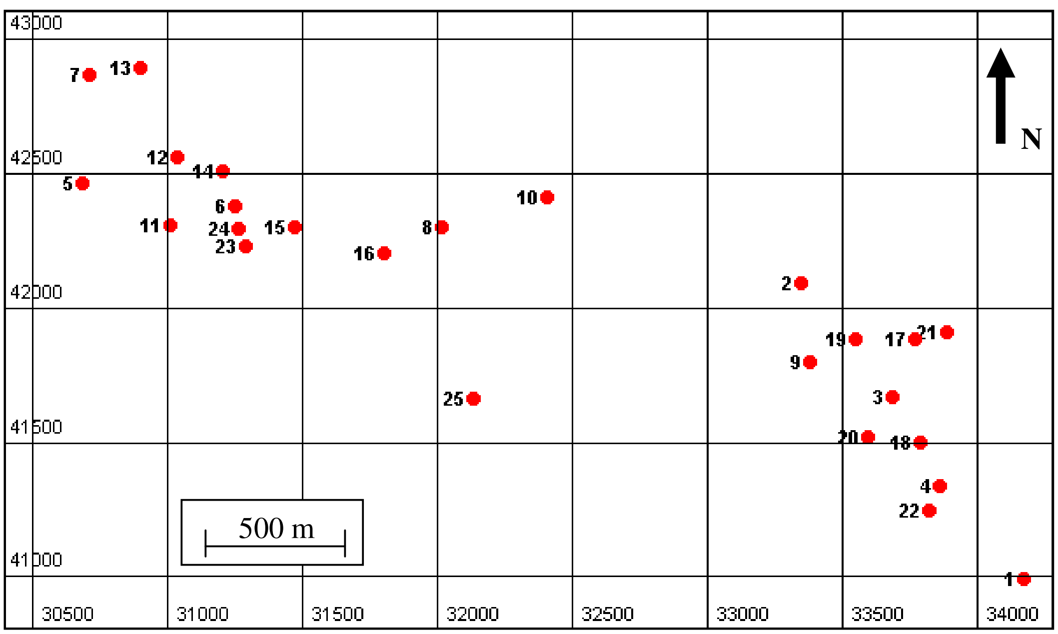


Figure 3.2 Locations of Drillholes

Table 3.1 Exploration Drillings (Doğan, 1987)

DRILLING ID	EASTING	NORTHING	ELEVATION	DEPTH, m
DS1	34176.50	40988.90	466.48	198.40
DS2	33351.80	42086.50	458.75	273.45
DS3	33688.40	41668.10	443.42	241.60
DS4	33866.70	41335.30	445.95	558.55
DS5	30686.20	42462.10	485.00	501.50
DS6	31249.90	42375.50	485.16	614.20
DS7	30712.50	42865.50	477.46	392.60
DS8	32020.90	42293.60	489.27	346.30
DS9	33385.80	41795.40	432.98	309.70
DS10	32406.10	42409.30	473.02	325.50
DS11	31008.40	42305.50	486.68	365.70
DS12	31034.00	42555.30	512.16	500.00
DS13	30901.70	42889.30	487.53	205.95
DS14	31204.20	42504.20	503.08	504.11
DS15	31469.60	42296.30	489.14	350.25
DS16	31804.20	42199.80	462.01	388.80
DS17	33771.87	41877.96	449.93	290.10
DS18	33796.45	41497.98	419.03	352.20
DS19	33552.79	41877.84	449.66	349.40
DS20	33596.21	41516.37	415.60	312.70
DS21	33892.21	41905.35	426.85	260.50
DS22	33826.14	41244.10	434.28	319.20
DS23	31289.14	42222.15	462.49	144.40
DS24	31267.08	42288.10	469.15	184.70
DS25	32135.62	41662.10	449.04	384.10

3.3 Reserve Estimation and Evaluation of Dereköy Copper Deposit

Average metal content is determined using the average grade of the orebody. Therefore, determining the average grade of the orebody is very important step. Two methods were applied to determine the average grade. In the first method, the average grade of the deposit was calculated statistically using the data obtained from 25 drillholes. In second method, the average grade was estimated by an estimation method which is inverse distance method. Then, the results obtained from the calculation and the estimation was compared.

3.3.1 Reserve Calculation Considering Grade Samples

In this method grade is estimated using drillhole data. The cores and sludge of each drillhole were already analyzed in detail at MTA Laboratories in Ankara (Doğan, 1987). In this study, the grades at each 10 m were calculated for each drillhole. First to conduct statistical analysis, grade histogram and frequency distribution is obtained using the data obtained from 25 drillholes as illustrated in Figure 3.3.

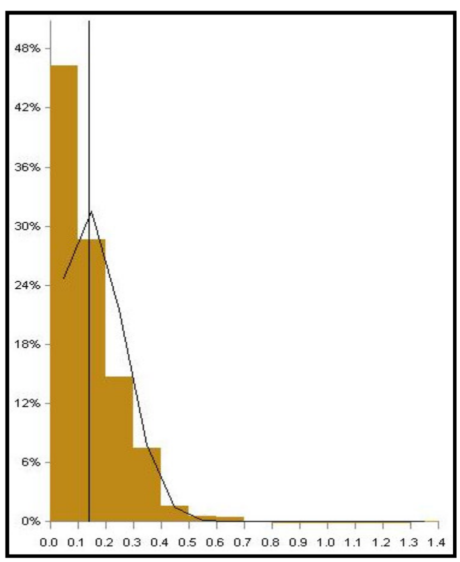


Figure 3.3 Histogram of Grades from 25 Drillholes

From frequency distribution in Figure 3.3 it is seen that the grade has lognormal distribution. The average copper grade as shown on the figure is about 0.14 % Cu.

3.3.2 Reserve Calculation Considering Blocks

In developing a mining project, an orebody model is generated from drillhole data to represent the deposit, usually through the application of geostatistical techniques (Ramazan *et al.*, 2005), such as inverse distance weight and nearest neighbor method. Inverse distance weight method was applied for modeling of Dereköy copper deposit. 3-D model of the orebody is generated to apply block model to estimate the average grade and the grade distribution of the deposit. The raw data from 25 drillholes are used for the modeling of Dereköy copper deposit. The deposit is divided to 40 m – 40 m – 20 m dimensions blocks and 4 m – 4 m – 4 m dimensions sub-blocks when the block model is applied. The sizes of the blocks are shown in Figure 3.4. The total number of blocks and sub-blocks together is 1,833,120.

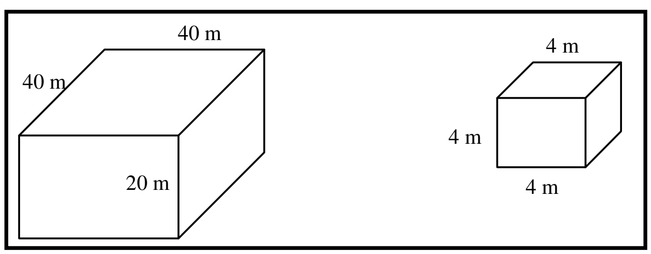


Figure 3.4 Dimensions of Blocks and Sub-Blocks

With the inverse distance weighting method, average grade of each block and grade distribution of the deposit were estimated. 3-D grade distribution of Dereköy copper deposit is shown in Figure 3.5.

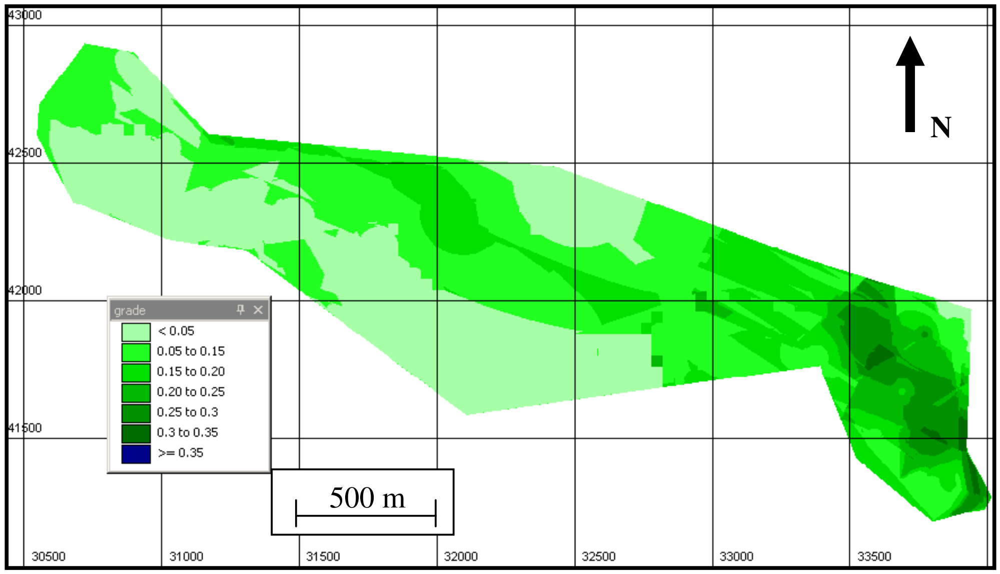


Figure 3.5 Grade Distribution of Dereköy Copper Deposit

The reserve amount is estimated as 1,740,805,517 tons. After block model application, average grade of the reserve is estimated as 0.106%. Detailed analysis of the deposit is given in Table 3.2 and Figure 3.6. Topographic map of the location is also presented in Figure 3.7.

Table 3.2 Analysis of Dereköy Copper Reserve

Grade From, %	**Grade To, %**	**Volume, m^3**	**Tonnes, ton**	**Grade. %**	**Cumulative Volume. M^3**	**Cumulative Tonnes, ton**	**Cum. Average Grade, %**
0	0.05	152,305,536	411,224,947	0.031	152.305.536	411,224,947	0.031
0.05	0.15	344,626,688	930,492,058	0.091	496.932.224	1,341,717,005	0.073
0.15	0.20	71,947,008	194,256,922	0.173	568.879.232	1,535,973,926	0.085
0.20	0.25	39,109,696	105,596,179	0.221	607.988.928	1,641,570,106	0.094
0.25	0.30	22,900,992	61,832,678	0.275	630.889.920	1,703,402,784	0.101
0.30	0.35	10,784,128	29,117,146	0.318	641.674.048	1,732,519,930	0.104
0.35	100	3,068,736	8,285,587	0.397	644.742.784	1,740,805,517	0.106

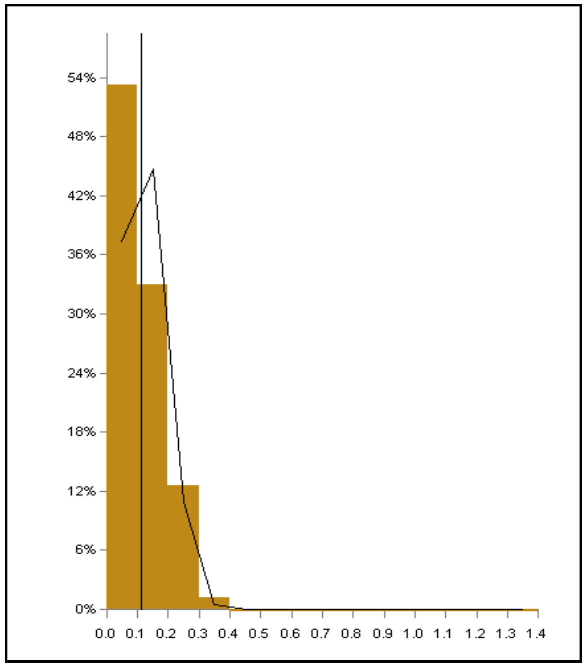


Figure 3.6 Grade Distribution by Block Model Estimation

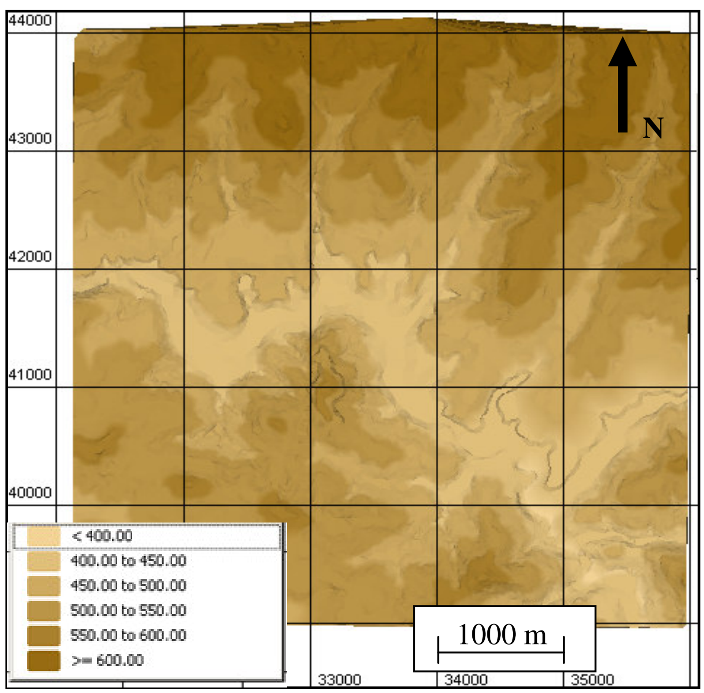


Figure 3.7 3-D Topography of the Location

3.3.3 Comparisons of the Applied Methods

Exploration drillings were conducted to investigate an orebody. Average grade of the orebody was determined by the grade data from drillholes. During the exploration stage optimum distance was not applied. Therefore, the grade data may not represent all orebody. If the optimum distance is selected, the difference between calculated average grade by the grade data from drillholes and estimated average grade with block model method is low.

The calculated average grade of the copper deposit is 0.14 %Cu but the estimated average grade is 0.106 %Cu. There is a difference between them. The main reason of this difference is the much distance between the drillholes. This difference also indicates that more than 25 drillholes are required for this location for more accurate results. The average distance between the drillholes is 225 m. The estimated result by block model is considered for the reserve amount estimation because of this reason.

3.3.4 Evaluation of Dereköy Copper Deposit

As seen in Figure 3.6 and in Table 3.2, 77.1% of the reserve has a grade lower than 0.15% Cu. Figure 3.5 also indicates that the northwestern part of the reserve has low grade and there are Dereköy town and some international roads are located on the northwestern part of the deposit. Because of these reasons, the feasibility of the southeastern part of the reserve is evaluated at first. A detailed reserve analysis was applied in the area between 33300E and 34200E and between 41000N and 42100N as indicated in Figure 3.8. The details of the reserve analysis are given in section 3.4.

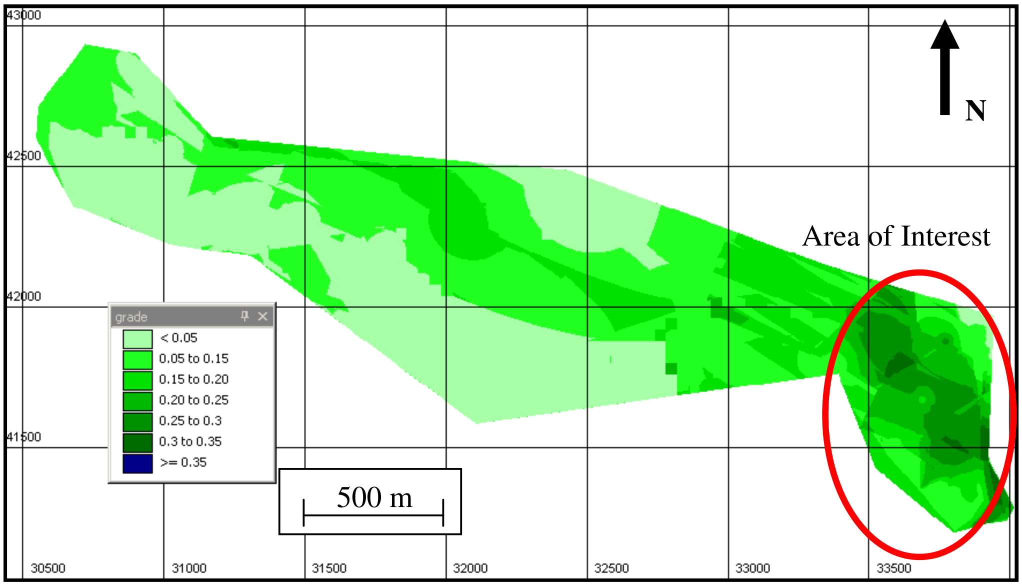


Figure 3.8 The Southeastern Part of the Reserve

3.4 Evaluation of the Southeastern Part of Dereköy Copper Deposit

3.4.1 Grade and Reserve Estimation using Polygonal Method

Polygonal method was applied to the southeastern part of the deposit to estimate the amount of ore and overburden. The polygons and orebody boundary are shown in Figure 3.9.

The polygonal method of computing ore reserves is based on the area of polygons constructed around each drillhole to define the area of influence of that drillhole orientation and configuration of the polygons are somewhat arbitrary, but drillholes that define the polygon should be selected so that they surround the central drillhole with as uniform radius as possible. The total volume of the polygonal prism defined by the polygon and central drillhole is determined by calculating the area of the polygonal and multiplying this area by the central drillhole depth which crosses the orebody.

In this study, a digital planimeter was used to find the polygon areas. To reduce the measurement errors, each polygon's area was measured three times and their average was accepted as the polygon area. The measurements with the digital planimeter were shown in Table 3.3.

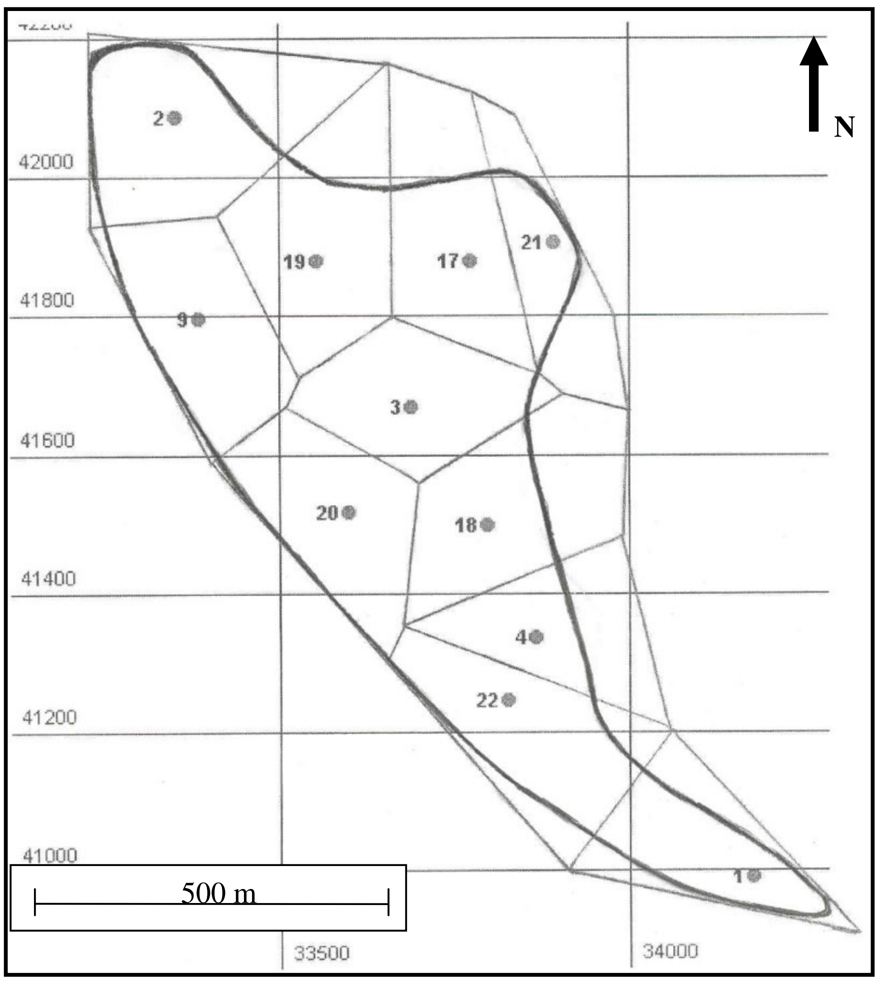


Figure 3.9 Polygons and Orebody Boundary for the Southeastern Part of Dereköy Copper Deposit

Amount of ore and average grade of deposit are estimated with polygonal method. They are shown in Table 3.4 and 3.5 respectively. Many geometrical considerations are necessary when using the polygonal method of computing ore reserves. An irregular boundary line is one of them. The main geometrical problem in usually the polygonal method of computing ore reserves is to make the sides of the polygons coincident with the boundary lines of the deposit. In this study, the boundary line is constructed by considering the position of the polygons and the influence area of drillholes.

Table 3.3 Areas of Polygons

Planimeter Measurements				
Polygon Name	**Measurement 1**	**Measurement 2**	**Measurement 3**	**Area, m^2**
1	27,563	26,438	27,000	27,000
2	48,973	47,813	50,625	49,137
3	48,938	48,375	50,063	49,125
4	20,813	21,375	19,125	20,438
9	54,910	53,465	54,910	54,428
17	59,245	59,968	58,523	59,245
18	38,293	36,848	34,680	36,607
19	46,963	49,853	49,853	48,890
20	49,853	48,408	49,130	49,130
21	18,063	17,340	19,509	18,304
22	45,518	44,795	44,073	44,795
TOTAL				**457,100**

As indicated in Table 3.4 and Table 3.5, amount of the orebody and the grade are 212,105,421 tons and 0.255 %Cu respectively. When average grade of the deposit is estimated with polygonal method, weighed average method is applied with consider reserve amount of each polygons as seen in Table 3.5.

Table 3.4 Reserve Estimation Obtained by Polygonal Method

Polygon Name	Area, m^2	Ore Thickness, m	Ore Volume, m^3	Density, ton/m^3	Ore Amount, ton
1	27,000	70	1,890,023	2.7	5,103,063
2	49,137	180	8,844,660	2.7	23,880,582
3	49,125	190	9,333,813	2.7	25,201,296
4	20,438	440	8,992,573	2.7	24,279,948
9	54,428	70	3,809,983	2.7	10,286,955
17	59,245	140	8,294,347	2.7	22,394,736
18	36,607	340	12,446,380	2.7	33,605,226
19	48,890	300	14,666,900	2.7	39,600,630
20	49,130	70	3,439,123	2.7	9,285,633
21	18,304	80	1,464,320	2.7	3,953,664
22	44,795	120	5,375,440	2.7	14,513,688
TOTAL	**457,100**		**78,557,563**		**212,105,421**
AVERAGE		**181.82**		**2.7**	

Table 3.5 Grade Estimation Obtained by Polygonal Method

Polygon Name	Ore Amount, ton	Average Grade, %	Cu Content, ton
1	5,103,063	0.229	11,686
2	23,880,582	0.199	47,522
3	25,201,296	0.372	93,749
4	24,279,948	0.232	56,329
9	10,286,955	0.222	22,837
17	22,394,736	0.108	24,186
18	33,605,226	0.325	109,217
19	39,600,630	0.283	112,070
20	9,285,633	0.236	21,914
21	3,953,664	0.158	6,247
22	14,513,688	0.240	34,833
TOTAL	**212,105,421**		**540,591**
AVERAGE		**0.255**	

3.4.2 Grade and Reserve Estimation using Software

There are 11 drillholes, namely number 1, 2, 3, 4, 9, 17, 18, 19, 20, 21, and 22 which are located in the southeastern part of the mineralized zone. These drillholes are used for modeling and statistical analysis of the grade.

It is difficult to obtain accurate result with the given 11 drillholes due to the fact that the distance between the drillholes is very high and the number drillholes is limited. Therefore, in this study grade estimation is made by block modeling for the southeastern part of the deposit.

The part is modeled in detail to get more accurate grade estimation. Different sections are used to create a solid orebody. During block model application, dimensions of blocks are reduced to 10 m – 10 m – 5 m and dimensions of sub-blocks are reduced to 2 m – 2 m – 2.5 m. 144,936 blocks and 522,352 sub-blocks are created by block modeling. Inverse distance weight method is applied to estimate the grade of the blocks and sub-blocks.

Average grade of the deposit is found by block model with weighted average method as 0.244 %Cu. Detail analysis of the southeastern part is shown in Figure 3.10, Figure 3.11 and Table 3.6. Table 3.6 indicates that only 11.9% of the southeastern part is lower than 0.15 %Cu.

Final pit design and final pit limit of the southeastern part of the deposit is also found by Micromine software. They are shown in Figure 3.12 and Figure 3.13 respectively.

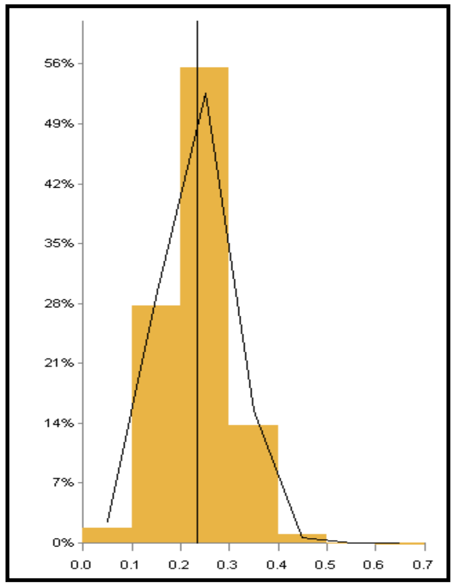


Figure 3.10 Frequency Distribution of Estimated Grades

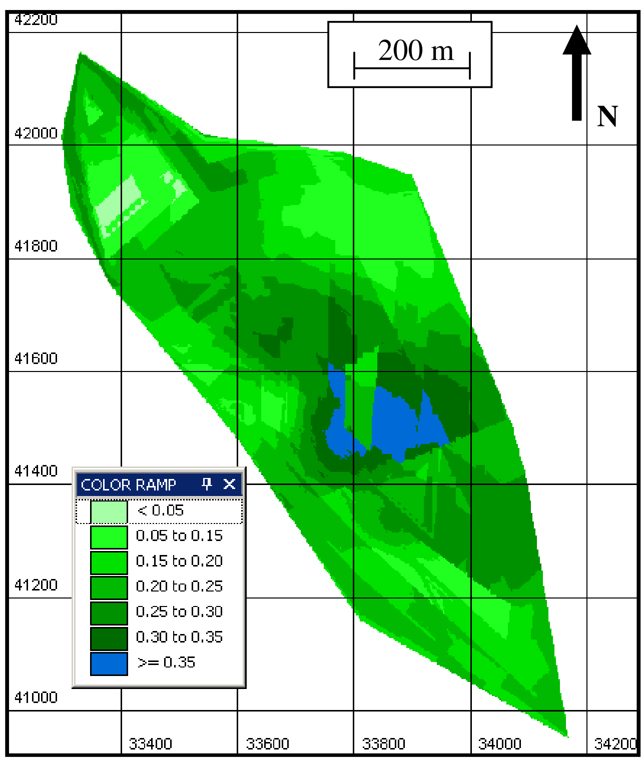


Figure 3.11 Grade Distribution of the Southeastern of Dereköy Copper Deposit

Table 3.6 Analysis of the Southeastern Part of Dereköy Copper Reserve

Grade From, %	Grade To, %	Volume, m^3	Tonnes, ton	Average Grade, %	Cumulative Volume, m^3	Cumulative Tonnes, ton	Cum. Average Grade, %
0	0.15	8,784,450	23,718,015	0.124	8,784,450	23,718,015	0.124
0.15	0.20	14,519,640	39,203,028	0.178	23,304,090	62,921,043	0.158
0.20	0.25	20,539,540	55,456,758	0.226	43,843,630	118,377,801	0.190
0.25	0.30	17,305,750	46,725,525	0.271	61,149,380	165,103,326	0.213
0.30	0.35	9,303,630	25,119,801	0.324	70,453,010	190,223,127	0.228
0.35	100	7,238,510	19,543,977	0.406	77,691,520	209,767,104	0.244

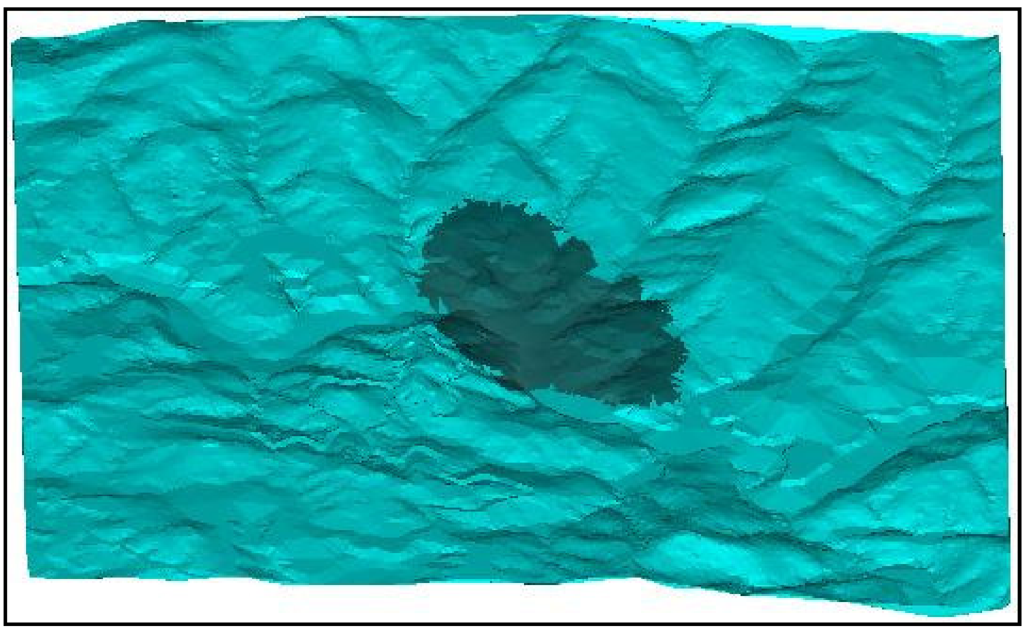


Figure 3.12 Final Pit Limit View

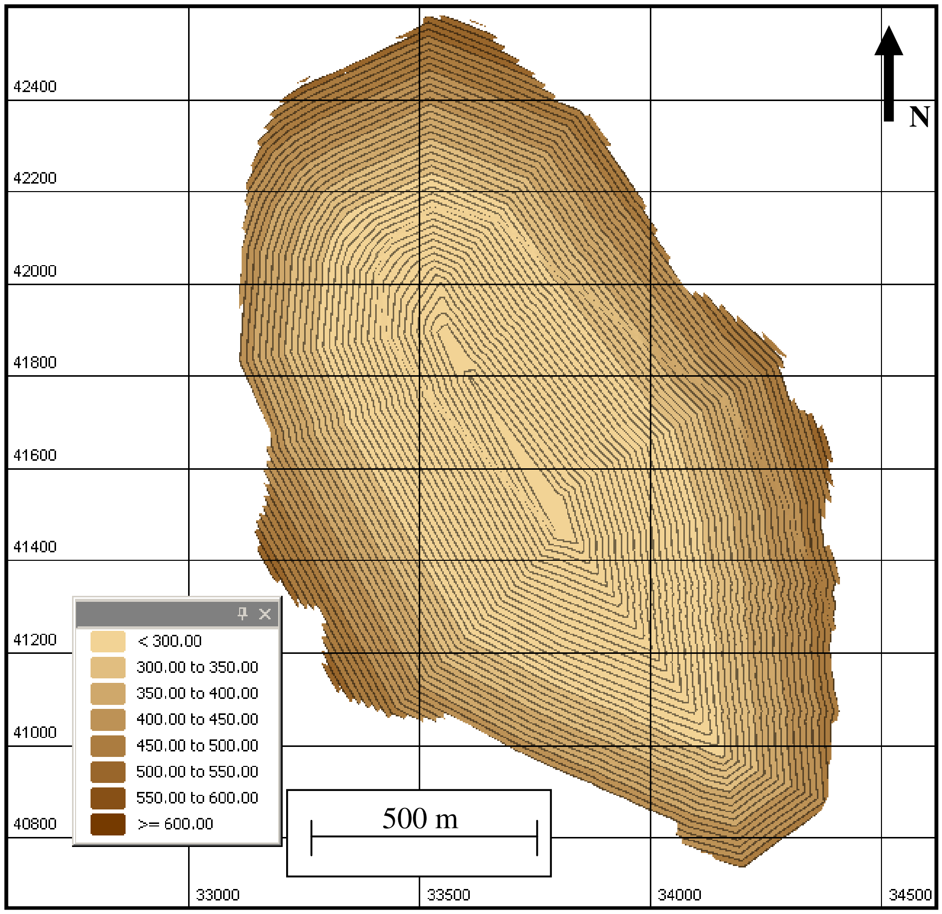


Figure 3.13 Final Pit Design and Elevation Legend

3.4.3 Comparison of Estimations by Micromine and Polygonal Method

The estimated amount of ore was 209,767,104 tons by Micromine but 212,105,421 tons ore was estimated by polygonal method. There is 2,338,317 tons difference between them. Estimation of average grade of the deposit is almost same. Micromine estimates average grade as 0,244 %Cu and 0,255 %Cu is estimated by polygonal method. Outputs of them are listed in Table 3.7.

Table 3.7 Outputs of Micromine and Polygonal Method

	Reserve Amount, ton	**Grade, %Cu**
Micromine	209,767,104	0.244
Polygonal Method	212,105,421	0.255

Although there is not a big difference between Micromine and polygonal method, Micromine result is more accurate than polygonal method because polygonal method is more likely to include an error due to the fact that the boundary of the deposit was defined manually and the calibration of the planimeter was not known. Results of Micromine were validated using polygonal method results. The slight difference indicated that the results are consistent.

3.4.4 Reserve Investigation of the Deposit

Dereköy copper deposit is explored with drillholes and amount of deposit is estimated with Micromine and polygonal method. Therefore, Dereköy copper deposit is a visible reserve in terms of traditional reserve classification. The estimated proven reserve and average grade of the deposit are 209,767,104 tons and 0.244 %Cu respectively.

CHAPTER 4

RISK MODELLING OF THE DEREKÖY COPPER DEPOSIT

Risk management plays a vital role for economical sustainability of the project. Underestimated risks are the main causes of the failure of the projects. The risks should be estimated precisely before the start of projects. The most common risk estimation method is simulation. Simulation is a procedure in which random numbers are generated according to probabilities assumed to be associated with a source of uncertainty, such as a new product's sales, stock prices, interest rates, exchange rates or commodity prices. Outcomes associated with these random drawings are then analyzed to determine the likely results and the associated risk. Monte Carlo simulation is a legitimate and widely used technique for dealing with uncertainty in many aspects of business operations (Chance, 2008).

Traditionally, when a model is created to estimate the fix value of NPV, an exact value for each input parameter is used. After the calculations, a fix value is predicted as an output. However, there is a risk for each input and the use of a single exact value may not be appropriate considering the risk bound around each input. Instead of using a fix value for an input, a distribution of each input should be used. Each distribution contains maximum likelihood and variability. Simulation should be applied to able to use all uncertainties in the model. @Risk 4.5.7 (Professional Edition, Palisade Corp.) was used to perform Monte Carlo simulation (Robert and Casella, 2004).

In mining industry, the inputs can be estimated but may not be appropriate without considering risks. The interval of the uncertainty can be estimated with engineering assumptions and some data related with the ore deposit and market. These estimation intervals can be indicated as probability distributions. The probability distributions are the inputs of the Monte Carlo simulation. Monte Carlo simulation technique can make a simulation with evaluating huge number of hypothetical scenarios. The main advantage of the Monte Carlo simulation is that the uncertainties are considered. Therefore, almost all scenarios can be evaluated. If the number of iteration increases, the reliability of the simulation increases. The algorithm of the Monte Carlo simulation method is presented in Figure 4.1.

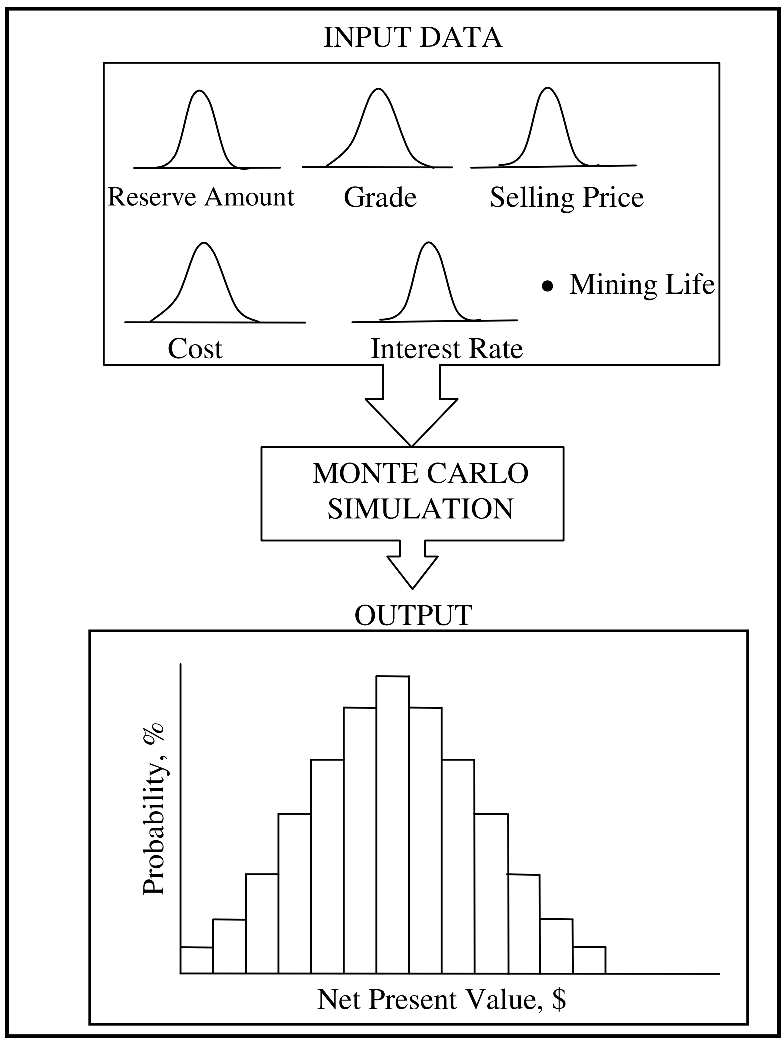


Figure 4.1 Evaluating NPV with Monte Carlo Simulation (Modified from Hawly, 1983)

At the first stage of applying Monte Carlo technique to Dereköy copper ore reserve, independent variables such as grade, density and selling price are determined. Then, the dependent variables are defined. The dependent variables are functions of the independent variables. After deciding all variables as probability distributions, a model was created. The model calculates the annual gross profit of the Dereköy copper deposit annually. After annual gross profit estimation, the model estimates the NPV of the deposit with considering interest rate and number of the related year. Finally, the probability distribution and cumulative density functions of NPV of Dereköy copper deposit were estimated.

After the simulation, the effects of the variables on the investment are assessed. Then, the investment must be evaluated to check feasibility of the project. Therefore, weak aspects of the investment can be detected and preventive measures can be taken before mining operation starts. In this study, an evaluation technique namely Monte Carlo Simulation technique was successfully applied to evaluate the copper ore deposit.

4.1 Use of Software (@Risk 4.5.7) in Monte Carlo Simulation

@RISK performs risk analysis using Monte Carlo simulation to show how many possible outcomes in Microsoft Excel spreadsheet exist and informs how likely they are to occur. This means investor can judge which risks to take and which ones to avoid, allowing for the best decision making under uncertainty. With @RISK, you can answer questions like, what is the probability of profit exceeding $150,000,000? or what are the chances of losing money on this venture (Palisade Corporation, 2008)?

@Risk 4.5.7 links directly to Microsoft Excel 2007 (Microsoft Corp.) to add risk analysis capabilities as an “add-in” to Microsoft Excel. The @Risk 4.5.7 system provides all the necessary tools for setting up, executing and viewing the results of risk analyses. @Risk 4.5.7 works in the style of Microsoft Excel menus and function. Therefore, its usage is familiar to Microsoft Excel users. An uncertainty can be defined as a probability distribution in Excel cells with the help of @Risk.

Classification of the independent and dependent inputs should be done before starting the simulation. The base point of Monte Carlo simulation is that the inputs must be independent. The dependent variables are created after the simulation as "RiskOutput". The probability distribution of an input can be defined manually with using "Define Distribution" function as illustrated in Figure 4.2. If a data set, such as historical data of an input, is available, @Risk 4.5.7 can select the most appropriate probability distribution for this data set with the module of "Fit Distribution" (Figure 4.3). "Fit Distribution" uses some tests, such as Kolmogorov-Simirnov (Damodaran, 2007), when defining the best distribution. After defining the probability distributions, a function is assigned by @Risk 4.5.7 to the selected Excel cell. The formula contains all needed information about the probability distribution. Some of the formulas are illustrated below.

- RiskLognorm(10,25)
- RiskUniform(15,40)
- RiskNormal(80,5)
- RiskTriang(A3/2.01,A4,A5)

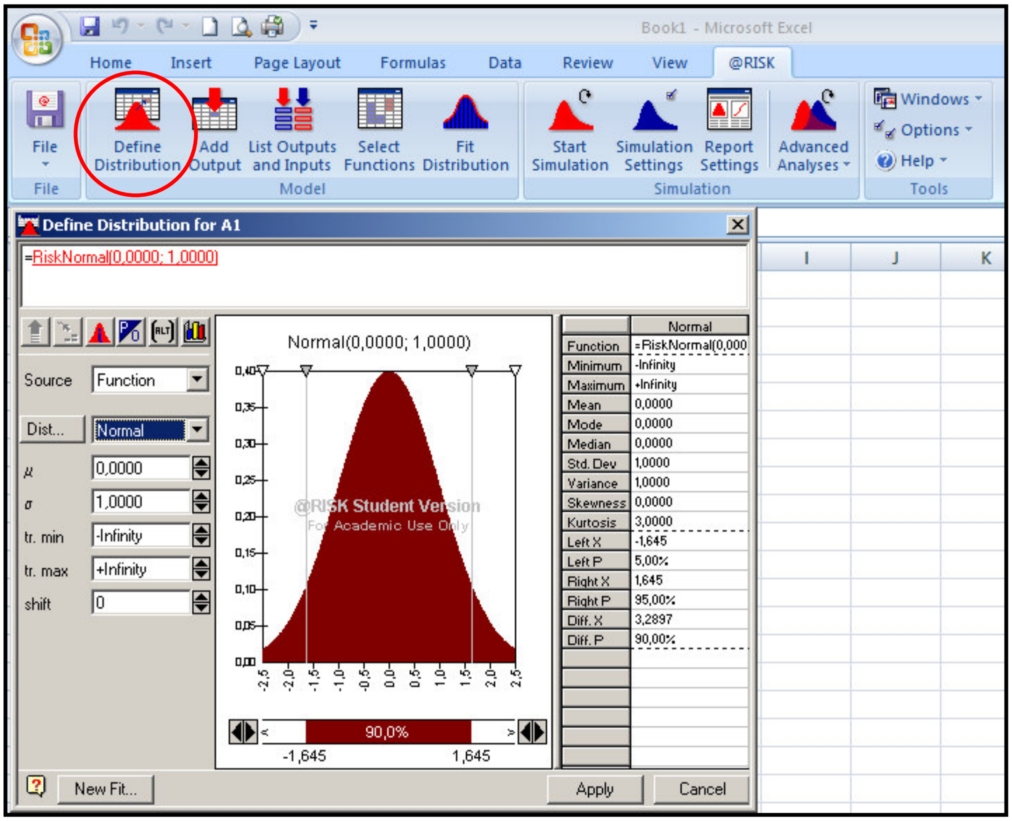


Figure 4.2 Defining a Probability Function for an Input

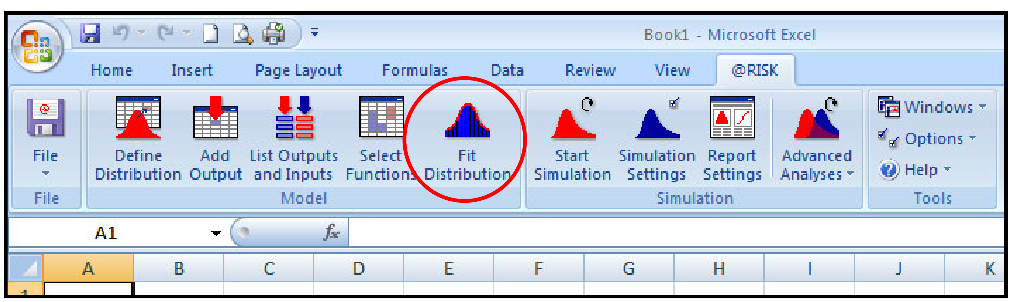


Figure 4.3 Defining a Probability Function for a Data Set with Fit Distribution

Simulation can be started after defining the formula and calculations between dependent and independent variables. Simulation settings can also be adopted by user for each project. The simulation settings can be adjusted by "Simulation Settings" (Figure 4.4).

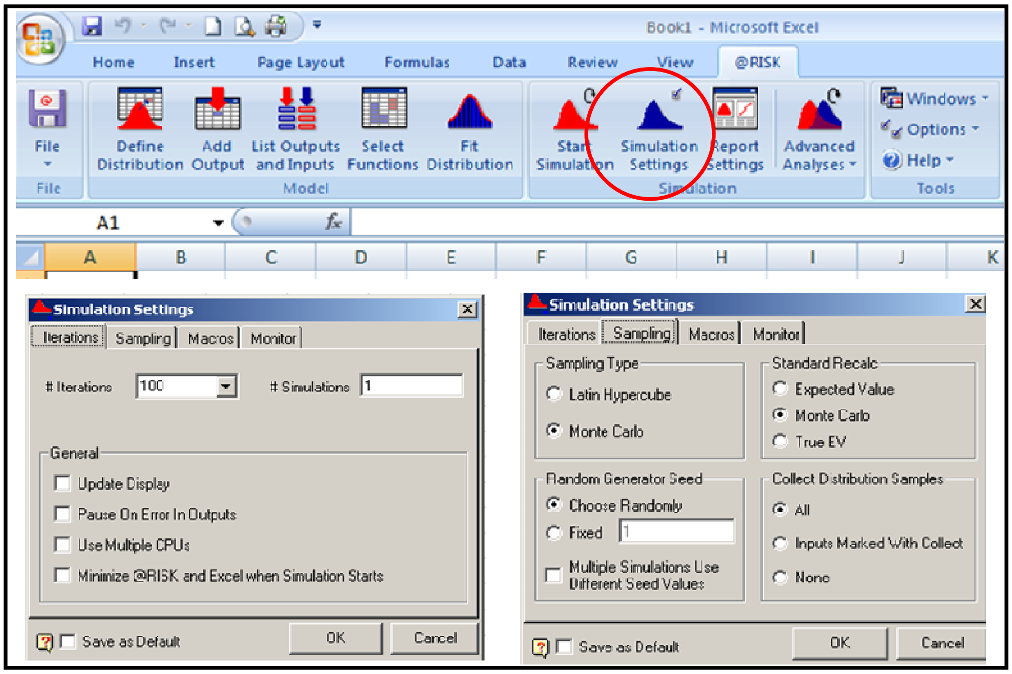


Figure 4.4 Simulation Settings

After defining probability distributions and selection of the simulation settings, simulation can be started with "Start Simulation" button on the toolbar (Figure 4.5). @Risk 4.5.7 gives all data about dependent variables after simulation as outputs. A detailed statistics and graphical representation and probability distributions of each independent and dependent variable can be supplied by @Risk 4.5.7. The statistical and graphical outputs are illustrated in Figure 4.6 and Figure 4.7 respectively. Sensitivity analysis is also provided by @Risk 4.5.7. Sensitivity analysis result is illustrated by a Tornado Graph. A sample sensitivity analysis is shown in Figure 4.8.

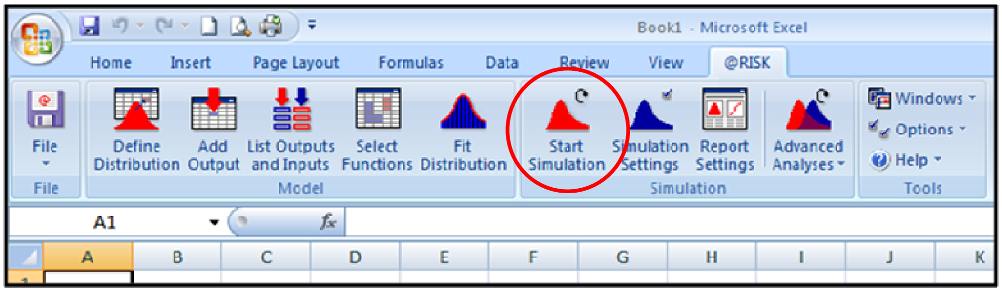


Figure 4.5 Start Simulation Button

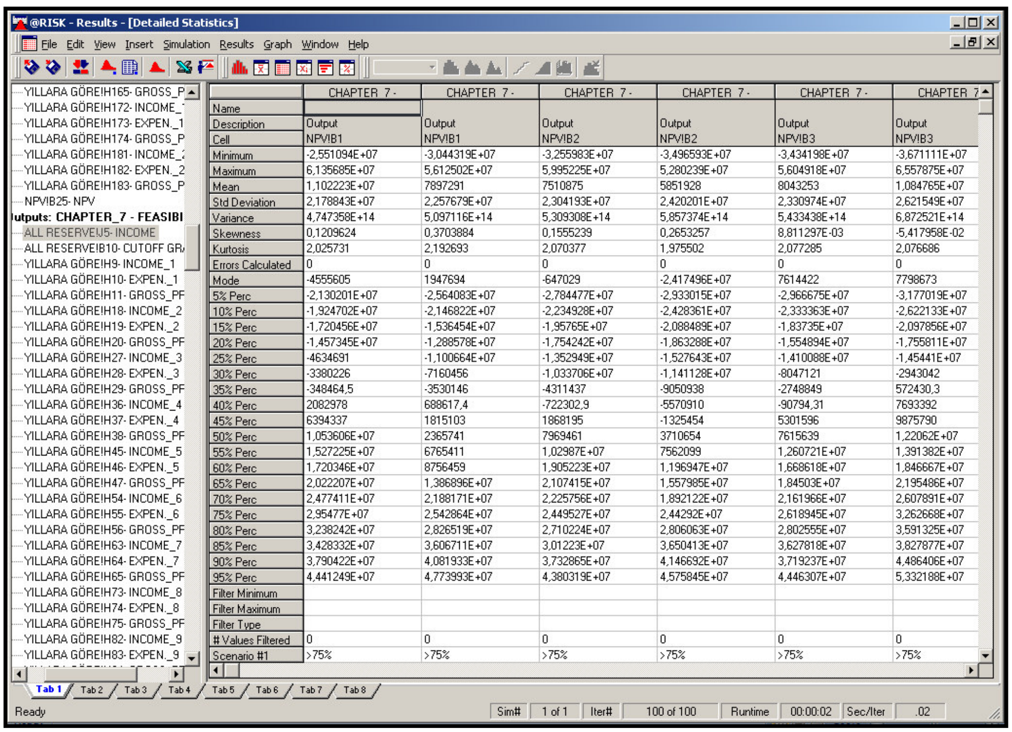


Figure 4.6 Statistical Evaluations of Outputs

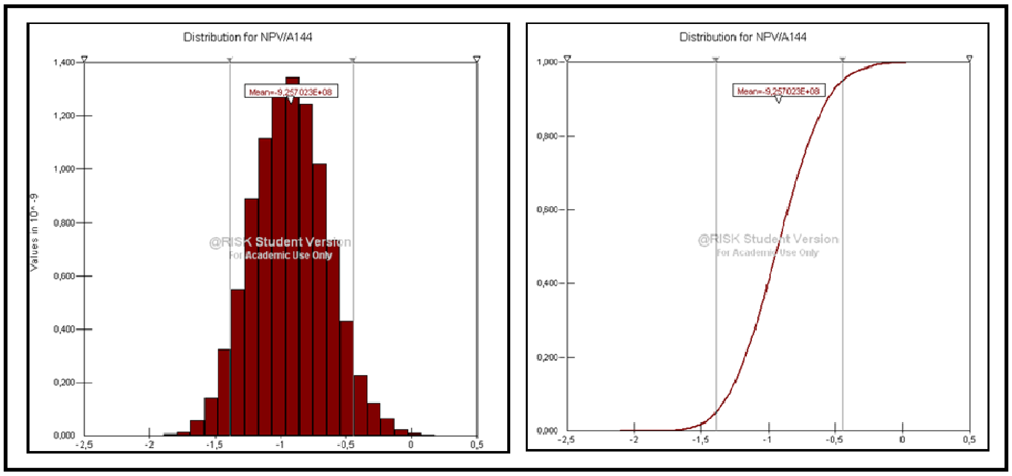


Figure 4.7 Graphically Evaluation of Outputs

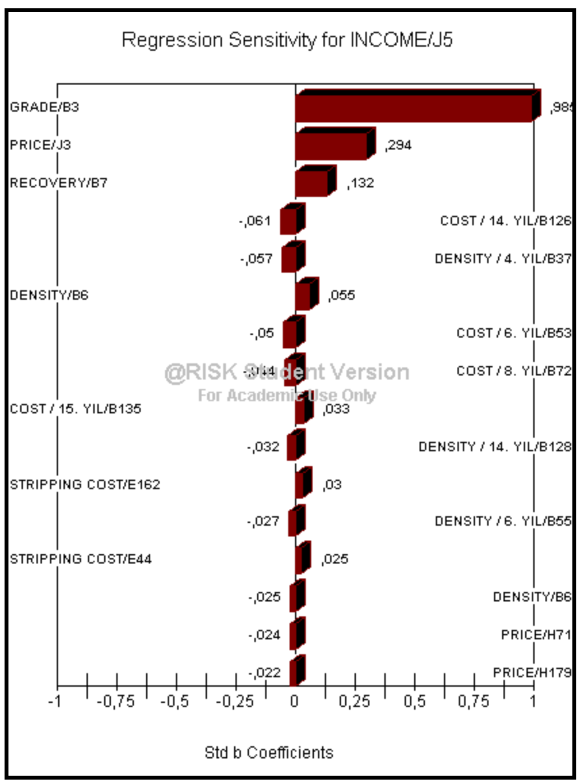


Figure 4.8 Tornado Graph

4.2 Developing a Risk Assessment Model for Dereköy Copper Ore Deposit

An economical model must be constructed to estimate the risk of a mining project. Determination of the economical value of a mining project can be done by evaluation of capital cost and cash flow. The aim of evaluation of the cash flow is to investigate the profitability of the mining project with related risks. The economical value of a mining project is determined by NPV which is obtained from cash flow (Nasuf and Orun, 1990).

A risk assessment model was created to simulate the NPV of the deposit. The principal behind the model is estimating revenue and cost per year. Annual gross profit has two important variables namely annual income and annual cost (equation 4.1). Equations of annual income and annual cost for simulation are presented in equation (4.2) and equation (4.3) respectively. Both equations have constants, independent and dependent variables. Estimation of annual cost is more difficult than estimation of annual income because annual cost estimation includes more uncertain variables. In other words, estimation of annual cost has more risks to be considered. Therefore, annual cost is divided into three main parts namely annual mining cost, annual processing cost and annual metallurgical cost. Their estimation equations in the model are illustrated in equation (4.4), equation (4.5) and equation (4.6) respectively. Estimations of the three costs were done independently from each other in the model. After the estimations, their summation gives the annual cost of the Dereköy copper mine.

Feasibility of the project is determined by the NPV of the reserve. NPV of the reserve was found by the summation of the present value (PV) of the annual gross profit. PV of yearly gross profit is calculated using single payment present worth factor (SPPWF). The formula for single payment present worth is given in equation (4.7). The expression in the bracket called the SPPWF. The future value, F, represents annual gross profit. It can be concluded that time value of money is one of the important factors for estimation of the NPV of the deposit.

Finally, a single equation, which includes all variables related to estimation of PV of annual gross profits, is concluded as indicated in equation (4.9). NPV of the deposit is obtained by extraction of PV of total capital cost from PV of summation of PV of annual gross profits. The steps of the construction of the equation for risk assessment model are presented in from equation 4.1 to equation 4.8. As indicated before, there are 20 inputs for the estimation of NPV for each year. The only three of them are defined as constant. They are mine life, ore grade after processing and grade of the blister copper. The list of the all defined variables in the equation (4.10) is presented in Table 4.1.

Table 4.1 The Variables of the Model

Constant Variables	**Independent Variables**	**Dependent Variables**
Mine Life	Density	Yearly Ore Mining
Ore Grade After Processing	Yearly Interest Rate	Yearly Stripping
Grade After Metallurgy	Mining Cost	Yearly Mining Cost
	Mining Recovery	Yearly Processing Cost
	Metallurgical Cost	Yearly Metallurgical Cost
	Metallurgical Recovery	Production at Processing
	Ore Grade	Production at Metallurgy
	Processing Cost	Yearly Income
	Processing Recovery	
	Stripping Cost	
	Selling Price	
	Total Overburden	
	Total Ore Volume	

$$Annual\ Gross\ Profit = Annual\ Income - Annual\ Cost \tag{4.1}$$

$$Annual\ Income = \frac{MR \times PR \times Met.R \times TOV \times D \times OG \times SP}{ML \times GAM \times 100^3} \tag{4.2}$$

In equation (4.2):

D = Density, ton/m^3

GAM = Grade After Metallurgy, % (It was defined as 99.99%)

ML = Mine Life, year (It was defined as 20 years)

MR = Mining Recovery, %

Met.R = Metallurgical Recovery, %

OG = Ore Grade, % (in-situ)

PR = Processing Recovery, %

SP = Selling Price, $/ton

TOV = Total Ore Volume, m^3, (in-situ)

$$Annual\ Cost = TMC + TPC + TMet.C \tag{4.3}$$

In equation (4.3):

TMC = Yearly Total Mining Cost, $

TMet.C = Yearly Total Metallurgical Cost, $

TPC = Yearly Total Processing Cost, $

$$Annual\ Mining\ Cost = \frac{[TOV \times D \times MR \times MC] + [TO \times SC \times 100]}{ML \times 100} \tag{4.4}$$

In equation (4.4):

D = Density, ton/m^3

MC = Mining Cost, \$/ton

ML = Mine Life, year (It was defined as 20 years)

MR = Mining Recovery, %

SC = Stripping Cost, \$/m^3

TO = Total Overburden, m^3

TOV = Total Ore Volume, m^3, (in-situ)

$$Annual\ Processing\ Cost = \frac{TOV \times D \times MR \times PC}{ML \times 100} \tag{4.5}$$

In equation (4.5):

D = Density, ton/m^3

ML = Mine Life, year (It was defined as 20 years)

MR = Mining Recovery, %

PC = Processing Cost, \$/m^3

TOV = Total Ore Volume, m^3, (in-situ)

$$Annual\ Metallurgical\ Cost = \frac{PR \times TOV \times D \times MR \times OG \times Met.C}{OGAP \times ML \times 100^2} \tag{4.6}$$

In equation (4.6):

D = Density, ton/m^3

ML = Mine Life, year (It was defined as 20 years)

MR = Mining Recovery, %

Met.C = Metallurgical Cost, $/ton

OG = Ore Grade, % (in-situ)

OGAP = Ore Grade after Processing, % (It was defined as 20%)

PR = Processing Recovery, %

TOV = Total Ore Volume, m^3, (in-situ)

$$P = F \times \left[\frac{1}{(1+i)^n}\right] \tag{4.7}$$

In equation (4.7):

P = Present Value

F = Future Value

i = Interest Rate

n = Number of Year

$$PV\ of\ Capital\ Costs = CCP + \left[CCF \times \frac{1}{(1+i)^n}\right] \tag{4.8}$$

In equation (4.8):

CCP = Capital Cost at Present

CCF = Capital Cost at Future

i = Interest Rate

n = Number of Year

$$PV\ of\ Annual\ Gross\ Profit = \begin{bmatrix} \dfrac{MR \times PR \times Met.R \times TOV \times D \times OG \times SP}{ML \times GAM \times 100^3} \\ - \dfrac{(TOV \times D \times MR \times MC) + (TO \times SC \times 100)}{ML \times 100} \\ - \dfrac{TOV \times D \times MR \times PC}{ML \times 100} \\ - \dfrac{PR \times TOV \times D \times MR \times OG \times Met.C}{OGAP \times ML \times 100^2} \end{bmatrix} \times \left[1 + \frac{i}{100}\right]^{-n} \tag{4.9}$$

In equation (4.9):

D = Density, ton/m^3

GAM = Grade After Metallurgy, % (It was defined as 99.99%)

i = Yearly Interest Rate for US Dollar in Turkey, %

MC = Mining Cost, \$/ton

ML = Mine Life, year (It was defined as 20 years)

MR = Mining Recovery, %

Met.C = Metallurgical Cost, \$/ton

Met.R = Metallurgical Recovery, %

n = Number of the Year

OG = Ore Grade, % (in-situ)

OGAP = Ore Grade After Processing, % (It was defined as 20%)

PC = Processing Cost, \$/ton

PR = Processing Recovery, %

SC = Stripping Cost, \$/m^3

SP = Selling Price, \$/ton

TO = Total Overburden, m^3, (in-situ)

TOV = Total Ore Volume, m^3, (in-situ)

$$NPV = \left\{ \sum_{i=1}^{20} (PV\ of\ Annual\ Gross\ Profit)_i \right\} - (PV\ of\ Total\ Capital\ Costs) \tag{4.10}$$

In this study, 10,000 successive iterations were done; this means that 10,000 random scenarios were evaluated for the established risk assessment model by @Risk 4.5.7 software. When iteration was conducted, the values were selected randomly from the input data distributions and one output was found. Distribution of these outputs was defined and mean and standard deviation of the distribution was found. Therefore, properties of NPV distribution and its properties were investigated.

4.3 Inputs (Variables) in the Model

4.3.1 The Inputs Related to Mining

The purpose of investment analysis is to determine the profitability of the project. Therefore, distribution of each input data should be decided with using the historical data of inputs and engineering assumptions. The southeastern part of Dereköy copper deposit was evaluated with considering uncertainty of variables. There are some effective variables in the risk assessment model in equation (4.10). These effective variables have important roles on the risk assessment. The effective variables will be discussed in sections 4.3.1 - 4.3.6.

4.3.1.1 Ore Grade

Grade distribution and average grade of the Dereköy copper deposit is estimated by Micromine software package with block model method. The deposit is divided to 667,288 blocks. Therefore, there are 667,288 different grades for the evaluation.

This number is so high to apply the “Fit Distribution” module. Therefore, 667,288 grade data are evaluated by Microsoft Excel and its mean and standard deviation are found with descriptive statistics. Mean grade and standard deviation are calculated as 0.233 %Cu and 0.06771 %Cu respectively. In the previous sections, when the block model technique is applied using Micromine, average grade of the deposit is found as 0.244 %Cu. The difference in the grade calculated using MS Excel and Micromine is due to the estimation methods applied. Excel makes use of simple averaging technique and Micromine makes use weighted average due to the fact that sizes of the blocks and sub-blocks are not the same. However, each blocks and sub-blocks grade weighted equally when the probability distribution for grade was defined. Because the difference between the average grade and weighted average grade of the blocks is only 0.011%Cu. As the difference is too small, both techniques can be used for the calculations.

4.3.1.2 Selling Price of Copper

Selling price of copper is another important factor for the evaluation of an investment. Price of copper is determined by London Metal Exchange (LME). Average monthly copper price between January 1990 and September 2008 are taken from LME. Average monthly copper prices between January 1990 and September 2008 are given in Table B.1 (see Appendix B). Copper price has increased since April 2006 dramatically. Therefore, when the copper price distribution is found, the price data which is between April 2006 and September 2008 are used. Best Fit module of @Risk 4.5.7 software package was used when the distribution was defined. Kolmogorov-Simirnov test was applied to find the best distribution for the selling price. Type of price distribution was found as Weibull distribution as indicated in Figure 4.9. The selling price distribution is constructed with average monthly copper prices of the last three years data. Monthly prices are considered due to the fact that construction of average annual copper price distribution with only three data is not reliable and also the “Fit Distribution” module works with at least five data.

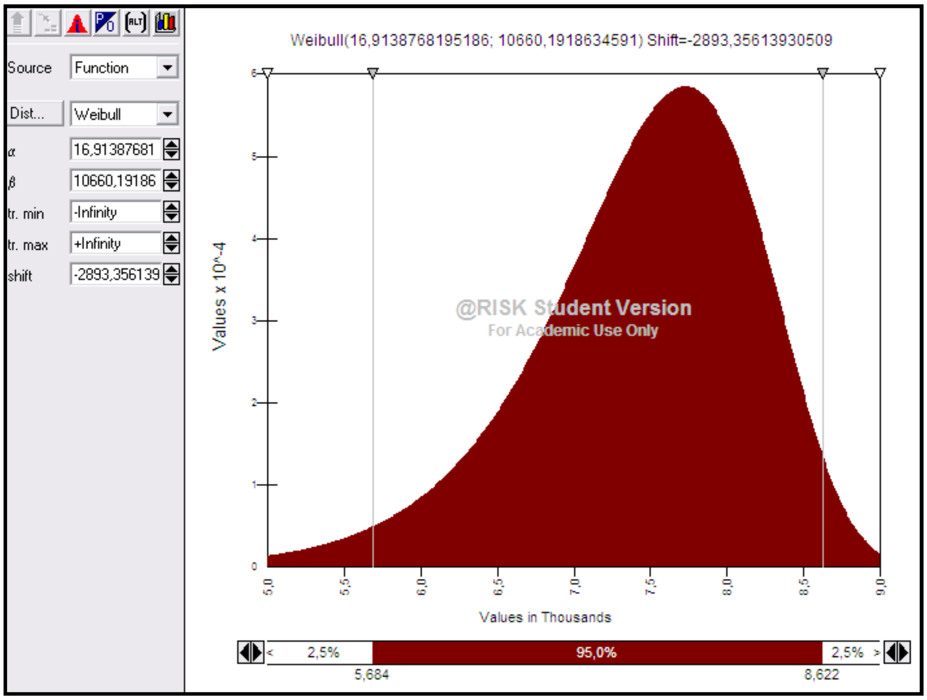


Figure 4.9 Distribution for Selling Price of Copper

4.3.1.3 Reserve Amount, Overburden Amount and Density of Ore

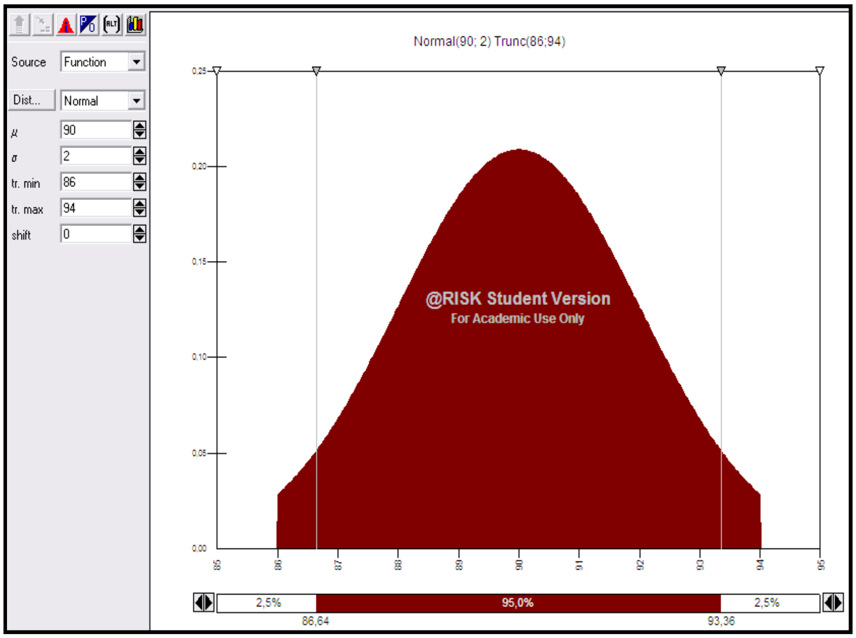
Reserve and overburden amounts are found as 77.69×10^6 m^3 and 140×10^6 m^3 respectively with Micromine. Reserve volume and overburden amount are defined as normal distributions. From the probability distribution of reserve estimation mean and standard deviation are found as 77.69×10^6 and 2×10^6 m^3 respectively while the probability distribution of overburden has mean and standard deviation as 140×10^6 m^3 as mean and 5×10^6 m^3 respectively. Model founds the tonnage by multiplying selected density with selected volume from related distributions. At the prefeasibility report of MTA, average density of ore is taken as 2.7 ton/m^3. The orebody can not have a constant density. Therefore, a normal distribution is considered for orebody density. In this distribution, mean and standard deviation are accepted as 2.7 t/m^3 and 0.1 ton/m^3respectively. Normal distribution was used to allow the model to select data around the mean value with high probability.

4.3.1.4 Mining, Processing and Metallurgical Recoveries in the Model

Three recoveries were defined in the model. They are mining recovery, processing plant recovery and metallurgical plant recovery. The information about the three recoveries is indicated in Table 4.2. Related recoveries can be adjusted by engineers in mine operations and plants. Therefore, the lowest limit and highest limit were defined in model for the three recoveries. The lowest and highest limits were accepted as $\bar{x}\pm2\sigma$. Therefore, the model selects random recovery values between the lowest and highest limits. The distributions of the three recoveries are illustrated in Figure 4.10.

Table 4.2 Information about the Recoveries

	Mean	Standard Deviation	The Lowest Limit	The Highest Limit
Mining Recovery	90%	2%	86%	94%
Processing Recovery	90%	2%	86%	94%
Metallurgical Recovery	93%	2%	89%	97%


Source
Function
Dist...
Normal
μ
90
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86
tr. max
94
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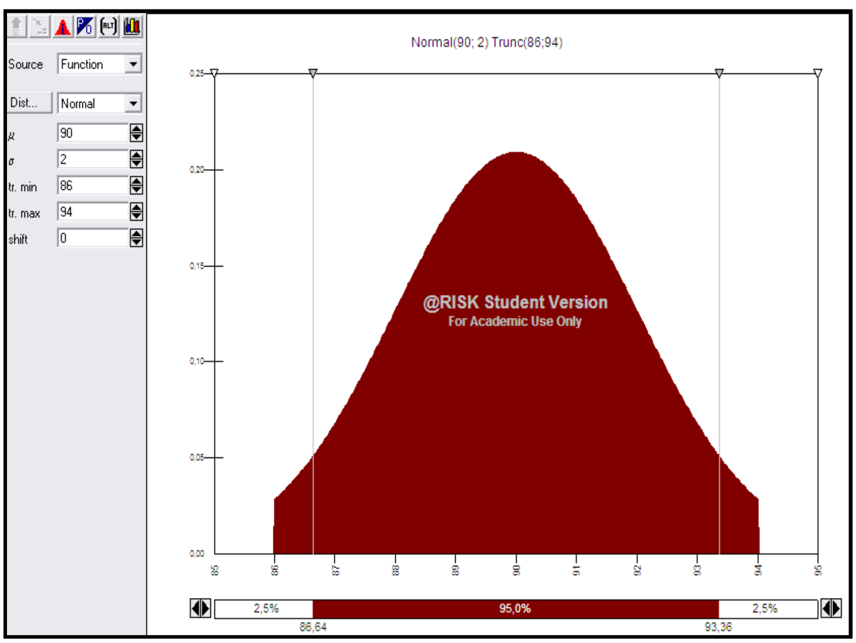
Normal(90; 2) Trunc(86;94)
@RISK Student Version
For Academic Use Only
2,5%
95,0%
2,5%
86,64
93,36

(a)


Source
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For Academic Use Only
2,5%
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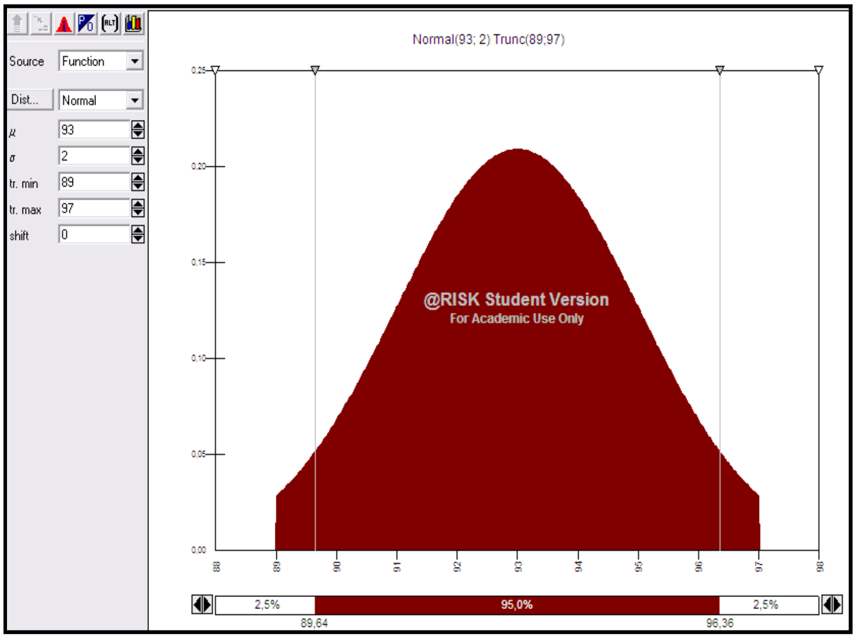
(b)



(c)

Figure 4.10 Probability Distribution of Mining (a), Processing Plant (b) and Metallurgical Plant (c)

4.3.1.5 Cost

Cost is one of the main input data for the estimation of revenue as indicated in equation (4.1). Two main types of costs namely capital cost and operating cost were considered in model. The distributions were also described for stripping and mining costs. Costs of stripping per m^3 and cost of mining per ton are assumed to be equal. Mean value and standard deviation of the cost are accepted as \$3.25/ton(m^3) and \$0.25/ton(m^3) respectively. Operating cost of processing plant is shown as a probability distribution. Mean and standard deviation are \$4.50/ton and \$0.25/ton respectively. Metallurgical cost's distribution parameters are \$100.00/ton mean, \$10.00/ton standard deviation. Total capital cost of mining and processing plant was considered as \$150,000,000 as constant at present. At the end of 10th year, \$100,000,000 is added as capital cost to the cash flow.

4.3.1.6 Interest Rate

Interest rate is very important for the calculation of NPV of the deposit. Annually profits and interest rates are essential parts of the estimation. Historical data of the interest rate between 2001 and 2008 for U.S. dollar is supplied from official webpage of Central Bank of the Republic of Turkey (Table 4.2). Probability distribution of the interest rate for U.S. dollar is defined as exponential distribution with the help of "Fit Distribution" module of @Risk 4.5.7 as indicated in Figure 4.11. Kolmogorov-Simirnov test was applied during the operation.

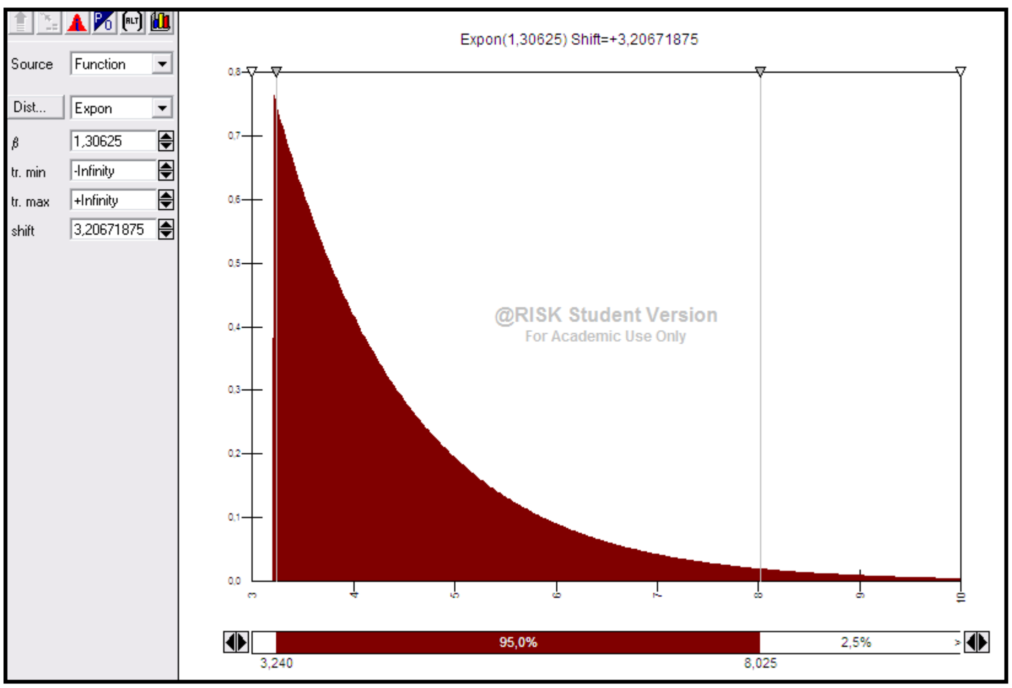


Figure 4.11 Distribution of Interest Rate between 2001 and 2008

Table 4.3 Weighted Annually Average Interest Rates for USD FX Deposits by Banks (Central Bank of the Republic of Turkey, 2008)

Year	Interest Rate, %	Year	Interest Rate, %
2001	8.70	2005	3.37
2002	4.56	2006	4.34
2003	3.68	2007	4.74
2004	3.46	2008	4.56

4.3.2 Simulation Inputs

The simulation inputs are related to running of the model. Simulation inputs are not defined as a probability distribution. They are given for one time at the beginning of the simulations. Number of iteration is one of the simulation inputs. Iteration number defines the number of random scenarios. The higher iteration number, the more accurate estimation but the more processing time with computer. In this study, iteration number is defined as 10,000 which is the highest number for iteration in @Risk 4.5.7.

4.4 Basic Principals Behind the Model

When the model runs, the software selects random variables. With the selected variables model makes the calculations which are already defined. In other words, the model proceeds the iteration. This procedure is repeated until it reaches to number of iteration which is defined at the beginning (10,000 times). The iterations are repeated for each variable related with the orebody, market and location. Finally, 10,000 random NPVs are performed with the related calculations. The results of the model are indicated as probability distribution and cumulative density curve of NPV and Tornado graph for sensitivity analysis.

CHAPTER 5

RESULTS AND DISCUSSIONS

5.1 Estimation of NPV without Uncertainty Assessment

Traditionally, NPV is estimated without uncertainty assessment using the model which is presented in equation (4.10). NPV of the southeastern part of Dereköy copper deposit was estimated by making use of the average grade, selling price, interest rate, density, mining, processing and metallurgical recovery, mining, striping, processing and metallurgical costs. It is also considered that amount of annual stripping and production amounts are constant. Above mentioned values which are used the input data are given in Table 5.1.

The grade used in grade calculation is estimated by block modeling. Weighted average of grades of the blocks and sub-blocks is considered. In determining the selling price, average of the last 30 months' are considered. The interest rate is found by taking the average of last eight years' rates. Annual production and stripping amounts are estimated using Micromine software. Total capital cost of mining and processing plant is estimated as $\$150\times10^{6}$ with today's money. It is also considered that at the end of 10^{th} year, additional $\$100\times10^{6}$ capital cost is added to the cash flow.

Estimation of the annual gross profits is done by conventional method and inflation rate is not considered. The uncertainties are not considered at the estimation of NPV at conventional method. Fix values are used as variables in the model. The used fix inputs are listed in Table 5.1. Selling price of copper in Table 5.1 is found with the arithmetic mean of the selling price between April 2006 and September 2008. Taking the variables explained above into consideration, annual gross profit is found as $36,986,193 using the equation (4.2), (4.3), (4.4), (4.5) and (4.6). The annual gross profits and capital costs are shown as a cash flow in Table 5.2 and Figure 5.1.

Table 5.1 The Fix Inputs which are used in Conventional Method

Input	**Average Value**	**Input**	**Average Value**
Grade	0.244 %Cu	Mining Cost	\$3.25 / ton
Density	2.7 ton/m^3	Stripping Cost	\$3.25 / m^3
Selling Price	\$7,434.01 / ton	Processing Cost	\$4.50 / ton of ore
Interest Rate	4.68%	Metallurgical Cost	\$100 / ton of concentrate
Mining Recovery	90%	Yearly Ore Mining	9,439,520 ton
Processing Recovery	90%	Yearly Stripping	7,021,899 m^3
Metallurgical Recovery	93%		

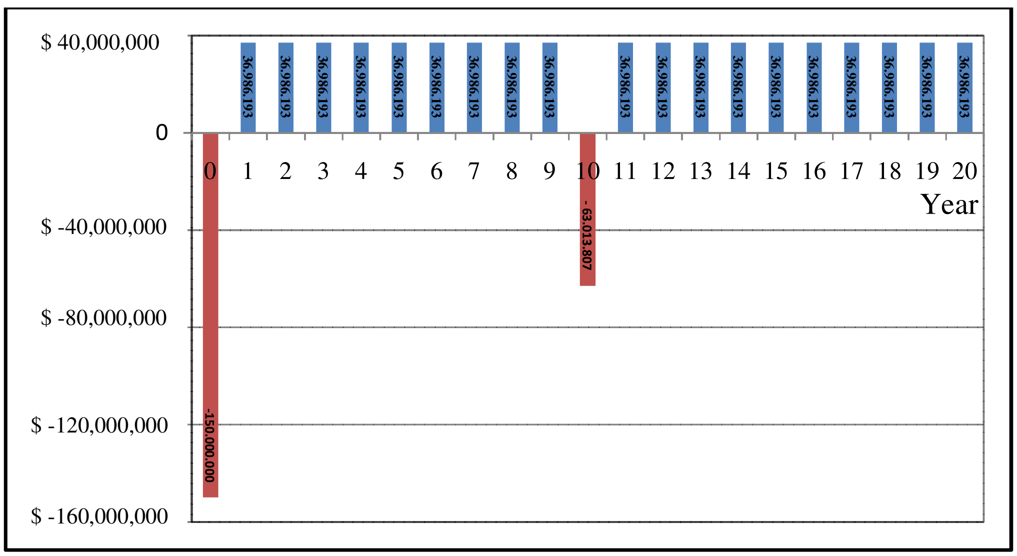


Figure 5.1 Cash Flow Diagram of the Southeastern Part of the Dereköy Copper Deposit without Uncertainty Assessment

Table 5.2 Cash Flow of the Southeastern Part of the Dereköy Copper Deposit without Uncertainty Assessment

Year	Gross Profit, $
0	-150,000,000
1	36,986,193
2	36,986,193
3	36,986,193
4	36,986,193
5	36,986,193
6	36,986,193
7	36,986,193
8	36,986,193
9	36,986,193
10	-63,013,807
11	36,986,193
12	36,986,193
13	36,986,193
14	36,986,193
15	36,986,193
16	36,986,193
17	36,986,193
18	36,986,193
19	36,986,193
20	36,986,193

After the estimation of the cash flow, NPV can be determined by using time value of money. NPV of the deposit is obtained from PV of the annual gross profits and capital costs. NPV of the deposit is found as $260,402,962 from equation (4.10). This means that the southeastern part of the Dereköy copper deposit will be feasible for the current situation. The feasibility of the deposit is also checked using internal rate of return (IRR) method. As mentioned in Chapter 2, IRR value of the investment must be greater than minimum attractive rate of return (MARR) for a feasible project. 15% MARR was defined for this investment. IRR value of the southeastern part of the Dereköy copper deposit was calculated as 22.2%. IRR value (22.2%) is greater than MARR value (15%), indicating that the project is feasible.

5.2 Estimation of NPV with Uncertainty Assessment

Unlike the traditional method explained above here the evaluation of the southeastern part of the Dereköy copper ore deposit will be done considering the uncertain variables in the model as indicated in equation (4.9) and equation (4.10). In the estimation 10,000 iterations are conducted and the results of iterations are saved by @Risk 4.5.7 software. Probability distribution of NPV of the deposit is estimated using the results obtained by iterations. In the model, inflation rate is considered as zero. It is also accepted that the deposit will be operated by the government and annual gross profit will be net profit because there will be no related tax paid to the government.

In the model, equation (4.9), the present values (PV) of annual gross profits of the deposit is estimated independently considering each year separately. In other words, selection of an input value is not affected by the other years selected values. For example, in the same scenario (in a single iteration) the first year's interest rate may be selected as 4.06% while the second year's interest rate is selected as 3.98% from the distributions. Therefore, verity of simulation is increased. Annual gross profit, annual income and annual operating and capital costs were estimated by the model in the similar manner.

5.2.1 Estimation of PV of Total Income

Estimation of annual income is found using equation (4.2). PV of total income is estimated by the summation of present values of annual incomes. 10,000 iterations are conducted for annual income for each year. Therefore, each year had different annual income distribution. 10,000 iterations were also done to estimate the PV of total income and distribution of PV of total income is indicated in Figure 5.2. Mean of the total income distribution was found as \$$1.800\times10^9$ PV of total income is found between \$$1.537\times10^9$ and \$$2.070\times10^9$ with 95% probability. The lowest and the highest values of PV of total income are found as \$$1.330\times10^9$ and \$$2.332\times10^9$ respectively.

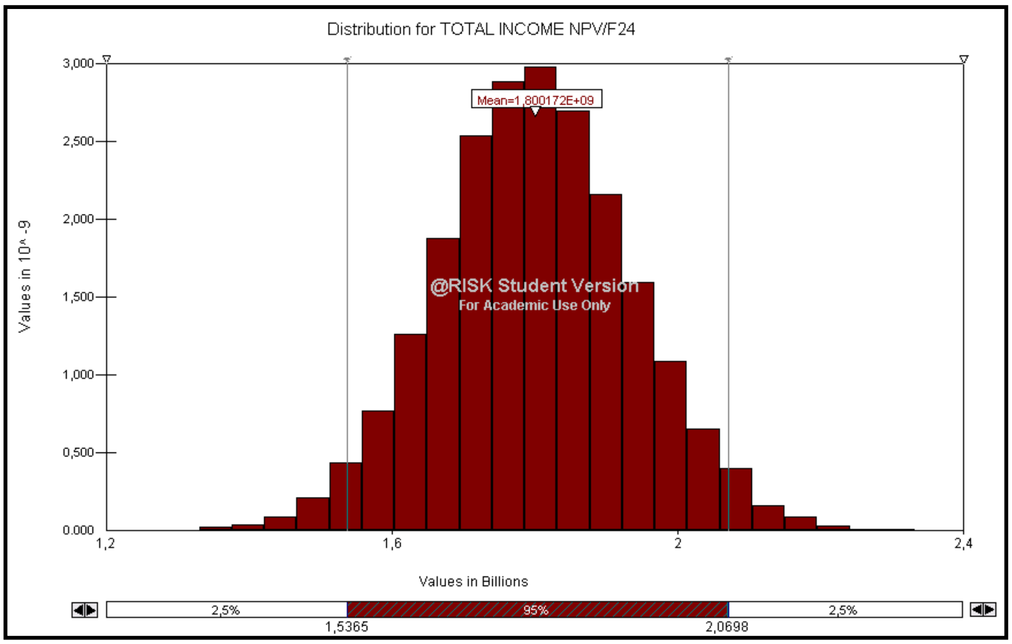
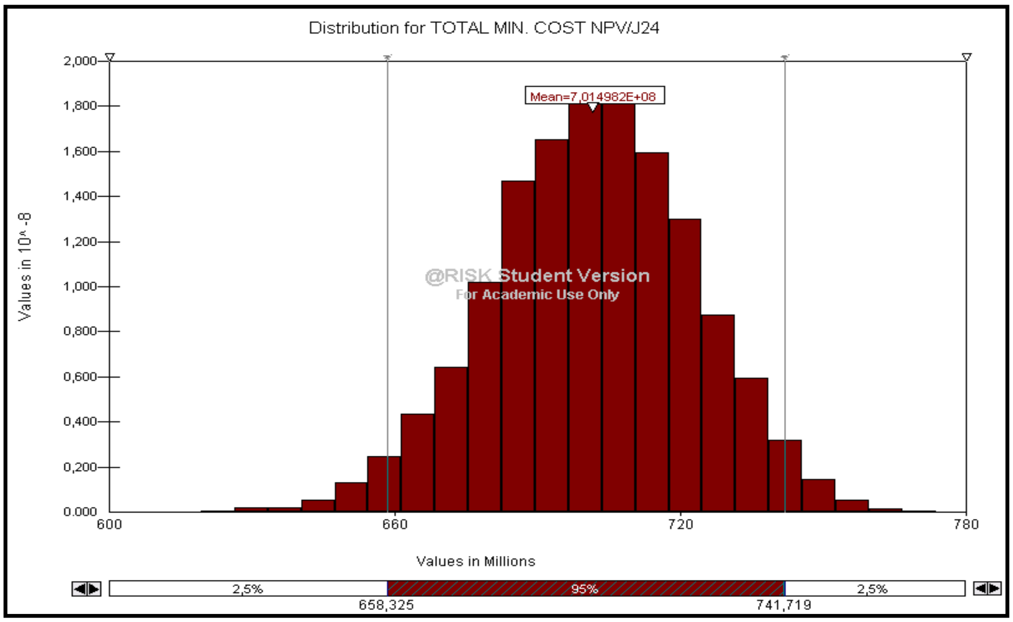


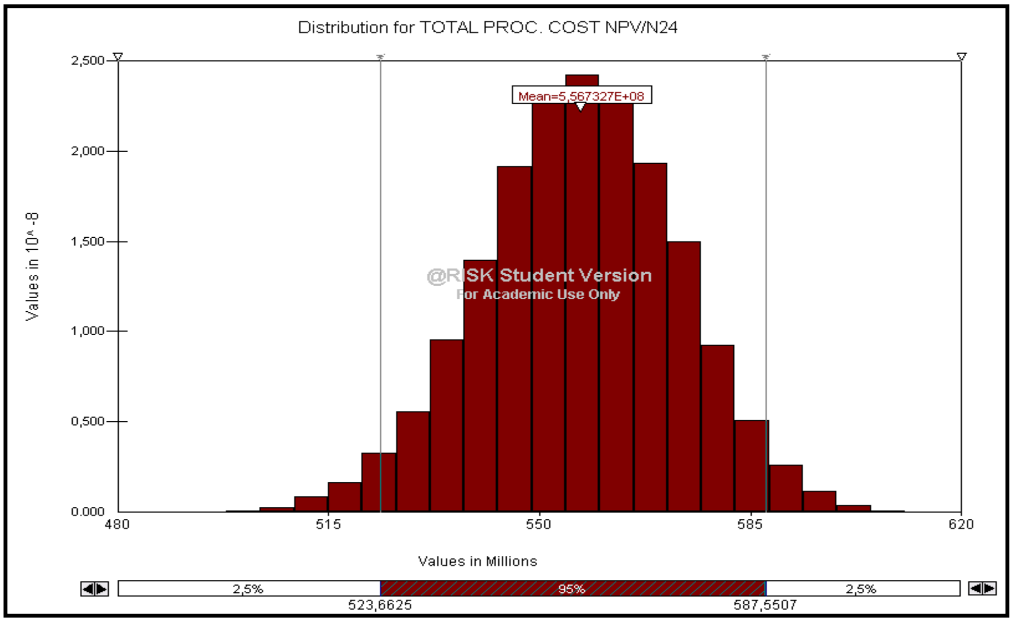
Figure 5.2 Distribution of PV of Total Income

5.2.2 Estimation of PV of Total Cost

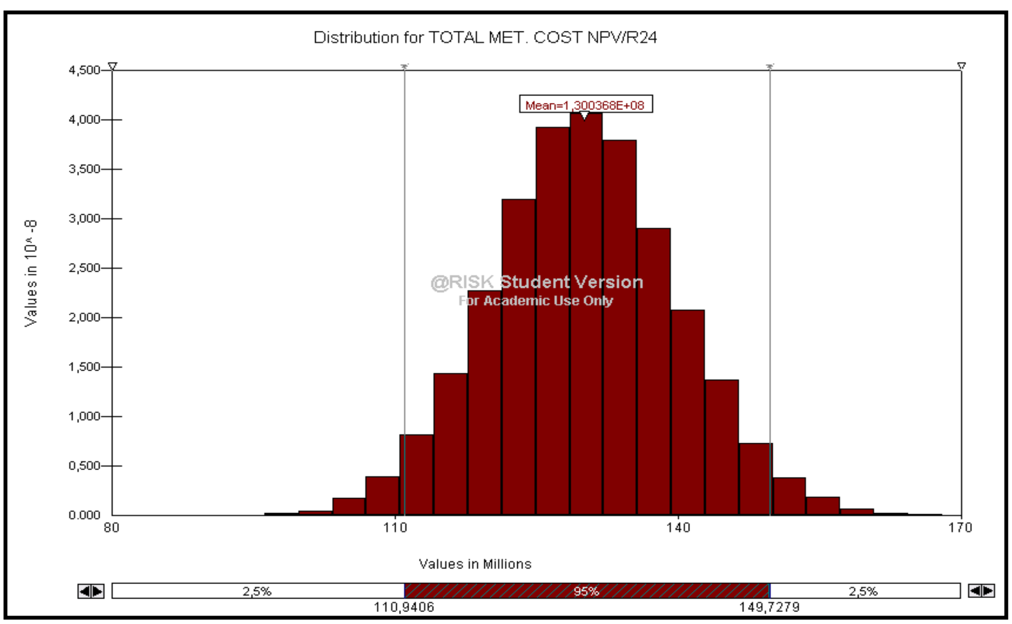
As it is mentioned in the previous section, the estimation of cost is more difficult than the estimation of income because there are more uncertain variables in estimation of costs. Therefore, operating cost is divided into three parts namely, mining cost, processing cost and metallurgical cost. Estimation of costs is done by the model using equations (4.3), (4.4), (4.5) and (4.6). Firstly, annual operating costs were estimated and then PV of total operating cost was estimated. Distributions of PV of total mining cost, total processing cost and total metallurgical cost are listed in Figure 5.3. PV of total operating cost distribution is also shown in Figure 5.4.



(a)



(b)



(c)

Figure 5.3 Distribution of Total Mining Cost (a), Total Processing Cost (b) and Total Metallurgical Cost (c)

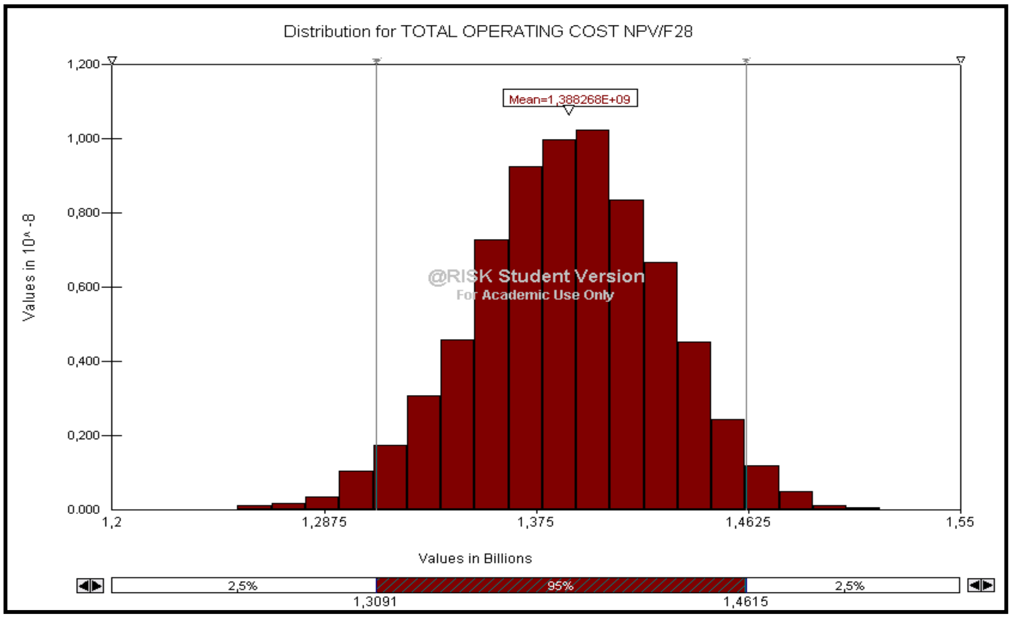


Figure 5.4 Distribution for Total Operating Cost

Besides operating costs, total capital cost is also estimated by the model using equation (4.8). As mentioned previously, two types of capital costs are defined in the model. They are initial cost of $150,000,000 at present and $100,000,000 at the end of the 10th year. PV of total capital cost is estimated by considering time value of money. Therefore, interest rate is selected from the interest rate distribution at the 10th year. Capital costs are defined as constants but PV of total capital cost is not constant because of the changes in the interest rate. 10,000 iterations are performed to select interest rate at the 10th year. Therefore, 10,000 random PV of total capital costs are estimated. The distribution of the estimated PV of total capital costs are presented in Figure 5.5.

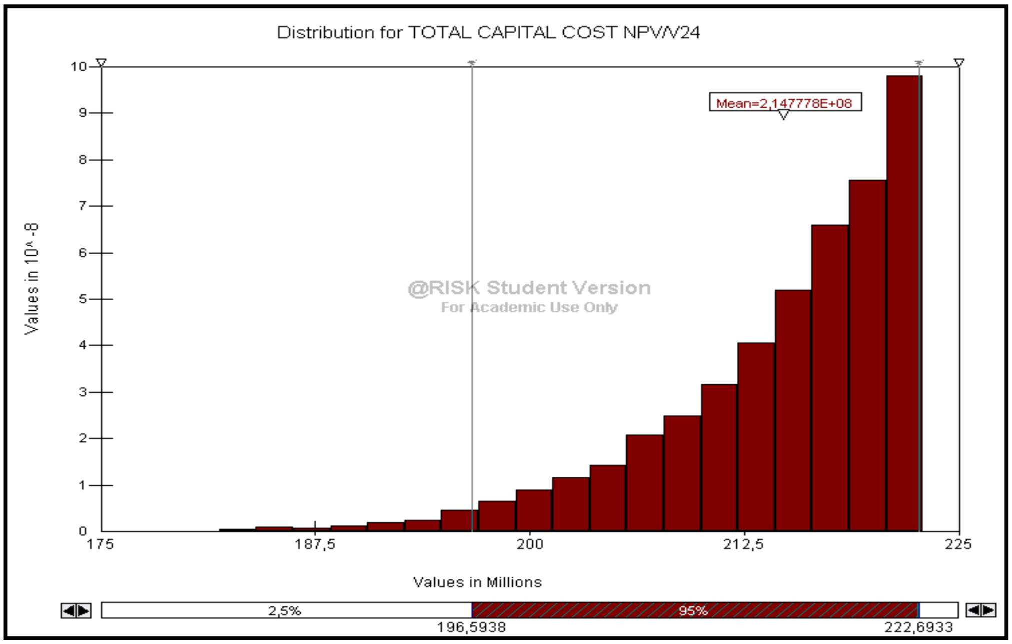


Figure 5.5 Distribution for Total Capital Cost

The model estimated the present value (PV) of the total cost after estimating the PV of total operating cost and capital cost. The model considered 10,000 random PV of total cost. Then, the probability distribution was established with 10,000 random values as presented in Figure 5.6. The probability distribution indicates that PV of the total cost is between $\$1.520\times10^9$ and $\$1,680\times10^9$ with 95% probability. Mean of the distribution is $\$1.603\times10^9$.

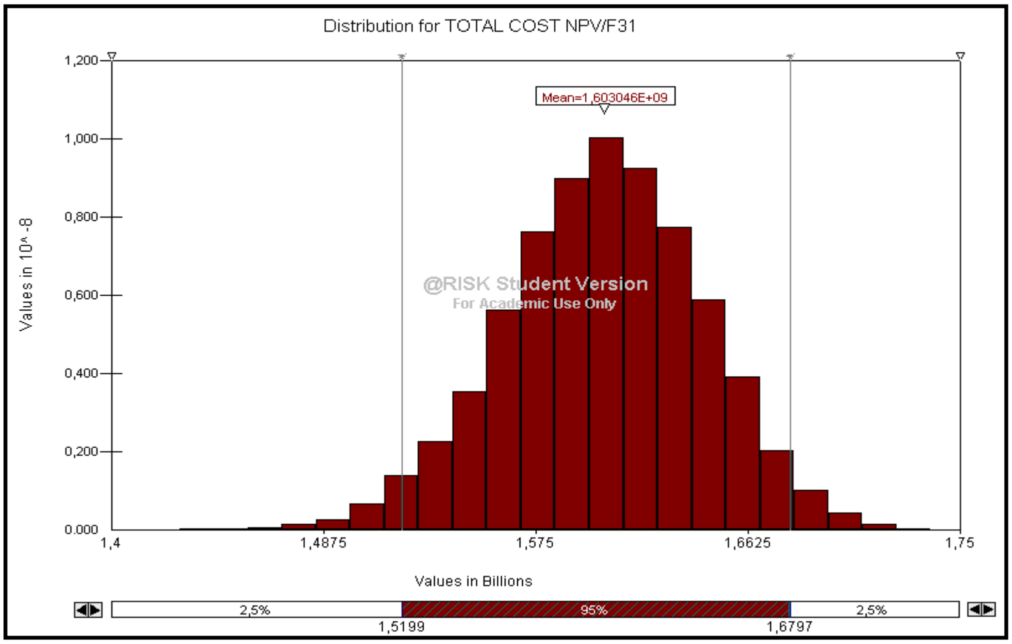


Figure 5.6 Distribution for Total Cost

5.2.3 Estimation of NPV of the Southeastern Part of Dereköy Copper Deposit

NPV of the deposit was estimated by the model as indicated in equation (4.10). The equation has many uncertain variables and each of them has some risks. The risks of them were defined and NPV of the deposit was estimated as a probability distribution. The simulation model estimated 10,000 random scenarios for each year to estimate the annual gross profits. The means of annual gross profit distributions of each year are presented as a cash flow in Table 5.3 and Figure 5.7. After constructing the distribution, PVs of annual gross profits were estimated. The PV of the annual gross profit distribution is very important because NPV of the deposit was estimated with the summation of the PV of annual gross profits.

Table 5.3 Cash Flow of the Southeastern Part of the Dereköy Copper Deposit with Uncertainty Assessment

Year	Annual Gross Profit, $
0	-150.000.000
1	31.565.550
2	31.431.660
3	31.310.470
4	31.593.580
5	31.476.720
6	32.087.910
7	31.132.620
8	31.504.230
9	31.137.480
10	-68.748.390
11	31.254.820
12	31.708.480
13	31.343.560
14	31.159.320
15	31.480.090
16	31.206.720
17	30.814.160
18	31.651.640
19	31.526.770
20	31.883.330

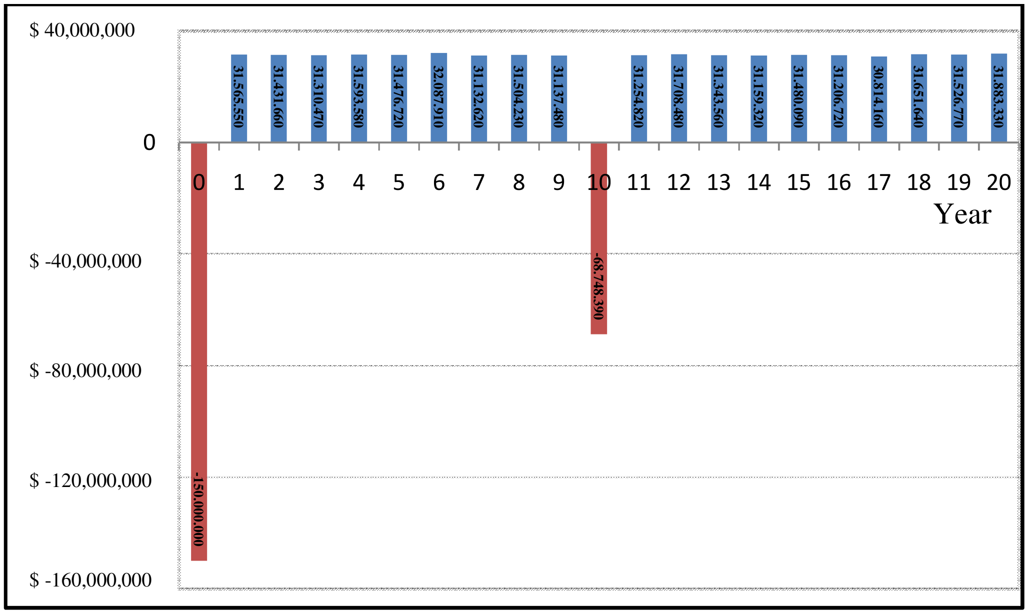


Figure 5.7 Cash Flow Diagram of the Southeastern Part of Dereköy Copper Deposit with Uncertainty Assessment

The model, finally, estimates 10,000 random NPV for the southeastern part of Dereköy copper deposit. With using these random values, @Risk 4.5.7 established a probability distribution and cumulative density curve to indicate the probability of the profit for the deposit. The probability distribution and cumulative density curve are presented in Figure 5.8 and Figure 5.9 respectively. Type of the NPV distribution was checked with "Fit Distribution" module of @Risk 4.5.7. The output of the fit distribution is presented in Figure 5.10. The output indicates that the NPV probability distribution is a normal distribution.

After the checking of the type of the distribution, the properties of it were investigated. Mean of the NPV probability distribution is $197,126,000 and standard deviation (σ) of the distribution was found to be $120,709,600. The risks can be evaluated by making use of the NPV probability distribution. One standard deviation interval ($\pm\sigma$ = 68.27%) and two "standard deviation interval ($\pm 2\sigma$ =

95.45%) were evaluated on the probability distribution. In this case NPV will be in the range of \$77.97 and \$318.78 million with the probability of 68.27% as it seen in Figure 5.11. Considering the $\bar{x}\pm2\sigma$, the range of NPV will be in the range of -\$45.37 to \$443.54 million with 95.45% probability as seen in Figure 5.12.

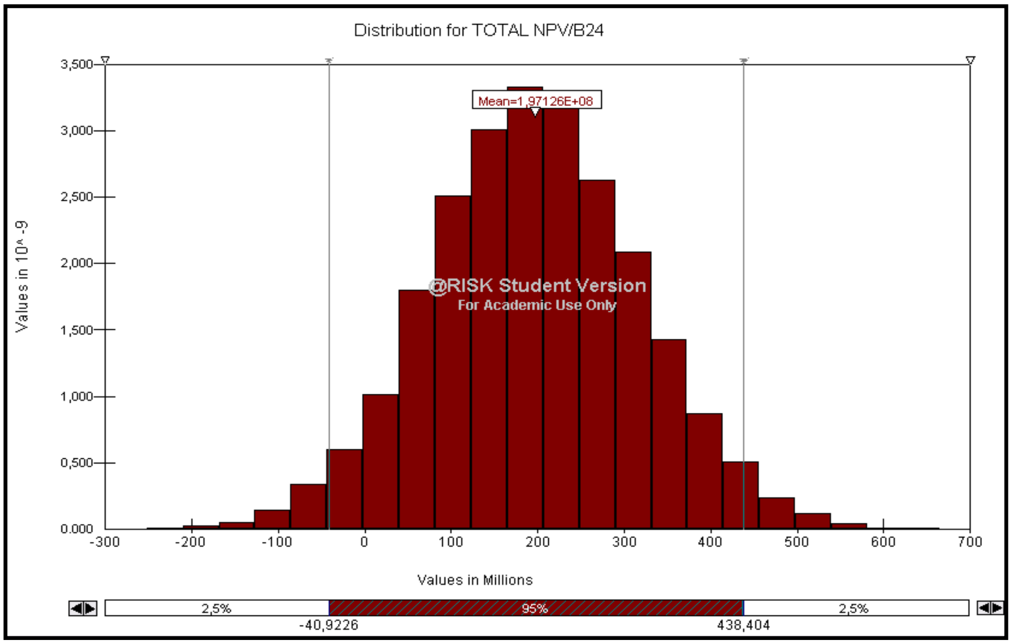


Figure 5.8 Frequency Distribution for NPV of the Deposit

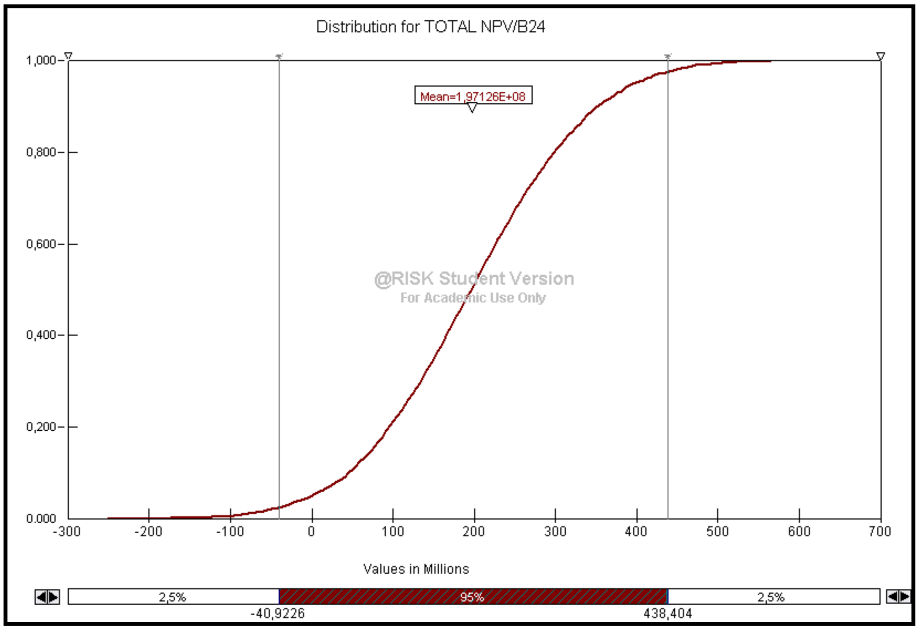


Figure 5.9 Ascending Cumulative Line for NPV with 95% Probability

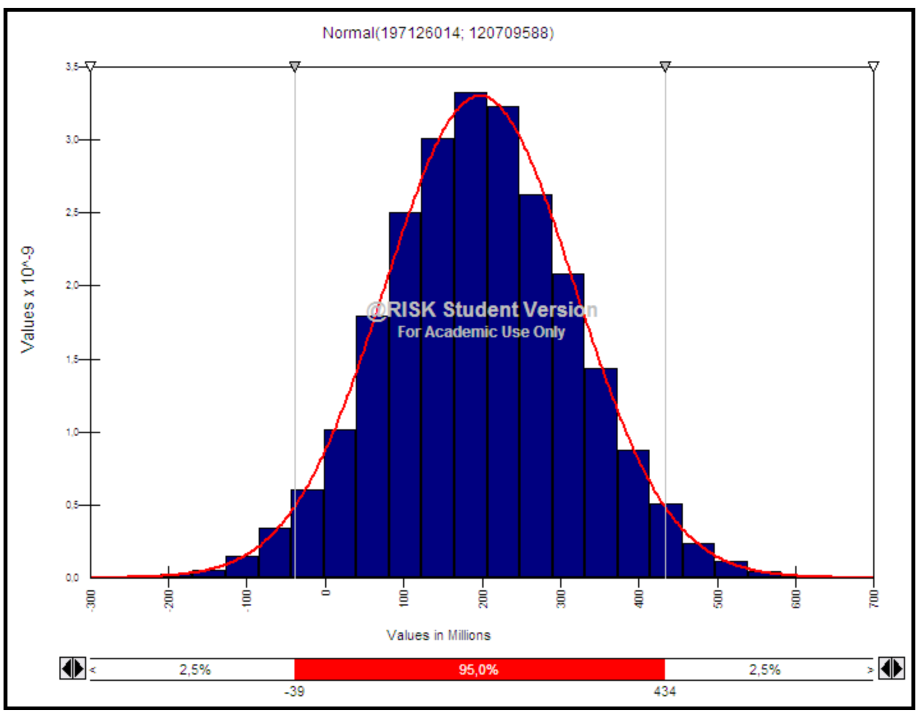


Figure 5.10 Fit Distribution Output for Estimated NPV Values

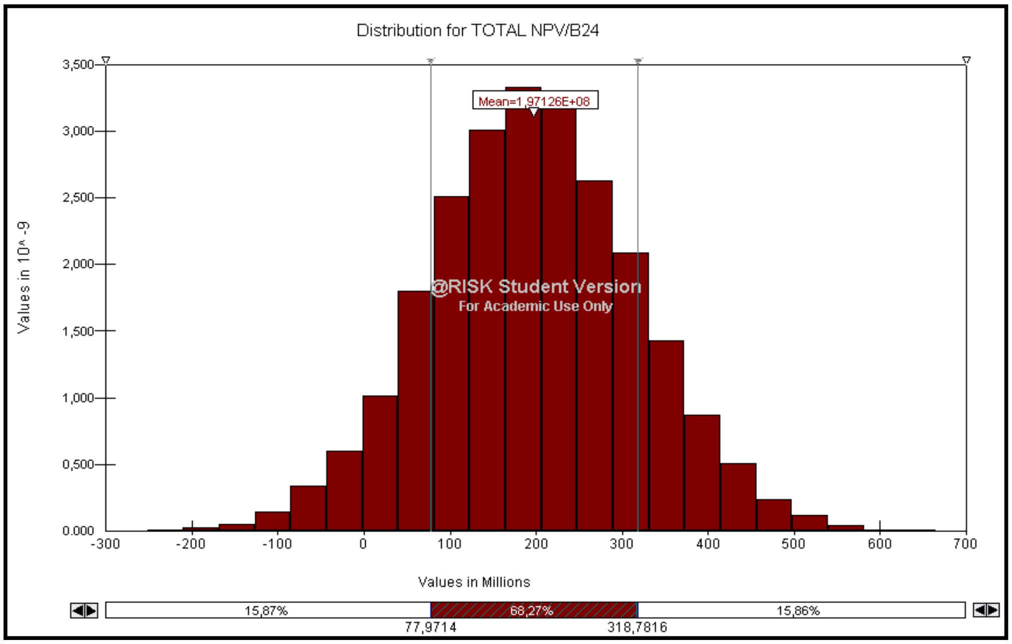


Figure 5.11 Probability of the NPV with 68.27% ($\bar{x}$±σ)

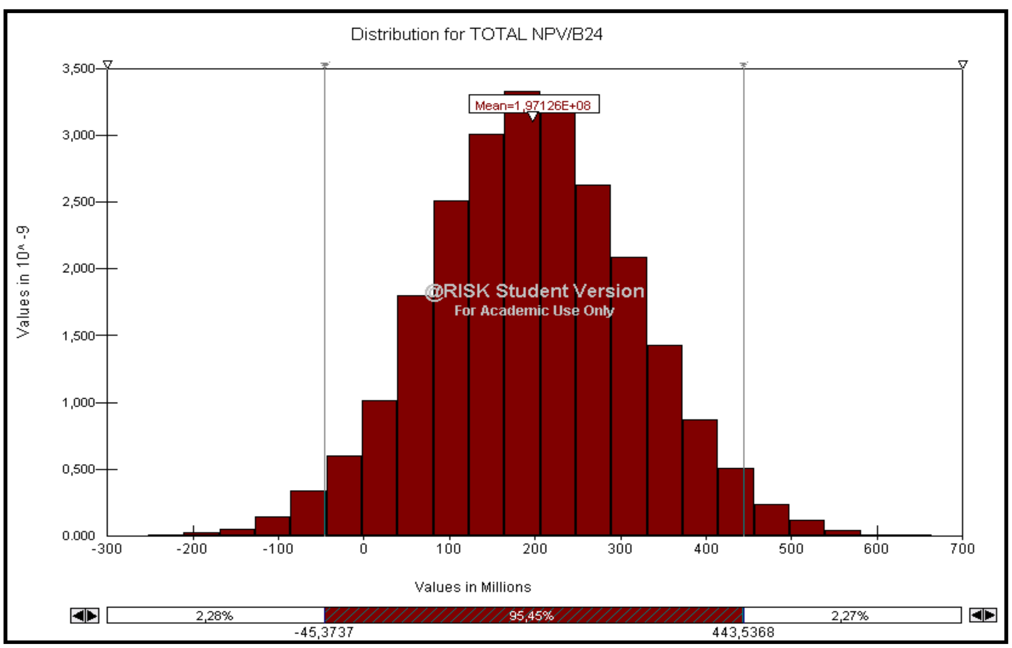


Figure 5.12 Probability of the NPV with 95.45% ($\bar{x}$±2σ)

When the case of 95.45% probability ($\bar{x}$±2σ) was investigated, it is obvious that the possibility of losing money is very low. The probability of getting positive NPV (profit) is 94.95% and probability of loss of money is only 5.05% as indicated in Figure 5.13.

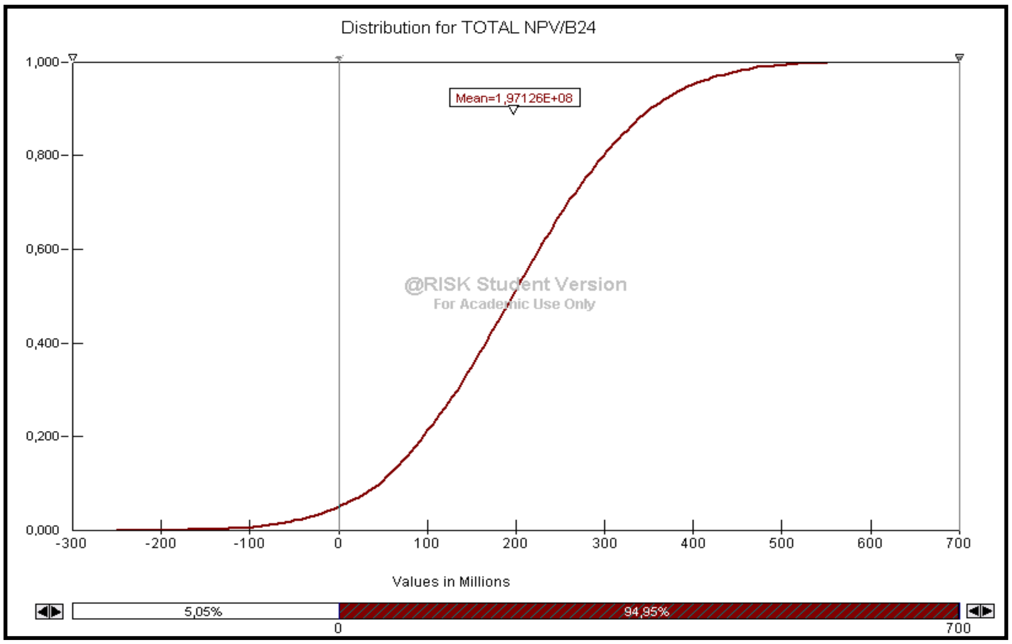


Figure 5.13 Ascending Cumulative Density Curve Indicating Positive NPV

IRR value of the investment analysis with uncertainty assessment was also conducted to check the feasibility of the investment. The model calculates the possible IRR values and then cerate a probability distribution with these values. The probability distribution for IRR is shown in Figure 5.14. Mean value of the distribution is 19.2%. IRR value is higher than defined MARR value with 62.73% as seen in Figure 5.14. The IRR method indicates that the project is feasible with 62.73%.

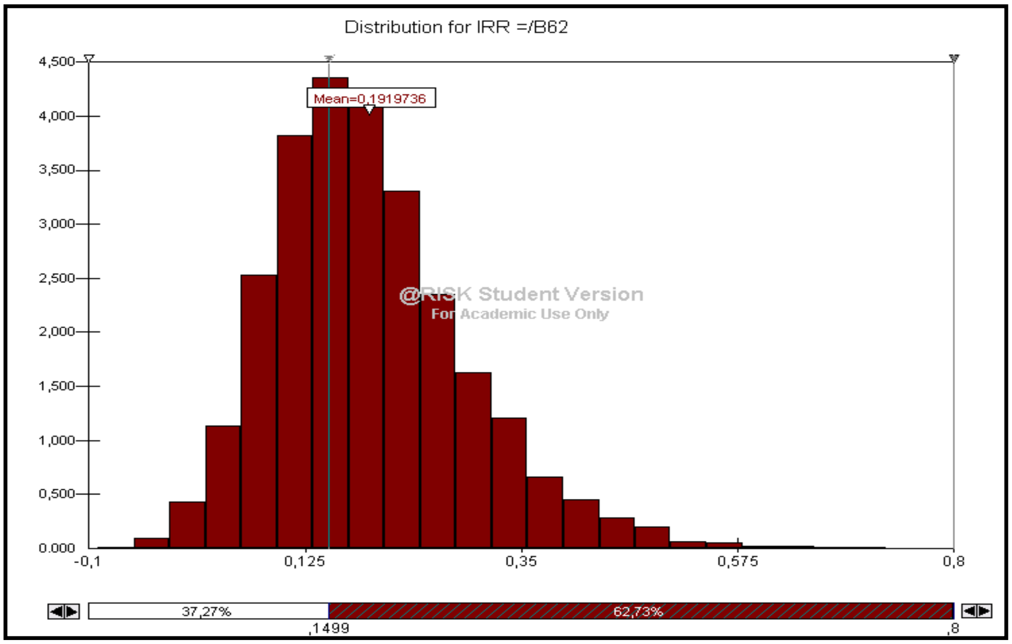


Figure 5.14 Probability Distribution for IRR

5.3 Sensitivity Analysis for NPV Estimation

Sensitivity analysis is applied to the project to determine which variable affects the estimation of the NPV of the deposit. It is found that NPV was the most sensitive to grade and secondly it was sensitive to selling price of copper. The results of the sensitivity analysis are shown in Figure 5.15. this analysis says that grade data should be upgraded when new data are available during the operation. The market conditions have also a significant impact on the NPV estimation because of the effect of the selling price.

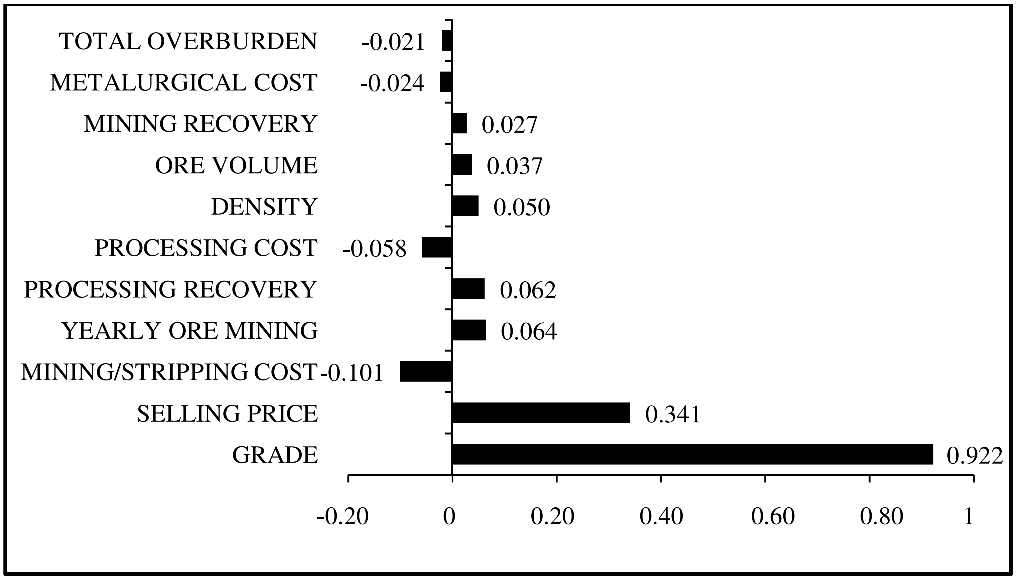


Figure 5.15 Results of Sensitivity Analysis

5.4 Discussions

There are many uncertainties as variables in the evaluation of the mineral reserves. A good financial model should be created to evaluate the ore reserves. Each uncertainty related with mining should be assessed carefully. In this case, accurate results can be obtained by the financial model. The decision of the mining investment is mostly related to NPV of the project. A financial model construction needs accurate estimations of income and costs. Estimation of the revenue and costs includes many uncertainties. Therefore, simulation method is the best method to estimate them. Simulation method can provide many hypothetical scenarios related with the project. The success of the financial modeling simulation depends on the estimation of the uncertainties accurately.

NPV of the southeastern part of the deposit was estimated with two methods. In the first method, estimation of the NPV was computed without uncertainty (risk) assessment. Therefore, the input data were given to the model as constants and a fix value was obtained as NPV ($260,402,962). IRR value was estimated as 22.2% while MARR is accepted as 15%. Positive NPV and satisfactory IRR value mean that the project is feasible.

However, this method does not consider the uncertainty of the variables. In other words, risk assessment of investment is not considered. Therefore, investor can not answer the questions like, what is the probability of NPV exceeding $100,000,000? or what are the probability of losing money? Mining is one of the risky operations. When a mining project is evaluated, the risks should be investigated and the risks should be included in the calculations and estimations.

In this study, the southeastern part of Dereköy copper deposit was evaluated considering related risks in this thesis. After the evaluation of the deposit, a probability distribution was estimated instead of a fix value and the type of the distribution was investigated as a normal distribution. Mean of the distribution is found as $197,126,000 and the probability of losing money is found as only 5.05% and the probability of NPV exceeding $150,000,000 (capital cost at present) is 65.05% as seen in Figure 5.16.

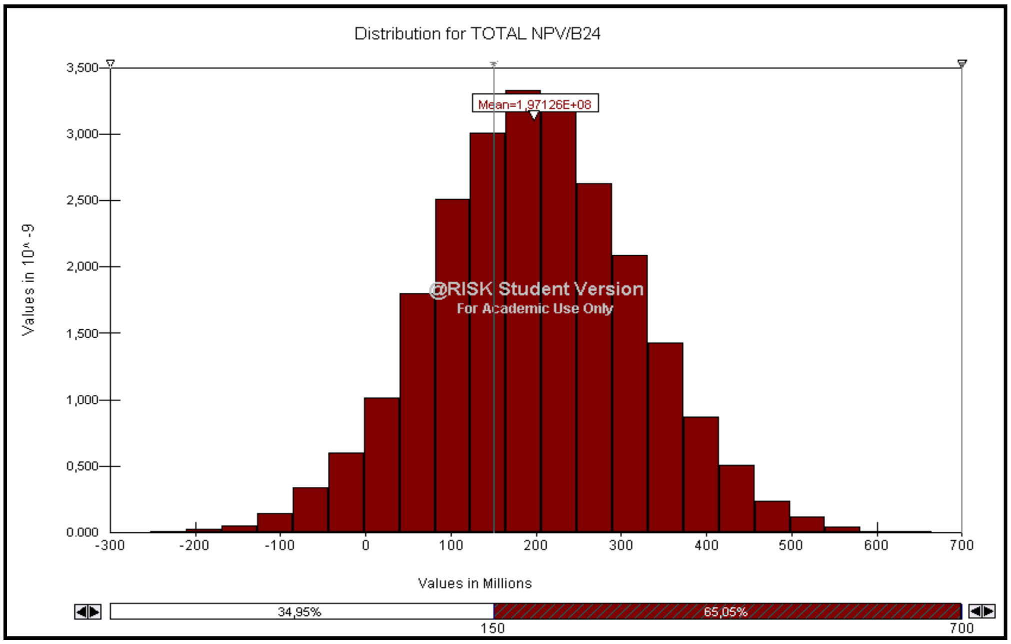


Figure 5.16 The Probability of NPV Exceeding $150,000,000

In traditional estimation method, a fix value was estimated. This is not a precise approach to mining having high risks in different stages. The probability of the realization of NPV equal and more than $260,402,962 (estimated by model without risk assessment) is 29.44% as indicated in Figure 5.16. When the IRR values are analyzed, traditional method estimates more IRR value than the other method. It means that the defined risks affected the estimation of the NPV because traditional model made an overestimation. The overestimation may cause some problems, such as loosing money, for investor because future of the investment is defined by result of NPV estimation. Therefore, NPV estimations should be conducted with considering related risks.

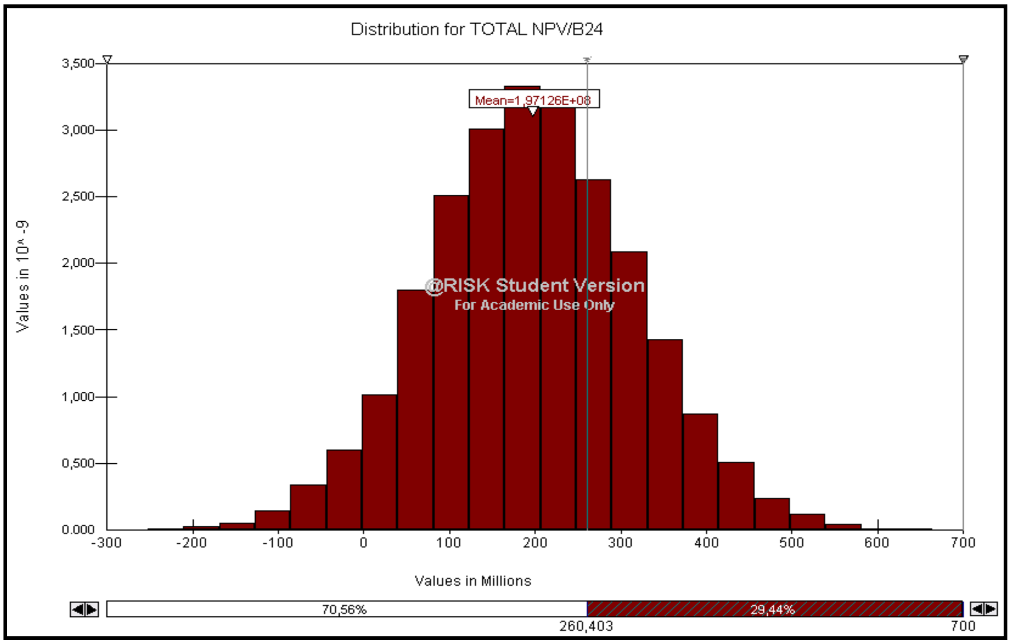


Figure 5.17 The Probability of NPV Exceeding NPV Estimation without Risk Assessment

CHAPTER 6

CONCLUSIONS AND RECOMMENDATIONS

The main conclusions obtained in this thesis are:

1. As it is stated in the text there are so many factors affecting the feasibility of mining investment. Traditionally, up to now effective factors are taken as constant although some uncertainties are involved in these factors. Since fix values are considered as inputs in the estimations, the risks involved in the estimation can not be defined. For healthy decision, the risk involved in the project should be determined before investment has been made. In this study, such a model which uses a simulation technique has been prepared.

2. The model constructed in this thesis has been successfully applied to a poor grade copper deposit which was accepted as mineral resources having no economic value.

3. In the study, not only the uncertainties exist in the reserve estimation but also uncertainties involved in the economical analysis were considered.

4. The application of Micromine software give the chance to evaluate complete deposit and part of it which has little higher grade than the whole body of deposit. The grade and tonnage of complete deposit are 0.106% and 1,740,805,517 ton respectively.

5. The grade and the tonnage of the part of reserve as mentioned in the previous item are 0.244% and 209,767,104 ton respectively using Micromine.

6. In the reserve estimation, the grade distribution of the copper deposit is found as lognormal distribution by fit distribution module of @Risk 4.5.7. This fact is important because in traditionally reserve estimation methods grade distribution is accepted as normal distribution and simple arithmetical is taken as the mean grade of the deposit.

7. In this study, some variables such as grade, density, selling price, interest rate, mining cost, and stripping cost are involved into the calculations as the probability distribution function. This application increased the accuracy of the result.

8. At the end of the NPV estimation of the southeastern part of Dereköy ore reserve, the NPV is between $–\$45.37\times10^6$ and $\$443.54\times10^6$ with 95.45% probability ($\bar{x}\pm2\sigma$) and between $\$77.97\times10^6$ and $\$318.78\times10^6$ with 68.27% ($\bar{x}\pm\sigma$) probability. The probability of positive NPV from the reserve is 94.95%. Therefore, the project will be feasible for the situations and parameters in this study.

The main recommendations related to this study are:

1. When a mining project is evaluated, all relevant uncertainties should be evaluated in detailed to get more accurate results. Assessment of the uncertainties is very important because accurate estimation of the uncertainties have a vital role on accuracy of the simulation of the financial model.

2. Number of drillhole is not enough when compared the area of the study location. Therefore, number of the drillholes may be increased with new drillholes.

3. Time series analysis can be conducted for the commodity price. Therefore, 20 years' or more commodity price can be evaluated to get more accurate results.

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APPENDIX A

COPPER RESERVES

A.1 World Copper Reserves

It is indicated in SPO Report (2006) that there are two types of copper ore namely sulfide type and oxide type. In the world, 80% of copper reserve is sulfide type. In the world, the most important copper zone passes through west of America, Chile, Peru, Mexico, Arizona, New Mexico, Nevada, Utah, and Canada. This copper zone represents the 50 percent of copper production of the western-world. There are two important zones for porphyry type copper mineralization. The first one is located in Indonesia, Papua New Gina, Philippines. The other starts from the southeastern Europe and continues up to Iran and Pakistan. Moreover, big sulfide type copper reserves exist at the eastern Canada, the northern America, Spain, Namibia, South Africa, and Australia (SPO, 2001).

It is estimated that proven copper reserve of world is 650 million tons. Percent distribution of this reserve among countries is shown in Table A.1 and Table A.2 respectively (SPO, 2001).

Table A.1 Percent Distribution of World Copper Reserves (SPO, 2001)

	Percentage, %
Developed Countries	24
OEDC	26
CIS and East Europe	16
Developing Courtiers	34
TOTAL	100

Table A. 2 Country Distribution of World Copper Reserves (SPO, 2001)

Country	**Proven Reserve, 10^6 tons**	**Probable Reserve, 10^6 tons**
USA	45	90
Australia	7	23
Canada	10	23
Chile	88	160
China	18	37
Indonesia	19	25
Kazakhstan	14	20
Mexico	15	27
Peru	19	40
Hungary	20	36
Russia	20	30
Zambia	7	13
Others	50	105
TOTAL	332	629

World copper reserves are geologically composed of porphyry, sulfide and sedimentary type copper reserves. U.S. Geological Survey estimates that total copper reserve of world (proven + probable + possible) is 1.6 billion tons (SPO, 2001).

It is known that the economically operated porphyry type copper deposits exist in And mountains (west America), Philippines and Alp orogenic zones. The And zone consists of near continent type porphyry copper reserve but the Philippines zone consists of island arc type porphyry copper reserve. The Alp zone consists of both types of porphyry copper reserves. Table A.3 and Table A.4 show the important copper reserves in these zones and it contains some porphyry type copper reserves in Turkey (SPO, 2001).

The part of Alp orogenic's zone between Samsun and Georgia is named as Pondit metelogenic zone. On the contrary Istranca Massive, this zone composes Philippines type island arc zone. This island arc zone continues up to Caspian Sea. In the Philippines zone, there are some porphyry type reserves such as Santo Thomas II, Dizon, Atlas-Lotopan Sipalay, Canabit. There are some porphyry types copper reserves at Black Seas region and their ore properties of these reserves are same with the Philippines porphyry type reserves. Their average grades are lower than Philippines's porphyry type copper reserves.

Table A.3 The Important Porphyry Copper Reserves and Their Geotectonic Locations, Near Continent Type (SPO, 2001)

Near Continent Type Porphyry Copper Reserves			
Zone Name	**Location**	**Reserve, 10^6 tons**	**Average Grade, %**
And Zone	Chuqicamata, Chile	>500	1,7 Cu
And Zone	Broden, Chile	>500	2,25 Cu 0,05 Mo
And Zone	El Salvador, Chile	>500	1,5 Cu
And Zone	Toquepala, Peru	>500	0,9 Cu
And Zone	Bingham, Utah	>500	0,75 Cu 0,05 Mo
And Zone	Ray, Arizona	>500	0,8 Cu
And Zone	Inspiration, Arizona	>500	0,9 Cu
Alp Zone	Bor, Yugoslavia	90	<1 Cu
Alp Zone	Maydenpek, Yugoslavia	500	0,6 Cu
Alp Zone	Medet, Bulgaria	>150	0,66 Cu
Alp Zone	Sar Çaşme, Iran	450	0,4
Alp Zone	Dereköy, Kırklareli	200	0,27 Cu+Mo
Alp Zone	Bakırçay, Merzifon	200	0,2 Cu+Mo

Table A.4 The Important Porphyry Copper Reserves and Their Geotectonic Locations, Island Arc Type (SPO, 2001)

Island Arc Type Porphyry Copper Reserves			
Zone Name	**Location**	**Reserve, 10^6 tons**	**Average Grade, %-gr/ton**
Philippines Zone	Santo Thomas II	328	0,34 Cu 0,61 Au 1,5 Ag
Philippines Zone	Dizon	105	0,43 Cu 0,003 Mo 0,93 Au 2,5 Ag
Philippines Zone	Tapian (closed at 1991)	177	0,52 Cu 0,12 Au 0,4 Ag
Philippines Zone	Atlas, Lotopan	860	0,42 Cu 0,027 Cu 0,31 Au
Philippines Zone	Sipalay, Canabit	740	0,49 Cu 0,015 Mo 0,050 Au 1,5 Ag
And Zone	Güzelyayla, Maçka	154	0,3 Cu+Mo
And Zone	Ulutaş, Ispir	20	0,4 Cu
And Zone	Balcılı, Yusufeli	140	0,2 Cu
And Zone	Kafan, Armenia	145	0,25 Cu+Mo

A.2 Copper Reserves in Turkey

In Turkey, there are almost 650 outcrops studied by MTA, national and international companies. Generally, origin of the copper reserves is magmatic. Copper and pyrite reserves generally emergence as copper-pyrite ore or Cu-Pb-Zn-Pyrite ore.

Copper reserves of Turkey can be classifies as;

- Porphyry copper reserves
- Massive sulfide copper reserve
- Hydrothermal veins and Contact metamorphic reserves

There are many hydrothermal veins and contact metamorphic reserves but their copper content is not high. Porphyry type copper reserves are not suitable for mining if copper price is low because their grade is low. Massive sulfide copper reserves are most important reserves for copper ore mining. Murgul, Çayeli-Madenköy, Lahanos, Ergani, Siirt-Madenköy, Cerattepe and Küre are massive sulfide type copper reserves (SPO, 2001).

Turkey's porphyry copper reserves can be studied in Alp orogenic zone. This zone starts from Balkans and pass from Istranca massive, Black Sea and Iran and ends with Himalayas. There are some operated porphyry type copper reserves in this zone. Some of them are Bor and Maydenpek (Yugoslavia), Medet (Bulgaria), and Sar Çaşme (Iran).

Dereköy and Bakırçay (Merzifon) porphyry copper reserves show almost the same properties with And type near continent porphyry copper reserves. Average grade of Dereköy and Bakırçay reserves are lower than the porphyry type copper reserves in Balkans.

As it is determined in 2000, the proven copper reserve of Turkey is about 1,697,204 tons as Cu content is considered (SPO, 2001). The detailed information about the copper reserves of Turkey is given in Table A.5.

Table A.5 Feasible Copper Reserves of Turkey (SPO, 2001)

City	Town	Location	Reserve, 10^3 tons	Cu, %	Zn,%	Au, g/t	Ag, g/t	Copper, tons
Artvin	Murgul	Damar	2,503	1.24	-	-	-	31,037
Artvin	Murgul	Çakmakkaya	5,714	0.84	-	-	-	47,998
Artvin	Murgul	Akerşen	582	2.24	4.70	-	219	13,037
Artvin	Merkez	Cerattepe	3,900	5.20	-	1,23	25,3	202,800
Artvin	Merkez	Seyitler	2,465	1.41	-	-	-	34,757
Ç.kale		Arapuçuran	1,230	1.25	-	-	-	15,375
Elazığ	Ergani	Anayatak	600	1.71	-	-	-	10,260
Giresun	Espiye	Lahanos + Kızılkaya	2,402	2.40	2.42	-	-	57,648
Giresun	Tirebolu	Harköy	498	1.90	-	-	-	9,462
Kastamonu	Küre	Bakibaba + Aşıköy	12,339	2.05	-	-	-	252,950
Rize	Çayeli	Madenköy	10,900	4.61	7.50	-	-	502,490
Sivas	K.hisar	Kan	964	1.73	-	-	-	16,677
Trabzon	Of	Kotaraktepe	963	1.31	-	-	-	12,615
Trabzon	Yomra	Kanköy	3,310	1.11	-	-	-	36,741
	TOTAL		**48,370**	**2.57**	**-**	**-**	**-**	**1,243,847**

APPENDIX B

COPPER PRICES BETWEEN JAN 1990 and SEP 2008

Table B.1 Price of Copper between Jan-1990 and Sep-2008

Date	Price	Date	Price	Date	Price
Jan-90	2365.56	May-92	2217.85	Sep-94	2505.93
Feb-90	2361.15	Jun-92	2297.21	Oct-94	2547.67
Mar-90	2623.50	Jul-92	2522.09	Nov-94	2802.45
Apr-90	2687.43	Aug-92	2513.27	Dec-94	2985.30
May-90	2740.34	Sep-92	2420.67	Jan-95	3008.93
Jun-90	2581.61	Oct-92	2253.12	Feb-95	2877.65
Jul-90	2766.80	Nov-92	2158.32	Mar-95	2924.04
Aug-90	2951.99	Dec-92	2209.03	Apr-95	2903.50
Sep-90	3029.15	Jan-93	2257.53	May-95	2773.31
Oct-90	2742.55	Feb-93	2213.44	Jun-95	2994.64
Nov-90	2586.02	Mar-93	2151.71	Jul-95	3075.67
Dec-90	2491.22	Apr-93	1951.09	Aug-95	3036.84
Jan-91	2447.13	May-93	1794.56	Sep-95	2915.52
Feb-91	2444.102	Jun-93	1854.09	Oct-95	2813.55
Mar-91	2416.26	Jul-93	1926.84	Nov-95	2977.36
Apr-91	2469.17	Aug-93	1946.68	Dec-95	2926.26
May-91	2374.38	Sep-93	1862.90	Jan-96	2616.41
Jun-91	2215.64	Oct-93	1646.85	Feb-96	2537.71
Jul-91	2231.08	Nov-93	1629.21	Mar-96	2561.02
Aug-91	2231.08	Dec-93	1724.01	Apr-96	2595.78
Sep-91	2321.46	Jan-94	1805.35	May-96	2658.26
Oct-91	2361.15	Feb-94	1866.40	Jun-96	2173.40
Nov-91	2376.58	Mar-94	1914.107	Jul-96	1985.57
Dec-91	2213.44	Apr-94	1881.82	Aug-96	2008.57
Jan-92	2140.69	May-94	2150.60	Sep-96	1941.45
Feb-92	2204.62	Jun-94	2364.20	Oct-96	1961.17
Mar-92	2228.87	Jul-94	2458.19	Nov-96	2230.86
Apr-92	2215.64	Aug-94	2406.23	Dec-96	2268.08

Table B.1 Price of Copper between Jan-1990 and Sep-2008 (cont'd).

Date	Price	Date	Price	Date	Price
Jan-97	2434.103	Jan-00	1843.98	Jan-03	1647.66
Feb-97	2405.85	Feb-00	1800.83	Feb-03	1683.80
Mar-97	2421.29	Mar.00	1739.39	Mar-03	1658.98
Apr-97	2391.18	Apr-00	1678.75	Apr-03	1587.48
May-97	2514.33	May.00	1785.62	May-03	1648.28
Jun-97	2612.62	Jun-00	1753.18	Jun-03	1686.50
Jul-97	2450.46	Jul-00	1799.36	Jul-03	1710.00
Aug-97	2251.2	Aug-00	1855.86	Aug-03	1760.28
Sep-97	2107.3	Sep-00	1960.40	Sep-03	1789.52
Oct-97	2052.26	Oct-00	1898.59	Oct-03	1920.54
Nov-97	1917.45	Nov-00	1795.11	Nov-03	2055.43
Dec-97	1762.33	Dec-00	1850.55	Dec-03	2201.29
Jan-98	1688.45	Jan-01	1787.50	Jan-04	2423.57
Feb-98	1664.10	Feb-01	1765.65	Feb-04	2759.53
Mar-98	1747.98	Mar-01	1738.77	Mar-04	3008.72
Apr-98	1800.9	Apr-01	1664.16	Apr-04	2948.73
May-98	1732.53	May-01	1682.21	May-04	2733.50
Jun-98	1660.52	Jun-01	1608.45	Jun-04	2686.70
Jul-98	1651.04	Jul-01	1525.20	Jul-04	2808.43
Aug-98	1620.93	Aug-01	1464.43	Aug-04	2846.10
Sep-98	1647.64	Sep-01	1426.33	Sep-04	2894.106
Oct-98	1586.39	Oct-01	1377.28	Oct-04	3012.24
Nov-98	1573.95	Nov-01	1427.73	Nov-04	3122.80
Dec-98	1473.57	Dec-01	1471.74	Dec-04	3145.45
Jan-99	1431.18	Jan-02	1503.95	Jan-05	3170.00
Feb-99	1410.78	Feb-02	1561.90	Feb-05	3253.70
Mar.99	1378.35	Mar-02	1604.108	Mar-05	3379.60
Apr-99	1466.00	Apr-02	1590.33	Apr-05	3394.48
May-99	1511.16	May-02	1595.68	May-05	3249.10
Jun-99	1422.48	Jun-02	1647.53	Jun-05	3524.07
Jul-99	1640.00	Jul-02	1589.46	Jul-05	3614.21
Aug-99	1647.62	Aug-02	1479.55	Aug-05	3797.75
Sep-99	1750.34	Sep-02	1478.71	Sep-05	3857.84
Oct-99	1724.12	Oct-02	1483.76	Oct-05	4059.76
Nov-99	1727.55	Nov-02	1582.29	Nov-05	4269.34
Dec-99	1764.75	Dec-02	1595.68	Dec-05	4576.78

Table B.1 Price of Copper between Jan-1990 and Sep-2008 (cont'd).

Date	Price	Date	Price	Date	Price
Jan-06	4734.33	Dec-06	6691.78	Nov-07	6966.7
Feb-06	4982.4	Jan-07	5669.66	Dec-07	6587.67
Mar-06	5102.85	Feb-07	5676.45	Jan-08	7059.18
Apr-06	6387.78	Mar-07	6452.48	Feb-08	7886.24
May-06	8045.86	Apr-07	7766.47	Mar.08	8437.13
Jun-06	7197.61	May-07	7682.17	Apr-08	8683.09
Jul-06	7712.1	Jun-07	7475.88	May-08	8380.33
Aug-06	7695.66	Jul-07	7973.91	Jun-08	8259.43
Sep-06	7602.36	Aug-07	7509.57	Jul-08	8412.35
Oct-06	7500.39	Sep-07	7648.98	Aug-08	7632.93
Nov-06	7029.18	Oct-07	8008.43	Sep-08	6988.43