

**THE CONTRIBUTION OF GEOMECHANICS AND ENGINEERING
GEOLOGY TO MINE ENTERPRISE VALUE**

by

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Abstract

The objective of this thesis is to identify the value of geomechanics and engineering geology to mine enterprise value for hardrock underground mines. It was decided that the most effective way to highlight the value of geomechanics and engineering geology was by identifying an increase in expenditure that could be economically justified in the present to mitigate the cost of a future event, thus providing a means for showing the economic value of the work performed.

Cost models were generated for several events based on the direct cost, value of ore lost and decline in value of ore due to the event. A cost associated with fatalities was also included. Six rockburst events were developed into cost models from publicly available information. A further 13 were developed from confidential information provided by mining companies, bringing the total number of events analyzed to 19.

A probabilistic approach was then taken to identify the probability of a rockburst with a certain magnitude occurring and, if an event occurs, the probability it will cause damage. The former is based on the Gutenberg-Richter Frequency-Magnitude relationship while the latter was derived from Unusual Occurrence Reports provided by the Ontario Ministry of Labour. Three case studies were then developed to show how to use the average cost of a rockburst event in conjunction with the probability analysis to arrive at an increase in expenditure above baseline spending. The first case study is based on the Unusual Occurrence Reports provided by the Ontario Ministry of Labour and analyzes Ontario operations as a whole. The second and third are based on information provided by mining companies and are defined as Mine A and Mine B respectively.

It was found that the total average cost of a rockburst based on the 19 events analyzed from 13 mines in 4 different countries for events occurring between 1984 and 2009 is \$35.4 million (2010 CAD) with a range of \$1.1 to \$263.5 million (2010 CAD). Using the probabilistic method outlined above and cost models from the specific region involved, the increase in expenditure for the Ontario hard rock underground case study, Mine A and Mine B was found to be \$12.1 million (2010 CAD), \$5 million (2010 CAD) and \$4.0 million (2010 CAD) respectively.

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Glossary of Terms

Nuttli Magnitude: Referred to as M_n , a magnitude scale of base 10 developed by Otto Nuttli that is used to define seismic events that occur in eastern North America

Richter Magnitude: Referred to as M_L or local magnitude, developed by Charles Richter to define California seismicity, a scale of base 10, used to define seismic events that occur in Western North America and around the world

Canadian Dollars: Referred to as CAD, currency used in Canada. All foreign currencies discussed in this report are converted to Canadian Dollars

Australian Dollars: Referred to as AUD, currency used in Australia. Average annual exchange rates from the Bank of Canada are used to convert Australian dollars to Canadian dollars

United States Dollars: Referred to as USD, currency used in the United States of America. Average annual exchange rates from the United States Federal Reserve are used to convert United States dollars to Canadian dollars

Tonnes: Refers to metric tonnes, mt. 1 tonne is equal to 1000 kilograms (kg)

Tons: Refers to empirical tons or short tons, t. 1 ton is equal to 0.907185 tonne

Gold: referred to as Au

Zinc: referred to as Zn

Copper: referred to as Cu

Gearing: Refers to a company's financial leverage. Defined as a company's debt as a percentage of its equity.

Chapter 1

Introduction

A phenomena first discussed by Agricola in 1556, rockbursting has long been identified as a risk to underground mining operations. As mines progress deeper, the in-situ stresses increase, due primarily to the increasing weight of overburden and tectonic activities. This leads to an increased risk of rockbursting events and the severity associated with them.

When an excavation is created, the pre-existing equilibrium within the rock is altered, and a new equilibrium is formed with an increased stress around the excavation opening. If the stress around the opening remains less than the strength of the rock, equilibrium is regained. If the stress exceeds the strength, the stress is released either violently or non-violently, depending on the properties of the rock (Hedley, 1992).

Geomechanics studies can be performed to assess the rock quality prior to excavation and support systems can be designed and installed to either reinforce the rock, thus increasing the strength to a point greater than the stress that is being applied, or to retain the damage that might be caused by a rockburst. Support systems often combine both elements and are held in place by linking reinforcement and retaining elements to solid rock past the failure zone (Kaiser, et al., 1996).

The challenge is in the economics of mine operations. Mines are not designed to last forever, as they are built with the understanding that they will eventually fail. Archibald (2002) notes that

“Eventually, all forms of introduced support will fail or degrade.” The key is designing an opening that will last long enough to safely extract the ore. This leads to a culture that encourages not spending more than absolutely necessary to perform the operation safely.

The technical nature of geomechanics engineering and engineering geology can make it difficult to show the economic value of the work being performed, thus making it challenging to show the need for an increase in funding or justification for continued funding levels.

1.1 Problem Statement

During a workshop held by the Centre for Excellence in Mining Innovation (CEMI) there was a discussion on geomechanics and engineering geology through the mine life cycle (pre-feasibility to production). Concerns were raised that, while in the initial stages of mine design, a heavy focus is placed on the development of models and interpretation of technical data related to geomechanics and engineering geology, once the mine enters into production the focus switches to maintaining profits and meeting production targets.

This switch in focus leads to geomechanics and structural geology models being treated as static designs instead of dynamic models. This can lead to inadequate support systems being designed based on inappropriate and outdated models which can ultimately result in costly ground failures and production delays. Furthermore, it forces geomechanics engineers and engineering geologists to be reactive instead of proactive, only being brought in during crisis situations.

1.2 Research Objectives

The principal objective of this thesis is to attempt to define geomechanics and engineering geology in terms of economic value to mine enterprise.

The value of geomechanics and engineering geology can be assessed by analyzing the costs associated with rockbursting events and identifying the probability that an event will occur.

Therefore, the objectives of this research are:

- To identify the financial parameters that are impacted by rockbursting
- To arrive at a generalized average cost per rockbursting event
- To investigate an appropriate method for calculating the probability that a rockburst will occur
- To determine the probability that if a rockburst event of a certain magnitude occurs, it will cause damage
- To develop a method for identifying an appropriate increase to baseline spending in the present to mitigate the future risk of a rockburst event based on the average cost of a rockburst event, the probability the event will occur and the probability damage will be caused as a result of an event

1.3 Scope

The scope of this thesis is limited to hard rock underground mining operations. There is a predominant focus on mines in Ontario. This thesis makes several assumptions to allow for the comparison of various events. These assumptions will be highlighted when they are made.

1.4 Thesis Organization

The following is a brief summary of the work presented in each chapter.

Chapter 2 – Literature Review

Several concepts will be reviewed including discount rates, cost of life, probability of rockbursting and geomechanics. First, the use of inflation will be discussed and an appropriate discount rate will be identified based on common rates used in industry. Next, the average cost of a fatality, direct and indirect, to a mining company will be identified by reviewing studies from other industries. An approach to identifying the probability of a rockburst event will then be discussed. Lastly, a general overview and background of geomechanics will be presented including the history of rockbursting and geomechanics and rockburst classification systems.

Chapter 3 – Identifying the Cost of Major Rockburst Events

A methodology will be presented to develop cost models for rockbursting events based on direct and indirect cost categories. Several cost models will be developed from data obtained by public documentation. Summary results of cost models developed from data provided by mining companies will also be presented.

Chapter 4 – Estimating an Appropriate Increase in Upfront Expenditure

The minimum magnitude at which rockbursting events cause significant damage will first be discussed, followed by the identification of the probability that if a certain magnitude event occurs, damage will occur. The latter is based on data provided by the Ontario Ministry of

Labour. Three case studies will then be developed to identify the increase in upfront expenditure that should be made to mitigate the risk of a future rockburst event based on the average cost of rockbursting developed in Chapter 3 and the probability of a rockburst event occurring and of the event causing damage if it occurs.

Chapter 5 – Limitations

The limitations of the methodology used to develop the cost models and identify the increase in upfront expenditure will be presented.

Chapter 6 – Conclusion and Recommendations

The final chapter will present a summary of the results, discuss the main conclusions from the research and recommend future areas of work to further build on this thesis.

Chapter 2

Literature Review

2.1 Inflation and Discount Rates

2.1.1 Inflation

Inflation refers to the devaluing of a currency over time or a decrease in the purchasing power of a currency over time. A cash flow which includes the effects of inflation from year to year is said to be in current dollars. A cash flow which has had the effects of inflation removed from year to year is referred to as being in constant dollars (Smith, 1987).

Heath, Kalov and Inns (1974) commented that discounted cash flows carried out in constant dollars will have a constant value throughout the life of the cash flow and therefore values from each year can be directly compared. However, a net present value calculated from a current dollar's cash flow will be misleading, since the terms in each year have inflation built in and thus cannot be directly compared. Furthermore, they state that if "...The rate of inflation is the same for revenue, capital and working costs, and where there are no tax and gearing complications, the [Discounted Cash Flow] yield may be correctly obtained by performing the whole calculation without regard to inflation, i.e. in constant money terms (Heath, et al., 1974)."

Smith (2000) notes that the use of a "bare bones" case, where inflation and debt are assumed to be 0% and constant commodity prices are assumed, allows for a common reference point between

multiple projects, essentially removing assumptions that could change the net present value and internal rate of return of a project.

The cost models developed in this thesis are from mines located in different regions around the world and from different time periods. For ease of calculations, all cash flows are presented on a before tax basis. Therefore, based on accepted economic practices referenced above, inflation is assumed to be zero and all discounted cash flows are calculated in 2010 Canadian Dollars (CAD). To ensure the proper conversion of all cash flows, cash flows that are originally in currencies other than Canadian dollars are first converted to Canadian dollars based on the average exchange rate for the applicable year and then converted to 2010 Canadian dollars based on the Canadian Consumer Price Index (CPI), Table 1.

The consumer price index was selected because it is generally considered an acceptable indicator of monthly and annual inflation rates in Canada (Smith, 2000). The CPI is based on the amount of Canadian dollars required to purchase a set basket of goods from year to year. The percentage increase in the price required to purchase said basket from period to period is the inflation rate between those periods. This is translated into an index, Table 1, with the year 2002 set at 100. Equation 1 is used to convert from dollars in year y to dollars in year x given the CPI for years x and y.

$$Dollars_x = Dollars_y \times \frac{CPI_x}{CPI_y} \quad (1)$$

Table 1: Canadian CPI from 1990 to 2010 (Statistics Canada, 2011)

Year	CPI (2002 = 100)	% Change from Previous Year
1990	78.4	4.8
1991	82.8	5.6
1992	84.0	1.4
1993	85.6	1.9
1994	85.7	0.1
1995	87.6	2.2
1996	88.9	1.5
1997	90.4	1.7
1998	91.3	1.0
1999	92.9	1.8
2000	95.4	2.7
2001	97.8	2.5
2002	100.0	2.2
2003	102.8	2.8
2004	104.7	1.8
2005	107.0	2.2
2006	109.1	2.0
2007	111.5	2.2
2008	114.1	2.3
2009	114.4	0.3
2010	116.5	1.8

2.1.2 Discount Rates

The discount rate has the greatest impact on a discounted cash flow. A hypothetical cash flow with losses of \$1000 for the first 5 years and profits of \$1000 for years 6 through 30, with varying discount rates, is illustrated in Figure 1. It can be seen that, as the discount rate increases, the value of the project decreases since a higher discount rate places more value on cash flows occurring closer to the present. Equation 2 identifies the formula for calculating the Net Present

Value (NPV) where R_t is the net cash flow at time t , i is the discount rate and t is the time of the cash flow.

$$NPV = \frac{R_t}{(1 + i)^t} \quad (2)$$

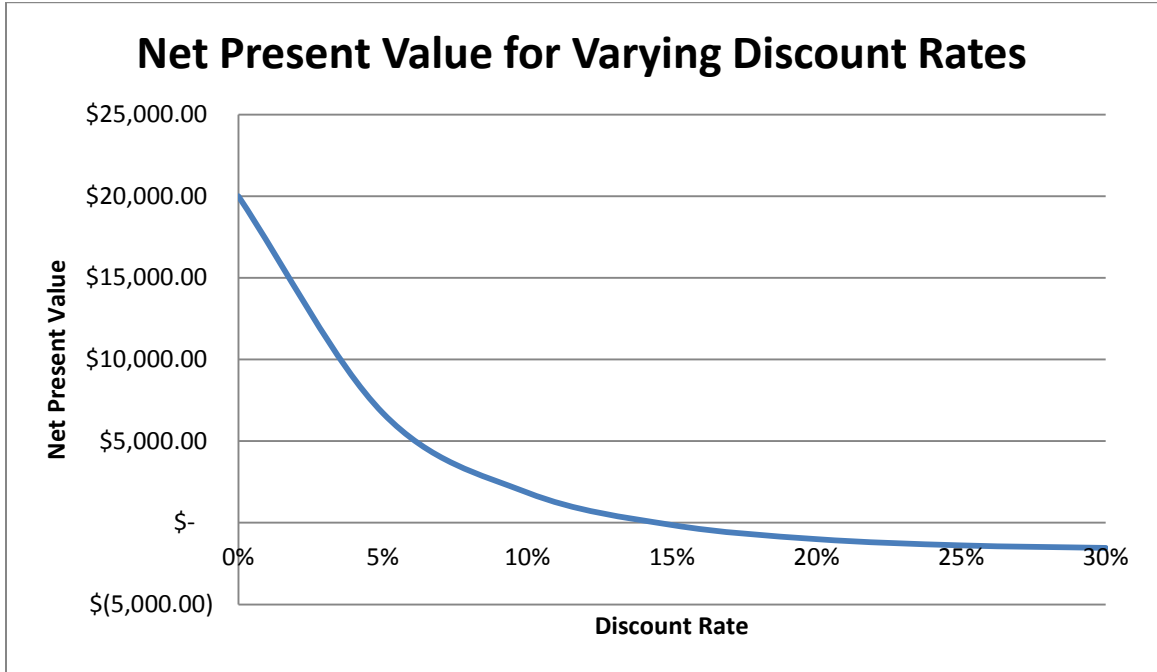


Figure 1: Net Present Value for varying discount rates

Therefore, it is crucial to select an appropriate discount rate. The main method for selecting a discount rate is based on a company's financial position by calculating a company's Weighted Average Cost of Capital (WACC) (3). This is based on how the company has financed itself and what the expected returns are from those that financed the company. This includes equity, debt and preferred shares.

$$r_{WACC} = r_e p_e + r_d p_d + r_p p_p \quad (3)$$

where r_{wacc} is the weighted average cost of capital expressed as a percentage, $r_{e,d,p}$ refers to the proportional costs of equity, debt and preferred stocks expressed as a percentage and $p_{e,d,p}$ is the amount of equity, debt and preferred stock that make up the total capital with their values summing to 1.

The Capital Asset Pricing Model (CAPM) (4) can be used to identify the cost of equity capital from Equation 3. CAPM compares the return of the company's stock to the stock market as a whole.

$$r_e = f + R \Xi \quad (4)$$

where r_e is the expected return of the common stock, f is the risk-free return, R is the risk premium of market returns above long term risk free rates and Ξ is the beta factor for the common stock. The beta factor is used to express the variability of the common stock to the variability of the market, with the beta of the market being 1.

Smith (2000) notes that, while this method is appropriate for calculating the discount rate of a company, it is not necessarily appropriate for selecting a discount rate for a project, since the method does not take into account project specific variations that may impact the discount rate. Smith therefore proposes that, based on discussions with mining companies, published evaluations by mining analysts, direct experience in studies undertaken for mining companies and

various published references, a discount rate of 10% for a feasibility study level project is considered acceptable by industry standards. This value is based on the risk-free long-term interest rate, mining project risk and country risk. As a project advances from early exploration to operating mine, the mining project risk decreases as knowledge of the operation increases. Thus as a project advances from exploration to production the discount rate used decreases, Figure 2. It is worth noting that gold projects often utilize lower discount rates because gold companies are often granted lower interest rates and have a much lower cost of equity capital.

For this report, one common discount rate was selected. This allows for net present values from different projects to be directly compared. Given that the majority of projects analyzed are base metal mines in the feasibility to production stage, a discount rate of 10% was deemed appropriate.

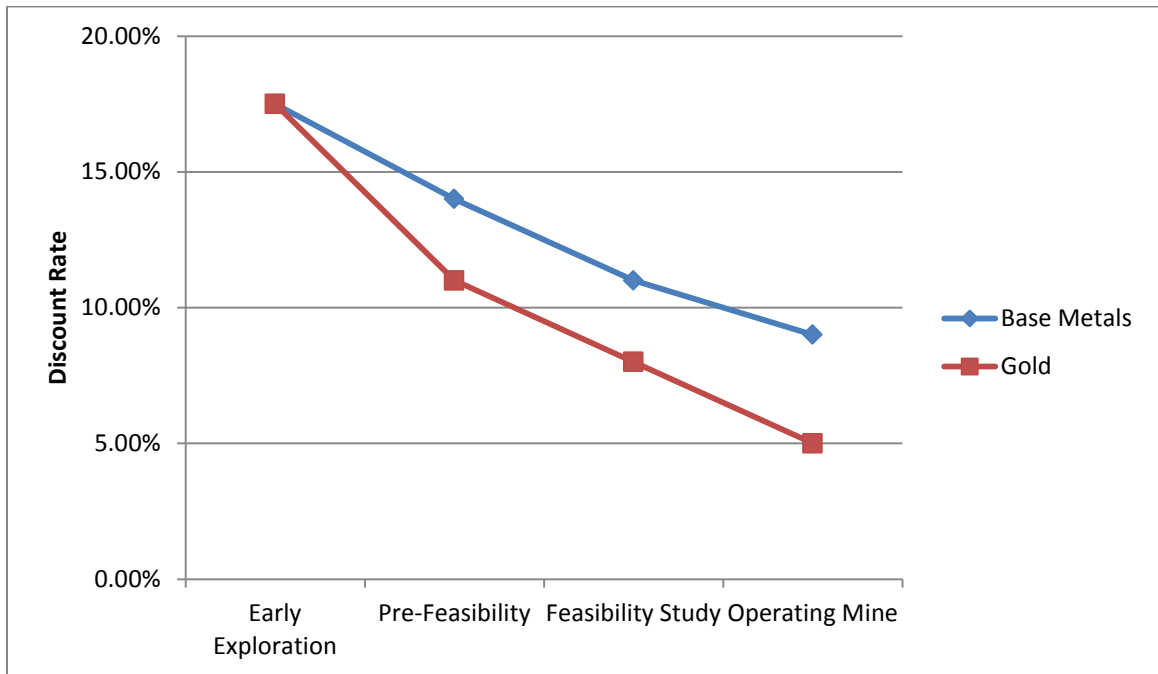


Figure 2: CIM Mineral Economics Society members survey for discount rate used as various stages of a mining project (Smith, 2000)

2.2 Value of Life

One of the main consequences that are associated with a rockburst or fall of ground is the potential for loss of life (Stacey, et al., 2007). Adams (2005) notes that “The cost of a fatality is different in every case and only rigorous consideration of the circumstances of each case can determine the exact cost.” Adams goes on to note that “...there is some merit in attempting to arrive at some average costs of a fatality to demonstrate, amongst other things, the loss that a company or society suffers as a result of such an incident.” Therefore, to properly analyze the economic impact of a rockburst event to a company, it is important to attempt to identify the average cost of a fatality to the mining company.

Adams (2005) used a study by the National Safety Council on the cost of a motor vehicle fatality in the U.S., which valued fatalities at \$1.12 million (2005 USD), for a mining study. While this value appears to be low, the practice of using values from other industries is sound. Several detailed economics studies have been published on the cost of workplace injuries and fatalities in the United States which can be drawn upon and applied to the mining industry.

Miller and Galbraith (1995) estimated that the annual cost of workplace injuries and fatalities in the United States in 1990 was \$140 billion (1990 USD). They found that while workplace fatalities accounted for only 0.1% of all cases, they comprised 20.5% of the total cost. This translates to an average cost of \$2.5 million (1990 USD) per workplace fatality. The cost breakdown is outlined in Table 2.

**Table 2: Cost breakdown for workplace fatalities suffered in the United States in 1990
(Miller, et al., 1995)**

Cost Category	Cost (1990 USD)	Cost (1990 CAD)	Cost (2010 CAD)
Medical/EMS	13,000	15,168	22,540
Wage/Fringe	520,000	606,736	901,591
Household Work	110,000	128,348	190,721
Work Disruption	9,900	11,551	17,165
Legal & Admin	18,000	21,002	31,209
Subtotal	670,900	782,806	1,163,226
Quality of Life	1,900,000	2,216,920	3,294,275
Total	2,500,900	2,999,726	4,457,501

From Table 2 it can be seen that the average cost per fatality is approximately \$4.5 million (2010 CAD) with the “Quality of Life” component, defined as “...reduced quality of life and pain and suffering of workers and their families... (Miller, et al., 1995)” representing an indirect cost, accounting for three quarters of the total cost. Similarly, Leigh et al. (2000) found that direct costs represent approximately 33% of the total cost compared to indirect costs which comprise of 67% of the total cost of a fatality.

A study by Viscusi (1996) investigated 26 labour market studies on the cost of fatalities. The results are outlined in Table 3. It can be seen that the value of life ranges from \$1 million to \$28 million (2010 CAD) with an average value of \$10.5 million (2010 CAD). Viscusi notes that some of the studies were unreliable and consequently omitted. Therefore, Viscusi identifies a more appropriate range of \$5.2 to \$12.1 million (2010 CAD) with a midpoint of \$8.7 million (2010 CAD).

It is worth noting that some studies by the same authors have varying results due to a different sample set or time period of interest being analyzed (Leigh and Folsom (1984), Moore and Viscusi (1988), Moore and Viscusi (1990), Kniesner and Leeth (1991)).

Table 3: Summary of labor market studies of the value of life (Viscusi, 1996) (all values in millions of dollars)

Study	Value of Life (1990 USD)	Value of Life (1990 CAD)	Value of Life (2010 CAD)
Smith (1974)	7.20	8.40	12.48
Thaler and Rosen (1976)	0.80	0.93	1.39
Smith (1976)	4.60	5.37	7.98
Viscusi (1978,1979)	4.10	4.78	7.11
Brown (1980)	1.50	1.75	2.60
Viscusi (1981)	6.50	7.58	11.27
Olson (1981)	5.20	6.07	9.02
Marin and Psacharopoulos (1982)	2.80	3.27	4.85
Arnould and Nichols (1983)	0.90	1.05	1.56
Butler (1983)	1.10	1.28	1.91
Leigh and Folsom (1984a)	9.70	11.32	16.82
Leigh and Folsom (1984b)	10.30	12.02	17.86
Smith and Gilbert (1984)	0.70	0.82	1.21
Dillingham (1985)	3.90	4.55	6.76
Leigh (1987)	10.40	12.13	18.03
Herzog and Schlottman (1987)	9.10	10.62	15.78
Moore and Viscusi (1988a)	2.50	2.92	4.33
Moore and Viscusi (1988b)	7.30	8.52	12.66
Garen (1988)	13.50	15.75	23.41
Cousineau, Lacroix and Girard (1988)	3.60	4.20	6.24
Viscusi and Moore (1989)	7.80	9.10	13.52
Moore and Viscusi (1990a)	16.20	18.90	28.09
Moore and Viscusi (1990b)	16.20	18.90	28.09
Kniesner and Leeth (1991a)	7.60	8.87	13.18
Kniesner and Leeth (1991b)	3.30	3.85	5.72
Kniesner and Leeth (1991c)	0.60	0.70	1.04
High	16.20	18.90	28.09
Low	0.60	0.70	1.04
Average	6.05	7.06	10.50

A 2008 report by the Center for Construction Research and Training (The Center for Construction Research and Training, 2008) found that the total cost of fatal and nonfatal injuries in the construction industry was approximately \$13 billion annually (2002 USD). Of this, fatalities were estimated to comprise 40%. This translated to a loss in value of \$7.3 million (2010 CAD) per construction worker fatality. This estimate includes both direct and indirect costs.

It can be seen that there is a wide range of values calculated for the cost of life. Viscusi (1996) notes that this can be attributed to the different models used and the source of the data. For this report, a value of \$8 million (2010 CAD) per fatality will be used. This is based on the average value of \$7.3 million (2010 CAD) per fatality drawn from the 2008 report by the Centre for Construction Research and Training (2008) and the \$8.7 million (2010 CAD) per fatality average drawn from Viscusi (1996).

Two important details need to be noted. First, many mining companies will internally treat near miss events as fatalities. Due to limited information for near misses for some of the events discussed in Chapter 3, near misses have not been included in the total cost. Secondly, in mining, especially in more tightly regulated regions such as Canada, the cost of a fatality can easily be much greater than \$8 million (2010 CAD). If a fatality leads to a Ministry of Labour stop work order, the value of lost production can be attributed to the cost of the fatality. If the fatality leads to a long term work stoppage, an example being Macassa Mine, discussed later, the cost of a fatality to a mining company can easily exceed \$50 million (2010 CAD).

2.3 A Brief Background on Geomechanics

This section will present a brief background on geomechanics. Definitions and concepts pertaining to geomechanics necessary to understand the information presented later in this thesis are outlined in this section.

2.3.1 History of Rockbursting

The earliest rockburst event discussed in literature occurred at the Altenberg Mine in present day Germany in 1545. Agricola (1556) describes the event by stating that "...Part of the mountain of Altenberg, which had been excavated, became loose and sank, and suddenly crushed six miners..." Ortlepp (2005) notes that the Altenberg region and nearby Ostrava-Karvina and Upper Silesian coalfields represents some of the earliest underground hard-rock and coal mining regions, and, given the "...Tectonic complexity of the region, it is not surprising that problems of a geotechnical nature would have become apparent early on."

Major occurrences of rockbursting in hard rock mines however, and the study of geomechanics, is a relatively new phenomenon. Rockbursting in the modern era first occurred in the Kolar Gold Field in India and in the South African Witwatersrand gold region in the early part of the 1900's (Ortlepp, 2005).

The first rockburst event in America is believed to have occurred at the Atlantic Copper Mine in Michigan in 1904. Around the same time the Coeur d'Alene lead-zinc silver mining district in Idaho also started experiencing rockbursting events (Ortlepp, 2005).

In Canada, rockbursts were first reported in the Kirkland Lake gold mining district and the nickel mines in the Sudbury Basin in the 1930's. The gold mining district in Red Lake began experiences rockbursting in the 1960's and mines in Elliot Lake and New Brunswick in the 1980's (Blake, et al., 2003).

In Australia, rockbursting was first experienced in the early 1900's, however not to the scale seen in Ontario and South Africa. It wasn't until the 1990's that Australia had major rockbursts¹. Similarly, South America has had some rockbursting, primarily limited to the El Teniente copper mine in Chile, starting in the 1990's (Ortlepp, 2005).

2.3.2 History of Geomechanics

The earliest literature that discusses rockbursting and ground support is Agricola's *de re Metallica* in 1556. Agricola (1556) mentions the need to place support to prevent caving and to protect the shafts, tunnels and drifts to prevent falling pieces of rock. However, it wasn't until the early 1900's that geomechanics was extensively studied. In South Africa commissions were appointed in 1908, 1915 and 1924. These commissions were initially set up to investigate damage to houses on surface, however, their mandate was broadened to further investigate rock mechanics. These studies led to the use of longwall mining and the reduction in the number of pillars and remnants in shallow dipping ore bodies (Hartman, 1992).

In North America, geomechanics wasn't studied until the 1930's. Potvin and Hudyma (2001) note that the US Bureau of Mines began studying microseismicity in the late 1930's. In Canada

¹ 6 seismic events between M_L 2.5 and M_L 4.3 were recorded at the Mount Charlotte Mine (Ortlepp, 2005)

the First Canadian Rockburst Research Committee was formed in 1939 involving mines from Kirkland Lake and was headed by R.G.K. Morrison which followed an increase in events in the Kirkland Lake region, including several events that were recorded by sensors of the National Seismograph in Ottawa, over 450 km away. Morrison released a detailed report in 1942 on “The Report on the Rockburst Situation in Ontario Mines (Morrison, 1942).” In the report, Morrison outlined the importance of sequencing in steep dipping ore bodies to reduce stress build up and to minimize rockbursting (Hartman, 1992).

In June of 1984, a major rockburst event occurred at the Falconbridge Mine in Sudbury that tragically killed 4 miners. This was followed one month later by one of the largest mine-induced events ever recorded in Canada, a 4.0 M_n rockburst at the Creighton Mine (Blake and Hedley, 2003). These 2 events lead to the creation of the Canadian Rockburst Research Program (CRRP) which from 1985 to 1990 worked on improving monitoring of rockbursts and seismicity in Canadian mines. From 1990 to 1995 the group focused on understanding how and why rockbursts occur.

The Australian Centre for Geomechanics, which is based out of the University of Western Australia and was formed in 1992, is the leading geomechanics research group in Australia. In South Africa, the Safety in Mines Research Advisory Committee, formed in 1991, coordinates all research pertaining to geomechanics. In the United States, since 1996, the National Institute for Occupational Safety and Health has been responsible for geomechanics research (Ortlepp, 2005).

2.3.3 Rockburst Definition and Classification

Before discussing an appropriate classification system for different types of rockbursts, it is important to first develop an acceptable definition for the overarching term “rockburst”. Ortlepp (2005) notes that there is no common internationally accepted definition for a rockburst, however there are many journal papers and government publications that attempt to define rockbursts. The U.S. Bureau of Mines (1968) states that a rockburst is “That phenomena which occurs when a volume of rock is strained beyond the elastic limit, and the accompanying failure is of such a nature that accumulated energy is released instantaneously.” Similarly, the Mine Safety and Health Administration (1984) defines it as “A sudden and violent failure of a large volume of overstressed rock, resulting in the instantaneous release of large amounts of accumulated energy.” Scott (1997) stated that “...a rockburst is defined as the sudden and sometimes violent release of accumulated energy when a volume of rock is strained beyond its elastic limit.”

For this report, a rockburst will be defined as outlined above.

There are several different classification systems in literature that are used to define different types of rockbursts. Ortlepp (2005) notes that there is a wide range of rock failures that are classified under the umbrella term of “rockburst.” He also notes that different terms are used in different countries to describe the same type of event and that in some instances the same word is used to describe different types of events by different people. These classification systems are based on varying factors including intensity of damage, location of the event in terms of mine geometry, failure mechanisms and magnitude of the event (in terms of seismic energy).

Blake and Hedley (2003) discuss the classification system brought into effect in Ontario following major bursting in the 1930`s. The system used was based on the amount of damage, but also took into consideration the amplitude and duration of the vibrations. The system broke rockbursting into 3 categories:

- Light – 10 tonnes of displaced rock
- Medium – 10 to 50 tonnes of displaced rock
- Heavy – greater than 50 tonnes of displaced rock

Blake and Hedley (2003) go on to note that rockbursts can also be classified as large seismic events or small seismic events. For a large seismic event, the event is associated with a loud, distinct blast-like noise underground near the source. Furthermore, the event and its corresponding seismic wave causes damage in the area of the event and can also cause a shake-down of loose rock and slabbing off of large pieces of rock near the event. The seismic waves generated by large seismic events can typically be detected on surface by seismographs at a distance of up to 100 km while some events can be detected at distances of greater than 1000 km. Conversely, small seismic events are associated with popping noises near the event location. They usually result in rock failure of less than 2 cubic meters and generally have a detection range of 50 m.

Gill et al. (1993) defined a rockburst as “A sudden rock failure characterized by the breaking up and expulsion of rock from its surroundings, accompanied by a violent release of energy.” They

state that rockbursts induced by mining are caused by either a slip on a pre-existing discontinuity (geological structure) or fracturing of the rock mass. Based on this they classify rockbursts as Type I or Type II. Type I rockbursts are fault-slip events where as Type II rockbursts are associated with the failure of the rock mass, such as strain bursts and pillar bursts.

Misich and Lang (2001) discuss a more detailed rockburst classification system developed in South Africa in the late 90`s. The system defines rockbursts by the source mechanism as well as by the damage mechanism that occurs at the excavation boundary and is paraphrased below. The source mechanism classification, paraphrased from Misich and Lang, is outlined below:

- Source Mechanism Classification
 - Strain Bursts – The violent failure of intact rock surrounding excavations caused by a high build up of stress around the opening.
 - Face-parallel bursts – Violent compression of already fractured rock resulting in rapid expansion of the rock ahead of the face, causing ejection or buckling of slabs on the face, violent peeling of rock from new fracture surfaces in the hanging wall and severe shake down of fractured rock from the hanging wall due to the proximity of the seismic event
 - Pillar bursts – When the stress placed on a pillar exceeds the strength of the pillar it becomes over stressed and fails violently. An increase in extraction leads to an increased risk of failure

- Pillar Foundation Failures – A seismic event which occurs below a pillar caused by the pillar `punching` through softer footwall rock leading to stope closure and shakedown damage
- Slip, on geological structures (fault-slip) – Redistribution of stress caused by mining activity which causes geological structures such as faults or dykes to slip
- Damage Mechanism Classification
 - `Near-field` rockbursts – Most severe rockbursts which are associated with medium to large seismic events resulting from shear of intact rock or slip of a geological structure. This event often results in extensive damage to support and widespread falls of ground and collapse of excavations
 - Violent shakedown bursts – Lower intensity than near-field bursts, this type of event involves severe and prolonged shaking of the whole stope or drift. This results in rock being shaken loose
 - Shakedown bursts – Caused by seismic waves from a nearby small event or a far away large event, this type of event causes loose rock to be shaken down but does not cause damage to support or result in excavation closures

Blake and Hedley (2003) define a similar classification system to Misich and Lang`s (2001) source mechanism classification while adding that strain bursts generally have a local magnitude of less than 2 and damage less than 100 tonnes, pillar bursts have a local magnitude of less than

3.5 and damage of several hundred tonnes and fault-slip events can have a local magnitude greater than 4 and damage of several thousand tonnes.

For this thesis, the events discussed will be classified as strain bursts, pillar bursts or fault-slip events as defined by the Misich and Lang (2001) source mechanism classification system.

2.4 Probability of Rockbursting

In order to identify the increase in upfront expenditure that should be made to mitigate rockbursting, the probability of rockbursting must first be identified. Gutenberg and Richter (1949) identified the relationship between seismic magnitude and frequency of occurrence (5) which is referred to as the Gutenberg-Richter frequency-magnitude relationship.

$$\log N = a - b M \quad (5)$$

where N is the number of events greater than or equal to magnitude M that occur in a year, a is the y intercept and b is the slope. This relationship has been observed to apply to both earthquakes and mining-induced seismic events as illustrated in Figure 3. It should be noted that this relationship applies to higher magnitude events. At lower magnitudes, as can be observed from Figure 3, the relationship has a distinct roll-off. This roll-off point is denoted as M_{\min} and represents the minimum magnitude that can be accurately forecasted. The roll-off is caused by limitations in microseismic monitoring systems to accurately detect all rockburst events at a lower magnitude.

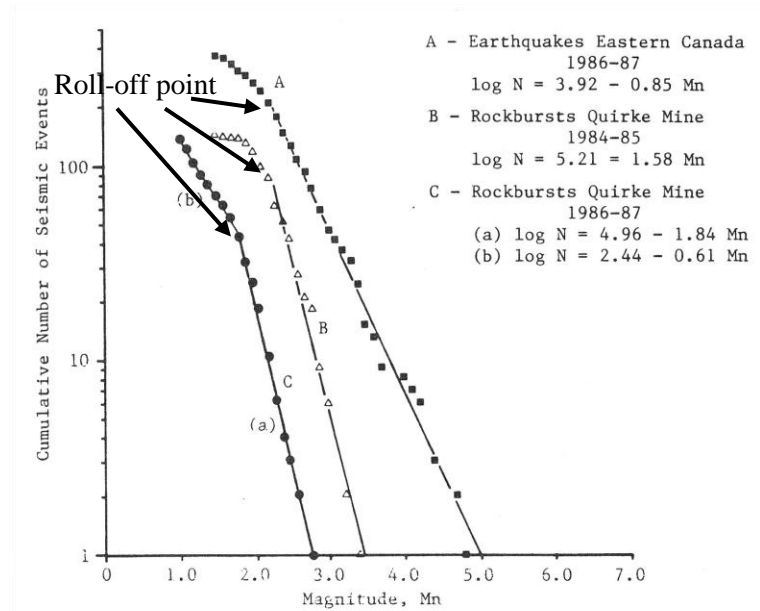


Figure 3: Frequency of occurrence of earthquakes and rockbursts using the Gutenberg-Richter frequency-magnitude relationship adapted from Hedley (1992)

2.4.1 Calculating the Maximum Potential Magnitude

In order to identify the probability of occurrence, it is important to identify the maximum potential magnitude that can be expected. There are several methods for calculating this. Kijko and Funk (1994) reviewed three such methods and noted that Robson and Whitlock's (1964) statistical method for calculating the M_{max} (6) is useful in situations involving limited seismic data.

$$M_{max} = M_{max}^{obs} + (M_{max}^{obs} - M_{n-1}) \quad (6)$$

where M_{max}^{obs} is the largest seismic magnitude observed and M_{n-1} is the second largest seismic magnitude. However, this method does not take into consideration that, as the observed time period and number of samples increases, there is a greater confidence that M_{max} will not deviate greatly from M_{max}^{obs} . Kijko and Funk (1994) discuss another method for calculating M_{max} which was first introduced by Kijko and Sellevoll (1989) (7).

$$M_{max} = M_{max}^{obs} + \frac{E_1(n_2) - E_1(n_1)}{\beta \exp(-n_2)} + M_{min} \exp(-N) \quad (7)$$

where β is defined as $b \ln 10$, N is the number of seismic events in the dataset larger than M_{min} and n_1 , n_2 and $E_1(x)$ for $1 \leq x < \infty$ are defined below.

$$n_1 = \frac{N}{(1 - \exp[-\beta(M_{max} - M_{min})])} \quad (8)$$

$$n_2 = n_1 \exp[-\beta(M_{max} - M_{min})] \quad (9)$$

$$E_1(x) = \frac{x^2 + a_1 x + a_2}{x(x^2 + b_1 x + b_2)} \exp(-x) \quad (10)$$

Equation 10 is derived from Abramowitz and Stegun (1972) and is an approximation of the exponential integral function with constants a_1 , a_2 , b_1 and b_2 having the values: $a_1 = 2.33$, $a_2 =$

0.25, $b_1 = 3.33$ and $b_2 = 1.68$ and bound by the limit $1 \leq x < \infty$. For cases where x is less than 1, equation 11, also derived from Abramowitz and Stegun, can be used to calculate $E_1(x)$.

$$E_1(x) = a_0 + a_1x + a_2x^2 + a_3x^3 + a_4x^4 + a_5x^5 - \ln(x) \quad (11)$$

where $a_0 = -0.58$, $a_1 = 1.00$, $a_2 = -0.25$, $a_3 = 0.06$, $a_4 = -0.01$ and $a_5 = 0.001$ and bound by the limit $0 \leq x \leq 1$.

It can be observed that the calculation of n_1 (8) and n_2 (9) which form part of equation 7, include the M_{\max} term, which equation 7 is solving for. Kijko (2004) notes "...When $M_{\max} - M_{\min} \leq 2$, and $N \geq 100$, the parameter M_{\max} in n_1 and n_2 can be replaced by M_{\max}^{obs} ." When the 2 parameters outlined by Kijko are not met, M_{\max} can be found by taking multiple iterations using $M_{\max}^{estimate}$ for m_{\max} in equations 8 and 9 and solving until $M_{\max}^{estimate} = M_{\max}$.

Brink et al. (2000) note that the Kijko-Sellevoll estimator of M_{\max} (7) has been used extensively by the scientific community to calculate the maximum regional earthquake magnitude in several regions including China, Canada, France, Iran, India, Romania, Greece, Algeria, Italy, Spain, Turkey and the West Indies. Since the Kijko-Sellevoll estimator more accurately forecasts the maximum possible magnitude that can be expected by taking into account the sample size, equation 7 will be used to calculate M_{\max} . In the event that M_{\max} is found to be greater than 4.2

M_n , the largest seismic event ever recorded in a Canadian Mine², it will be assumed that M_{max} is $4.2 M_n$.

2.4.2 Double Truncated Gutenberg-Richter Frequency Magnitude Relationship

Kaiser et al. (1996) note that, given the slope b and M_{max} equated above, a variation of the double truncated Gutenberg-Richter frequency-magnitude relationship as derived by Cosentino et al. (1977) can be applied to calculate the probability that a seismic event E has a magnitude greater than M (12).

$$P[E > M] = \left[\frac{(e^{-\beta M} - e^{-\beta M_{max}})}{(e^{-\beta M_{min}} - e^{-\beta M_{max}})} \right] \quad (12)$$

Applying equation 12 to Benjamin's (1968) seismic hazard function (13), the probability of a seismic event, E , being larger than magnitude M , during a future time period Δt can be calculated (Kaiser et al. 1996).

$$P[E > M, \Delta t] = 1 - \left[\frac{t_r}{(t_r + \Delta t P[E > M])} \right]^{n+1} \quad (13)$$

where t_r is the monitoring period, n is the number of seismic events in the data set and Δt is the future time period of interest. Using equation 13 in conjunction with seismic data, a magnitude-probability-time plot can be generated, Figure 4. This type of graph easily shows the probability of an event with magnitude M occurring over a time period Δt .

² Hedley and Udd (1989) note that Wright-Hargreaves Mine in the Kirkland Lake Mining district in 1964 had the largest recorded rockburst event in Canadian history which Blake and Hedley (2003) defined as a fault-slip event with a magnitude of $4.2 M_n$

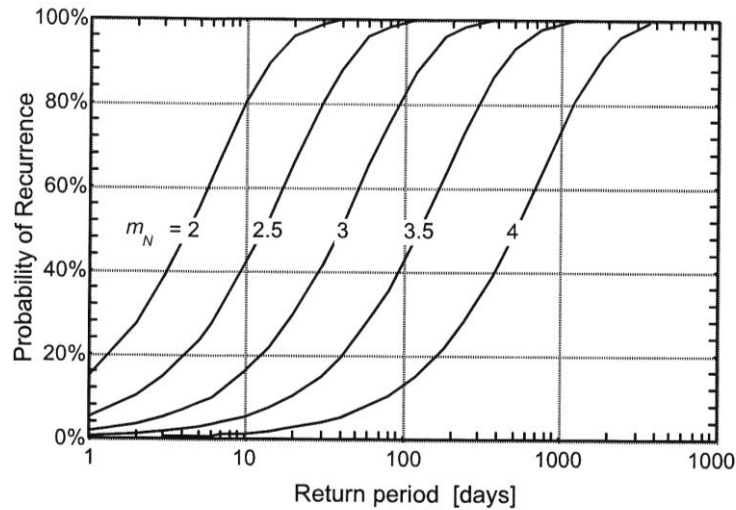


Figure 4: Magnitude-probability-time plot for a large Canadian Mine (Kaiser, et al., 1996)

It is important to note that, while this approach can be used for regional seismology, the probabilistic approach in terms of mining is very site specific and can even vary between mining zones within the same operation (Hedley, 1992). From Table 4 it can be seen that, over similar time periods, Creighton Mine, Macassa Mine and Quirke Mine all had different a and b values meaning that each operation had a different probability that a certain magnitude event would occur and a different maximum potential magnitude event. Furthermore, McGarr (1976) notes that this approach is applicable only where the rate and geometry of mining does not significantly change from year to year. In many instances, major rockbursting events can lead to an overall change in stress distribution. This makes historical measurements inaccurate. For the case studies developed in Chapter 4, it is therefore assumed that no major changes in geometry or mining have occurred during the time period that is being analyzed.

Table 4: Guttenberg-Richter frequency-magnitude relationship constants a and b for different locations, adapted from Kaiser et al. (1996)

Location (year)	a	b
Creighton Mine (1984 to 1991)	2.95	0.97
Macassa Mine (1984 to 1993)	2.93	1.23
Quirke Mine (1984 to 1985)	4.91	1.58
Quirke Mine (1986 to 1987)	4.66	1.84
Eastern Canada Earthquakes (1986 to 1987)	3.62	0.85

Chapter 3

Identifying the Cost of Major Rockburst Events

This chapter will attempt to outline the cost of major rockbursting events. Events from Ontario, Turkey, South Africa and Australia will be discussed and analyzed. It is important to note that while this chapter will draw upon an average cost for rockbursting, the actual cost of a rockburst will vary depending on location of the event, support methods in place, mining method, extraction ratio etc. The average cost identified should not be taken out of context. The average cost of rockbursting identified in this chapter is drawn from a small sample set. Further data and research is required to improve on the findings presented.

3.1 Methodology

In order to quantify the cost of rockbursting, cost categories need to be identified. Brummer and Andrieux (2008) identified three cost categories:

- Loss of production and rehabilitation costs
- Public relations costs
- Loss in share value

While measuring the loss of production and rehabilitation costs presents some challenges, it is relatively straight forward, especially when compared to public relations and loss in share value. Brummer and Andrieux (2008) note that, while it is not easy to estimate, the public relations cost can be very significant, pointing out that an increase in adverse publicity can result in increased scrutiny and trigger additional costs. With regards to share value loss, Brummer and Andrieux

comment that events that cause losses or delays on production and increased capital costs in rehabilitation often lead to a drop in share prices noting that "...recent incidents have resulted in immediate drops in share price in the range of 5%."

Due to the challenges with identifying the public relations cost, this aspect is deemed outside the scope of this thesis and will not be considered in developing a cost for rockbursting. Furthermore, since the thesis focuses on the cost to mines and not the overall mining company, the loss in share value will also not be considered. While neither of these costs are considered in this thesis, both are real and will adversely affect an operation. An event which leads to a fatality will lead to an increase in public scrutiny and most likely a drop in share price which will affect the market capitalization of a company.

Martin (2010) undertook a similar study and utilized the direct cost, value of ore lost due to the event and decline in value of ore from postponed production due to the event to comprise the total cost. Martin noted that the cost of a fatality and near miss would impact the total cost and should be included.

Based on Martin (2010) and Brummer and Andrieux (2008), the cost categories that will be used to develop a total event cost are the direct cost, the value of ore lost due to the event, the decline in value of ore from postponed production due to the event and the cost of fatalities due to the event.

Two other costing categories that were considered for inclusion were near misses and loss in exploration potential. These categories were ultimately omitted since information for both categories was not available for all cost models.

The direct cost is the most straightforward to calculate and includes rehabilitation costs, replacement costs for damaged equipment, cost of updating or changing support methods etc.

The value of ore lost due to the event is defined as the profit of the tonnes lost based on operating costs and value of the ore at the time of the event (i.e. metal prices and operating costs at the time of the event). The value is discounted at 10% per year over the number of years it would have taken to mine the lost tonnes based on the mining rate at the time of the event.

The decline in value of ore from postponed production due to the event is based on the time value of money and the delay in opportunity. First, the profit that would have been achieved had the event not occurred will be calculated. The tonnes of delayed production will then be moved to the end of the mine life and discounted at 10% annually. The difference between the profit if the event had not occurred and the realized profit at the end of mine life represents the decline in value of the ore due to the event.

The cost of fatalities due to the event is calculated based on the number of fatalities caused by the event and the cost of a fatality to a company calculated in Section 2.2, which was found to be \$8 million (2010 CAD).

It is important to note that some operations altered the mining method used in high stress ore zones or changed sequencing or stope sizes to mitigate stress in the mining block as the result of a rockburst. These changes would generally lead to a decrease in production and an increased operating cost. In some instances, the decrease in production in the affected mining zone was offset by increasing mining from another zone. For example, following the 2002 ore pillar burst at Çayeli Mine, the stope size was decreased. In order to increase production, the mine increased the number of stopes being mined.

The increase in operating cost as a result of an increase in the number of stopes can be hard to identify if it is not specifically stated since operating costs can fluctuate annually based on several factors including fuel costs, inflation, labour costs and maintenance costs. Therefore, unless specified, a change in operating cost was not included. This is discussed in more detail in Section 5.2. Furthermore, due to the fact that mining companies only publicize short term production targets, i.e. 1 to 2 year forecasts, only short term production short falls due to a rockburst event were calculated while longer term decreases in annual production were not. Without detailed knowledge of the operation, it is impossible to know whether a decrease in annual production is the result of a past rockburst event or if it is the result of mining a lower grade ore zone.

3.2 Data Acquisition

The data used to develop the cost models for the rockburst events was obtained from both the public domain and from mining companies. Events that have been developed based on data obtained from the public domain will be discussed in more detail and will be used to show the calculations carried out to identify the individual event costs. Information on specific events

provided by mining companies will not be discussed, but general trends in event costs will be analyzed and discussed. The same methods used to develop the cost models for the events obtained from public data were used to generate cost models for events developed from confidential data provided by mining companies.

It is important to note that there is an inherent bias in the dataset developed. The dataset should not be considered as a sample of all rockbursting events but rather the upper limit costs associated with major rockbursting events that occur in hardrock underground mines.

3.2.1 Public Domain Data Acquisition

Sources for publicly obtained data include quarterly reports, annual reports and letters to shareholders by companies as well as scientific journal articles and conference proceedings where rockbursting costs were discussed. The type of data obtained and confidence in the data varies greatly for each of the six rockburst cost models developed from public data.

Data obtained from quarterly and annual reports and letters to shareholders included capital and expensed costs, insurance claims, operating costs, reserves (tonnes and grade) and metal sales prices realized. Data from scientific journal articles included capital costs, insurance claims and total event costs.

One of the factors affecting the confidence in the data is that mining companies have the option of expensing or capitalizing events. In instances when the event was capitalized, it was easy to identify the direct cost of the event. However, when the event was expensed, it resulted in a

higher operating cost for the year in which the event occurred. It was assumed that the higher operating cost was attributed to just the rockburst event; however, other factors may have led to an increased operating cost. This assumption could result in a higher direct cost being calculated compared to what was actually observed by the company.

3.2.2 Data Provided by Mining Companies

Three mining companies provided information pertaining to the cost of rockbursting events at 8 different large scale, hardrock, underground mines located in Canada. Data provided included direct costs of the event, delays in production, tonnes of sterilized ore and total costs. The information provided by the companies includes both realized costs and engineering estimates for costs. When calculating the value of ore for delayed production and sterilized ore costs, if the average grades of the ore mined, operating cost or commodity price was not provided, the values were drawn from the company's annual report for the applicable year. If no annual commodity price was present in the annual report, the United States Geological Survey Commodity Reports were used. All of the data was standardized and the methods discussed in the following section were used to develop the cost models for the confidential data.

In general, the cost models developed were based on events occurring at an operation in one calendar year, however, some cost models were developed based on multiple events in the same ore zone over a several year period. It was decided that the events over multiple years were the result of the stress regime in the particular zone and therefore was assumed to be one continuous event.

3.3 Development of Cost Models

This section will discuss the process used to calculate the rockburst events and show the method using the events derived from publicly obtained data. In total 19 events were analyzed: 6 from public data and 13 from data provided by mining companies. The 19 events are from 13 mines located in 4 countries. The public events are outlined in Table 5.

Table 5: Events derived from publicly obtained data

Mine	Location	Date	Event Type	Magnitude
Macassa Mine	Ontario	26-Nov-93	Sill Pillar Burst	2.5 M_n
Macassa Mine	Ontario	12-Apr-97	Fault Slip	3.8 M_n
Williams Mine	Ontario	29-Mar-99	Sill Pillar Burst	3.0 M_n
Çayeli Mine	Turkey	25-Oct-02	Sill Pillar Burst	Unknown
DRD Gold, NWO	South Africa	09-Mar-05	Fault Slip	5.3 M_L
Strzelecki Mine	Australia	30-Jul-01	Fault Slip	2.0 M_L

3.3.1 Macassa Mine, Ontario, 1993

3.3.1.1 Event Description

On November 26, 1993 Macassa Mine suffered 3 rockbursts with magnitudes 2.5, 2.1 and 1.7 M_n , in quick succession, Figure 5. The first event was located in the sill pillar above the 6600 level. The event caused the ejection of 22,000 tonnes of material onto the backfill mat of the 6723 underhand stope, causing the mat to collapse, killing 2 miners and burying a scoop tram. The second burst occurred in the 6450 level and caused localized damage, and the third burst occurred above the 6300 level and caused no damage. Due to the large amount of material displaced into

the 6723 stope, it took workers 77 days to recover the bodies of the deceased miners (Blake and Hedley, 2003).

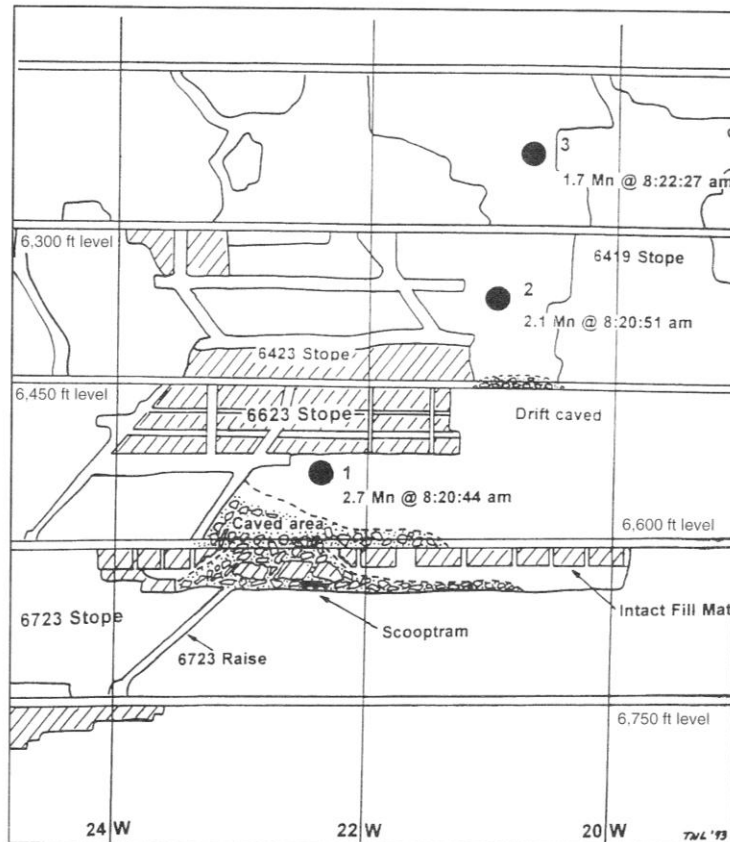


Figure 5: Longitudinal section view showing the location and damage of the November 26, 1993 rockburst at Macassa Mine (Blake and Hedley, 2003)

3.3.1.2 Aftermath

Investigation into the cause of the event concluded that the trigger was...”cumulative stress transfer effects which caused slippage along a crossover fault that ran through the pillar (Blake, et

al., 2003).” The event led to the mine switching from underhand cut and fill mining to long-hole mining below the 6450 level, to reduce the exposure of workers to active stopes. A paste backfill system was also implemented and a rockburst support system that allowed for dynamic loading was introduced.

3.3.1.3 Assumptions

The November, 1993 rockburst event at Macassa led to a shortfall in production in 1993, 1994 and 1995. There are several challenges associated with calculating the cost of the 1993 Macassa event: the acquisition of Lac Minerals by Barrick Gold in 1994 followed by the acquisition of Macassa Mine by Kinross in 1995, the expensing of the direct costs, and the production forecast for 1994 and 1995.

The event occurred while the mine was owned and operated by Lac Minerals Ltd. Macassa resumed production in May of 1994, however, Lac Minerals was acquired by Barrick Gold on September 6, 1994. Lac Minerals reported Macassa Mine and a nearby tailings reprocessing operation, Lake Shore Tailings, as separate operations. However, Barrick Gold treated Macassa Mine and Lake Shore Tailings as one operation. This leads to a difficulty in calculating the shortfall in production in 1994. This can be addressed by reviewing the Lac Minerals 1993 annual report which forecasted that 16,000 oz. of gold would be produced from the Lake Shore Tailings project in 1994. It is assumed that this forecast was realized. Therefore, the 1994 Macassa Mine production can be calculated from the 1994 Barrick Annual Report which stated that Macassa operations (Lake Shore Tailings and Macassa Mine) produced 45,400 ounces of gold, meaning Macassa Mine produced 29,400 ounces of gold.

Furthermore, the direct costs associated with the event, including rehabilitation of the damaged zone, recovery of the buried workers and improvement in the ground support system was expensed. This means the cost was not directly reported in the annual report but was rolled into the operating cost for the mine. In order to calculate the direct cost of the event, the operating cost for 1992, when delays were lowest, was compared to the operating costs for 1993 and 1994. The difference in the operating costs is assumed to represent the direct cost of the rockburst event.

Lastly, since the event occurred in November of 1993, by the time the 1993 annual report was released, the production forecast for Macassa for 1994 had been adjusted to account for delays caused by the event. In order to estimate the realized loss in production, it is assumed that the production forecast for 1993, 1994 and 1995 is the realized production from 1992.

3.3.1.4 Calculating the Cost of the Event

3.3.1.4.1 Direct Cost

The direct cost is based on the difference in operating costs between 1992 and 1993 and 1992 and 1994. Since the operating cost is stated in current USD per ounce of gold produced, it is first converted to CAD and then adjusted using the CPI ratio so that the 1992 operating cost can be compared with the 1993 and 1994 operating cost. The difference in operating costs is then calculated. This value is multiplied by the realized gold production (in ounces) for the year in question. Based on the reported operating cost in 1992 of \$262 (1992 USD/oz. Au), the direct

cost of the 1993 Macassa event was estimated to be \$6.6 million (2010 CAD) in 1993 and \$8.8 million (2010 CAD) in 1994 totaling \$15.4 million (2010 CAD), Table 6.

Table 6: Calculating the direct cost of the 1993 Macassa rockburst

	1993³	1994⁴
CAD to USD Exchange Rate	1.290	1.366
Operating Cost (USD/oz. Au)	303	397
Operating Cost (CAD/oz. Au)	391	542
CPI	85.6	85.7
CPI Ratio (1992 to current)	1.01905	1.02024
Estimated Operating Cost (CAD/oz. Au)	322.66	323.03
Difference in Op Cost (CAD/oz.)	68.26	219.43
Realized Production (oz. Au)	71,087	29,400
Expensed Cost (current CAD)	4,852,577	6,451,121
Expensed Cost (2010 CAD)	6,604,267	8,769,610

3.3.1.4.2 Decline in Value of Ore from Postponed Production

The decline in the value of the ore due to postponed production as a result of the event is calculated based on several assumptions as previously stated. It is first assumed that the forecasted production rate for 1993, 1994 and 1995 was 78,000 ounces of gold per year (Lac Minerals Ltd., 1993). It is also assumed that the average gold price for each year is the average gold price for that year reported by the USGS (Sehnke, 1997). The operating cost per year is assumed to be the estimated operating cost calculated above. Lastly, it is assumed that the delayed ore will be recovered as mining operations ramp down in 2001. This is based on 582,000

³ From Lac Minerals 1993 Annual Report

⁴ From Barrick Gold 1994 Annual Report

oz. gold reserves on December 31, 1993 (Lac Minerals Ltd., 1994) subtracted by the realized production in 1994 and 1995, divided by 78,000 ounces of gold which yields 6.5 years of remaining production from 1995.

Based on these assumptions, the decline in value of the ore due to postponed production was found to be \$0.8 million (2010 CAD) in 1993, \$7.2 million (2010 CAD) in 1994 and \$3.6 million (2010 CAD) in 1995 totaling \$11.6 million (2010 CAD), Table 7.

Table 7: Calculating the value lost due to deferred production⁵

	1993⁶	1994⁷	1995⁸
Forecasted Production (oz. Au)	78,000	78,000	78,000
Actual Production (oz. Au)	71,087	29,400	49,983
Delayed Production (oz. Au)	6,913	48,600	28,017
USD to CAD Exchange Rate	1.290	1.366	1.373
Gold Sale Price (USD/oz.)⁹	360.91	385.41	385.50
Gold Sale Price (CAD/oz.)	465.63	526.62	529.10
Estimated Operating Cost (CAD/oz.)	322.66	323.03	330.20
Revenue of Delayed Ore (CAD)	3,218,928	25,593,937	14,823,760
Operating Cost of Delayed Ore (CAD)	2,230,535	15,699,497	9,251,122
Profit of Delayed Ore (CAD)	988,394	9,894,440	5,572,638
Profit of Delayed Ore (2010 CAD)	1,345,185	13,450,435	7,411,100
NPV of Delayed Ore discounted to 2001 at 10% (2010 CAD)	570,490	6,274,727	3,803,066
Decline in value due to postponed production (2010 CAD)	774,695	7,175,708	3,608,034

⁵ All values in current dollars unless otherwise stated

⁶ From Lac Minerals 1993 Annual Report (Lac Minerals Ltd., 1994)

⁷ From Barrick Gold Corporation 1994 Annual Report (Barrick Gold Corporation, 1995)

⁸ From Kinross Gold Corporation 1995 Annual Report (Kinross Gold Corporation, 1996)

⁹ USGS Mineral Commodity Summary: Gold (Sehnke, 1997)

3.3.1.4.3 Cost of Fatalities

The 1993 Macassa event resulted in 2 fatalities. Given the assumed cost of a fatality to a mining company of \$8 million (2010 CAD), the cost of the 2 fatalities to Macassa is \$16 million (2010 CAD).

3.3.1.4.4 Total Cost of the Event

The total cost of the 1993 Macassa event is estimated to be \$43 million (2010 CAD), Table 8. The event not only led to a shortfall in production targets for 1993, 1994 and 1995, but cost the lives of 2 miners and played a major role in the Ministry of Labour's decision to suspend operations at Macassa Mine following the 1997 rockburst, which is discussed further in the thesis.

Table 8: Summary of costs of the 1993 Macassa event (all values in 2010 CAD)

Direct Cost	Decline in Value of Ore Due to Postponed Production	Cost of Fatalities	Total Cost
15,373,876	11,558,436	16,000,000	42,932,313

3.3.2 Macassa Mine, Ontario, 1997

3.3.2.1 Event Description

Following the 1993 rockburst at Macassa, the mine changed mining methods from underhand and overhand cut and fill to long-hole blasting. While this reduced miner's exposure to active stopes

and allowed miners to work under well supported ground, long-hole mining induced larger seismic events as a result of larger production blasts.

Seven minutes after a long-hole blast in the 6638 stope on April 12, 1997, a 3.8 M_n rockburst was recorded beside the blasted panel. During the subsequent 24 hours over 1700 seismic events were recorded including 3.7 M_n , 2.8 M_n , 2.6 M_n , 2.5 M_n and 2.2 M_n rockburst events, Figure 6. The events occurred between the 5725 and 6750 levels. It should be noted that 2 smaller events, at 1.8 M_n and 1.2 M_n , occurred near the Number 3 Shaft between the 6150 and 6450 levels (Blake and Hedley, 2003).

The 3.8 M_n event and subsequent aftershocks caused extensive damage between the 5300 and 6000 levels. Damage was particularly heavy between 5875 and 6025 levels, including damage to the walls of the Number 3 Shaft, stations and access drifts.

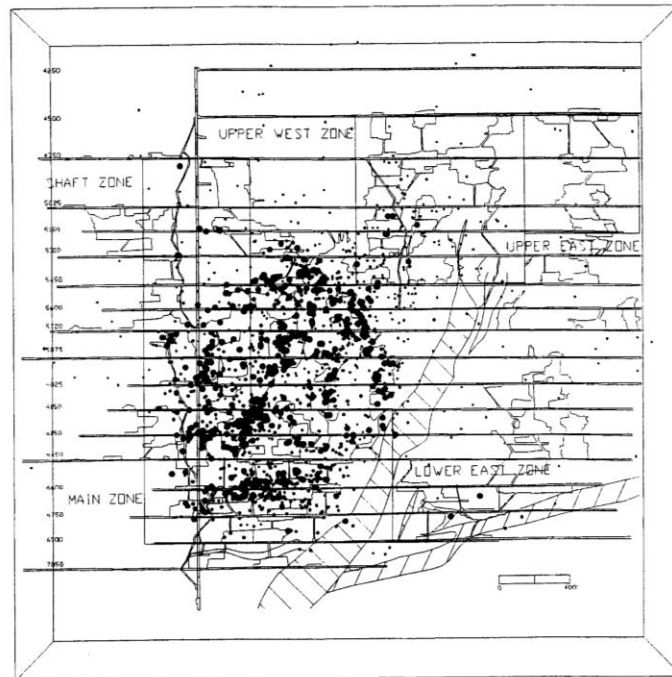


Figure 6: Seismic activity at Macassa Mine on April 12, 1997 (Blake, et al., 2003)

3.3.2.2 Aftermath

Due to an error in the microseismic system, no events were recorded between 6:00 am and 7:00 am on the morning of April 12. Since no seismic events were reported during this time period, a decision was made to send down crews at 7:00 am. As the crews were being sent down, a large seismic event was felt. The crews were immediately brought back to surface. 2 personnel were then sent back underground to check on the pumping system at the 6450 level. Shortly after releasing the cage the 3.7 M_n event occurred which did extensive shaft damage and blocked the Number 3 Shaft below the 5725 level. The 2 personnel had to climb the manway in the Number

2 Winze from the 6450 level to the 5000 level, where they were hoisted to surface (Blake and Hedley, 2003).

Due to the challenges associated with getting the two man crew back to surface following the 3.7 M_n aftershock, and, in part, due to the 1993 event at Macassa discussed previously, the Ministry of Labour issued a stop-work order, preventing access below the 4500 level. This was later reduced to the 5000 level (Blake, et al., 2003).

As a result of limited reserves above the 5000 level and a low gold price, Macassa was closed in 1999 and subsequently sold by Kinross in 2001. The mine has since been reopened but mining operations are still restricted to above the 5000 level (Blake, et al., 2003).

3.3.2.3 Calculating the Cost of the Event

3.3.2.3.1 Assumptions

Following the April 12, 1997 rockburst events, significant rehabilitation work was done. Furthermore, exploration work to identify new mining zones was conducted and a new loading station was built. All of this work was expensed, similar to the 1993 Macassa event. In order to calculate the direct cost of the event, the operating cost for 1996 was compared to the operating cost for 1997. The increase in operating cost from 1996 to 1997 is assumed to represent the direct cost of the rockburst event.

Secondly, in order to calculate the value of the reserves lost, it is assumed that the decline in reported reserves from December 31, 1996 and December 31, 1997 minus the production from

1997 represents the loss in reserves due to the event. The gold price is assumed to be the long term gold price stated by Kinross in the 1997 financial statements, \$350 (1997 USD/oz. Au). The pre event operating cost is assumed to be the operating cost from 1996, \$276 (1996 USD/oz. Au) or \$383 (1997 CAD/oz. Au). Lastly, in order to calculate the net present value of the reserves, an annual production rate of 91,000 ounces is assumed based on the 1997 production target stated in Kinross' 1996 Annual Report (Kinross Gold Corporation, 1997).

Similarly, in order to calculate the decline in value of ore from postponed production, the long term gold price stated by Kinross, \$350 (1997 USD/oz. Au) is used in conjunction with the 1996 operating cost. Lastly, since the mine closed in 1999, well before the anticipated end of mine life, the ounces of gold delayed were never recovered by Kinross. Therefore, decline in value of ore from postponed production due to the event will be the full value of the unrealized production in 1997.

3.3.2.3.2 Direct Cost

The direct cost of the event includes the cost to build a new loading station, stope development, exploration work to identify new mining regions and rehabilitation costs. As stated previously, these items were expensed and therefore the direct cost is based on the difference in operating costs between the period in which the items were expensed and a period in which the mine was running normal operations, Table 9. Given that Macassa produced 56,709 ounces of gold in 1997 (Kinross Gold Corporation, 1998), the direct cost of the event was estimated to be \$9.2 million (1997 CAD) or \$11.9 million (2010 CAD).

Table 9: Comparison between operating costs at Macassa Mine (Kinross Gold Corporation, 1998)

Period	Operating Cost (1997 CAD/oz. Au)
1996	383
1997	512
Difference	129

3.3.2.3.3 Value of Ore Lost Due to the Event

Analysis of Kinross' 1996 and 1997 annual reports shows a drastic decrease in Macassa's reserves. The reserves as of December 31, 1996 and December 31, 1997 are shown in Table 10 and Table 11.

Table 10: Reserves at Macassa Mine as of December 31, 1996 (Kinross Gold Corporation, 1997)

	Ore (mt)	Grade (g Au/mt)	Grade (oz. Au/mt)	Oz. Au
Proven	1,446,000	12.5	0.441	637,577
Probable	280,000	12	0.423	118,521
Total	1,726,000	12.42	0.44	756,097

Table 11: Reserves at Macassa Mine as of December 31, 1997 (Kinross Gold Corporation, 1998)

	Ore (mt)	Grade (g Au/mt)	Grade (oz. Au/mt)	Oz. Au
Proven	821,000	10.49	0.370	303,790
Probable	221,000	19.7	0.695	153,572
Total	1,042,000	12.44	0.44	457,362

Given the production in 1997 of 56,709 ounces of gold, the net reserves of gold lost due to the event is 242,027 ounces. Based on the assumption of 91,000 ounces of annual gold, this translates to a loss of 2.66 years of mining. Based on a discount rate of 10%, a long term gold price of \$484 (1997 CAD/oz. Au) and an operating cost of \$383 (1997 CAD/oz. Au) the value of ore lost can be calculated, Table 12. The discounted profit lost was found to be \$20.8 million (1997 CAD) or \$26.8 million (2010 CAD).

Table 12: Value of reserves lost due to the 1997 Macassa rockburst

Year	Oz. Au Mined	Profit (1997 CAD)	Discounted Profit (at 10%, 1997 CAD)
1998	91,000	9,319,856	8,472,596
1999	91,000	9,319,856	7,702,360
2000	60,027	6,147,678	4,618,841
total	242,027	24,787,390	20,793,798

3.3.2.3.4 Decline in Value of Ore from Postponed Production

Based on the production target of 91,000 ounces of gold in 1997 and the realized production of 56,709 ounces of gold, the loss in production due to the event is 34,291 ounces of gold. Based on the assumed gold price of \$484 (1997 CAD/oz.) and the operating cost \$383 (CAD/oz.) the lost production value is \$3.5 million (1997 CAD) or \$4.5 million (2010 CAD). As previously mentioned, since the mine closed in 1999, well before it was scheduled to, and the mine was ultimately sold in 2001, it is assumed that Kinross never recovered the 34,291 ounces of gold and therefore the entire value of the postponed ore is considered lost.

3.3.2.3.5 Total Cost of the Event

The total cost of the event was calculated to be \$43.2 million (2010 CAD), Table 13. This event led to the significant loss of reserves and the issuance of a stop-work order by the Ministry of Labour. The combination of a low gold price and the decrease in reserves resulted in the mine being closed in 1999 and sold to Foxpoint Resources for \$5 million (2001 CAD) in 2001 (Blake, et al., 2003) (Foxpoint Resources Ltd., 2001).

Table 13: Summary of costs of the 1997 Macassa event (all values in 2010 CAD)

Direct Cost	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Total Cost
\$11,873,738	\$26,797,317	\$4,525,905	\$43,196,961

3.3.3 Williams Mine, Ontario, 1999

3.3.3.1 Event Description

In 1999 the Williams Mine suffered its first major rockburst, a 3.0 M_n event, on March 29. Prior to this event, no rockburst larger than 1.0 M_n had been recorded at Williams Mine. The rockburst was located in the sill pillar and caused extensive damage to the 9415 and 9450 levels, Figure 7 (LeBlanc, et al., 2000).

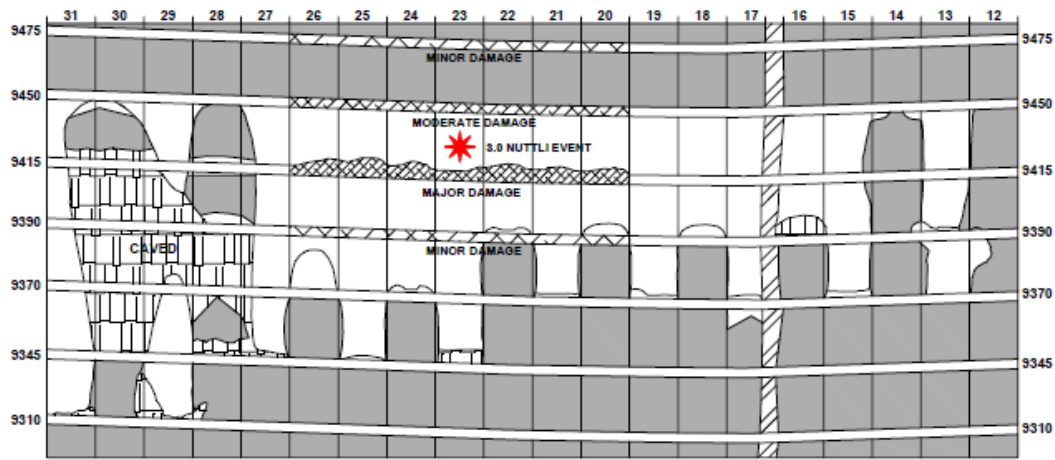


Figure 7: Longitudinal view looking North of the area affected by the March rockburst at Williams Mine (LeBlanc, et al., 2000)

3.3.3.2 Aftermath

Following the March, 1999 rockburst, a microseismic monitoring system was installed and the support system was improved to withstand a 3.0 M_n event, which included SMART cable bolts and yielding super swellex bolts.

On December 17, 1999, the Williams Mine suffered another major rockburst, 2.6 M_n , which occurred in the sill pillar of the 9415 level. Localized caving in the region of the rockburst on the 9415 level was reported, however, the affected area had been abandoned following the March rockburst. No other damage was reported, indicating the improved support system had worked as designed (LeBlanc, et al., 2000).

3.3.3.3 Calculating the Cost of the Event

3.3.3.3.1 Assumptions

While it is likely that the rockbursting caused the sterilization of some reserves and led to a decline in production from the affected area, review of Teck Resources 1999 and 2000 annual reports¹⁰ indicated no decline in reserves or shortfall in production in 1999. It is assumed that Williams Mine was able to make up shortfalls in production by activating other mining regions in the Williams Mine. Furthermore, it is assumed that any reserves impacted by the rockbursting were eventually recovered. Lastly, it is assumed that all costs stated by LeBlanc and Murdock (2000) are in 2000 Canadian Dollars.

3.3.3.3.2 Direct Cost

Following the March 29, 1999 blast, extensive rehabilitation work was done between the 9370 level and 9450 level. In addition, a microseismic monitoring system was installed and the ground support system was improved. This had a total cost of \$5.3 million (2010 CAD), Table 14 (LeBlanc, et al., 2000).

¹⁰ At the time of the rockburst the Williams Mine was operated as a joint venture between Teck Resources (50%) and Homestake Mining Corporation (50%) (Teck Corporation, 2001)

Table 14: Direct cost of the March, 1999 rockburst event at Williams Mine (LeBlanc, et al., 2000)

Area	Cost (2000 CAD unless otherwise stated)
9450 Rehab	500,000
9415 H/W Access Drift	1,300,000
9390 Rehab	600,000
9370 Rehab	300,000
West End Ramp	1,100,000
Micro Seismic System	560,000
Total	4,360,000
Total (2010 CAD)	5,324,319

3.3.3.3.3 Total Cost

While ore was most likely lost due to the rockburst, it is impossible to identify the tonnage of ore that was ultimately sterilized because the annual reports from 1999 and 2000 show no change in reserves. Therefore, it is assumed that the direct cost is the total cost, which is \$5.3 million (2010 CAD). This is most likely an underestimate of the actual cost of this event.

3.3.4 Strzelecki Mine, Australia, 2001

3.3.4.1 Event Description

On July 29, 2001 a rockburst event occurred on the 5926 level at the Strzelecki Mine, Australia, which caused subsequent damage to the 5938 and 5901 levels. The affected area was barricaded off pending a more detailed investigation. The following morning, while the underground manager, mine foreman and geotechnical engineers were investigating the rockburst damage, there was a series of large magnitude seismic events, Figure 8. The largest of the events was a

2.0 M_L burst. The event caused extensive damage between the 5715 and 5925 levels including damage to the main decline and damage to the mine communications system. The event also temporarily trapped 9 miners (Slade, et al., 2007).

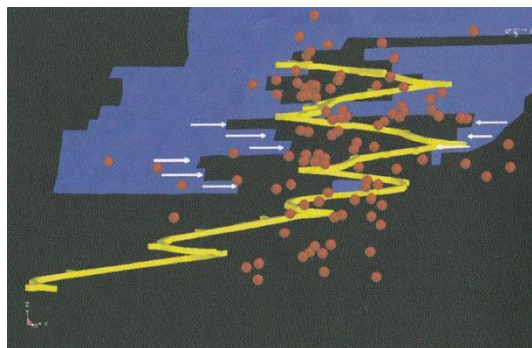


Figure 8: Location of rockbursting events at Strzelecki Mine on July 29 and 30, 2001 (Slade, et al., 2007)

3.3.4.2 Aftermath

It was determined that the 2.0 M_L burst was a fault-slip event that caused several smaller seismic events resulting in localized rockburst damage. As a result of the event, operations at the Strzelecki Mine were halted for 9 months while rehabilitation work was done and the support system was upgraded. The bursting sequence also led to a change in the extraction sequence, to eliminate the practice of leaving a central pillar (Slade, et al., 2007).

3.3.4.3 Calculating the Cost of the Event

3.3.4.3.1 Assumptions

The Strzelecki Mine is part of the Kundana Project, which is located in the Kalgoorlie West Camp in Western Australia, Figure 9. During the time of the event, in July of 2001, the mine was owned by Goldfields Ltd. In January of 2002, Delta Gold merged with Goldfields Ltd. to form AurionGold which was subsequently acquired by Placer Dome on October 31, 2002 (Placer Dome Inc., 2003). Further complicating the data set is the fact that Goldfields reported results for the Kundana Project and not for Strzelecki specifically, while Placer Dome reported results for Kalgoorlie West, which combines the Kundana Project with the Paddington Mill and associated mines. Furthermore, the production reported for Kalgoorlie West by Placer Dome for the 2002 operating year only represents the production for November and December of 2002. Since production resumed in May of 2002, the production loss for 2002 cannot be inferred from Placer Dome's 2002 annual report. Since AurionGold was purchased prior to the end of the 2002 fiscal year, there is no record of the production from the Kundana Project or Kalgoorlie West for January to October of 2002.

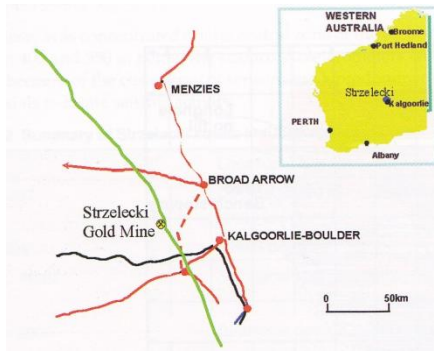


Figure 9: Location of the Strzelecki Gold Mine (Slade and Ascott, 2007)

Since data is available by fiscal quarter for Kundana for 2001,¹¹ the production loss for Strzelecki, which forms a part of the Kundana Project, can be assumed based on the difference between the average production for quarters 1 and 2, when Strzelecki was operating, and the average production for quarter 4, when Strzelecki was shut down.

Furthermore, it is assumed that the weighted average cash operating cost for quarters 1 and 2 of 2001 at Kundana represent the average cost for Strzelecki. The gold price for the shutdown period is assumed to be \$311 (2002 USD/oz. Au) based on the average 2002 gold price published by the USGS (George, 2008).

It is assumed that the affected ore zone will be mined in 5 years. This is based on the December 31, 2002 Kalgoorlie West reserves of 1.5 million ounces of gold with a 2002 production rate of 0.4 million ounces of gold resulting in 3.75 years of mining remaining (Placer Dome Inc., 2003).

¹¹ Q1 Jan. to March 2001, Q2 April to June 2001, Q3 July to Sept. 2001, Q4 Oct. to Dec. 2001

This means, based on production rates, operations can be sustained through to September 2006, 5 years and 2 months after the rockburst event. Lastly, it is assumed that the costs identified by Slade and Ascott (2007) are in 2002 AUD.

3.3.4.3.2 Direct Cost

The total cost of the rehabilitation program, which included the installation of 15,000 m of cable bolts, 2000 sheets of mesh, 2000 split sets and 9000 conebolts, was estimated to be \$3 million (2002 AUD) (Slade, et al., 2007). This translates to a direct cost of \$3 million (2010 CAD).

3.3.4.3.3 Decline in Value of Ore from Postponed Production

The operating costs for the Kundana operations are outlined in Table 15. The average operating cost was calculated based on the weighted average operating cost for quarter 1 and quarter 2 of 2001 and was found to be \$217 (2002 CAD/oz. Au). Given the assumed gold price of \$311 (2002 USD/oz. Au) the profit is \$272 (2002 CAD/oz. Au).

It can be observed that the average production per month for quarters 1 and 2 was 14,376 and 18,252 ounces respectively, Table 15. Therefore the average monthly gold production for quarters 1 and 2 is 16,314 ounces of gold, compared to the average production per month for quarter 4 of 7,354 ounces of gold. This decline of 8,960 ounces of gold per month is assumed to represent the loss in production due to the July rockburst. Given that the mine was shut down for 9 months, the lost production as a result of the event is 80,642 ounces of gold.

Table 15: Production results by quarter for 2001 for the Kundana operations (GoldAvenue, 2002)

	Quarter 1	Quarter 2	Quarter 3	Quarter 4
Ore Treated (mt)	186,000	195,000	196,000	210,000
Grade (g/mt)	7.6	8.9	4.8	3.3
Gold Produced per quarter (oz.)	43,129	54,756	28,757	22,062
Gold Produced per month (oz.)	14,376	18,252	9,586	7,354
Cash Cost (2001 USD/oz.)	139	135	190	202

The profit of the delayed production was found to be \$21.9 million (2002 CAD). When this is discounted at 10% for 5 years, the net present value is \$13.6 million (2002 CAD). The difference, representing the decline in value of the ore as a result of delayed recovery, is \$8.3 million (2002 CAD) or \$9.7 million (2010 CAD). The calculations are outlined in Table 16.

Table 16: Calculating the decline in value of ore from postponed production for the Strzelecki Mine

Weighted Average Operating Cost (2002 CAD/oz. Au)	216.83
Gold Price (2002 CAD/oz. Au)	488.38
Profit (2002 CAD/oz. Au)	271.55
Average Monthly Production for Q1 and Q2 (oz. Au)	16,314
Average Monthly Production for Q4 (oz. Au)	7,354
Production Loss per Month (oz. Au)	8,960
Production Loss (months)	9
Total Production loss (oz. Au)	80,642
Profit of Delayed Ore (2002 CAD)	21,898,340
NPV of Delayed Ore (discounted @10% over 5 years) (2002 CAD)	13,597,146
Decline in Value of Ore from Postponed Production (2002 CAD)	8,301,194

3.3.4.3.4 Total Cost

While the total cost calculated is \$12.7 million (2010 CAD), Table 17, the actual cost of this event is probably higher. Slade and Ascott (2007) note that besides the \$2.9 million (2010 CAD) rehabilitation cost and the loss of 9 months of production, “...substantial section of the remaining ore reserve may not be extracted.” However, since the Strzelecki operation was never reported as an individual mine in annual reports and due to the multiple changes in ownership of the operation between 2002 and 2006, it is impossible to identify lost reserves as a result of the 2001 fault-slip event.

The event also occurred in a region where several mine personnel, including the underground manager, mine foreman and geotechnical engineers were working. 9 miners were temporarily trapped as a result of blockage in the decline and communications were knocked out. Had this event resulted in fatalities, the cost would have been substantially higher.

Table 17: Summary of costs of the 2001 Strzelecki Mine event (all values in 2010 CAD)

Direct Cost	Decline in Value of Ore Due to Postponed Production	Total Cost
2,982,983	9,670,891	12,653,873

3.3.5 Çayeli Mine, 2002

3.3.5.1 Event Description

On October 25, 2002, Çayeli Mine, a copper zinc underground mine located on the Black Sea coast in North-eastern Turkey, majority owned and operated by Inmet,¹² experienced an ore pillar burst which led to the rapid redistribution of stresses leading to further ground failure. The event occurred in the south section of the mine which is mostly mined out and backfilled. There were no injuries or damaged equipment as a result of the event but significant damage was done to the ramp, Figure 10 (Inmet Mining Corporation, 2002).

¹² At the time of the event, Inmet owned 55% and were the operators of the mine

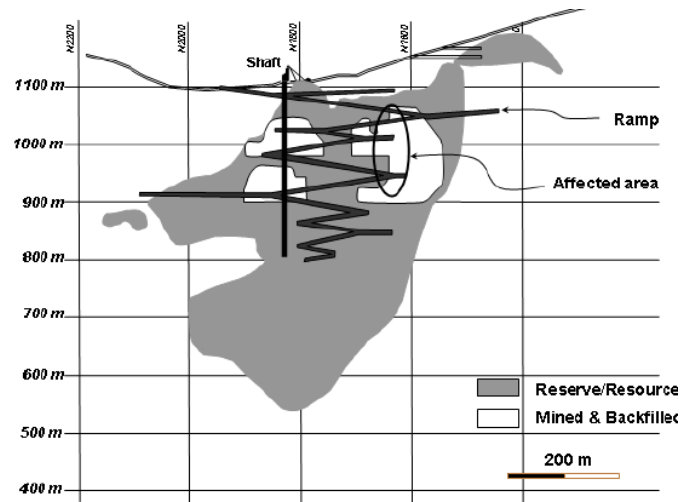


Figure 10: Longitudinal section of the Çayeli Mine with the location of the rockburst circled (Inmet Mining Corporation, 2002)

3.3.5.2 Aftermath

The damage to the ramp led to significant production delays while rehabilitation work was done. This included 42 days of no production, 22 days of 50% production, 3 months of 65% production and 3 months of 85% production (Inmet Mining Corporation, 2003). Furthermore, the event led to the implementation of smaller stopes, resulting in higher operating costs and a reduction in annual production from initial estimates.

3.3.5.3 Calculating the Cost of the Event

3.3.5.3.1 Assumptions

The Çayeli Mine had an insurance policy in place which covered the cost of production delays while the mine was in standby and rehabilitation costs associated with a rockburst event. It is

assumed that the insurance claim filed by Çayeli, stated on Inmet's 2003 annual report, represents the total cost of the rockburst event. It is also assumed that the only costs are rehabilitation and deferred production since the event occurred in a mined out section. Based on this, it is assumed that the direct cost can be found by subtracting the insurance claim from the calculated deferred production.

In order to calculate the tonnes of ore affected by the event, it is assumed that the production targets for Çayeli, as stated in the 2002 3rd quarter financial statements, represent the targets for both 2002 and 2003. The difference between the 2002 targets and the 2002 and 2003 realized production represents the shortfall in production. It is also assumed that the copper and zinc prices for 2002 are \$0.71 and \$0.36 (2002 USD/lb) and for 2003 are \$0.81 and \$0.38 (2003 USD/lb) respectively based on the 2004 USGS commodity reports (Edelstein, 2004) (Plachy, 2004). Lastly, it is assumed that the ore delayed as a result of the 2002 event will be mined in 2016 based on a stated mine life of 2017 (Inmet Mining Corporation, 2010).

3.3.5.3.2 Direct Cost

The direct cost for the Çayeli event is based on the difference between the insurance claim and the deferred production cost. The insurance claim reported by Inmet amounts to \$4.4 million (2002 USD) in the 4th quarter of 2002 and \$8.6 million (2003 USD) in the 1st quarter of 2003. This translates to a total claim of \$21.7 million (2010 CAD). Based on a decline in value of postponed production due to the event of \$13.3 million (2010 CAD) (calculated below), the direct cost of the event is \$8.4 million (2010 CAD).

3.3.5.3.3 Decline in Value of Ore from Postponed Production

As previously mentioned, the Çayeli pillar burst led to a 42 day work stoppage, 22 days of 50% production, 3 months of 65% production and 3 months of 85% production. The actual tonnage affected by the event can be calculated by analyzing the target production with the actual production.

Table 18: Comparison between 2002 target production and 2002 and 2003 realized production for Çayeli Mine

Production	Target¹³	2002 Realized¹⁴	2003 Realized¹⁵
Ore Milled (tonnes)	1,030,000	895,000	928,000
Copper Grade (%)	4.20	4.20	4.20
Zinc Grade (%)	5.00	5.10	5.10
Copper Recovery (%)	86	87	87
Zinc Recovery (%)	72	72	72
Copper (tonnes)	38,000	32,600	33,500
Zinc (tonnes)	38,000	33,100	33,600

From Table 18 it can be seen that the shortfall in production for 2002 was 5,400 tonnes of copper and 4,900 tonnes of zinc compared to 8,617 tonnes of copper and 5,622 tonnes of zinc in 2003. Given a 2002 and 2003 operating cost of \$0.66 (2002 USD/lb Cu) and \$0.71 (2003 USD/lb Cu) respectively, and the assumed metal prices (stated above) the decline in value of ore from postponed production is found to be \$13.3 million (2010 CAD), Table 19 and Table 20.

¹³ From Inmet Mining Corporation's 2002 3rd Quarter Report (Inmet Mining Corporation, 2002)

¹⁴ From Inmet Mining Corporation's 2002 Annual Report (Inmet Mining Corporation, 2003)

¹⁵ From Inmet Mining Corporation's 2003 Annual Report (Inmet Mining Corporation, 2004)

Table 19: Calculating the decline in value of ore from postponed production in 2002 for the Çayeli Mine

	2002 USD	2002 CAD	2010 CAD
Gross Copper Value	\$8,416,808	\$13,214,389	\$15,394,763
Gross Zinc Value	\$3,856,546	\$6,054,778	\$7,053,816
Total Operating Cost	\$7,857,275	\$12,335,922	\$14,371,349
Profit of Delayed Ore	\$4,416,080	\$6,933,245	\$8,077,230
NPV of Delayed Ore (discounted at 10% over 14 years)	\$1,162,892	\$1,825,740	\$2,126,987
Decline in Value of Ore from Postponed Production	\$3,253,188	\$5,107,505	\$5,950,243

Table 20: Calculating the decline in value of ore from postponed production in 2003 for the Çayeli Mine

	2003USD	2003 CAD	2010 CAD
Gross Copper Value	\$15,330,233	\$21,462,327	\$24,322,578
Gross Zinc Value	\$4,648,226	\$6,507,517	\$7,374,764
Total Operating Cost	\$13,487,566	\$18,882,592	\$21,399,047
Profit of Delayed Ore	\$6,490,894	\$9,087,251	\$10,298,295
NPV of Delayed Ore (discounted at 10% over 13 years)	\$1,880,181	\$2,632,253	\$2,983,049
Decline in Value of Ore from Postponed Production	\$4,610,713	\$6,454,998	\$7,315,246

3.3.5.3.4 Total Cost

While the total cost of the Çayeli event is assumed to be the value of the insurance claim, \$21.7 million (2010 CAD), the actual value is probably higher than this. The event led to the mine

utilizing smaller stope sizes, which has increased the operating cost and reduced the annual production capacity.

3.3.6 DRD Gold, 2005

3.3.6.1 Event Description

On March 9, 2005, DRD Gold's Northwest Operations, located in the Northwest Province of South Africa, suffered a 5.3 M_L event. The event caused widespread damage to the mine, including damage to the No. 5 shaft and surrounding infrastructure. In addition, 2 workers were killed and 3200 workers were evacuated from the mine. The event also caused substantial damage to the nearby town of Stilfonstein and injured 58 people in the town (Durrheim, et al., 2007).

3.3.6.2 Aftermath

Shortly following the event, the mine was permanently closed and went into liquidation. The event led to a major investigation into mining practices, design approaches, monitoring and management in South Africa. Investigation into the cause of the event found that historical mining lead to the slippage of a major fault (Durrheim, et al., 2007).

3.3.6.3 Calculating the Cost of the Event

3.3.6.3.1 Assumptions

DRD Gold had an insurance policy covering the direct cost and loss of production. It is assumed that the insurance claim filed by DRD Gold represents the total direct cost and the value of the postponed production as a result of the rockburst event, but does not include the fatality cost or

the value of the reserves lost. It is therefore assumed that the value of ore lost due to the event is the value of the 5.1 million ounces of gold reserves remaining at the Northwest Operations at the time of closure (DRD Gold Limited, 2006).

It is assumed that the discontinued operations results published by DRD Gold in the 2005 annual report can be fully attributed to the Northwest Operations. Therefore, the annual production rate used to calculate the value of ore lost is based on the 2004 results for discontinued operations. Production costs and gold price are assumed to be \$379 (2005 USD/oz. Au) and \$421 (2005 USD/oz. Au) respectively, based on the prices stated in DRD Gold's 2005 annual report. These prices represent the operating cost and realized gold price for all of DRD Gold's operations but are assumed to be the prices realized at the Northwest Operations.

3.3.6.3.2 Direct Cost

The direct cost for the Northwest Operations event is based on the difference between the insurance claim and the value of the postponed production due to the event. The direct cost includes rehabilitation work done prior to the mine permanently closing and damage to the nearby town of Stilfonstein. The insurance claim was reported to be 500 million (2005 Rand) (Durrheim, et al., 2007) which, based on a 2005 exchange rate of 1 Rand to \$0.1907 CAD (Financial Markets Department, 2006) corresponds to \$103.8 million (2010 CAD). Given that the value for lost production due to the event is \$7.5 million (2010 CAD) (calculated in the following section), the direct cost of the event is \$96.3 million (2010 CAD).

3.3.6.3.3 Value of Ore Lost Due to the Event

Based on the 2004 production of 341,861 ounces of gold and the stated loss of 5.1 million ounces of gold reserves due to the event, the mine life of the lost ounces of gold is found to be 14.9 years. Based on a discount rate of 10%, cash cost of \$379 (2005 USD/oz. Au) and realized gold price of \$421 (2005 USD/oz. Au) the net present value of the reserves lost is \$143.7 million (2010 CAD, Figure 11).

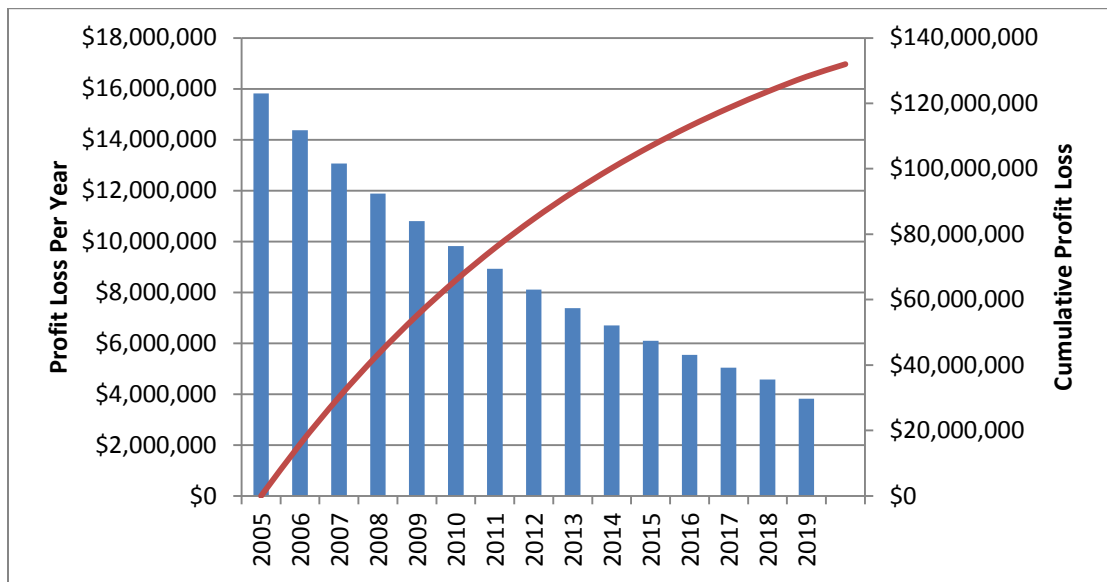


Figure 11: Annual profit of lost reserves discounted at 10%

3.3.6.3.4 Decline in Value of Ore from Postponed Production

For the 2005 fiscal year, DRD Gold reported a shortfall in production of 136,137 ounces which the company attributed to the loss of production at the Northwest Operations following the March 9, 2005 event. Based on an operating cost of \$379 (2005 USD/oz. Au) and a realized gold price of \$421 (2005 USD/oz. Au) the undiscounted value of ore from postponed production is \$5.7

million (2005 USD) or \$7.5 million (2010 CAD). Since, similar to the 1997 Macassa event, the mine closed shortly after the event occurred, this gold was never recovered and therefore the entire value of the postponed ore is considered lost.

3.3.6.3.5 Cost of Fatalities

The 2005 Northwest Operations event resulted in 2 fatalities. Based on the assumed cost of a fatality to a mining company of \$8 million (2010 CAD), the cost of the 2 fatalities is \$16 million (2010 CAD).

3.3.6.3.6 Total Cost

The total cost of DRD Gold’s Northwest Operations fault-slip event is \$263.5 million (2010 CAD). This is by far the most expensive event analyzed (including events provided by mining companies). The event has such a high cost due to the damage caused to the nearby town and the loss of 5.1 million ounces of reserves resulting from the premature closure of the mine following the event. The costs by category are outlined in Table 21.

Table 21: Summary of costs of the 2005 DRD Gold Northwest Operations event (all values in 2010 CAD)

Direct Cost	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Cost of Fatalities	Total Cost
\$96,272,752	\$143,699,465	\$7,542,902	\$16,000,000	\$263,515,120

3.4 Results and Analysis

3.4.1 Summary of Results from Public Data

The results from the 6 cost models developed from public documentation are outlined in Table 22. The average cost per event is \$64.9 million (2010 CAD) with a range of \$5.3 to \$263.5 million (2010 CAD) and a median cost of \$32.3 million (2010 CAD).

Table 22: Summary of results from cost models developed from public documentation (all values in 2010 CAD)

	Direct Cost	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Cost of Fatalities	Total Cost
Total	\$140,254,551	\$170,496,783	\$46,563,624	\$32,000,000	\$389,314,957
High	\$96,272,752	\$143,699,465	\$13,265,489	\$16,000,000	\$263,515,120
Low	\$2,982,983	\$0	\$0	\$0	\$5,324,319
Average	\$23,375,758	\$28,416,130	\$7,760,604	\$5,333,333	\$64,885,826
Median	\$10,150,311	\$0	\$8,606,896	\$0	\$32,312,343

3.4.2 Summary of Confidential Results from Mining Companies

The results from the 13 cost models developed from confidential data provided by mining companies is outlined in Table 23. The 13 events are from 8 mines and occurred between 1984 and 2009. All events are fault-slip type events from hard rock underground operations located in Canada. The average total cost is \$21.8 million (2010 CAD) with a range of \$1.1 to \$120.7 million (2010 CAD) and a median cost of \$5.4 million (2010 CAD).

Table 23: Summary of results from cost models developed from confidential data provided by mining companies (all values in 2010 CAD)

	Direct Cost	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Cost of Fatalities	Total Cost
Total	\$81,797,805	\$120,776,222	\$52,579,615	\$32,000,000	\$282,923,394
High	\$40,841,367	\$79,865,704	\$19,830,275	\$32,000,000	\$120,707,071
Low	\$0	\$0	\$0	\$0	\$1,140,559
Average	\$6,292,139	\$9,290,479	\$3,719,182	\$2,461,538	\$21,763,338
Median	\$1,425,699	\$0	\$1,277,115	\$0	\$5,430,673

3.4.3 Summary of all Results

The results from all 19 cost models are outlined in Table 24. The average cost per event is \$37.6 million (2010 CAD) with a range of \$1.1 to \$263.5 million (2010 CAD) and a median cost of \$12.8 million (2010 CAD).

Table 24: Summary of results from all 19 cost models developed (all values in 2010 CAD)

	Direct Cost	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Cost of Fatalities	Total Cost
Total	\$222,052,356	\$291,273,005	\$94,912,990	\$64,000,000	\$672,238,351
High	\$96,272,752	\$143,699,465	\$19,830,275	\$32,000,000	\$263,515,120
Low	\$0	\$0	\$0	\$0	\$1,140,559
Average	\$11,686,966	\$15,330,158	\$4,995,421	\$3,368,421	\$35,380,966
Median	\$2,895,392	\$0	\$4,163,261	\$0	\$12,829,870

Figure 12 shows the frequency distribution of the total cost of the 19 events analyzed. It can be seen that 11 events have a cost less than \$15 million (2010 CAD), 3 events range between \$20 and \$30 million (2010 CAD), 2 events are between \$40 and \$45 million (2010 CAD) and 3 events are more than \$50 million (2010 CAD). Of the 3 events more than \$50 million, 1 is \$62 million (2010 CAD), 1 is \$121 million (2010 CAD) and 1 is \$264 million (2010 CAD).

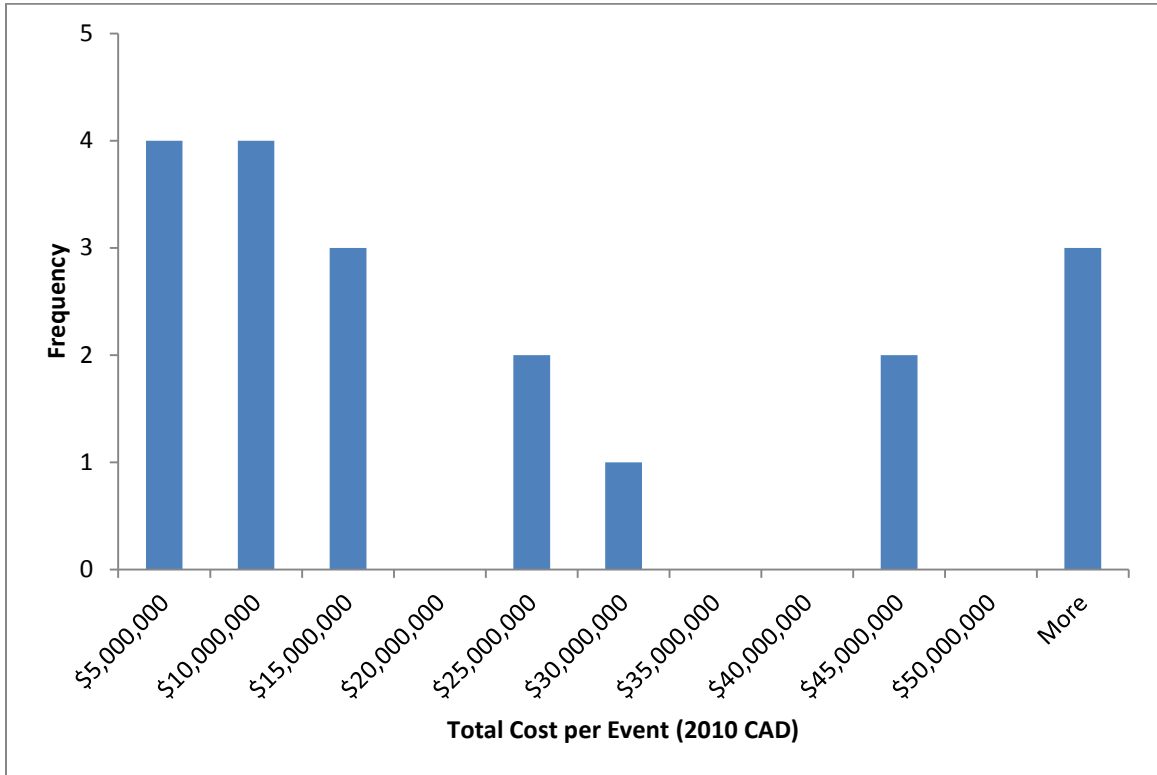


Figure 12: Frequency of the cost per event for the 19 events analyzed

3.4.4 Cost per Event for Case Studies

In Chapter 4 probability analysis will be carried out to identify an increase in expenditure that should be made to prevent future ground failure at 2 mines and for all of Ontario. These

calculations are based on the cost models developed in this chapter. The 2 mines that will be analyzed are based on cost models developed from data provided by mining companies. Therefore these operations will be called Mine A and Mine B.

Mine A had 3 major events over a 10 year period. The average cost per event was found to be \$10.6 million (2010 CAD) with a range of \$5 to \$21 million (2010 CAD). Mine B had 2 events in 1 year which had an average cost of \$11.6 million (2010 CAD). None of the events at either mine resulted in the sterilization of ore or fatalities. All of the events resulted in a direct cost and a delay in production.

Of the 19 events summarized in table 24, 16 are from mines located in Ontario. These events range from 1984 to 2009 and have an average cost of \$23.3 million (2010 CAD) with a range of \$1.1 to \$120.7 million (2010 CAD) and a median of \$9.1 million (2010 CAD). For the Ontario Case Study an average cost per event of \$23.3 million (2010 CAD) will be used.

3.5 Discussion

3.5.1 Public vs. Private Data

Review of Tables 22 and 23 show that there is a substantial difference between the average cost per event for the public data (\$64.9 million 2010 CAD) and the average cost per event for the confidential data (\$21.8 million 2010 CAD). This can be attributed to two main factors, the high cost of the DRD Gold event and the types of events analyzed.

First, the DRD Gold event, which had an average cost of \$263.5 million, is more than six times greater than the Macassa Mine 1997 event which is the next highest cost model from the public data. With only six events analyzed from the public domain, one event with such a high cost greatly increases the average cost of the six events. Removing the DRD Gold event, the average cost drops to \$25.2 million (2010 CAD).

Secondly, only rockburst events with significant financial impact are disclosed by mining companies in public documentation. Since the majority of the public events were derived by analysis of annual reports by mining companies, it is logical that these events would have a higher cost than the cost models developed from confidential information since mining companies would not publish the costs associated with smaller, less significant events. In fact, of the 11 events with a total cost less than \$15 million (2010 CAD), Figure 12, only 2 are from public information, while the other 9 were developed from confidential information provided by mining companies.

It is important to note that this doesn't mean that the public cost models are unreliable, this simply shows the wide range in cost associated with rockbursting events. The events chosen range in severity and damage caused. The DRD Gold event is an example of the extreme high end cost associated with rockbursting while the lower costs per event represent the cost associated with smaller more frequent events.

3.5.2 Cost Categories

3.5.2.1 Direct Cost vs. Indirect Cost

Of the 19 events analyzed, 18 had a direct cost while 7 events led to the loss of ore and 14 events resulted in production delays. From Table 24 it can be seen that the highest single cost was related to the value of ore lost due to an event, the second highest single cost was the direct cost associated with a rockburst event. Similarly, the highest average cost category is the value of ore lost due to an event and the second highest average cost is the direct cost. This shows that while the direct cost associated with a rockburst event can have a major financial impact, the indirect costs can be substantially higher and therefore should not be negated.

3.5.2.2 Fatalities and Near Misses

While an average cost of \$8 million (2010 CAD) was applied per fatality for all events, it is important to note that this value can be significantly higher. Of the 19 events analyzed, 3 resulted in fatalities, these being DRD Gold's Northwest Operations, Macassa Mine and Mine X.¹⁶ When looking at the total cost of the 3 events, each has a higher cost than the \$20.0 million (2010 CAD) average derived above. Furthermore, all 3 events led to the premature closure of the mines. While the fatalities at DRD Gold's Northwest Operations played a lesser role in the decision to close the mine, the 1993 fatalities at Macassa and the fatalities at Mine X both played a significant role in the premature closing of the mines. The 1993 Macassa event was a significant factor in the Ministry of Labour's decision to issue a stop work order below 5000 feet following the 1997 event. This decision ultimately led to the closure of Macassa Mine in 1999 due to the

¹⁶ Note that the cost model for Mine X was developed based on information provided by a mining company and therefore the mine cannot be disclosed

loss of 242,027 ounces of gold from reserves. Mine X had 4 fatalities as a result of a major rockburst, which led to the immediate closure of the mine and the loss of approximately 1.2 million tonnes of ore.

It can therefore be seen that while a cost of \$8 million (2010 CAD) per fatality was used to show the direct cost of a fatality to a company, the actual cost, at least in the 3 events reviewed that involved fatalities, can be much greater. It should be noted that both Macassa Mine and Mine X are located in Canada which is regarded as having one of the best mining safety standards in the world. Therefore, a fatality is viewed much more critically. It can be concluded that a fatality in a Canadian mine would have a more significant financial impact than a fatality in a mine located in a country with less stringent regulations.

3.5.3 Location of the Event

Analysis into the events used for the cost models showed that the location of the event had the biggest impact on the total cost. When events occurred outside the ore zone, away from infrastructure, the costs were the lowest, but where events occurred in key infrastructure, i.e. impacted the shaft, ramp or key intersections, the cost was greatest. This is logical, since an event impacting infrastructure will have a rehabilitation cost, cause delay in recovery of ore and could lead to the sterilization of ore.

An excellent example of the potential high cost of an event based on its location occurred at Mine Y.¹⁷ Multiple fault-slip events occurred at the face of a drift that was being driven to access a potential new resource. The resource had a 23 year mine life and a net present value (discounted at 10%) of \$660 million (2010 CAD). The inability to safely access the resource and the near misses that occurred during the exploration phase resulted in the company walking away from this project. Had the fault slip event occurred in a different part of the mine, such as a mined out stope or unused drift, the impact would have been minimal, but due to the event occurring in the access drift, the loss of the resource was substantial.

¹⁷ Note that the cost model for Mine Y was developed based on information provided by a mining company and therefore the mine cannot be disclosed

Chapter 4

Estimating an Appropriate Increase in Upfront Expenditure

This chapter will develop 3 case studies based on the results for the average cost of rockbursting for Ontario, Mine A and Mine B calculated in the previous chapter and discussed in Section 3.4.4. The case studies will identify an increase in expenditure that should be made on top of the baseline spending to mitigate the risk of a potential rockburst. While this chapter will calculate an average annual expenditure to minimize rockburst risk in Ontario, this is simply done to highlight the cost of rockbursting and show an approach to calculating the cost. While the trend in rockbursting can be analyzed, more specific conclusions cannot be drawn from this case study since the probability of a rockburst will vary greatly depending on local conditions.

Furthermore, it is important to note that while this chapter will identify an increase in expenditure from baseline spending, this value is based on historical average rockbursting costs. As noted in the previous chapter, rockburst costs can range greatly depending on location, magnitude, support structures in place, etc. To properly apply the methodological approach outlined in this chapter, these factors must be considered when determining the average cost per event.

4.1 Methodology

The approach used to calculate an upfront increase in expenditure is based on the average cost per event developed from the cost models in the previous chapter, the probability that an event will occur and the probability that if an event occurs it will cause damage.

As outlined in Section 2.4 (Probability of Rockbursting), the double truncated Gutenberg-Richter frequency magnitude relationship can be used with Benjamin's (1968) seismic hazard function to calculate the probability of a seismic event occurring based on historical seismic data. McGarr (1976) notes that this approach is applicable only where the rate and geometry of mining does not significantly change from year to year. It is therefore assumed that no major changes in geometry or mining have occurred during the time period that is being analyzed for the 3 case studies developed. For the two individual mines analyzed, this is considered an acceptable approach.

The minimum magnitude at which rockbursts are likely to cause severe damage will be found based on research done by Hedley (1992). Next, the probability that an event will cause damage will be calculated based on rockbursting data provided by the Ontario Ministry of Labour. The dataset provided by the Ministry of Labour includes 382 events from 1986 to 1999 based on Unusual Occurrence Reports submitted to the Ministry. These reports are submitted if "A rockburst occurs causing damage to equipment or the displacement of more than five tonnes of material (Ontario Ministry of Labour, 2007)." The trend between tonnage displaced and magnitude of event will be analyzed to identify at what magnitude severe damage is caused and what the likelihood that an event of that magnitude will cause damage based on the number of events that caused damage versus the number of events that did not. This is discussed in detail in the next section.

Given the average cost per event and the probability an event will cause damage, the increase in upfront expenditure will be found for a 50%, 80% and 100% probability of an event occurring. This will be done by analyzing the magnitude-probability time plots developed using the method

outlined above. Based on the number of years for 50%, 80% and 100% probability of occurrence, the average cost multiplied by the probability an event will cause damage and the probability an event will occur, will be discounted at a rate of 10% as outlined in Section 2.1.2. This will yield a net present value that represents the potential cost of a rockburst event to a company today (14).

$$NPVC_{max} = \frac{(AC_{event} \times P_{damage} \times P_{event})}{(1 + i)^t} \quad (14)$$

where $NPVC_{max}$ represents the net present value of the average cost of an event, AC_{event} is the average cost of an event, P_{damage} is the probability the event will cause damage, P_{event} is the probability the event will occur (50%, 80% or 100%), i is the discount rate (10%) and t is the return period in years. This value can be used to identify an increase in expenditure that should be made today to mitigate a potential rockburst event in the future.

4.2 Key Parameters

4.2.1 Identifying a Minimum Magnitude that Rockbursting Causes Damage

It must first be noted that damage can occur at any magnitude depending on the support system in place, mining method, extraction ratio, etc. However, in order to arrive at an appropriate increase in expenditure to mitigate rockburst damage, a magnitude range that causes damage must be identified.

A 1984 to 1990 study by Hedley (1992) analyzed 391 recorded seismic events occurring in Ontario mines of magnitude 2.0 M_n or greater. Hedley found that there were 3 general categories

that the events fell into based on Nuttli magnitude, Figure 13. Minor damage of a few tonnes of displaced material was observed for events ranging from 2.0 to 2.4 M_n , some damage, tens of tonnes of displaced material, was observed for events ranging between 2.5 and 2.9 M_n and severe damage, hundreds to thousands of tonnes of displaced material, was observed for events greater than 3.0 M_n . Of the 391 events, 65% were between 2.0 and 2.4 M_n , 30% were between 2.5 and 2.9 M_n and 5% were greater than 3.0 M_n . Hedley did note that while this was the general trend, 2 events of 2.0 M_n caused 800 tonnes of material to be displaced in a drift at Strathcona Mine.

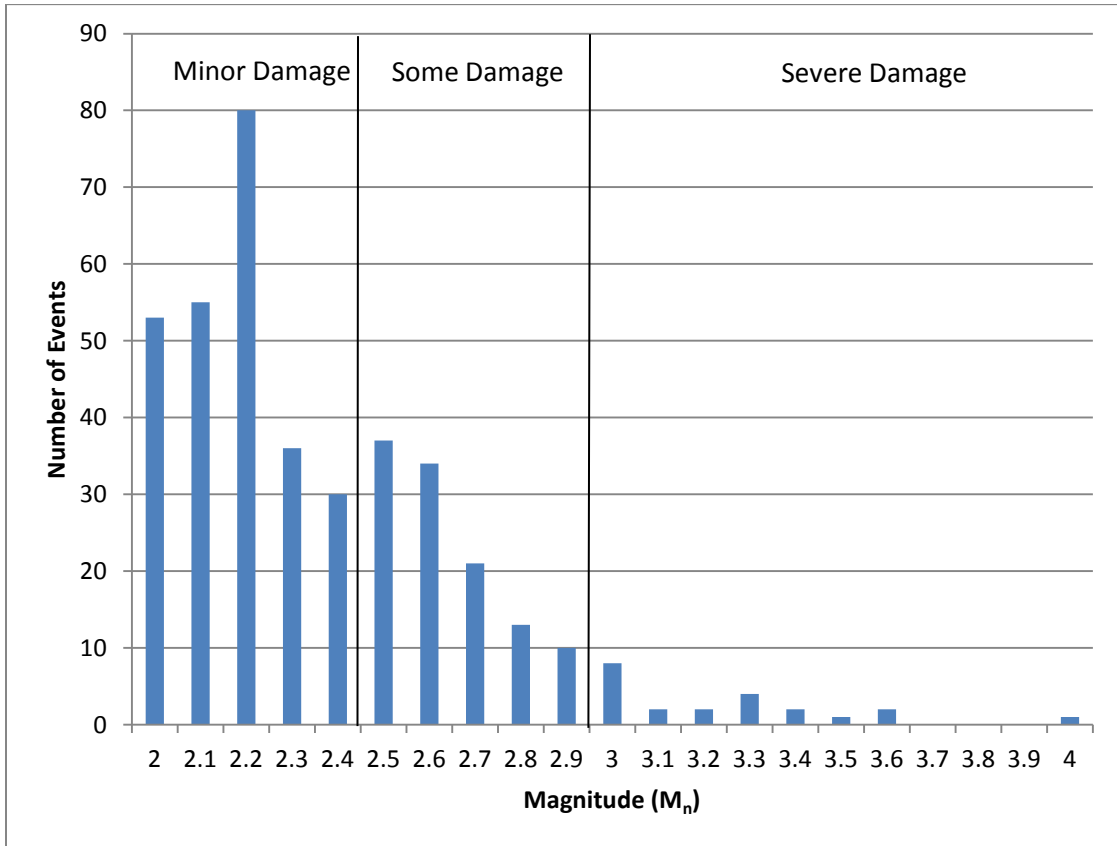


Figure 13: Distribution of seismic events in Ontario from 1984 to 1990 (Hedley, 1992)

For the case studies presented later in this chapter, it is assumed that an event of 3.0 M_n or greater will cause severe rockburst damage. Furthermore, severe damage will be defined as 100 tonnes or more of displaced material. This is based on Hedley's definition for severe damage.

4.2.2 Identifying the Probability that a Rockburst event will Cause Damage

By analyzing the Unusual Occurrence Reports data set provided by the Ontario Ministry of Labour, the probability that an event of a given magnitude will cause significant damage (100 tonnes or more) can be found. Of the 382 events in the Ministry of Labour data set, 134 include both displaced material as a result of the event and a magnitude for the event. Of these, 94 events have a magnitude of 2.0 M_n or greater, which is displayed in Figure 14.

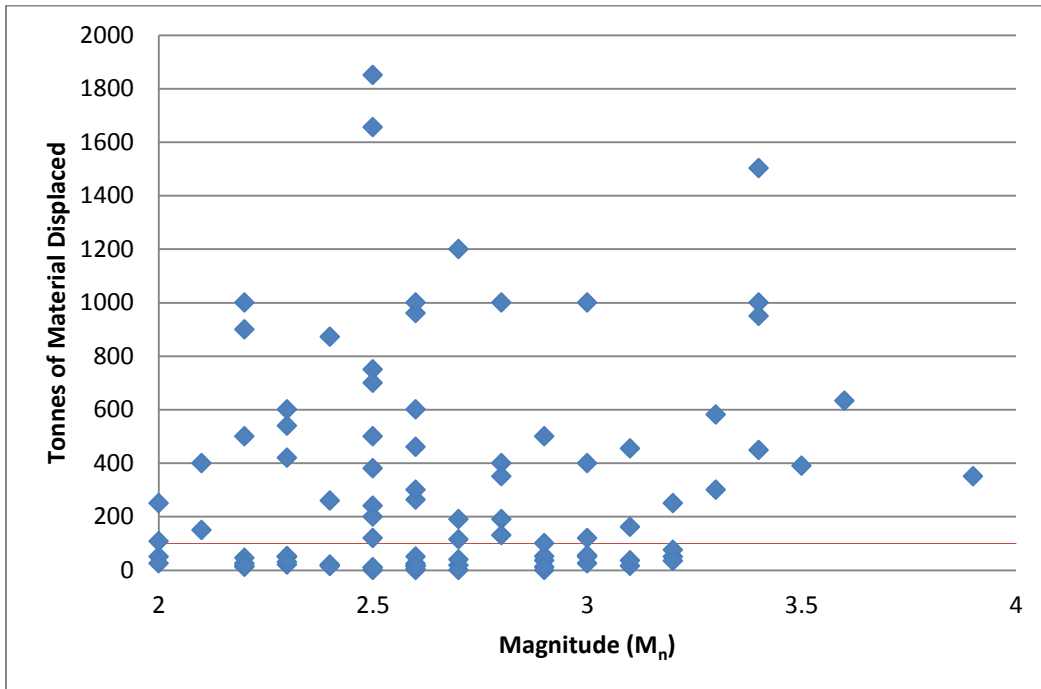


Figure 14: Tonnes of material displaced versus magnitude for events greater than 2.0 M_n , the red line indicates the severe damage threshold (Kat, 2010)

Figure 15 outlines the distribution of tonnes of material displaced for the 94 events with a magnitude of 2.0 M_n or greater. It can be seen that the cumulative percentage of events that have caused a displacement of material of less than 100 tonnes is 43%. This means that the probability that an event of 2.0 M_n or greater causing damage of 100 tonnes or more, based on the Ontario Unusual Occurrence Reports filed with the Ministry of Labour from 1986 to 1999, is 57%.

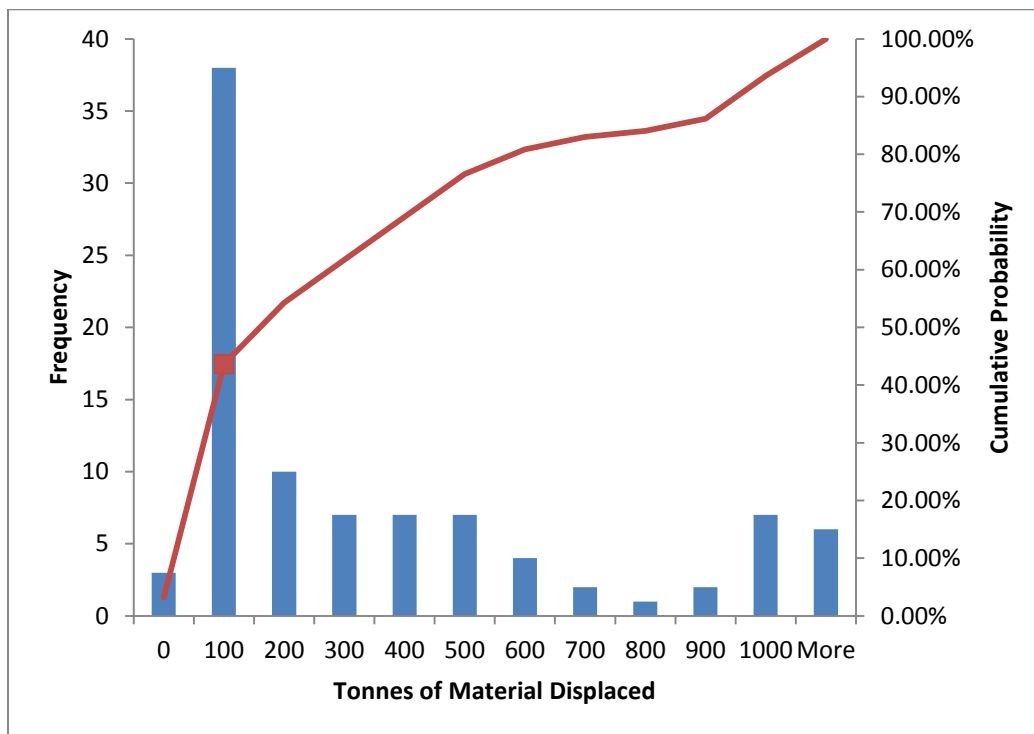


Figure 15: Distribution of tonnes of material displaced for events with a magnitude of 2.0 M_n or greater, the red line is the cumulative percentage

4.3 Case Study – Ontario Mines

4.3.1 Background

Based on a data set developed from Unusual Occurrence Reports from 1986 to 1999 provided by the Ontario Ministry of Labour and the average cost of a rockburst event for Ontario mines developed in the previous chapter, the appropriate increase in expenditure for Ontario mining operations can be calculated. It is imperative to stress that this is done for illustrative purposes to highlight the costs associated with rockbursting and the methodology that can be applied for investment decisions. Every operation will have a different probability of an event occurring, average event cost and probability an event will cause damage, based on site specific parameters including mining method, extraction ratio, support system, rock mass characteristics, local and regional stresses etc.

4.3.2 Data Acquisition

The Ontario Ministry of Labour provided a data set of 382 reported events which occurred between January 1, 1986 and December 31, 1999. As previously discussed, these events were developed from Unusual Occurrence Reports which were filed for any rockburst events that caused the ejection of more than 5 tonnes of material or caused damage to equipment.

4.3.3 Calculations

4.3.3.1 Gutenberg-Richter Frequency-Magnitude Relationship

Before analysis on the dataset could be performed, errors in the dataset were removed. This involved making corrections for the largest recorded event category which was used to find the

Gutenberg-Richter Frequency-Magnitude Relationship. Events that were comprised of multiple bursts and had a larger first magnitude event than biggest magnitude event were adjusted so that the largest magnitude event was equal to the first magnitude event.

Given the cleaned dataset, the cumulative number of seismic events was calculated and graphed, Figure 16. From the graph, it can be observed that the roll-off point, M_{\min} , is 2.5. Furthermore, it can be seen that the semi-log graph is most linear between 2.5 and 3.4 M_n . The Gutenberg-Richter Frequency Magnitude Relationship was found to be best represented by equation 15. The b-value, the slope of the best fit line between 2.5 and 3.4 M_n was found to be 1.84. The number of seismic events greater than or equal to M_{\min} , denoted as N, is 69.

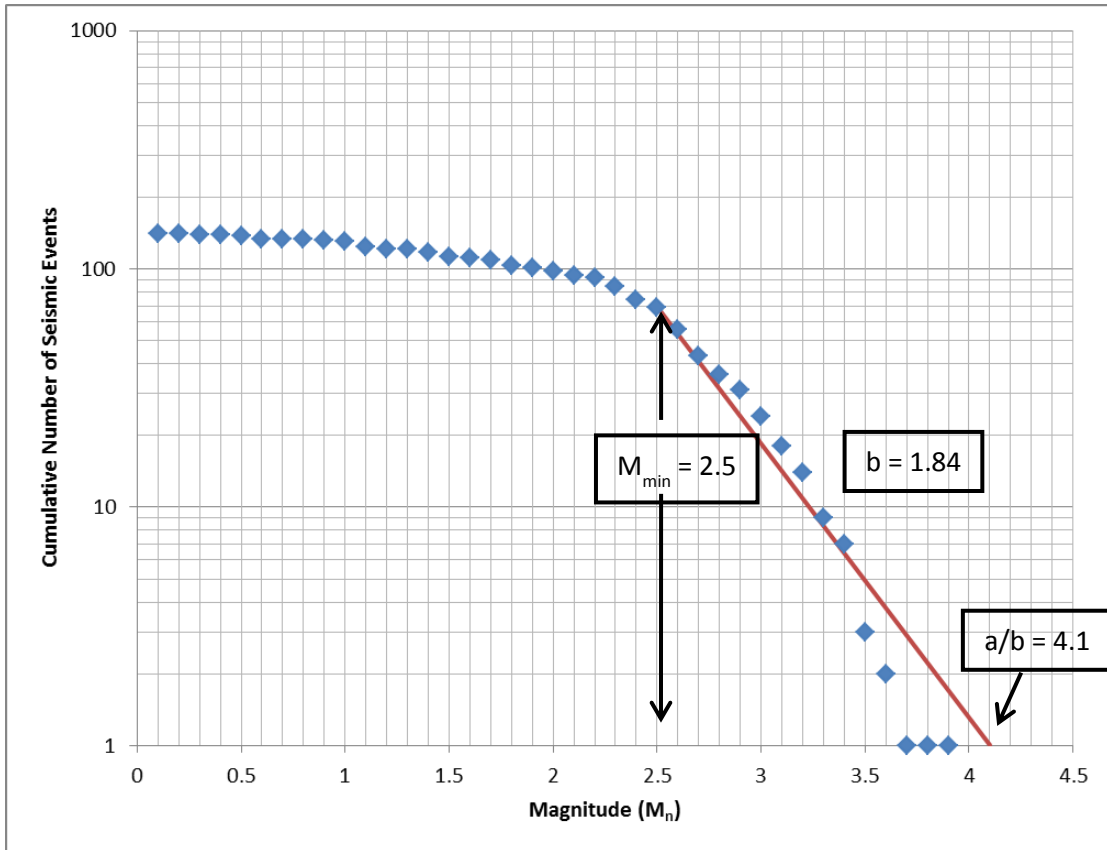


Figure 16: Frequency of occurrence of rockbursts in Ontario from 1986 to 1999

$$\text{Log } N = 4.70 - 1.15M_n \quad (15)$$

4.3.3.2 M_{\max} Calculation

M_{\max} was calculated based on equation 7 outlined in Section 2.4.1. While $M_{\max} - M_{\min}$ is less than 2, the dataset is less than 100, meaning M_{\max}^{obs} cannot be assumed to be M_{\max} in equations 8 and 9.

Therefore, multiple iterations were run changing $M_{max}^{estimate}$ until it equaled M_{max} . Calculations were performed based on the known parameters identified in Table 25. The results from multiple iterations are outlined in Table 26. Based on this, the maximum potential magnitude for an event, M_{max} , was found to be 4.14.

Table 25: Known parameters for Ontario Ministry of Labour Data

Parameter	Description	Value
M_{min}	Roll-off point	2.5
M_{max}^{obs}	Maximum observed magnitude	3.9
N	Number of seismic events $\geq M_{min}$	69
b	slope of best fit line	1.15
β	$\ln 10$	2.65

Table 26: Results for M_{max} calculation based on varying $M_{max}^{estimate}$

$M_{max}^{estimate}$	M_{max}
3.9	4.05
4.05	4.10
4.10	4.12
4.12	4.13
4.13	4.14
4.14	4.14

4.3.3.3 Double Truncated Gutenberg-Richter Frequency-Magnitude Relationship

The double truncated Gutenberg-Richter frequency-magnitude relationship was found from equation 12, discussed in Section 2.4.2 for the probability of an event with magnitude M greater than 2.5, 3, 3.5 and 4. The parameters and results are outlined in Table 27.

Table 27: Calculation of the probability that an event with magnitude E will be larger than magnitude M for Ontario dataset

Parameter	P[E>2.5]	P[E>3]	P[E>3.5]	P[E>4]
M	2.5	3	3.5	4
b	1.15	1.15	1.15	1.15
β	2.65	2.65	2.65	2.65
M_{\min}	2.5	2.5	2.5	2.5
M_{\max}	4.14	4.14	4.14	4.14
$e^{-\beta M}$	1.34E-03	3.57E-04	9.50E-05	2.53E-05
$e^{-\beta M_{\max}}$	1.75E-05	1.75E-05	1.75E-05	1.75E-05
$e^{-\beta M_{\min}}$	1.34E-03	1.34E-03	1.34E-03	1.34E-03
P[E>M]	1.000	0.257	0.059	0.006

4.3.3.4 Magnitude-Probability-Time Plot

Given $P[E>M]$ for M_n 2.5, 3, 3.5 and 4, identified in Table 27, a magnitude-probability-time plot can be generated for the Ontario dataset based on equation 13 discussed in Section 2.4.2, Figure 17. The results for 50%, 80% and 100% probability of a 3.0 M_n event are outlined in Table 28.

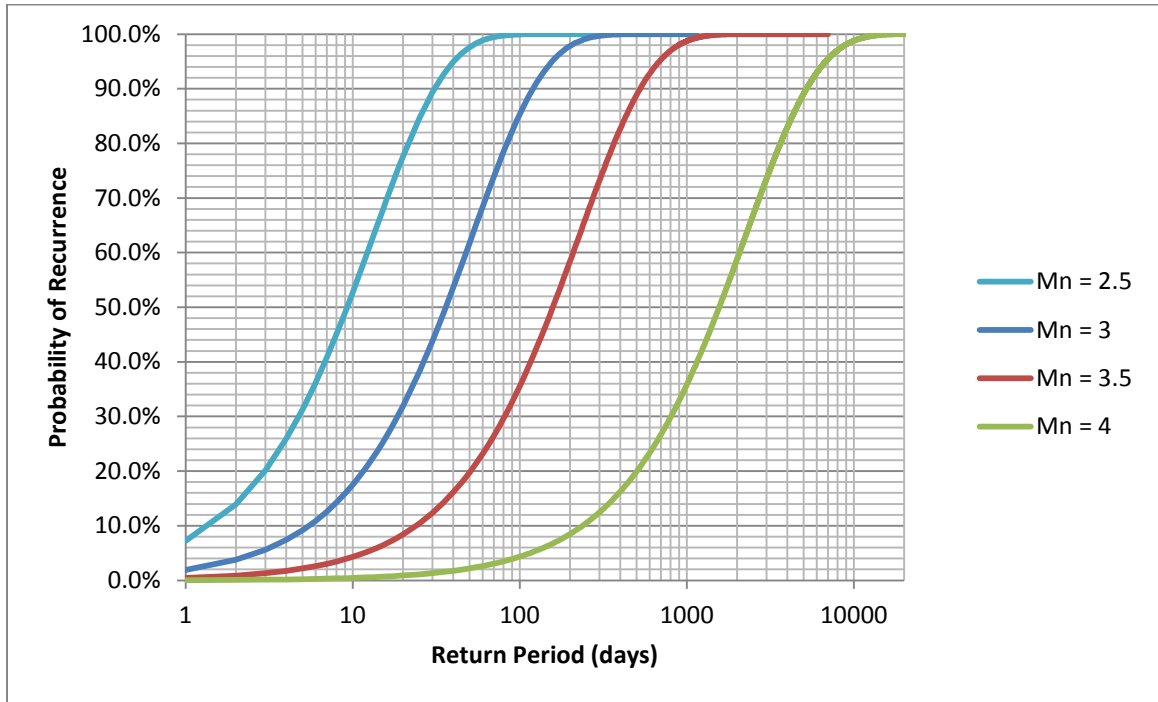


Figure 17: Magnitude-Probability-Time plot for Ministry of Labour dataset

Table 28: Return period for 50%, 80% and 100% probability of a 3.0 M_n event at an Ontario mine

Probability of Recurrence for 3.0 M_n event	Return Period (Days)	Return Period (years)
50%	36	0.01
80%	84	0.23
100%	400	1.10

4.3.4 Results

Since the time periods for 50% and 80% probability of a 3.0 M_n event occurring in Ontario are both less than half a year, the upfront expenditure for those probabilities does not need to be

discounted because the event is occurring in the same year as the start period (year 0). The 100% probability has a return period of 400 days meaning the event occurs in year 1, resulting in a 1 year discount being applied to the cost of the event at a rate of 10%.

Given the return period for 50%, 80% and 100% probability of a 3.0 M_n event calculated above and the probability a 3.0 M_n event will cause damage, 57%, calculated in the previous section, the increase in expenditure that should be made to mitigate the rockburst risk can be found by using equation 14. Based on the average cost per event of \$23.5 million (2010 CAD) calculated in Section 3.4.4, the increase in expenditure for each of the three probabilities of occurrence analyzed is outlined in Table 29.

Table 29: Increase in expenditure for varying probabilities of a 3.0 M_n event based on Ontario Ministry of Labour Dataset

Probability of 3.0 M_n Event	Increase in Upfront Expenditure
50%	\$6,678,758
80%	\$10,686,013
100%	\$13,357,516
100% (discounted 1 year at 10%)	\$12,143,197

4.3.5 Discussion

Given that there is a 100% probability that a 3.0 M_n event will occur in Ontario in the next 400 days and the mine life for Ontario mines extends well past 400 days, the 100% probability

discounted at a rate of 10% for 1 year should be used to calculate the appropriate increase in expenditure that should be made in Ontario. Based on this, the increase that should be made today by Ontario mines to mitigate the risk of a 3.0 Mn rockburst occurring in the next year is \$12.1 million (2010 CAD).

4.4 Case Study – Mine A

4.4.1 Background

The case study for Mine A is based on a hard rock underground nickel mining operation that averages 2700 tonnes of ore per day. The average cost for rockburst events at Mine A is based on 3 clusters of events from 1998 to 2007 and was found to be \$10.6 million (2010 CAD). The events at Mine A resulted in both direct costs and production delays, however the value lost as a result of production delays averaged 11.5 times the direct cost of the event.

4.4.2 Data Acquisition

The costing data for Mine A was provided by a mining company. The seismicity data used was adopted from Morissette et al. (2011). Mine A is similar in scale, mining method and rock quality to the mine discussed by Morissette et al. and therefore it was deemed acceptable to use the seismic data presented by Morissette et al. for the case study on Mine A. The Gutenberg-Richter Frequency-Magnitude Relationship is adopted from the adjusted data set which took into consideration the use of different macroseismic systems. The total number of events in the set and the total number of events equal to or greater than M_{\min} were inferred from the frequency-magnitude graph, Figure 18.

4.4.3 Calculations

4.4.3.1 Gutenberg-Richter Frequency-Magnitude Relationship

As previously mentioned, the Gutenberg-Richter Frequency-Magnitude graph, Figure 18, was produced by Morissette et al. (2011). Mine A utilized two different macroseismic systems during the time period analyzed and therefore Morissette et al. adjusted the values to minimize the discrepancy between the results from the two systems. The monitoring period for the dataset was 10 years and 6 months or 3832.5 days which recorded 3500 events. The original dataset and adjusted dataset are outlined in Figure 18. The adjusted dataset will be used for all calculations.

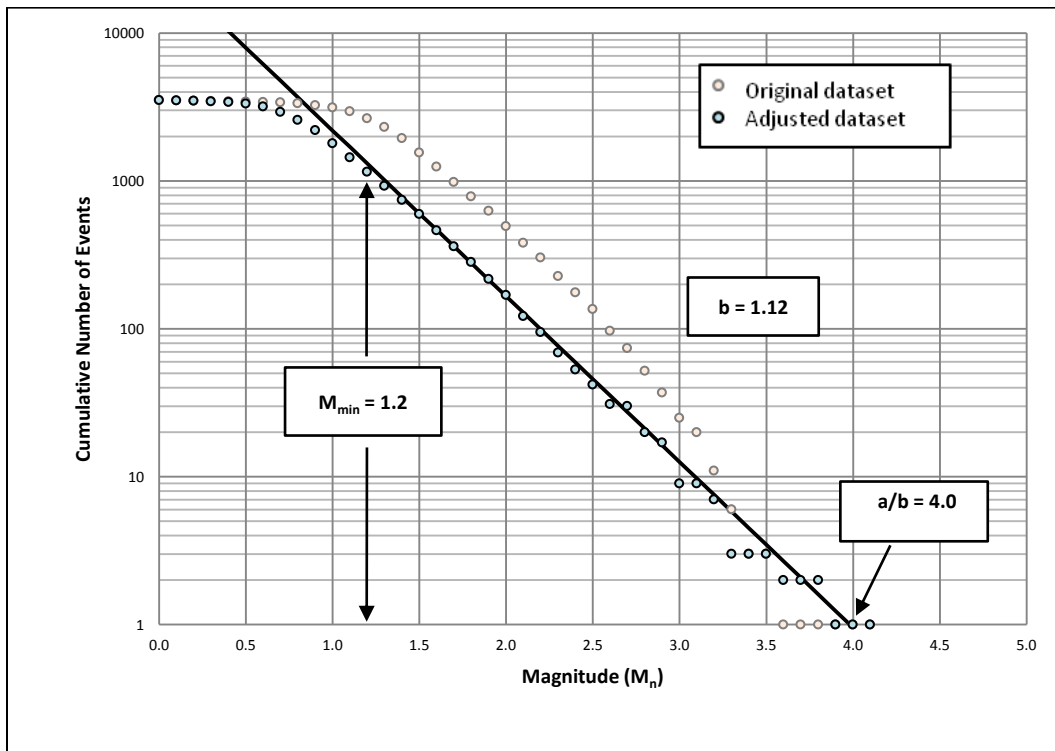


Figure 18: Frequency of occurrence of rockbursts at Mine A (Morissette, et al., 2011)

Based on Figure 18, M_{\min} , the roll-off point, was found to be $1.2 M_n$. The number of seismic events in the dataset greater than or equal to M_{\min} , N , is 1100. It can be observed that the Gutenberg-Richter relationship is best represented between 1.2 and $3.2 M_n$. The best fit line between these 2 points produces a b value of 1.12 . The Gutenberg-Richter Frequency-Magnitude relationship was found to be best represented by equation 16.

$$\text{Log } N = 4.30 - 1.075M_n \quad (16)$$

4.4.3.2 M_{\max} Calculation

In order to calculate M_{\max} using equation 7, n_1 and n_2 were first calculated based on equations 8 and 9 as described in Section 2.4.1, Table 30. It can be observed that n_2 is less than 1, meaning equation 11 must be used to calculate $E_1(n_2)$ while equation 10 can be used to calculate $E_1(n_1)$. While the number of events equal to or larger than M_{\min} , parameter N , is substantially greater than 100, $M_{\max}^{obs} - M_{\min}$ is greater than 2, meaning M_{\max}^{obs} cannot be assumed to be M_{\max} . Therefore, multiple iterations were run, changing the variable $M_{\max}^{estimate}$ until it equalled M_{\max} . 26 iterations were run to calculate M_{\max} which was found to be $5.3 M_n$. Since the calculated M_{\max} is significantly larger than $4.2 M_n$, M_{\max} is assumed to be $4.2 M_n$.

Table 30: List of parameters and results for calculating n_1 and n_2 for Mine A

Parameter	Value
M_{\min}	1.2

M_{max}^{obs}	4.1
N	1100
b	1.12
β	2.65
n_1	1100.62
n_2	0.62

4.4.3.3 Double Truncated Gutenberg-Richter Frequency-Magnitude Relationship

The double truncated Gutenberg-Richter Frequency Magnitude relationship was found by using equation 12, discussed in Section 2.4.2, for the probability of an event with nuttli magnitude M greater than 2, 2.5, 3, 3.5 and 4. The parameters and results are outlined in Table 31.

Table 31: Calculation of the probability that an event with magnitude E will be larger than magnitude M for Mine A

Parameter	P[E>2]	P[E>2.5]	P[E>3]	P[E>3.5]	P[E>4]
M	2	2.5	3	3.5	4
b	1.12	1.12	1.12	1.12	1.12
β	2.58	2.58	2.58	2.58	2.58
M_{min}	1.2	1.2	1.2	1.2	1.2
M_{max}	4.2	4.2	4.2	4.2	4.2
$e^{-\beta M}$	5.75E-03	1.58E-03	4.37E-04	1.20E-04	3.31E-05
$e^{-\beta M_{max}}$	1.98E-05	1.98E-05	1.98E-05	1.98E-05	1.98E-05
$e^{-\beta M_{min}}$	4.53E-02	4.53E-02	4.53E-02	4.53E-02	4.53E-02
P[E>M]	1.27E-01	3.46E-02	9.21E-03	2.22E-03	2.95E-04

4.4.3.4 Magnitude-Probability-Time Plot

Given P[E>M] for M_n 2, 2.5, 3, 3.5 and 4, identified in Table 31 above, a magnitude-probability-time plot was developed for Mine A based on equation 13 of Section 2.4.2, Figure 19. The return

period for the 50%, 80% and 100% probability of a 3.0 M_n event occurring are outlined in Table 32.

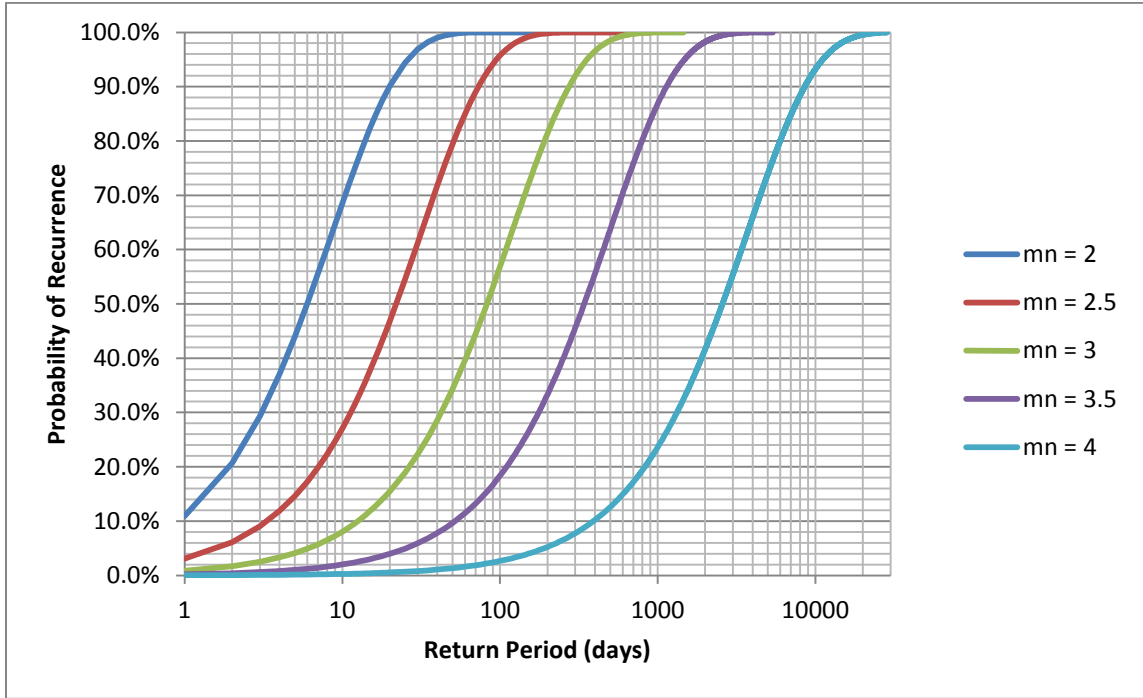


Figure 19: Magnitude-Probability-Time plot for Mine A

Table 32: Return period for 50%, 80% and 100% probability of a 3.0 M_n event at Mine A

Probability of Recurrence for 3.0 M_n event	Return Period (Days)	Return Period (years)
50%	83	0.23
80%	191	0.52
100%	905	2.48

4.4.4 Results

As with the previous case study, the return period for 50% and 80% probability of a 3.0 M_n event occurring is less than a year. Therefore, the cost of both of these events is assumed to occur in year 0, meaning a net present value has no impact on the cost. However, 100% probability has a return period of 2.38 years. Therefore, the cost associated with this event will be discounted 2 years at a rate of 10%.

Assuming the probability that a 3.0 Mn event will cause significant damage is 57%, as calculated above and based on an average cost of a rockburst event at Mine A of \$10.6 million (2010 CAD), the increase in funding to mitigate rockburst damage can be found from equation 14. The results based on the varying probabilities of a 3.0 M_n event occurring are outlined in Table 33.

Table 33: Increase in expenditure today based on the average cost of a rockburst, probability of a rockburst and probability the event will cause damage for Mine A

Probability of 3.0 M_n Event	Increase in Upfront Expenditure
50%	\$3,054,802
80%	\$4,887,683
100%	\$6,109,604
100% (discounted 2 year at 10%)	\$5,049,259

4.4.5 Discussion

Given that there is a 100% probability that a 3.0 Mn event will occur in the next 2 years and that Mine A has a mine life that extends long past 2 years, it is recommended that the 100% probability discounted at 10% for 2 years be used to identify an appropriate increase in expenditure that should be made to mitigate the rockburst risk. Therefore, based on the results

outlined in Table 33, the increase to baseline spending that should be made for Mine A today to reduce the risk of a 3.0 Mn event that has a 100% probability of occurring in the next 2 years, was found to be \$5.1 million (2010 CAD).

4.5 Case Study – Mine B

4.5.1 Background

The case study on Mine B is based on a hard rock underground copper-zinc mining operation that averages 7000 tonnes of ore per day. The average cost of rockbursting at Mine B, as discussed in Section 3.4.4, is based on 2 fault-slip events over a 1 year period that had an average cost of \$11.6 million (2010 CAD). Both events resulted in a direct cost and loss in the value of ore as a result of production delay.

4.5.2 Data Acquisition

All data for Mine B, including costing and seismicity data, was provided by a mining company. The Gutenberg-Richter Frequency-Magnitude relationship was adopted from a graph provided by the mining company, with values adjusted from a local (site specific) magnitude to Nuttli magnitude. M_{min} , the total number of events and the number of events greater than or equal to M_{min} were found by analyzing the adjusted frequency-magnitude graph, Figure 20.

4.5.3 Calculations

4.5.3.1 Gutenberg-Richter Frequency-Magnitude Relationship

As previously mentioned, the Gutenberg-Richter Frequency-Magnitude graph, Figure 20, is adapted from a figure provided by Mine B. The magnitude values were adjusted from Mine B's

local scale by 1.5 to generate a Nuttli magnitude scale. The dataset is based on a monitoring period of 5 years and 3 months or 1915 days and 70 recorded events.

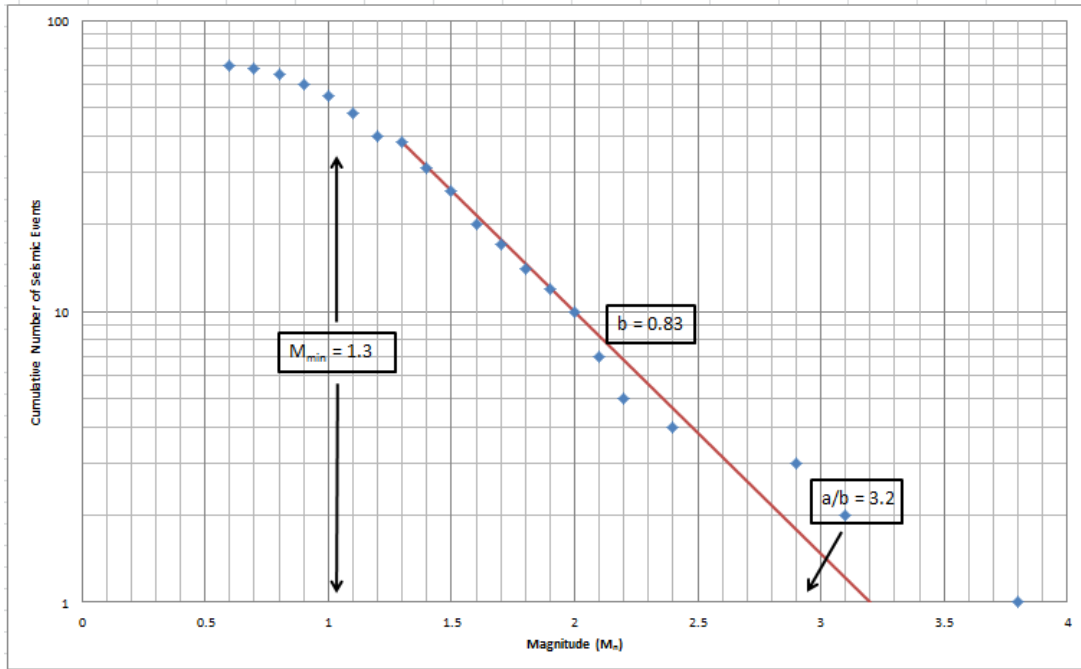


Figure 20: Frequency of occurrence of rockbursts at Mine B

From Figure 20 it can be observed that M_{\min} , the roll-off point, is $1.3 M_n$. Furthermore, the number of seismic events greater than or equal to M_{\min} in the data set, N , is 38. The Gutenberg-Richter Frequency-Magnitude relationship can be seen to be best represented between 1.3 and $2.0 M_n$, with some scatter above $2.0 M_n$. The best fit line between 1.3 and $2.0 M_n$ yields a b value of 0.83 and is best defined by equation 17.

$$\text{Log } N = 2.66 - 0.83M_n \quad (17)$$

4.5.3.2 M_{\max} Calculation

Before attempting to calculate M_{\max} using equations 7, $M_{\max}^{obs} - M_{\min}$ was analyzed. It can be seen that this value is above 2, meaning there is a substantial spread in the dataset. Furthermore, the number of seismic events greater than or equal to M_{\min} , parameter N, is substantially less than 100. The combination of these 2 factors results in low confidence in equations 7 being able to produce a valid M_{\max} . Multiple iterations of equation 7 were run, based on the parameters in Table 34, changing the variable $M_{\max}^{estimate}$ until it equalled M_{\max} to see if a reasonable value could be found, however, the 2 values do not meet. Therefore, $4.2 M_n$ will be used as M_{\max} .

Table 34: List of parameters and results for calculating n_1 and n_2 for Mine B

Parameter	Value
M_{\min}	1.3
M_{\max}^{obs}	3.8
N	38
b	0.83
β	1.91
n_1	38.32
n_2	0.32

4.5.3.3 Double Truncated Gutenberg-Richter Frequency-Magnitude Relationship

Based on equation 12 from Section 2.4.2, the double truncated Gutenberg-Richter Frequency-Magnitude relationship was calculated for the probability of an event with Nuttli magnitude M greater than 2, 2.5, 3, 3.5 and 4. The parameters used and results are outlined in Table 35.

Table 35: Calculation of the probability that an event with magnitude E will be larger than magnitude M for Mine B

Parameter	P[E>2]	P[E>2.5]	P[E>3]	P[E>3.5]	P[E>4]
M	2	2.5	3	3.5	4
b	0.83	0.83	0.83	0.83	0.83
β	1.91	1.91	1.91	1.91	1.91
M_{min}	1.3	1.3	1.3	1.3	1.3
M_{max}	4.2	4.2	4.2	4.2	4.2
e^{-βM}	2.18E-02	8.39E-03	3.22E-03	1.24E-03	4.76E-04
e^{-βM_{max}}	3.25E-04	3.25E-04	3.25E-04	3.25E-04	3.25E-04
e^{-βM_{min}}	8.32E-02	8.32E-02	8.32E-02	8.32E-02	8.32E-02
P[E>M]	2.59E-01	9.73E-02	3.50E-02	1.10E-02	1.83E-03

4.5.3.4 Magnitude-Probability-Time Plot

Given P [E>M] for 2, 2.5, 3, 3.5 and 4 M_n, Table 35, a magnitude-probability-time plot was developed based on equation 13 discussed in Section 2.4.2, Figure 21. It should be noted that the total number of events in the dataset, n, was 70 and the monitoring period, t_r, was 1915 days. The return period for the 50%, 80% and 100% probability of a 3.0 M_n event occurring are outlined in Table 36.

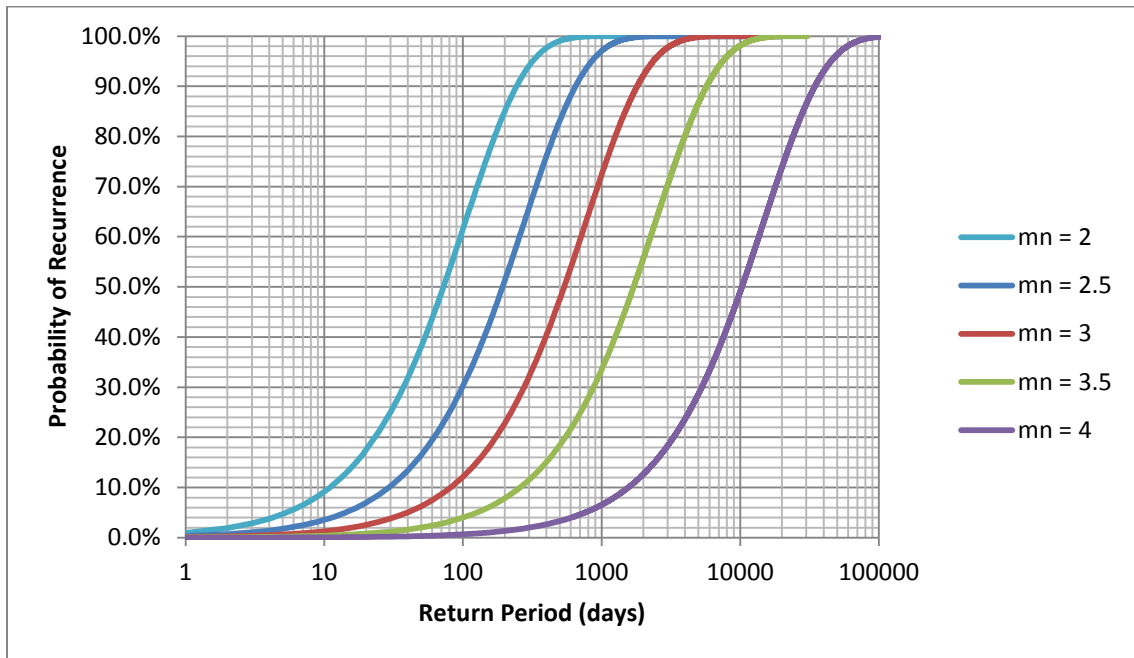


Figure 21: Magnitude-Probability-Time plot for Mine B

Table 36: Return period for 50%, 80% and 100% probability of a 3.0 M_n event at Mine B

Probability of Recurrence for 3.0 M_n event	Return Period (Days)	Return Period (years)
50%	537	1.47
80%	1255	3.44
100%	6190	16.95

4.5.4 Results

From Table 36, it can be seen that the return period for 50% probability of a 3.0 M_n event falls within year 1, while the return period for 80% probability falls within year 3. Given that the return period for 100% probability is 16.95 years, it is assumed that 100% probability of a 3.0 M_n

event occurs in year 17. Therefore, the average cost for the 50%, 80% and 100% probability of a 3.0 M_n event occurring will be discounted at a rate of 10% for 1, 3 and 17 years respectively. Based on the assumption that a 3.0 Mn event will cause significant damage of more than 100 tonnes, 57% of the time and given that the average cost of a rockburst event at Mine B is \$11.6 million (2010 CAD), the appropriate increase in funding to mitigate rockburst damage can be found by using equation 14. The results are outlined in Table 37.

Table 37: Increase in expenditure today based on the average cost of a rockburst, probability of a rockburst and probability the event will cause damage for Mine B

Probability of 3.0 M_n Event	Increase in Upfront Expenditure	Increase in Upfront Expenditure (discounted @ 10%)
50%	\$3,328,511	\$3,025,919
80%	\$5,325,618	\$4,001,215
100%	\$6,657,022	\$1,317,056

4.5.5 Discussion

From Table 37 it can be seen that the largest cost is associated with the 80% probability of a 3.0 M_n event occurring. Since the return period for 100% probability is 17 years, the discounted value is substantially less than both the 50% and 80% probability. It is therefore recommended that Mine B base their appropriate increase in expenditure to mitigate the potential risk of a 3.0 M_n event on 80% probability of a 3.0 Mn event. This yields an increase in expenditure today of \$4.0 million (2010 CAD).

4.6 Discussion on the Methodological Approach

The method presented allows for the identification of an increase from baseline spending to mitigate the risk of a future rockburst event using a probabilistic approach. It is important to first note that since every operation is unique in regards to mine design, ground support, ground monitoring, local stress regime, extraction rate etc., the mitigation work performed will vary from operation to operation and can even vary from mining zone to mining zone within an operation. Therefore there is no discussion on how the increase in spending should be allocated. The approach is designed to identify an appropriate increase in spending that can be justified based on historical information and probability.

It should also be noted that if a method is identified which will reduce the risk of a rockburst event causing damage, but does not reduce the risk to zero, than a probability of successful mitigation can be built into equation 14 as shown in equation 18 below.

$$NPVC_{max} = \left[\frac{(AC_{event} \times P_{damage} \times P_{event})}{(1 + i)^t} \right] \times P_{mitigate} \quad (18)$$

where $P_{mitigate}$ represents the probability that the mitigating effort will be successful. For the previous calculations that found the increase in expenditure that is justified, it was assumed that $P_{mitigate}$ was equal to 100%.

Chapter 5

Limitations

This chapter will discuss limitations associated with the cost models and case studies. This will include a discussion on the confidence in the results, operating costs associated with changing the mining method of an operation and the impact of tax on the increase in expenditure discussed in Chapter 4.

5.1 Confidence in the Results

As outlined in Table 24 the average cost per rockburst event was found to be \$35.4 million (2010 CAD) with a range of \$1.1 to \$263.5 million (2010 CAD) and a median of \$12.8 million (2010 CAD). As previously mentioned, a confidence in terms of percentage cannot be calculated for the results. It can be assumed that the direct cost category has a higher confidence than the indirect cost categories. Furthermore, the 4 cost models involving insurance claims by the mining companies have a higher level of confidence than the other cost models.

5.1.1 Sample Size

For the cost models, the average total cost was derived from 19 rockbursts. Analysis of the results showed a wide range in average costs with 11 events having a total cost less than \$15 million (2010 CAD), 3 events having a total cost between \$20 and \$30 million (2010 CAD), 2 events between \$40 and \$45 million (2010 CAD) and 3 events more than \$50 million (2010 CAD) (the highest being the DRD Gold event which cost \$263.5 million (2010 CAD)). The low

number of events and wide scatter of data reduces the confidence in the average cost per event. A larger dataset would result in an increase in confidence.

One of the challenges of increasing the sample size arises when trying to investigate events by analyzing public documentation from larger mining companies. Most large mining companies do not discuss specific operations, instead they report results by region. This makes it impossible to identify the impact an event might have had since a shortfall in production for one operation could have been minimized by increasing production from several other operations in the same region.

Another challenge is acquiring data from mining companies. While technical information pertaining to rockburst events is generally easy to come by both in literature and by request to mining companies, financial information is much harder to acquire. The financial data needed to calculate the impact of a rockburst event (operating costs, production targets, long term metal prices, ore grades etc.) is very sensitive material and therefore mining companies are naturally hesitant to provide it.

5.1.2 Accounting Practices

One of the biggest challenges with developing the cost models was the varying accounting practices used by mining companies. Every mining operation will account for the direct cost of a rockburst event differently. From observation of the 19 events analyzed, smaller events will generally be expensed, in other words, the cost of the event will be rolled into the operating cost, leading to an increase in operating cost for the year in which the event occurred. In order to

calculate the direct cost of the event, the difference in operating costs between the year the event occurred and the previous full operating year can be used, however, an assumption must be made that the difference in operating costs is only related to the event. Several factors can affect the operating cost from year to year including fuel costs, inflation, labour costs, exchange rates and maintenance costs, meaning this assumption likely overestimates the direct cost.

When costs are capitalized or when a contractor is brought in to do the rehabilitation work, there is a much higher confidence in the costs, since it is easier to identify. For the 13 cost models developed from information provided by mining companies, 5 were based on engineering estimates, 2 were based on insurance claims and 6 were based on contract work. The varying accounting practices make it hard to directly compare the results with a high degree of confidence.

5.2 Change in Mining Method

While it was attempted to analyze the impact of a change in mining method, it was deemed that not enough information was present in each case study to properly assess it. However, an example can be used to illustrate the potential impact on the annual operating cost by changing the mining method. This does not include any capital expenditures required to change from one method to another.

Assume a hypothetical mine utilizing a shaft to skip 2000 tonnes of ore per day, operating on a 350 day per year schedule, were to change from vertical crater retreat (VCR) to cut and fill as a result of a rockburst event. The increase in operating cost per tonne would be \$12.60 (2010

CAD) (InfoMine USA Inc., 2008). This would translate to an increase in operating cost of \$9.2 million (2010 CAD) per year. If the same operation were to switch from sublevel longhole to cut and fill, the increase in annual operating cost alone would be \$22.4 million (2010 CAD).

Table 38: Comparing operating costs for different mining methods extracting 2000 tonnes of ore per day (InfoMine USA Inc., 2008)

Mining Method	Operating Cost (2010 CAD/tonne ore)
Sublevel Longhole	24.00
Vertical Crater Retreat	40.58
Cut and Fill	52.16

While switching mining methods for an entire operation is typically a last resort, it did occur following the 1993 Macassa event. Some operations will change mining methods in specific regions of the mine where there is a higher stress regime and an increased risk of rockbursting. Typically, mining operations will change stope sequencing or stope size first, to try to mitigate rockburst damage by changing the stress regime. This was employed in several of the operations analyzed. Decreasing the stope size would result in a higher per tonne operating cost while altering the mining sequence could yield a temporary decrease in production while new stopes are brought on line and an increase in capital costs to develop new drifts to access the new stopes.

5.3 Effect of Tax on Results

As discussed in Section 2.1.1, the cost models were calculated on a before tax basis. It was decided that since the events used to develop the cost models occurred in different time periods and in different countries it was more appropriate to compare the event costs before tax. This

assumption has a major impact when analyzing the increase in expenditure that should be made today based on the cost a rockburst might have to a mining company at some future date. This section will analyze the impact of tax on the results for the average cost of rockbursting in Ontario and the probability of a rockburst occurring in Ontario developed in Sections 3.4.4 and 4.3. The increase in expenditure to mitigate a future risk of a rockburst event is assumed to be made in 2010.

5.3.1 Federal and Provincial Tax Regime

As of 2009, Ontario's corporate income tax, capital tax and corporate minimum tax have been administered by the Canada Revenue Agency, resulting in a harmonized corporate income tax. As of 2009, the Ontario corporate tax rate for mining companies is 12% while the federal corporate tax rate is 19% for a combined rate of 31%. The Canadian government has been incrementally decreasing the federal corporate tax rate from 19% in 2009, to 18% in 2010, to 16.5% in 2011 to 15% in 2012. However, for simplification the tax rate is assumed to be 31% for all years. Furthermore, the half year rule of depreciation has been omitted.

5.3.2 Depreciation Allowance

Whether a capital expenditure is made to reduce the risk of a rockburst or a capital expenditure is made to rehabilitate part of the mine following an event, the capital costs incurred can be depreciated to reduce the taxable income. Both capital expenditures fall under the Canadian Development Expense (CDE) which includes costs related to "...the acquisition costs of Canadian resource properties and the cost of mine shafts and main haulage ways or similar underground work incurred after coming into commercial production (PricewaterhouseCoopers, 2009)." These

costs are placed in the Cumulative Canadian Development Expense (CCDE) pool. The CCDE has a deduction rate of 30% per year. This means that a company can deduct 30% per year from the CCDE pool against the taxable income.

5.3.3 Cost Categories

The average cost of an event in Ontario was previously found to be \$23.3 million (2010 CAD). Of the four categories used to derive the total event cost, the direct cost category is the most sensitive to the taxation effect as it is depreciable if the cost of the event is capitalized. However, the decline in value due to postponed production and the cost of fatalities will also be impacted by tax. The before tax cost breakdown is shown in Table 39.

Table 39: Average cost of an event in Ontario by category

Direct Cost of Event	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Fatality Cost	Total Cost
\$7,000,228	\$9,223,346	\$4,027,107	\$3,000,000	\$23,250,681

5.3.4 Impact of Tax on the Direct Cost of an Event

Assuming that the direct cost of a rockburst is \$7.0 million (2010) and the cost is capitalized and depreciated using declining balance at a rate of 30%, with a tax rate of 31%, the result is a tax credit of \$2.2 million (2010 CAD) over a period of 15 years. When this value is discounted at a rate of 10% per year, the net present value drops to \$1.8 million (2010 CAD). The results are

shown in Figure 22. This translates to a decrease in the direct cost of a rockburst event by \$1.8 million (2010 CAD) to \$5.2 million (2010).

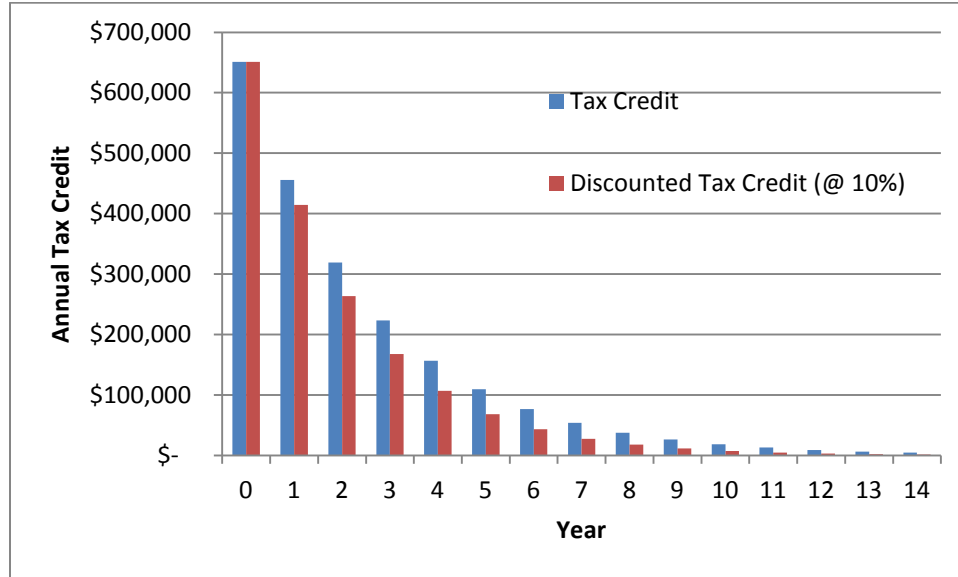


Figure 22: The tax credit for the direct cost category for Ontario mining operations based on a \$7.0 million capital expenditure in year 0

5.3.5 Impact of Tax on the Value of Ore lost as a Result of a Rockburst

The addition of tax will reduce the impact of the value of ore lost as a result of a rockburst. Since the value of ore lost represents a loss in revenue, the addition of tax will reduce the revenue taxable by the company. Assuming that the ore is lost in year 0, the result on the average value of lost ore of \$9.2 million (2010 CAD), based on a tax rate of 31%, is a reduction of \$2.9 million (2010 CAD). This yields a realized after tax value of lost ore of \$6.4 million (2010 CAD).

5.3.6 Impact of Tax on the Decline in Value of Ore from Postponed Production

The decline in value due to postponed production will also be impacted by the addition of tax. For Ontario operations the average value lost due to deferred production was found to be \$4.0 million (2010 CAD). Assuming the ore is recovered 10 years after the event (i.e. if the event occurs in 2010, year 0, ore is recovered in 2020) and based on a 10% discount rate, the profit in year 0 can be found to be \$6.6 million (2010 CAD). Therefore, based on a 31% tax rate for Ontario mining operations, the tax, where the ore mined in 2010 is \$2.0 million (2010 CAD), yielding an after tax profit of \$4.5 million (2010 CAD). Given a 10 year delay in recovery, the discounted after tax profit in 2020 of the delayed ore is \$1.7 million (2010), producing an after tax decline in value due to postponed production of \$2.8 million (2010 CAD). This represents a decrease of 45% from the pretax to post tax value. Figure 23 shows a comparison between the discounted before tax and after tax annual profits for 10 years.

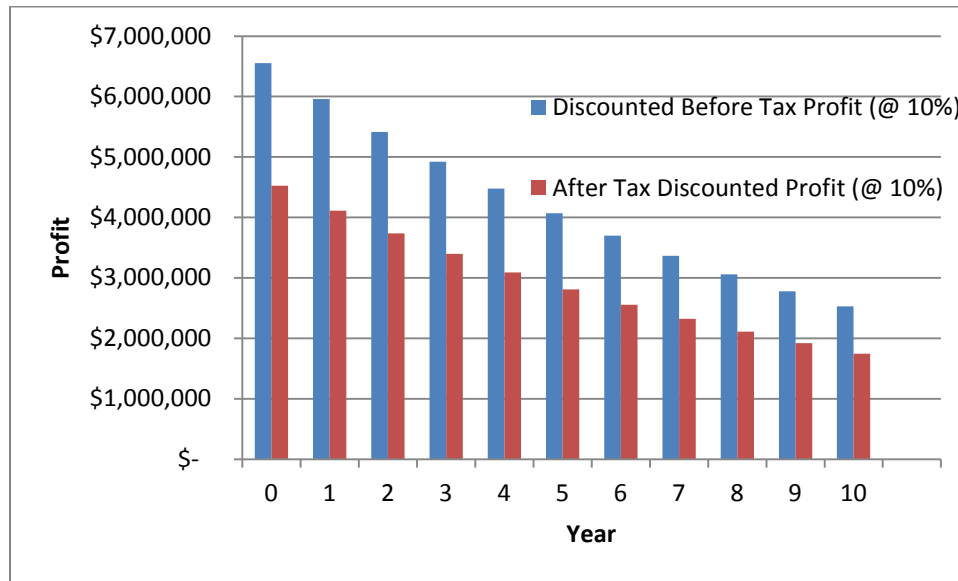


Figure 23: Impact of tax on the decline in value of ore due to postponed production

5.3.7 Impact of Tax on the Cost of Fatalities

The average cost of fatalities per rockburst event in Ontario was found to be \$3 million (2010 CAD). It is assumed that the cost of a fatality is expensed in the year the event occurs. Therefore, a \$3 million (2010 CAD) cost, will reduce the taxable income pool and yield a tax credit of \$0.9 million at a tax rate of 31%. This results in a realized average cost of \$2.1 million (2010 CAD).

5.3.8 Total Average Cost of an Event in Ontario After Tax

Based on the results found, the average after tax cost of a rockburst event in Ontario is \$16.4 million (2010 CAD). This is a reduction of \$4.4 million (2010 CAD) or approximately 21% from the pretax average cost. A breakdown of the average costs can be found in Table 40.

Table 40: Comparison between pretax and after tax average cost of an event in Ontario

Category	Direct Cost of Event	Value of Ore Lost Due to the Event	Decline in Value of Ore Due to Postponed Production	Fatality Cost	Total Cost
Pre Tax	\$7,000,228	\$9,223,346	\$4,027,107	\$3,000,000	\$23,250,681
After Tax	\$5,211,955	\$6,364,109	\$2,778,704	\$2,070,000	\$16,424,768
Difference	\$1,788,274	\$2,859,237	\$1,248,403	\$0,930,000	\$6,852,913

5.3.9 Impact of Tax on Increase in Expenditure

Based on the adjusted after tax average cost of an event, \$16.4 million (2010 CAD), a 57% probability a 3.0 M_n event will cause damage and 100% chance a 3.0 Mn event will occur in the next year, the adjusted before tax discounted at 10% increase in upfront expenditure can be found to be \$8.6 million (2010 CAD). This results in a reduction of \$3.6 million (2010 CAD) or a decline of approximately 29%.

Table 41: Adjusted upfront increase in expenditure for Ontario operations

Probability of a 3.0 M_n Event	Before Tax Increase in Expenditure	After Tax Increase in Expenditure	Difference
50%	\$6,678,758	\$4,718,015	\$1,960,744
80%	\$10,686,013	\$7,548,208	\$3,137,190
100%¹⁸	\$13,357,516	\$8,578,208	\$3,564,988

¹⁸ Values discounted by 1 year at 10%

Applying a 30% declining balance depreciation rate and a 31% tax rate to the \$8.6 million (2010 CAD) increase in expenditure to mitigate the risk of rockbursting, generates a tax credit of \$2.6 million (2010 CAD) over a 10 year period. When the tax credit is discounted at a rate of 10%, the tax credit reduces to a net present value of \$2.2 million (2010 CAD). The annual tax credit for both cases is shown in Figure 24.

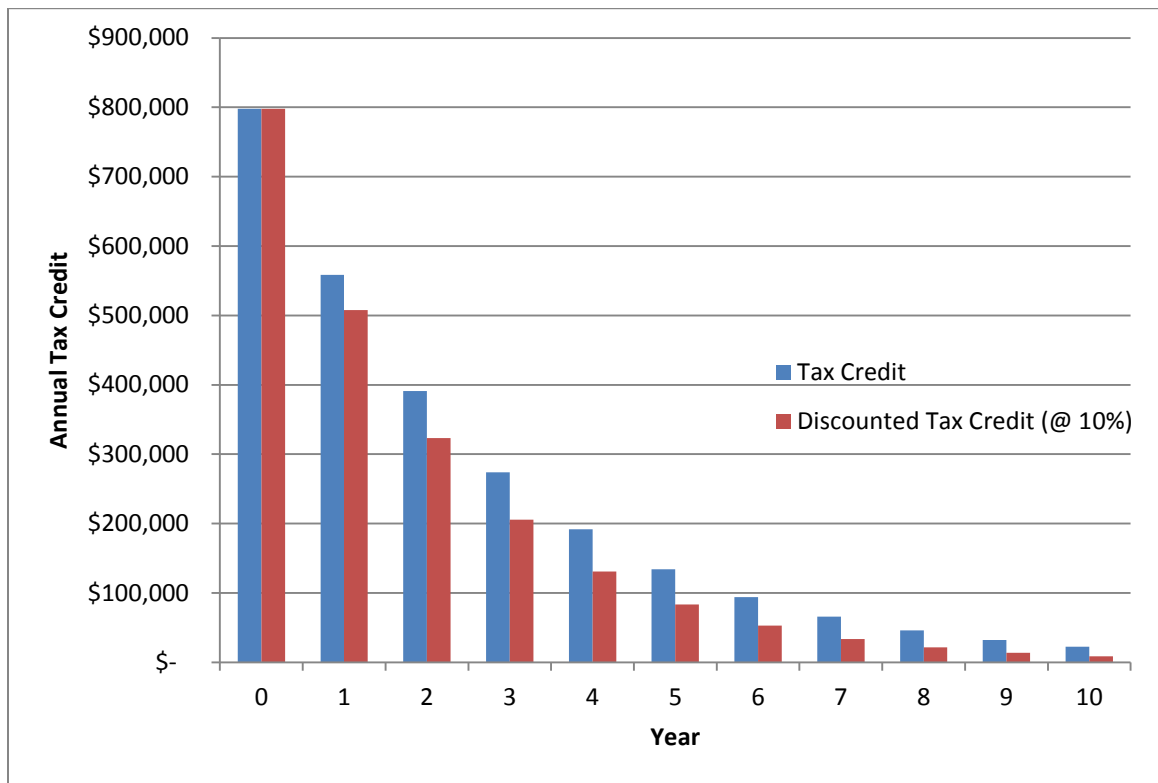


Figure 24: The tax credit for an upfront increase in expenditure of \$8.5 million (2010 CAD) to mitigate the risk of rockbursting made in year 0

5.3.10 Overall Impact of Tax on Results

Based on the previous sections it was found that a tax regime similar to the Ontario tax system results in a 29% reduction in the average future cost of the event. An increase in expenditure of \$8.6 million (2010 CAD) yields a tax credit, discounted at 10% per year for 10 years, of \$2.2 million (2010 CAD). Therefore, the realized cost of the increase in expenditure is reduced to \$6.4 million (2010 CAD), a 25% decline. It is important to note that the initial expenditure before tax remains \$8.6 million (2010 CAD) however the after tax, realized cost is reduced.

Chapter 6

Conclusion and Recommendations

In the previous chapters a method was developed to identify the cost of a major rockburst event and the increase in expenditure that should be made by a company to mitigate a rockburst event based on the probability an event will occur, the probability an event will cause damage and the expected cost if it causes damage. Cost models for six rockburst events derived from public data were discussed in detail to show the approach used to calculate the cost of a rockburst event. Three case studies were then developed to illustrate a method for calculating the increase in expenditure that should be made today, based on probability of occurrence, likelihood of damage and average cost. This chapter will identify the key findings of this report while discussing some of the challenges and potential areas of future work.

6.1 Summary

Based on the work done in Chapter 3, it was found that the before tax average cost of a rockburst event is \$35.4 million (2010 CAD). This value was calculated based on 19 cost models from rockburst events that occurred at 13 mines in 4 different countries between 1984 and 2009. The events ranged in total cost from \$1.1 million to \$263.5 million (2010 CAD) with 16 events having a cost less than \$50 million (2010 CAD). It is important to note that the 19 events were selected due to the high cost associated with the particular events. This study was performed to analyze the potential high end cost associated with rockbursting.

Based on the methodological approach developed in Chapter 3, the increase in expenditure above baseline spending that a mining company should make today to mitigate the risk of a rockburst at some future time period was identified in Chapter 4. The probability a rockburst would cause severe damage was found by analyzing Unusual Occurrence Reports provided by the Ontario Ministry of Labour for events that occurred in Ontario between 1986 and 1999. Next, the probability a seismic event would occur was found using the double truncated Gutenberg-Richter frequency-magnitude relationship developed by Cosentio et al. (1977) combined with a seismic hazard function derived by Benjamin (1968). Three case studies were developed, one based on data provided by the Ontario Ministry of Labour from Unusual Occurrence Reports and two from data provided by mining companies, identified as Mine A and Mine B.

For the Ontario case study, the average cost of a rockburst was derived from 16 cost models developed in Chapter 3 and found to be \$23.3 million (2010 CAD). Using data from the Unusual Occurrence Reports mentioned above, a magnitude-probability-time plot was generated based on Benjamin's (1968) seismic hazard function. This produced a 100% probability that a 3.0 M_n event will occur in Ontario in the next 400 days. Adjusting for the probability that if an event occurs, it will cause damage, multiplied by the cost of the event and discounted at 10% by 1 year generated an increase in expenditure of \$12.1 million (2010 CAD).

For Mine A, the average cost of a rockburst was found to be \$10.6 million based on 3 rockburst events that occurred over a 10 year period. Based on work done by Morissette et al. (2011) a magnitude-probability-time plot was developed using Benjamin's (1986) seismic hazard function. It was found that the return period for 100% probability of a 3.0 M_n rockburst occurring was 905

days or 2.5 years. Taking into account the probability an event will cause damage and the average cost of a rockburst, discounted at 10% by 2 years, the increase in expenditure for Mine A was found to be \$5.0 million (2010 CAD).

Mine B was found to have an average rockburst cost of \$11.6 million based on 2 events that occurred over a 1 year period. A magnitude-probability-time plot was developed from Benjamin's (1986) seismic hazard function based on seismic data provided by the mine site. This yielded an 80% probability that a 3.0 M_n event would occur in the next 1255 days or 3.5 years. Multiplying the probability of occurrence by the probability of damage and the average cost, discounted at 10% by 3 years generated an increase in expenditure of \$4.0 million (2010 CAD).

The increase in expenditure represents the increase from baseline spending that can be justified to mitigate the risk of a rockburst event based on the average cost of historical rockbursting events. Since every operation has unique stresses, support systems, rock quality, mining method, sequencing etc., no recommendations for the allocation of this increase in funding is recommended. This approach simply provides a means for mine management to identify the largest justified increase in geomechanics spending based on historical rockburst events and probability analysis.

6.2 Conclusions

As discussed in Chapter 5 there are limitations with the results. First, the relatively small dataset of 19 events leads to a wide range in results. Second, different accounting practices are employed by mining companies which results in event costs being treated differently. Third, all calculations

were done on a before tax basis. Lastly, several assumptions were made which ultimately resulted in a lower average cost per event.

The small dataset of 19 events yields a lower confidence in the average cost per event. Ideally, the dataset would be substantially larger. This would allow for analysis into costs associated with different mining methods, extraction rates, support systems etc. With only 19 events, the application of the dataset is limited.

Different accounting practices were employed by each operation. Some mining companies expensed the direct cost associated with an event while others capitalized it. Furthermore, some operations were covered by insurance policies which minimized some of the costs associated with the event. The latter two yield a higher confidence since it is easier to identify the direct cost. The different accounting practices make it challenging to directly compare events since the methods used to develop the costs were different. While every effort was made to standardize the results, different accounting practices is a source of error.

All of the cost models and case studies were calculated on a before tax basis, since events that were analyzed occurred in 4 different countries with varying tax regimes. However, a hypothetical example based on the Ontario tax system outlined that, while tax lowers the future cost of an event as expected, it also decreases the realized upfront cost due to tax credits resulting from the expenditure. This highlights the importance of considering tax when requesting funding for increased spending on rockburst mitigation.

The cost models excluded the public relations cost, loss in share value and impacts a rockburst has on the mill. Brummer and Andrieux (2008) noted that an adverse event, such as a rockburst resulting in an injury or fatality, could lead to increased public scrutiny, increasing the cost of an event. Brummer and Andrieux go on to note that investigation of recent incidents at mines resulted in a 5% drop in the company's share price. Lastly, a rockburst can greatly impact milling operations by impacting the blend of ore sent to a mill. This can result in the mill charging penalties to the mine, a decrease in recovery rates at the mill and increase in operating costs at the mill.

Overall, the approach outlined in this thesis, provides geomechanics engineers and engineering geologists with a method for identifying the value of their work by identifying the future cost of an event. This method takes into consideration both the probability an event will occur and the probability that if an event occurs it will cause damage. This, coupled with an average cost per event, is then discounted to the present to provide a probabilistic estimate of the increased expenditure to invest to mitigate the risk of an event based on the likelihood of a future rockburst event.

6.3 Application

While this work provides an approach for identifying the cost of a rockburst event and a method for identifying the increase in spending that can be justified to mitigate the risk of a future rockburst event, the greatest benefit is that it quantifies the potential financial risk associated with rockburst events.

The average cost of rockbursting events based on the 19 cost models developed was found to be \$35.4 million (2010 CAD). While this, in itself, is a large number, the potential high end cost, represented by the DRD Gold event, was found to be \$263.5 million (2010 CAD). This was a fault-slip event caused by historical mining in the region and resulted in the death of 2 miners and the premature closure of the operation.

Safety has improved over the years, particularly in Ontario where no fatality as a result of a rockburst has occurred since 1984, however, as mining has progressed deeper, rockbursting has occurred which has resulted in a significant direct cost, the sterilization of ore and production delays. Furthermore, 3 of the rockbursting events analyzed resulted in the premature closure of a mine and the loss of millions of dollars worth of reserves (2 of which occurred in Ontario). It is clear, based on the potential high financial impact of rockbursting that mines prone to bursting (i.e. deep mines in Canada and South Africa and other burst prone regions around the world) must continue to place a strong emphasis on geomechanics work once a mine enters into production mode.

6.4 Future Work and Recommendations

As identified above, there are areas where this study could be improved upon by further work. There are also changes that could be made in the industry as a whole that were brought to light while completing this report.

6.4.1 Future Work

6.4.1.1 Add to the dataset

Now that a methodological approach has been identified for calculating the cost of rockbursting events, the dataset needs to be further developed to improve the accuracy in the results. Furthermore, by increasing the dataset, work could be performed to analyze costing trends based on mining method, extraction ratio, support method, geological location, depth, rock mass properties etc. This would give a better indication of situations that justify an increase in spending based on the risk of rockbursting.

6.4.1.2 Investigate Impact of a Rockburst Event on a Vertically Integrated Company

As discussed above, other cost categories were identified but deemed outside the scope of this thesis. Identifying the impact of a rockburst event on a vertically integrated mining company (i.e. a mining company involved in production through to refined product) would give greater insight into the impact of a rockburst on a mining company, as opposed to the impact on a specific mine.

6.4.1.3 Investigate Impact of Different Tax Regimes to Rockbursting Costs

In this study, it was decided that cost estimates should be conducted on a before tax basis to allow for the comparison of rockbursting costs from different time periods and geographic locations. A hypothetical example based on the Ontario tax system showed that the after tax rockburst cost will be lower than the before tax cost, but that an increase in expenditure made to mitigate the risk of rockbursting would be lower as a result of tax credits. Investigating the impact of different tax regimes on rockbursting costs could potentially lead to policy changes by government officials to promote work safety through tax credits.

6.4.2 Recommendations

6.4.2.1 Standardize accounting practices

While working on this thesis, it became very apparent that every operation accounts for rockbursting costs differently. Many operations simply rolled the cost of an event into the operations budget, making it impossible to calculate the direct cost. A standardized accounting practice for rockburst events would allow for the direct cost to be easily identified and compared between events.

6.4.2.2 Include Financial Information to Develop Cost Models on Unusual Occurrence Reports

The Unusual Occurrence Reports provided by the Ontario Ministry of Labour were an excellent source of rockburst information. These reports included a great deal of valuable information that was used to develop some of the key concepts in this report. However, the Unusual Occurrence Reports in their current state are only used for technical purposes. By requiring mining companies to include information such as tonnes of ore sterilized and rehabilitation cost estimates, these reports could be valuable in further developing a rockburst cost database.

6.4.2.3 Upload unusual occurrence reports to a computer

During work on this thesis, it was discovered that, while mining companies are still required to file Unusual Occurrence Reports with the Ontario Ministry of Labour, as outlined in Ontario's Regulations for Mines and Mining Plants, these reports are no longer uploaded to a computer and stored on a central server. In fact, since January of 2000, Unusual Occurrence Reports have not been uploaded. There is a wealth of knowledge in these reports and this information should be stored electronically in an easy to access database and provided for research purposes.

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